CALCATREU GOLD PROJECT

Initial Feasibility Study
Volume 1: Executive Summary

5 April 2007

Prepared by

www.snowdengroup.com
Perth, Brisbane, Vancouver, Johannesburg, London

Summary

The Calcatreu Project is owned by Aquiline Resources Incorporated (AQI). At this time, legislative restrictions placed on the use of cyanide by the Rio Negro government mean no economic or practical processing route for treating the Calcatreu mineralization is available.

The scope of the Initial Feasibility Study (IFS) is to determine the potential for establishing a project at Calcatreu assuming no impact from the restrictions on the use of cyanide.

The project scope entails mining ore containing gold and silver from open-pit mines at a nominal rate of 750,000 tonnes per year and treating it in a conventional crushing, grinding, and carbon-in-leach (CIL) process plant to produce gold/silver doré. The IFS has been compiled with reference to USD currency throughout.

The mineral resource for the Vein 49 and Nelson zones at Calcatreu, at a cut-off of 0.55 g/t Au, is estimated to comprise:

- Indicated Mineral Resource – 6.155 Mt at 3.04 g/t Au and 28.1 g/t Ag
- Inferred Mineral Resource – 1.876 Mt at 2.10 g/t Au and 19.4 g/t Ag

The results of AQI's drilling campaigns conducted since August 2004 are yet to be included in estimating the mineral resource.

As no economic or practical processing route for treating the Calcatreu mineralization is available at present, no estimate of a mineral reserve can be made. However, an estimate has been made of the potential mining inventory which may become available in the event that the restrictions are lifted.

The potential mining inventory derived from the Indicated Mineral Resource amenable to treatment by cyanide tank leaching at a diluted cut-off of 1.12 g/t Au equivalent is estimated to be 3.50 Mt at 3.86 g/t Au and 33.22 g/t Ag for 435 koz Au and 3,739 koz Ag.

Gold recovery is predicted by testwork to be 90% and silver recovery is 74%.

Operating costs are $31.50/t of ore and capital costs are $79.15 M.

A mine life of 4.7 years is projected.

Financial analysis shows that the Calcatreu Project has the potential to offer a modest level of profitability using the assumptions and findings of the IFS. At a gold price of $500/oz and a silver price of $8.00/oz, the project delivers free cashflow of $16.26 M and an NPV of $6.07 M with an IRR of 14%. The financial analysis was also run at 6 month (to 31 December 2006) average gold and silver prices of $624/oz and $12.38/oz. Free cashflow, NPV and IRR of $73.54 M, $54.68 M and 74%, respectively, result.

Risk analysis using a multi-variate Monte Carlo simulation shows the project has a 37% probability of delivering a positive NPV result and a 24% probability of delivering the IFS NPV result.

Apart from resolving the acceptability of the use of cyanide, risk mitigation strategies should be incorporated into the project before a decision is made to proceed with development. In particular, the identification of additional mineral resources for conversion into mining inventory, through discovery or upgrading the confidence classification of existing mineral resources, provides an opportunity to significantly improve the project risk profile. It is understood that Aquiline is pursuing this strategy.

1 Project description

The Calcatreu Project is located approximately 1,250 km southwest of Buenos Aires, the national capital of Argentina, 165 km southeast (330 km by road) of the major regional centre of San Carlos De Bariloche and 60 km south of the town of Ingeniero Jacobacci (Jacobacci). The project area is about 1,300 m above sea level and consists of pampa (treeless grassland or plains) in wide flat valleys between low rocky hills, which support very low intensity sheep farming based on poor quality soil. Calcatreu is located in Rio Negro Province close to the northern boundary of Chubut Province.

The project entails mining material containing gold and silver from open-pit mines at a nominal rate of 750,000 tonnes per year and treating it in a conventional crushing, grinding, and carbon-in-leach (CIL) process plant to produce gold/silver doré. The project site is devoid of infrastructure except for an unsealed road to Jacobacci. Provision is made in the project scope for access, power, water, communications and on-site accommodation.

2 Mineral Resources

Precious metal-bearing quartz-calcite veining was discovered in the Calcatreu area in late 1997. Since discovery, vein systems at 11 prospects have been delineated within the project area. The Vein 49/Nelson system, which has a strike length of over 2 km and widths of up to 20 m, has been the most intensely investigated of these and is the significant discovery at Calcatreu. At certain locations the vein system remains open down dip and the structure hosting it remains open along strike. The possibility for the discovery of additional mineralization exists.

Previous drilling at Calcatreu was completed in four campaigns, by La Source and Normandy SA, between 1999 and 2001. Aquiline completed a brief due diligence drill program in 2003 and is continuing an in-fill definition and exploration drilling program at the property.

Micon International Limited (Micon) prepared a preliminary assessment study for Vein 49 and Nelson based on the results from drilling carried out up to August, 2004 ("A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project" October 2004) which was filed by AQI with the Toronto Stock Exchange (TSX) in October 2004.

As part of the preliminary assessment study, Micon estimated the mineral resource for the Vein 49 and Nelson zones, at a cut-off of 0.55 g/t Au, to comprise:

- Indicated Mineral Resource – 6.155 Mt at 3.04 g/t Au and 28.1 g/t Ag
- Inferred Mineral Resource – 1.876 Mt at 2.10 g/t Au and 19.4 g/t Ag

The results of AQI's drilling campaigns conducted since August 2004 are yet to be included in estimating the mineral resource at Calcatreu.

3 Mining and Mineral Reserves

At this time, no economic or practical processing route for treating the Calcatreu mineralization is available due to legislative restrictions placed on the use of cyanide by the Rio Negro government. Therefore no estimate of a mineral reserve can be made. However, an estimate has been made of the potential mining inventory which may become available in the event that the restrictions are lifted.

The potential mining inventory estimate is made based on a gold price of US$500 /oz and a silver price of US$8 /oz.

The potential high grade mining inventory amenable to treatment by cyanide tank leaching derived from the Indicated Mineral Resource at a diluted cut-off of 1.12 g/t Au equivalent is estimated to be:

- 3.50 Mt at 3.86 g/t Au and 33.22 g/t Ag for 435 koz Au and 3,739 koz Ag.

The potential mining inventory estimate is considered to be at a medium level of confidence due to the confidence level of the resource from which it was derived.

A potential low grade mining inventory which may be amenable to treatment by a passive leaching process was derived from the Indicated Mineral Resource at a diluted cut-off of 0.48 g/t Au equivalent and is estimated to be:

- 1.37 Mt at 0.64 g/t Au and 9.19 g/t Ag for 33 koz Au and 405 koz Ag.

The potential low grade mining inventory is of a low level of confidence due to the confidence level of the process and cost and cost assumptions used its determination.

A mine life of 4.7 years is achieved through mining and processing the potential high grade mining inventory at the nominated throughput rate of 750,000 per year.

Operating costs associated with the mining part of the operation are $1.45/t of material mined over the life of the project or $11.95/t of the high grade production material mined.

Capital costs associated with the mining operation are $12.7 M at start-up and $13.0 M over the mine life.

Mining at Calcatreu is expected to be simple with no especially challenging or high cost mining issues likely to be encountered.

Diligent grade control practices will be used to derive the greatest value from the project.

Control over the mining and dumping of pyritic waste will be exercised to manage the risk of acid mine drainage being generated.

Recommendations

The mineral resource estimate for Calcatreu should be updated using the results of the drilling campaigns conducted since 2004 to enable a more accurate and higher confidence estimate of the potential mining inventory to be made.

Further work should be undertaken to confirm or modify the preliminary conclusions regarding processing performance, hydrogeology and environmental matters which were relied upon during the estimation of the potential mining inventory.

Processing investigations should be initiated to identify a technically and economically feasible method for treating the low grade potential mining inventory estimated in the IFS.

4 Processing and site infrastructure

The project entails mining and processing mineralised material at a nominal rate of 750,000 tonnes per year and treating it in a conventional crushing, grinding, and carbon-in-leach (CIL) process plant to produce gold/silver doré.

A comprehensive metallurgical testwork programme demonstrated that the ore is “free milling” and that high gold and silver recoveries can be achieved using a conventional grinding and cyanidation process.

Grade versus leach residue relationships were developed and equate to a gold recovery of 90% at the average head grade of 3.86 g/t and a silver recovery of 74% at the average head grade of 33.2 g/t.

Cyanide detoxification tests were conducted on samples of leach residue tails and show that the cyanide level in the leach residue tails can be readily detoxified to meet the International Cyanide Management Code (ICMC) standard for the discharge of tailings into a Tailing Storage Facility.

Electrical power will be provided by an owner-operated, power station using high-speed diesel generator sets.

Site buildings will comprise process plant, mine workshop, reagent storage, core storage and laboratory. Process water will be supplied from a site borefield.

On-site camp accommodation will be provided for the site personnel, who would generally reside in Jacobacci.

The average annual cost for processing Calcatreu ore at a rate of 750,000 tonnes per year over the life of the mine is $19.55/t.

The project initial capital cost for the process plant and site infrastructure is estimated at $66.2 M.

The IFS also included a conceptual assessment of alternative process routes including a non-cyanide gold and silver recovery process. Whilst this process route eliminates the use of cyanide, the recoveries of gold and silver were significantly lower than those achieved using a cyanide leach.

Recommendations

During the next phase of developing the Calcatreu project, AQI should obtain environmental and mine permitting licenses, and conduct hydrological studies to determine the source and quality of the raw water for the proposed processing facility.

It is imperative to establish whether the decision by the Rio Negro government can be modified before embarking on the next phase of project development to make a definitive process route selection for optimization.

5 Financial analysis

The financial analysis shows that the Calcatreu Project offers a modest level of profitability using the assumptions and findings of the IFS. At a gold price of $500/oz and a silver price of $8.00/oz, the project delivers free cashflow of $16.26 M and NPV of $6.07 M with an IRR of 14%. Operating costs are $31.50/t of material processed and the capital cost is $79.15 M.

The project as it stands does not have an attractive risk profile. When analysed by multi-variate Monte Carlo simulation for the impact of identifiable significant risks, it is apparent that there is only a 37% probability of the project delivering better than a break-even financial result (i.e. zero NPV) and a 24% probability that the project will deliver the IFS NPV or greater. The NPV of the project is particularly sensitive to variations in the revenue factors (price, mined grade and recovery) over which AQI has limited control, and the size of the mining inventory.

The financial analysis was also undertaken using the 6 month average (to 31 December 2006) gold and silver prices of $624/oz and $12.38/oz respectively. Under these assumptions, the project delivers free cashflow of $73.54 M and NPV of $54.68 M with an IRR of 74%.

Recommendations

Apart from resolving the acceptability of the use of cyanide, risk mitigation strategies should be considered and incorporated into the project before a decision is made to proceed with development. In particular, the identification of additional mineral resources for conversion into mining inventory, through discovery or upgrading the confidence classification of existing mineral resources, provides an opportunity to significantly improve the risk profile. It is understood that AQI is pursuing this strategy.

CALCATREU GOLD PROJECT

Initial Feasibility Study
Volume 2: Mineral Resources

5 April 2007

Prepared by

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Executive Summary

Precious metal-bearing quartz-calcite veining was discovered in the Calcatreu area in late 1997. Since discovery, vein systems at 11 prospects have been delineated within the project area. The Vein 49/Nelson system, which has a strike length of over 2 km and widths of up to 20 m, has been the most intensely investigated of these and is the significant discovery at Calcatreu. At certain locations the vein system remains open down dip and the structure hosting it remains open along strike. The possibility for the discovery of additional mineralization exists.

Previous drilling at Calcatreu was completed in four campaigns, by La Source and Normandy SA, between 1999 and 2001. Aquiline Resources Incorporated (AQI) completed a brief due diligence drill program in 2003 and is continuing an in-fill definition and exploration drilling program at the property.

Micon International Limited (Micon) prepared a preliminary assessment study for Vein 49 and Nelson based on the results from drilling carried out up to August, 2004 ("A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project" October 2004) which was filed by AQI with the Toronto Stock Exchange (TSX) in October 2004.

As part of the preliminary assessment study, Micon estimated the mineral resource for the Vein 49 and Nelson zones, at a cut-off of 0.55 g/t Au, to comprise:

- Indicated Resource - 6.155 Mt at 3.04 g/t Au and 28.1 g/t Ag
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The results of AQI's drilling campaigns conducted since August 2004 are yet to be included in estimating the mineral resource at Calcatreu.

Tables

Figures

1 Introduction

Micon International Limited (Micon) prepared a preliminary assessment study for Vein 49 and Nelson based on the results from drilling carried out up to August, 2004 ("A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project" October 2004) which was filed by AQI with the Toronto Stock Exchange (TSX) in October 2004.

This report comprises extracts from Micon 2004 to provide context for the IFS prepared for AQI during 2006 and 2007.

2 Mineral Resources

The Calcatreu gold project (Calcatreu) is a largely contiguous collection of prospecting licenses and mining claims comprising approximately 2,930 km2 in the Somuncura Volcanic Massif of north central Patagonia, Argentina. It is located approximately 165 km (330 km by road) southeast of San Carlos de Bariloche (Bariloche) and 1,500 km southwest of Buenos Aires (see Figure 2.1). Calcatreu straddles the southern boundary of Rio Negro Province and the northern boundary of Chubut Province. The closest town is Ingeniero Jacobacci (Jacobacci), some 90 km away by road (see Figure 2.2). The project area covers rolling pampa and low hills, which support very low intensity sheep farming based on poor quality soil.

Calcatreu passed to Normandy Mining of Australia (Normandy) when it purchased La Source Developpement Argentine (La Source) from the French Government in 1998. Normandy, through its Argentine subsidiary Minera Normandy Argentina SA (Normandy SA), completed most of the exploration on the property.

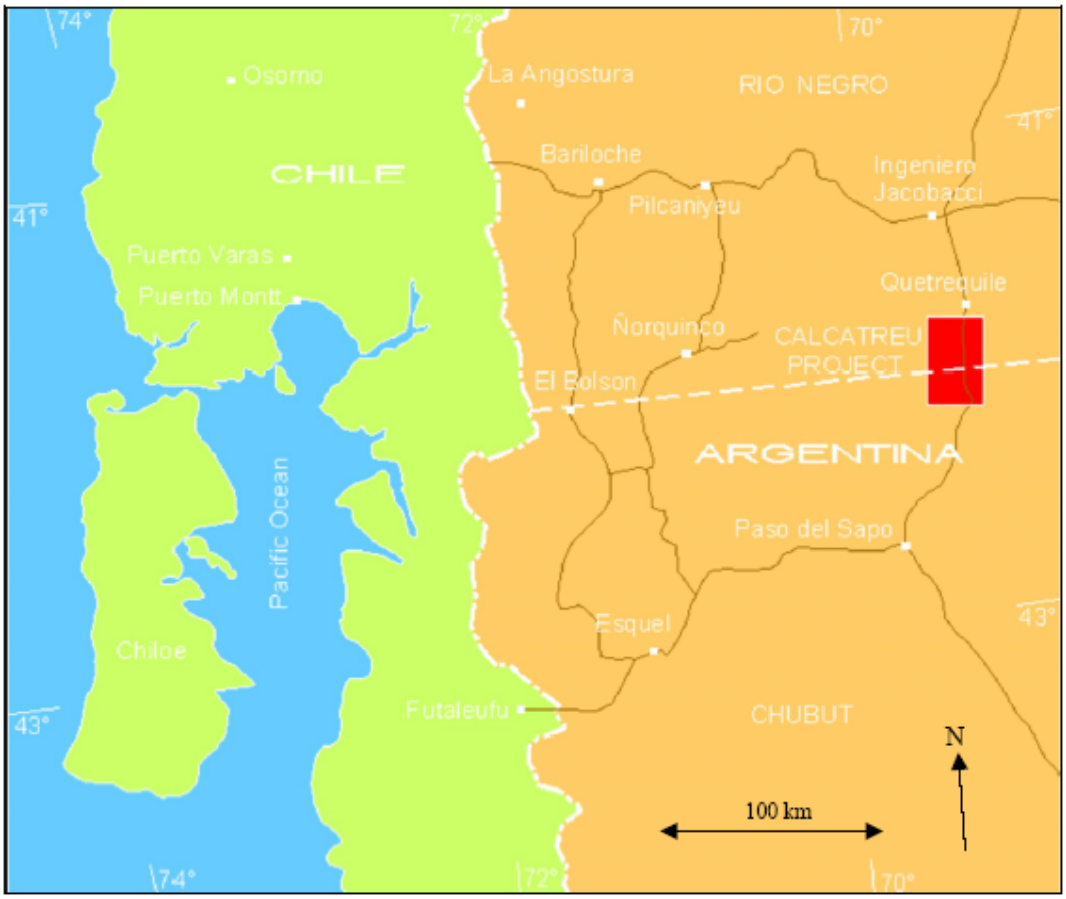
Newmont Mining Corporation (Newmont) purchased Normandy in 2002 and the project passed into its hands in the deal. It was reported that Newmont determined that Calcatreu did not meet corporate size objectives and they elected to dispose of Normandy SA through an auction process in September, 2002. AQI of Toronto, Canada was the winning bidder. The final agreement was signed by Aquiline in June, 2003 and allows it to purchase 100% of the property and acquire Normandy SA by paying Newmont US$2.05 million in a series of staged payments over three years. The project is subject to a 2.5% net smelter return royalty payable to Newmont on any production. Newmont also retains a back-in right over much of the property however, the central mining claim areas of Nabel, where Vein 49 and Nelson are located, are not subject to this right.

Geologically, Calcatreu occurs within bimodal calc-alkaline volcanic rocks of the Jurassic Taquetren Formation within the massif. Low sulphidation, epithermal, precious metal-bearing quartz-calcite veining was discovered in the Calcatreu area by La Source in late 1997 when a geologist sampled quartz float that he observed on the roadside while visiting a prospect further to the west. As a result of the assays received and the vein textures noted, two prospecting licenses were staked in 1997, and regional reconnaissance activities commenced the following year.

To date vein systems at 11 prospects have been delineated within the project area. The Vein 49/Nelson system, which has a strike length of over 2 km and widths of up to 20 m, has been the most intensely investigated of these and is the significant discovery at Calcatreu. It is a low sulphidation, quartz adularia gold deposit and its geological setting has strong similarities with the Cerro Vanguardia and El Desquite projects also located in southern Argentina. At certain locations it remains open down dip and the structure hosting it remains open along strike. The possibility for the discovery of additional mineralization exists.

Figure 2.1 Location map – Argentina

Figure 2.2 Location Map – Calcatreu project area



Previous drilling at Calcatreu was completed in four campaigns, by La Source and Normandy SA, between 1999 and 2001. AQI completed a brief due diligence drill program in 2003 and is continuing an in-fill definition and exploration drilling program at the property.

AQI retained Micon International Limited (Micon) to prepare a Preliminary Assessment Study for Vein 49 and Nelson based on the results from drilling carried out up to August, 2004. As part of the study, Micon updated its previous mineral resource estimate for the Vein 49 and Nelson zones, the methodology of which is described in the Micon 2004 report. A summary of that updated resource is presented in Table 2.1.

Table 2.1 Mineral Resource 0.55 g/t Au cut-off (Micon 2004B)

	INDICATED					INFERRED				
Zone/mineralisation type	Tonnes	Au g/t	Ag g/t	Au oz	Ag oz	Tonnes	Au g/t	Ag g/t	Au oz	Ag oz
Vein 49 Oxide										
Quartz vein	758,000	5.34	32.9	130,14	801,800	3,000	1.72	10.2	170	1,000
Mineralised andesite	410,000	0.77	9.9	10,150	130,500	0	0.00	0.0	0	0
Vein 49 Primary										
Quartz vein	2,998,00	3.79	37.8	365,32	3,643,50	997,000	2.76	25.5	88,470	817,400
Mineralised andesite	854,000	0.75	8.2	20,590	225,100	343,000	0.86	7.7	9,480	84,900
Subtotal Vein 49	**5,020,00**	**3.26**	**29.8**	**526,20**	**4,800,90**	**1,343,00**	**2.27**	**20.9**	**98,120**	**903,300**
Nelson Area Oxide										
Quartz vein	338,000	2.83	20.5	30,750	222,800	53,000	1.54	12.8	2,620	21,800
Mineralised andesite	149,000	0.79	7.7	3,780	36,900	21,000	1.01	8.0	680	5,400
Nelson Area Primary										
Quartz vein	510,000	2.35	28.4	38,530	465,700	405,000	1.80	17.4	23,440	226,600
Mineralised andesite	138,000	0.74	9.7	3,280	43,000	45,000	0.73	6.4	1,060	9,300
Subtotal Nelson	**1,135,00**	**2.09**	**21.1**	**76,340**	**768,400**	**524,000**	**1.65**	**15.6**	**27,800**	**263,100**
Total Oxide	1,655,00	3.29	22.4	174,82	1,192,00	77,000	1.40	11.4	3,470	28,200
Total Primary	4,500,00	2.96	30.3	427,72	4,377,30	1,790,00	2.13	19.8	122,450	1,138,200
Grand Total	**6,155,00**	**3.04**	**28.1**	**602,54**	**5,569,30**	**1,876,00**	**2.10**	**19.4**	**125,920**	**1,166,400**

CALCATREU GOLD PROJECT

Initial Feasibility Study
Volume 3: Mining

5 April 2007

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Summary (NI 43-101 Item 3)

Setting

The Calcatreu Project is located approximately 1,250 km southwest of Buenos Aires, the national capital of Argentina, 165 km southeast (330 km by road) of the major regional centre of San Carlos De Bariloche and 60 km south of the town of Ingeniero Jacobacci. The project area is about 1,300 m above sea level and consists of pampa (treeless grassland or plains) in wide flat valleys between low rocky hills.

The project was originally owned by La Source Developpement Argentine and was acquired by Normandy Mining of Australia in 1998. Newmont Mining Corporation purchased Normandy in 2002 gaining project ownership. An agreement to purchase Normandy SA, the owning entity of the project, over a three year period, was reached with Aquiline Resources Incorporated (AQI) in 2003.

The initial discoveries at Vein 49 and Nelson were made by regional geological mapping and chip sampling. Trench and channel sampling with detailed geological mapping followed with a number of mapping, sampling and drill programs having been conducted since.

The mineral resource estimate used for this assessment was prepared in 2004. The estimate was prepared using the ordinary kriging method. At a 0.55 g/t Au cut-off, the mineral resource was estimated to contain an Indicated resource of 6.155 Mt at 3.04 g/t Au and 28.1 g/t Ag. The results of AQI's drilling campaigns conducted since August 2004 are yet to be included in estimating the mineral resource at Calcatreu.

Conclusions

At the time of reporting, no economic or practical processing route for treating the Calcatreu mineralization is available due to legislative restrictions on the use of cyanide. Therefore no estimate of a mineral reserve can be made. However, an estimate has been made of the potential mining inventory which may become available in the event that the legislative restrictions are lifted.

The potential mining inventory estimate is made based on costs estimated in US$, a gold price of US$500 /oz and a silver price of US$8 /oz.

The potential high grade mining inventory amenable to treatment by cyanide tank leaching is estimated to be:

- 3.50 Mt at 3.86 g/t Au and 33.22 g/t Ag for 435 koz Au and 3,739 koz Ag.

The potential mining inventory estimate is at a medium level of confidence.

A potential low grade mining inventory which may be amenable to treatment by a passive leaching process is estimated to be:

- 1.37 Mt at 0.64 g/t Au and 9.19 g/t Ag for 33 koz Au and 405 koz Ag.

The potential low grade mining inventory is of a low level of confidence.

A mine life of 4.7 years is achieved through mining and processing the potential high grade mining inventory at the nominated throughput rate of 750 ktpa.

Operating costs associated with the mining part of the operation are $1.45 /t of material mined over the life of the project or $11.95 /t of the high grade production material mined. Other parts of the operation including processing, engineering and administration will incur additional costs not estimated as part of this study.

Capital costs associated with the mining operation are $12.7 M at start-up and $13.0 M over the mine life. Other parts of the operation including processing, engineering and administration will incur additional capital costs not estimated as part of this study.

Mining at Calcatreu will be simple with no especially challenging or high cost mining issues likely to be encountered.

Diligent grade control practices will need to be employed to derive the greatest value from the project.

Control over the mining and dumping of pyritic waste will need to be exercised to manage the risk of acid mine drainage being generated.

Recommendations

The mineral resource estimate for Calcatreu should be updated using the results of the drilling campaigns conducted since 2004 to enable a more accurate and higher confidence estimate of the potential mining inventory to be made.

Further work should be undertaken to confirm or modify the preliminary conclusions regarding processing performance, hydrogeology and environmental matters which were relied upon during the estimation of the potential mining inventory.

Processing investigations should be initiated to identify a technically and economically feasible processing method for the low grade potential mining inventory estimated in this study.

Tables

Figures

Appendices

1 Introduction (Item 4)

AQI has retained Snowden Mining industry Consultants (Snowden) to prepare a mine plan as part of an Initial Feasibility Study AQI is conducting on its Calcatreu Project in Rio Negro province in Argentina, South America. The mine plan has been prepared with considerable reliance on information supplied by AQI in the form of electronic documents and data sources including:

- Micon 2003. "A Mineral Resource Estimate for the Vein 49 and Nelson Zones at the Calcatreu Gold-Silver Project, North-Central Patagonia, Rio Negro Province, Argentina", November 2003*
- Micon 2004B: "A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project"; Micon International Ltd, October 2004*
- Golder 2005: "Pit Slope Design for Nelson and Vein 49 Open Pits Calcatreu Project Argentina"; Golder Associates Ltd, November 2005
- "Block Model Data.zip", "Drill Data.zip", "Topo Pit and Mineralized Domains.zip", "Whittle Data.zip"; AQI, January 2006.

* Denotes documents previously lodged in accordance with NI 43-101.

AQI's requirement is for the mine plan to satisfy the requirements of Canadian National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (NI 43-101) for technical reports, feasibility studies and mineral reserve estimation.

The Calcatreu Project site was visited between 24 April 2006 and 26 April 2006 by Snowden Divisional Manager Peter Myers who inspected the site and AQI's offices in Jacobacci, conducted enquiries with regard to issues relevant to the mining aspects of the project.

2 Reliance on Other Experts (Item 5)

In preparing this report, Snowden has relied on work undertaken and reported by experts in fields other than the subject of this report. Those experts and the references supplied by them are listed in Table 2.1. Snowden has not confirmed the accuracy or correctness of the information provided in the expert references and accepts no responsibility for outcomes resulting from any errors or inaccuracies contained in that information.

Table 2.1 Expert references

Field	Expert	Reference
Project overview	Micon International Limited	"A Review of the Calcatreu Gold-Silver Project, North-Central Patagonia, Rio Negro Province, Argentina", October 2004*
	Micon International Limited	"A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project", October 2004*
Geology and resource estimation	Micon International Limited	"A Mineral Resource Estimate for the Vein 49 and Nelson Zones at the Calcatreu Gold-Silver Project, North-Central Patagonia, Rio Negro Province, Argentina", November 2003*
	Gene Puritch	"Block Model Data.zip", "Drill Data.zip", "Topo Pit and Mineralized Domains.zip", "Whittle Data.zip"; electronic data
Geotechnical and hydrological engineering	Golder Associates Ltd	"Pit Slope Design for Nelson and Vein 49 Open Pits Calcatreu Project Argentina"; November 2005
	Water Management Consultants Chile Ltd	"Preliminary dewatering analysis and recommendations, Calcatreu Project", memorandum to AQI, February 2005
Process engineering	Ausenco Limited	Calcatreu Gold Project Metal Extraction Case Study, May 2006

*Denotes previously lodged in accordance with NI 43-101.

3 Property Description and Location (Item 6)

The property and its location have previously been described (Micon 2004B). A summary description of the property and its location is presented below.

The Calcatreu Project is located approximately 1,250 km southwest of Buenos Aires, the national capital, 165 km southeast (330 km by road) of the major regional centre of San Carlos De Bariloche (Bariloche) and 60 km south of the town of Ingeniero Jacobacci (Jacobacci). The Rio Negro – Chubut provincial border passes through adjacent exploration tenements 12 km to the south of the project.

Jacobbaci is a town of 6,000 inhabitants and is located on the railway line connecting Viedma, the Atlantic coastal capital of Rio Negro, with Bariloche.

Figure 3.1 Calcatreu project location plan

The coordinates of the centre point of the project area are reported by AQI to be 2,460,000E, 5,370,000N as defined in the Gauss Krugger Projection, Zone 2 with the Campo Inchauspe Datum.

There are two principal means of securing mineral rights in Argentina. The Cateo is an exploration permit which is granted in up to 20 units of 500 ha each. There are various time related conditions for size reduction and relinquishment.

A Cateo can be converted to a Mina or mining right after the declaration of the discovery of a mineralized occurrence through a “Manifestacion de Descubrimiento” (manifestation of discovery) or MD. Minas are granted for an unlimited time though are subject to fulfilling expenditure and agreed work program commitments. The deposits which are the focus of the Calcatreu Project Initial Feasibility Study, Vein 49 and Nelson, are reported by AQI to be contained within the Nabel 4 and Rebecca Minas.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography (Item 7)

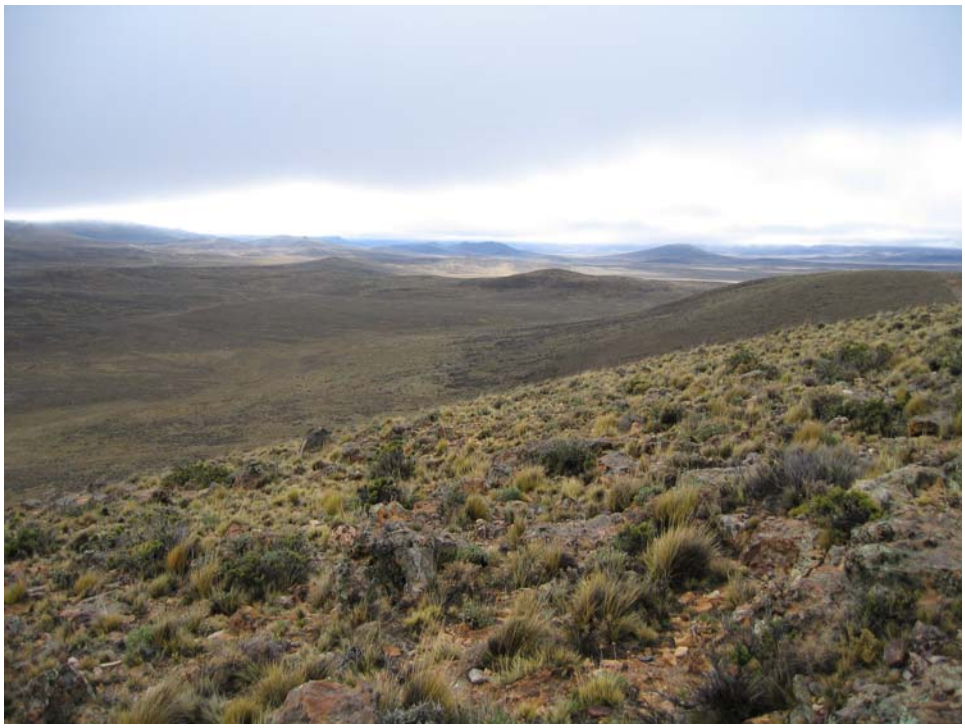
The accessibility, climate, local resources, infrastructure and physiography of the project have been fully described in the Micon 2004B. A summary description of these aspects of the project is presented below.

The Project area has well established access. The site is reached from Jacobacci by the wide unsealed but maintained Ruta 314 which connects Jacobacci with Paso del Sapo in Chubut Provence to the south. Road distance between Jacobacci and Calcatreu is 90 km.

Jacobacci is accessed by the railway line connecting Viedma with Bariloche or by unsealed but maintained roads from Bariloche and Viedma. Bariloche and Viedma are served by commercial jet services from Buenos Aires and Bariloche has access to international jet services from Santiago (Chile) via Mendoza.

The project area is about 1,300 m above sea level and consists of pampa (treeless grassland or plains) in wide flat valleys between low rocky hills. The area includes a moderately well developed network of slow draining creeks, streams and swampy areas, partly the result of the deposition of a high loading of wind borne dust including ash from active volcanoes in Chile to the west. The topography and vegetation typical of the project area are shown in Figure 4.1.

Figure 4.1 Topography and vegetation typical of the project area



The climate is classed as moderately cold continental and is characterised by the features shown in Table 4.1.

Table 4.1 Climatic features

Climatic feature	Magnitude
Average annual temperature	10° Celsius
Average January temperature	20° Celsius
Average July temperature	4° Celsius
Principal wind direction	From NW/W to SE/E
Average wind velocity	20 kph
Maximum wind velocity	>100 kph in spring
Annual rain/snow fall	200 mm

The dominant local land use is sheep grazing with some additional stocking of cattle and horses. The harsh climate, poor soils and increasing international competition has resulted in the regional agricultural industry becoming severely depressed. AQI and recent previous owners are reported to have maintained good relations with the land holders.

There is no local power or communications infrastructure at the project site, with the closest power and telecommunications located at Jacobacci.

5 History (Item 8)

The history of the project has been previously described in Micon 2004B. A summary description of the project history is presented below.

Anomalous gold mineralisation was first discovered about 15 km north of the Calc project in 1997 at what were to become the Nabel and Nabelon prospects. Regional follow-up in 1998 lead to the discovery of a number of prospects, including Vein 49 and Nelson which now form the Calcatreu Project.

The project was originally owned by French government entity, La Source Developpement Argentine (La Source) and was acquired by Normandy Mining of Australia (Normandy) with the purchase of La Source in 1998. Newmont Mining Corporation purchased Normandy in 2002 gaining project ownership. An agreement to purchase Normandy SA, the owning entity of the project, over a three year period, was reached with AQI in 2003.

The initial discoveries at Vein 49 and Nelson were made by regional geological mapping and chip sampling. Trench and channel sampling with detailed geological mapping followed with an initial drilling program undertaken in April 1999. A number of follow-up mapping, sampling and drill programs were completed by Normandy, the details of which are reported in Micon 2004B.

Since acquiring the project in 2003, AQI has collected further information through additional drilling. In 2004, AQI commissioned Micon International Limited (Micon) to undertake an economic assessment study, the results of which were reported in "A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project", October 2004.

Since 2004, AQI has conducted more exploration work and in 2005, commissioned an Initial Feasibility Study on the project, which is due for completion during 2006.

6 Geological Setting, Deposit Types, Mineralisation, Exploration, Drilling, Sampling and Data Verification (Items 9, 10, 11, 12, 13, 14, 15 and 16)

The details of the project geological setting, deposit types, mineralisation, exploration, drilling, sampling methods, approach, preparation, analysis and security, and data verification have previously been described in Micon 2004B. A summary of these aspects is presented below.

6.1 Geology, deposit types and mineralisation

The western margin of southern Argentina is dominated by the Andean Cordillera, a volcanic mountain belt. To the east of the Andean Cordillera lie the Somun Cura Massif in the north and the Deseado Massif to the south. These large volcanic massifs are separated and bordered by younger sediment-filled valleys containing major rivers draining the eastern slopes of the Andes

The Calcatreu project is hosted in the calc-alkaline, bimodal Jurassic Taquetren Formation volcanic rocks within the Somun Cura Massif. The local geology is complex and attempts to describe it have been made by a number of authors (Nullo, Hodgkin, Rivera, Franzeese, Cembrano) whose conclusions show less than complete agreement. Hodgkin and Rivera (2001) conclude that the alteration and mineralisation encountered in the project area is associated with a complex of intrusives and breccias emplaced at the end of the last of four volcano-sedimentary cycles.

The Vein 49 and Nelson deposits at Calcatreu are of the epithermal gold/silver bearing low sulphidation type. The mineralisation is hosted in altered (silicified and clay-altered) brecciated host rock. The host is interpreted to be a porphyritic andesite lava.

Gold occurs as electrum and as free gold. Grades average about 3 g/t Au with Vein 49 being somewhat higher and Nelson somewhat lower. Individual assays can be up to 60 g/t Au over widths of 1 m to 5 m. Mineralisation is largely restricted to quartz and calcite veins and stockworks. The zones of veining can be up to 20 m wide. Surrounding the quartz veining is a zone of argillic (altered) andesite. These rocks generally show low gold content, usually less than 0.5 g/t but locally in excess of 1.0 g/t, often when accompanied by silicification or quartz veinlets. The silver:gold ratio is approximately 10:1. The mineralised system has a very low sulphide content with minor pyrite and lesser galena and sphalerite present.

The mineralisation has been oxidised to an average depth of approximately 75 m, to a maximum depth of 120 m. The surrounding host rocks have been oxidised to an average of approximately 30 m depth.

The mineralized zone shows illite alteration and strong weathering near surface. The main clays associated with the mineralisation are illite and smectite. Kaolinite is very well developed in the hanging wall of the main Vein 49 structure and is interpreted to be the result of supergene weathering.

6.2 Exploration, drilling, sampling and data verification

Various exploration programs have been undertaken as the project has developed. After the initial discovery in May 1998 using regional mapping and chip sampling techniques,

detailed mapping has been undertaken to enable an improved understanding of the deposit geology to be gained.

Channel sampling in September 1998 returned encouraging results over an extended strike length. A soil sampling and trenching program over a 700 m strike length of Vein 49 and 300 m of Nelson were undertaken during 1999. A more detailed trenching program was undertaken in 2000 which defined zones of continuous mineralisation.

Geophysical surveys were undertaken in 1999 (airborne magnetic and radiometric), 2000 and 2001 (time domain induced polarisation). These surveys assisted in identifying targets for exploration drilling, some of which are yet to be explored.

To August 2004, six drilling campaigns had been conducted at the Calcatreu project and were used in the compilation of the resource estimate described in Micon 2004B. La Source/Normandy conducted four programs between 1999 and 2002. AQI completed a small due diligence program in 2003 and has since completed further programs of drilling at increasingly closer spacing. A summary of the drilling campaigns, undated to December 2005, is shown in Table 6.1.

The Normandy drilling comprised reverse circulation (RC) ore-collars with diamond drill (DD) coring below the collar. All quartz vein intercepts were with diamond coring. The AQI drilling comprised RC or combined RC drilling to the water table changing to DD at that point. Changes from RC to DD whilst in a vein were avoided.

Table 6.1 Drilling programs

Drill program	Drilled by	Date completed	No. holes	M drilled
1	La Source	April 1999	11	1188.6
2	Normandy	March 2000	20	1764.2
3	Normandy	March 2001	11	1084
4	Normandy	March 2002	9	1764
5	Aquiline	July 2003	14	1274.5
6	Aquiline	August 2004	185	19,583.75
7	Aquiline	December 2005	55	6228

Approximately 92% of drilling to December 2005 was focused on the Vein 49 and Nelson deposits, with the remainder focused on other prospects within the Calcatreu project area. From December 2005 to the time of writing, an additional 90 holes for approximately 8,000 m have been drilled in the nearby Belen, Puesto, Nelson, Castro Sur and Viuda de Castro prospects. The economic potential of these prospects has not been considered in this assessment.

Records have been made of drilling campaigns using standard field techniques including geological and geotechnical logging, photography and collar and down-hole surveys. A number of different hole naming conventions have been used over the different programs. AQI is taking care to avoid confusion emanating from the different naming conventions.

The results of AQI's drilling campaigns since August 2004 have been periodically reported but have not been used in the preparation of an updated estimate of the mineral resource at Calcatreu. The area of economic interest has now been drilled with drill lines at approximately 50 m spacings and mineralisation pierce points at 25 m to 50 m spacings.

6.3 Sampling Method and Approach

La Source/Normandy commenced sampling DD core 6 m outside obvious veining with 2 m length (3 m for La Source) samples. Within veining, samples of 1 m length were collected. Intervals of less than 1 m were allowed where vein widths were between 0.5 m and 1 m. The geologist in charge marked sample intervals and the recommended orientation for splitting during logging. RC drilling chips were automatically split at the rig and collected as 20 kg samples.

AQI commences sampling DD core 5 m to 10 m before the start of obvious veins, or veined or mineralised andesite, taking 2 m samples. Samples of 1 m length were taken within veins. Samples of less than 1 m length were allowed by the geologist to honour lithologic breaks. Sample intervals were marked by the geologist during logging. RC drilling chips were collected in plastic bags and a small sample taken for logging by binocular microscope examination. The logging geologist then determined which samples should be split to make duplicate 2 kg samples for analysis and storage. All reject cuttings are stored on site in an orderly fashion.

6.4 Sample Preparation, Analyses and Security

The same logging, storage and sample handling facilities in Jacobacci have been used by Normandy and AQI. The facilities include an office and storage shed in a walled yard, behind a locked gate. AQI's consultants or staff geologists collect the samples in the field, at the drill, and deliver them to the secure facility in Jacobacci. Diamond drill core is logged, sawn in half and sampled, and RC samples split, at this facility. Sample splits and half core are stored there or at another locked shed in town. Samples are stored ready for shipment in a shed in the yard at the Jacobacci office. Samples are transported from AQI's field office by dedicated courier truck, under supervision of contracted independent personnel, to the selected analytical facility, ALS Chemex, in Mendoza.

Normandy and AQI have used the services of reputable laboratory ALS Chemex to undertake sample preparation and analysis work. Normandy used ALS Chemex facilities in Mendoza, Argentina, for sample preparation and analysis, while AQI has used the Mendoza facilities for sample preparation and the ALS Chemex facilities in Santiago, Chile, for analysis. ALS Chemex offers a number of standard processes by which to undertake analysis. The selection of the appropriate process is at the discretion of the sample provider.

Normandy used a fire assay with AAS finish technique to determine gold concentration, and multi-acid digestion followed by AAS for base metals determination.

AQI's due diligence program used fire assay with AAS finish to determine gold and silver content. Aqua regia digestion with ICP AES (inductively coupled argon plasma atomic emission spectroscopy) was used to analyse for a large (34) suite of base metal and other elements.

For the ongoing exploration and definition programs, AQI has specified the use of a fire assay with AAS finish for gold and silver analysis. If high gold values are indicated (Au > 10 g/t), follow-up fire assay with gravity finish analysis is conducted. If high Ag values are indicated (Ag > 50 g/t), a multi-acid digest with AAS method is used. For higher Ag values (Ag > 350 g/t), a silver fire assay with gravity finish is used. A multi-acid ICP AES method is used to analyse for an additional 27 elements including base metals.

6.5 Data Verification

Normandy and AQI employed QAQC and data verification procedures during their respective sampling programs.

Normandy made use of standards, duplicates, check assaying and re-sampling to ensure reliable results. A process of database checking and verification was carried out and any modifications to the database were restricted to the geologist in charge.

For the due diligence program, AQI relied on standards, duplicates and blanks inserted into the sample stream by ALS Chemex. For the ongoing exploration program, AQI developed its own routine of inserting filed duplicates, field blanks and standards into sample batches for analysis by ALS Chemex, in addition to the QAQC procedures managed by ALS Chemex. A single company officer is responsible for managing the program and analysing its results. AQI has established an electronic database which incorporates automated querying of the data to identify erroneous entries and to identify data results of concern.

In 2003 AQI's consultant, Micon, conducted a due diligence program of check sampling and data validation as part of its consulting commission. Micon made some recommendations for improvement but concluded that the assay database presented by AQI was a suitable one for use in the estimation of a mineral resource.

7 Adjacent Properties (Item 17)

AQI is the dominant mineral tenement holder in the region and there have been no mineral discoveries of significance made in the immediate vicinity apart from those in AQI's portfolio.

The most significant occurrence of mineralisation with some proximity to Calcatreu is the Navidad project located in Chubut Province approximately 100 km southeast of Calcatreu. In July 2006, AQI gained ownership of Navidad through a court award.

Navidad comprises five deposits: Galena Hill, Connector Zone, Navidad Hill, Calcite Hill and Calcite Hill NW Extension. These deposits are located on the Navidad Trend and form nearly continuous, near-surface zones of mineralization over a strike length of 3.6 km. In 2006, Snowden estimated a total Measured and Indicated Resource at Navidad of 93.4 Mt at 102 g/t Ag and 1.41 % Pb reported at a 50 g/t Ag cut-off.

8 Mineral Processing and Metallurgical Testing (Item 18)

The details of mineral processing and metallurgical testwork undertaken on Calcatreu mineralization is fully described in the processing volume of the Initial Feasibility Study. A summary of the conclusions reached is presented below.

The practical and operational aspects of metal extraction have been considered by AQI's processing consultant, Ausenco Limited, and comparative capital and operating cost estimates developed for these three process options:

- Carbon-in-leach (CIL)
- Merrill Crowe
- A hybrid circuit, combining aspects of both CIL and Merrill Crowe.

The CIL option is estimated to have the lowest capital cost while Merrill Crowe has the lowest estimated operating cost. There is no significant "net present cost" (NPC) difference for the CIL and Merrill Crowe options, while the hybrid option has the highest. Metallurgical testwork indicated potential for higher silver extraction when leaching in the presence of carbon. This higher silver extraction has become more significant following the recent rise in the silver price. Further testwork is required to quantify any higher silver recovery, but this could offset the marginally lower NPC found for the Merrill Crowe option.

For the CIL and hybrid options, the ability to achieve design carbon loadings may be considered a process risk. The CIL option has been designed to handle a maximum silver grade of 53 g/t and if ores with higher silver grades are to be processed, then this will impact upon plant performance.

The solids settling rates of some of the variability samples were poor and this poses a significant process risk for Merrill Crowe or hybrid options.

On a technical basis, the advantages and disadvantages of the three options appear to balance out, giving no clear preferred option. Ausenco recommends the CIL option for the Calcatreu Gold Project for the following reasons:

- It has lower risk than Merrill Crowe or hybrid options when processing ores with poor settling characteristics.
- The "net present cost," is not significantly different to that for the Merrill Crowe option over the expected five year life of mine.
- It has some potential for higher extraction of silver in the presence of carbon.
- It is better able to operate with variations in feed rate than a Merrill Crowe circuit.
- It has a lower capital cost for the short project life.

While CIL is the recommendation from this assessment, Ausenco recommends repeating the process evaluation as a first step in any study of the project following on from this assessment. Changes to the mine plan, mine life, head grade and silver price, will all impact on selection of the optimum processing route.

Discussions with Ausenco concluded that plant feed grade variability is unlikely to present an unacceptable risk to processing performance given the scale of operations and Ausenco's planned provision of a one month run-of-mine (ROM) stockpile.

As this mining study was approaching completion, Ausenco advised that the most recent metallurgical testwork indicated process recoveries suitable for use in mining studies are 91% for Au and 75% for Ag. The mining study reported here has used earlier advised recoveries of 91% for Au and 66% for Ag. Applying the updated recoveries to the design through metal equivalency and cut-off grade factors results in a less than 2% variation in the high grade production inventory. This variation is within the accuracy of the existing study .and so does not justify a re-estimation of the mining inventory considered in the study for scheduling and costing purposes.

9 Mineral Resource and Mineral Reserve Estimates (Item 19)

9.1 Mineral Resource

The mineral resource estimate and the methods used to prepare it are reported fully in Micon 2004B. A brief summary of the methods employed and the results is presented below.

Micon reports that the mineral resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council August 20, 2000.

Micon associate, qualified person Eugene Puritch, prepared an ordinary Kriged mineral resource estimate for Calcatreu in a block model comprising blocks measuring 2.5 m in the X direction, 5 m in the Y direction and 5 m in the Z direction. Measured and Indicated interpolations were performed in two consecutive search ellipsoid passes. Inferred resources were interpolated in 2 subsequent passes (Inferred 1 and 2) in order to fill the model. Blocks not filled after each pass were interpolated in the subsequent ones. After the first pass, search ellipse ranges were relaxed and after the second pass minimum required samples for interpolation were relaxed. The Nelson and Vein 49 areas, as well as the quartz and andesite domains, were interpolated separately in order to prevent use of data outside of each vein or domain. Maximum search distances at Nelson were selected so as to prevent data from Nelson East affecting the Nelson West vein and vice versa.

Under the CIM definitions, a mineral resource must be potentially economic in that it must be “in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction.” Puritch used a cut-off grade of 0.55g/t Au for the reporting the mineral resources at the Calcatreu Project. This cut-off grade is reported to have been based upon a simple review of the deposit geometry, local topography and the assumption that open pit mining with processing by heap leaching would be employed to exploit the marginal resource. A 0.55 g/t cut-off would represent US$7.00 per tonne contained gold (at a price of US$396/oz) and, at reasonable heap leach type recoveries, should render enough cash flow to approximately cover the cash costs of production. The metallurgical and mining cost assumptions used were considered reasonable for the reporting of a mineral resource at the time.

The block model was reported using the 0.55 g/t Au cut off to produce the mineral resources reported in Table 9.1. The majority of the blocks were coded as indicated resources and the geological domain model of the quartz vein showed good continuity from hole to hole and section to section as well as good agreement between sectional interpretation and the surface mapping. As a result, a majority of the mineralization reported above cut-off is in the indicated category.

The results of AQI’s drilling campaigns conducted since August 2004 are yet to be included in estimating the mineral resource at Calcatreu.

Table 9.1 Mineral Resource 0.55 g/t Au cut-off (Micon 2004B)

	INDICATED					INFERRED				
Zone/mineralisation type	Tonnes	Au g/t	Ag g/t	Au oz	Ag oz	Tonnes	Au g/t	Ag g/t	Au oz	Ag oz
Vein 49 Oxide										
Quartz vein	758,000	5.34	32.9	130,14	801,800	3,000	1.72	10.2	170	1,000
Mineralised andesite	410,000	0.77	9.9	10,150	130,500	0	0.00	0.0	0	0
Vein 49 Primary										
Quartz vein	2,998,00	3.79	37.8	365,32	3,643,50	997,000	2.76	25.5	88,470	817,400
Mineralised andesite	854,000	0.75	8.2	20,590	225,100	343,000	0.86	7.7	9,480	84,900
Subtotal Vein 49	**5,020,00**	**3.26**	**29.8**	**526,20**	**4,800,90**	**1,343,00**	**2.27**	**20.9**	**98,120**	**903,300**
Nelson Area Oxide										
Quartz vein	338,000	2.83	20.5	30,750	222,800	53,000	1.54	12.8	2,620	21,800
Mineralised andesite	149,000	0.79	7.7	3,780	36,900	21,000	1.01	8.0	680	5,400
Nelson Area Primary										
Quartz vein	510,000	2.35	28.4	38,530	465,700	405,000	1.80	17.4	23,440	226,600
Mineralised andesite	138,000	0.74	9.7	3,280	43,000	45,000	0.73	6.4	1,060	9,300
Subtotal Nelson	**1,135,00**	**2.09**	**21.1**	**76,340**	**768,400**	**524,000**	**1.65**	**15.6**	**27,800**	**263,100**
Total Oxide	1,655,00	3.29	22.4	174,82	1,192,00	77,000	1.40	11.4	3,470	28,200
Total Primary	4,500,00	2.96	30.3	427,72	4,377,30	1,790,00	2.13	19.8	122,450	1,138,200
Grand Total	**6,155,00**	**3.04**	**28.1**	**602,54**	**5,569,30**	**1,876,00**	**2.10**	**19.4**	**125,920**	**1,166,400**

9.2 Mineral Reserve estimation

The purpose of this part of the Initial Feasibility Study is to describe the methods used and to report the results of preparing the mineral reserve estimate. However, it is apparent at the time of reporting that no economic or practical processing route for treating the Calcatreu mineralization is available. This is due to legislative restrictions having been placed on the use of cyanide in Rio Negro province. Under these circumstances, no estimate and statement of a mineral reserve can be made. Under direction from AQI, Snowden has completed the work required to estimate the potential mining inventory which may become available in the event that the restrictions on the use of cyanide are lifted. The remainder of this report describes the methods used and results of the estimation of the potential mining inventory.

The potential mining inventory has been estimated using a structured process of analysing the mineral resource estimate, identifying an appropriate mining method, preparing an optimum ultimate mining outline, preparing a detailed mine plan consistent with the optimum ultimate mining outline, estimating levels of dilution and mineral loss expected with the chosen mining method, calculating a cut-off grade to use to report potentially economic and waste materials in the pit design, selecting the material resources (labour, equipment and materials) needed to execute the mining plan, preparing mining and waste dumping schedules, and preparing capital and operating schedules for the planned mining operation.

The mineral resource estimate used to prepare the mine plan and potential mineral inventory estimate is described in Section 9.1 above, and fully in Micon 2004B. Most of the mineral resource is classified as Indicated and none is classified as Measured. Therefore any estimate of a potential mining inventory can only be made to a medium level of confidence.

The mineral resource estimate is represented by a three dimensional block model which includes separate models representing quartz and andesite mineralization. The estimate of potential mining inventory is included in the mineral resource estimate.

9.3 Mining method

The Calcatreu project has been the subject of a preliminary assessment (Micon 2004B) which described the potential for economic open pit mining of the Vein 49, Nelson East and Nelson West mineralized zones. Micon 2004B also considered extraction of the mineralisation which fell below the optimum pit shell, as determined by Micon, by underground methods. The potential mining inventory estimate prepared by Snowden has only considered mineralization which is accessible by open pit methods. Further studies will be required to estimate a potential mining inventory suitable for extraction by underground methods.

Micon 2004B considered project production rates of 1,500 tpd, 2,000 tpd and 2,500 tpd and concluded that 2,000 tpd delivered the best balance of capital expenditure, operating cost and mine life.

This study assumes mining by open pit methods with production material being delivered to a ROM pad which has one month's feed storage capacity. Any requirements for feed blending will be managed by short term in-pit scheduling and ROM draw-down strategies. There is no requirement for any other significant mining, stockpiling or feed blending activity.

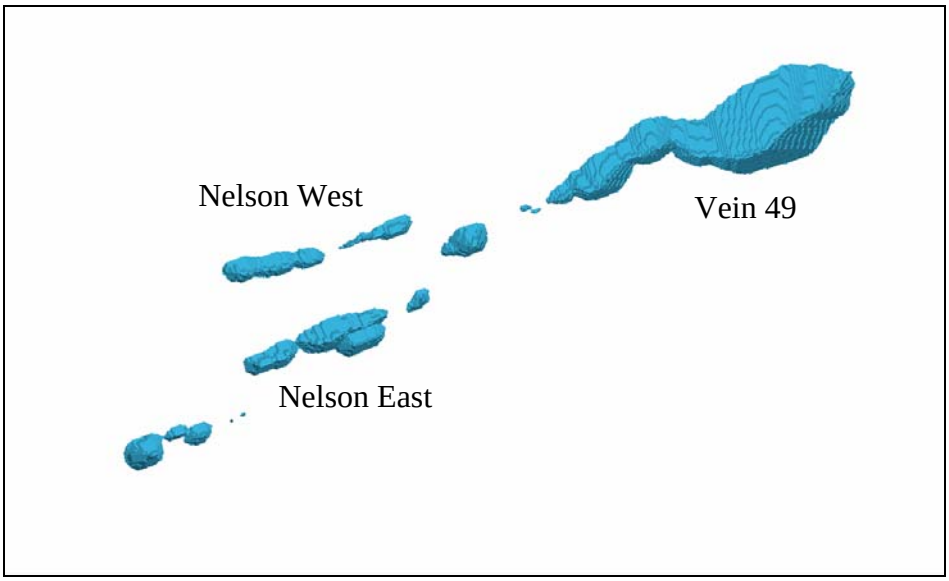
9.4 Pit Optimisation

The optimum ultimate pit outline was identified using the well recognised Whittle 4X optimization software (Whittle). Cyanide tank leaching was selected as the primary processing method for application in the optimisation. The optimization process adopted using Whittle is more fully described in Appendix A.

The Whittle optimisation routine identifies the combination of resource blocks that can be exposed and recovered to deliver the greatest NPV in an open cut mine for a given set of design, operating and economic assumptions. The resultant pit outline is thus the optimum for the criteria used. The optimum pit shell can then be used as the basis for preparing the detailed mine plan. The optimum pit shell is shown in Figure 9.1.

The pit shell selected as the basis for the detailed design is Shell 25 of the optimisation sequence which contains 24.8 Mt of waste and 2.9 Mt of ore grading 4.56 g/t Au and 38.89 g/t Ag.

Figure 9.1 Optimum pit shell (Shell 25)



The optimum shell shows a number of small pits in the Nelson area, in addition to the large Vein 49 pit. Where possible, these will be incorporated into the detailed design.

9.5 Cut-off grade calculation

A cut-off grade is required to identify material within the detailed pit design that can be processed and sold economically. The methodology used to calculate the cut-off grade is described in detail in Appendix A.

The cut-off grade is calculated by normalising the recovered silver value to a recovered gold equivalent and determining the cut-off in terms of the gold equivalent.

Based on metal prices of US$500 /oz for Au and US$8.00 /oz for Ag, the gold equivalency formula for the cyanide leach process is:

- AuEq g/t = Au g/t + (Ag g/t/86.2).

The cut-off grade is 1.12 g/t AuEq.

For the low grade stock for passive leaching, the gold equivalency formula is:

- AuEq g/t = Au g/t + (Ag g/t/117.2).

The cut-off grade is 0.48 g/t AuEq.

9.6 Pit design and mining inventory

Pit designs were prepared using Surpac software, based on the Whittle optimum Shell 25, block model data imported from a gold equivalent model constructed using Gemcom software, and geological ore solids and terrain contours supplied by AQI.

9.6.1 Pit Design

General Design method

The design process involves viewing the Whittle shell in horizontal sections at increments common with mining benches spaced at intervals determined by geotechnical guidelines. Floor outlines are initiated at the lowest mining depth at a minimum practical mining width. Life-of-mine pit ramps are simulated to find the best starting point on the lowest pit floor. Benches and associated ramps are then designed progressively bottom-up from toe to crest and bench to bench using the following guidelines:

- Geotechnical pit wall slopes and berm widths as provided in Golder 2005
- Andesite and quartz sections used in conjunction with block model grades to design maximum above cut-off mineralisation and minimal waste within the optimum Whittle shell
- Minimum mining width of 10 m to incorporate the chosen mining fleet
- General ramp design at 1 in 10 gradient
- Single lane ramp of 12 m width to accommodate the Cat 777F, clearance either side, and a bund at axle height
- Ramp outside turning radius of 17 m or greater in accordance with Cat recommendations
- Designs must be practical
- Final design volumes and material types and grades must be within an acceptably low variance of the Whittle Shell 25
- There are no special requirements for blending ex-pit
- Groundwater inflows are not expected to be significant based on the preliminary analysis reported in Water Management Consultants 2005 and require no specific provisions within the detailed designs.

Pit wall angles

The inter-ramp pit slope angles (excluding the ramps) are measured in terms of the slope angle from bench crest to crest and are:

- 55 degrees for fresh material
- 48 degrees for oxide material

These angles are derived from the specific final pit bench and berm dimensions:

- Bench height in fresh material is 20 m
- Berm width in fresh material is 8 m
- Bench heights in oxide material or material less than 30m below surface is 10 m
- Berm widths in oxide material or material less than 30m below surface is 6 m.

Narrow sections of the pits

Special design considerations are made for the design of the narrow sections at the base of the open pits. The following methods were used to access as much mineralized material at the base of the pits without widening the overall design and incurring excessive overburden waste material:

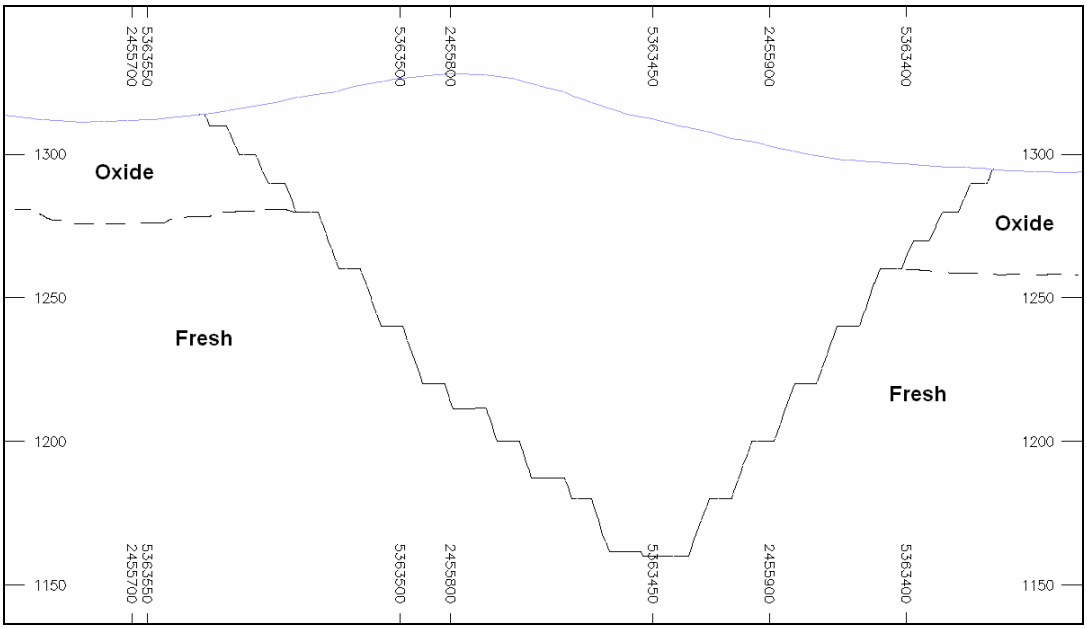
- Ramping down on ore for the final benches
- Retreating by taking a final cut down with the excavator with dimensions 5 m deep and 5 m or greater mining width.

Vein 49 pit

The main Vein 49 pit final ramp has been located to the foot wall of the mineralized zone for the bulk of its length to allow for a push back on the hanging wall and to be in a geotechnically favourable position. The ramp switches over to the hanging wall side of the mineralized zone, past the pit saddle, to shorten haulage distances to the run-of-mine (ROM) pad and waste dump.

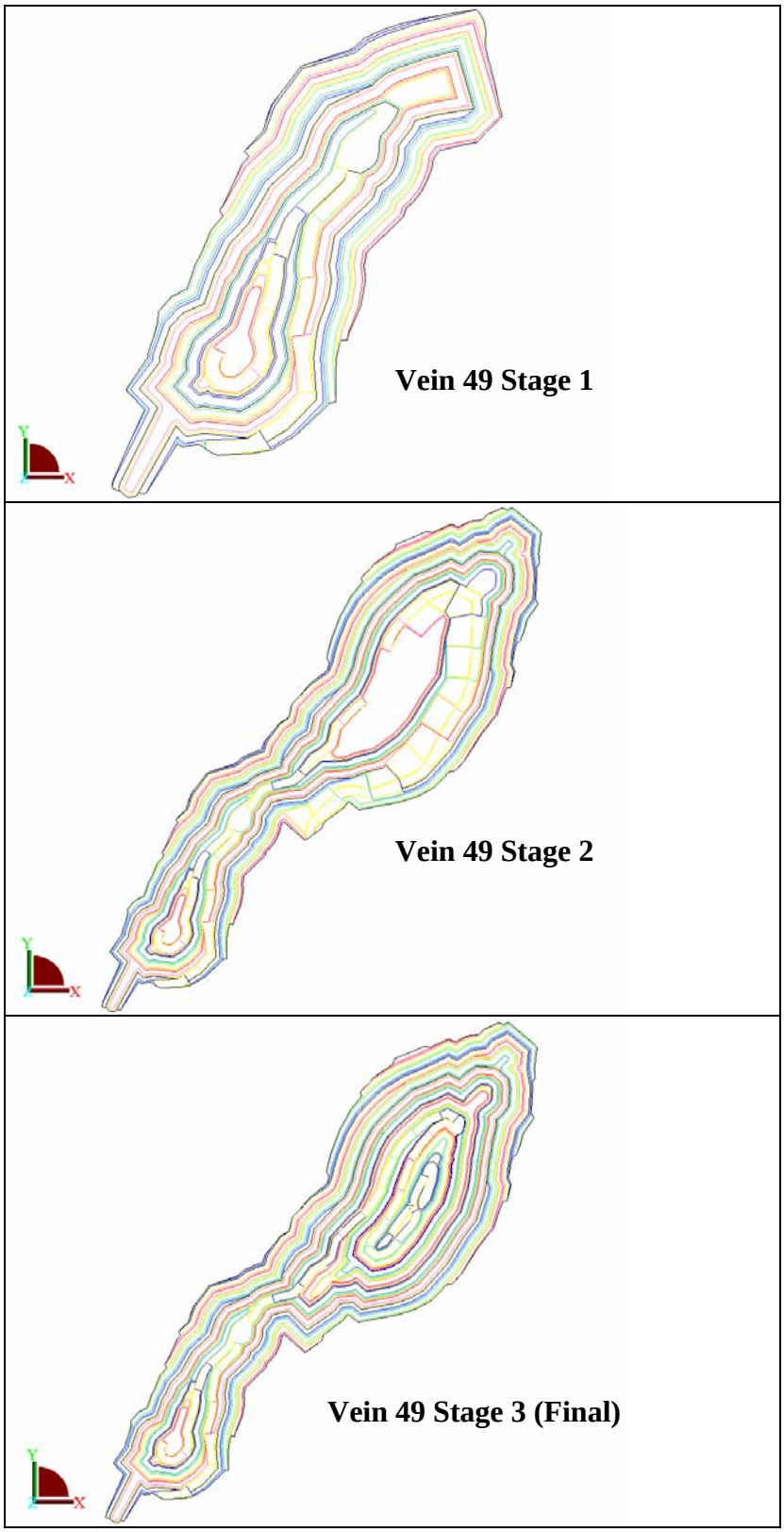
The vertical extent of the Vein 49 final pit ranges from 1300 mRL to 1160 mRL, a depth of .140 m. The pit is 1,000 m long and is 300 m wide at the crest. Overall slope angles are 48 degrees in oxide ore and 55 degrees in fresh ore. A typical cross section is shown in Figure 9.2

Figure 9.2 Typical vein 49 final pit cross section looking northeast



Vein 49 final pit is divided into three stages (Stage 1, Stage 2, and Stage 3, the Final Stage) for efficient scheduling. These stages are based on the Whittle shells preceding Shell 25, maximising the early extraction of more profitable material. The Vein 49 pit design stages are shown in Figure 9.3.

Figure 9.3 Vein 49 pit design stages

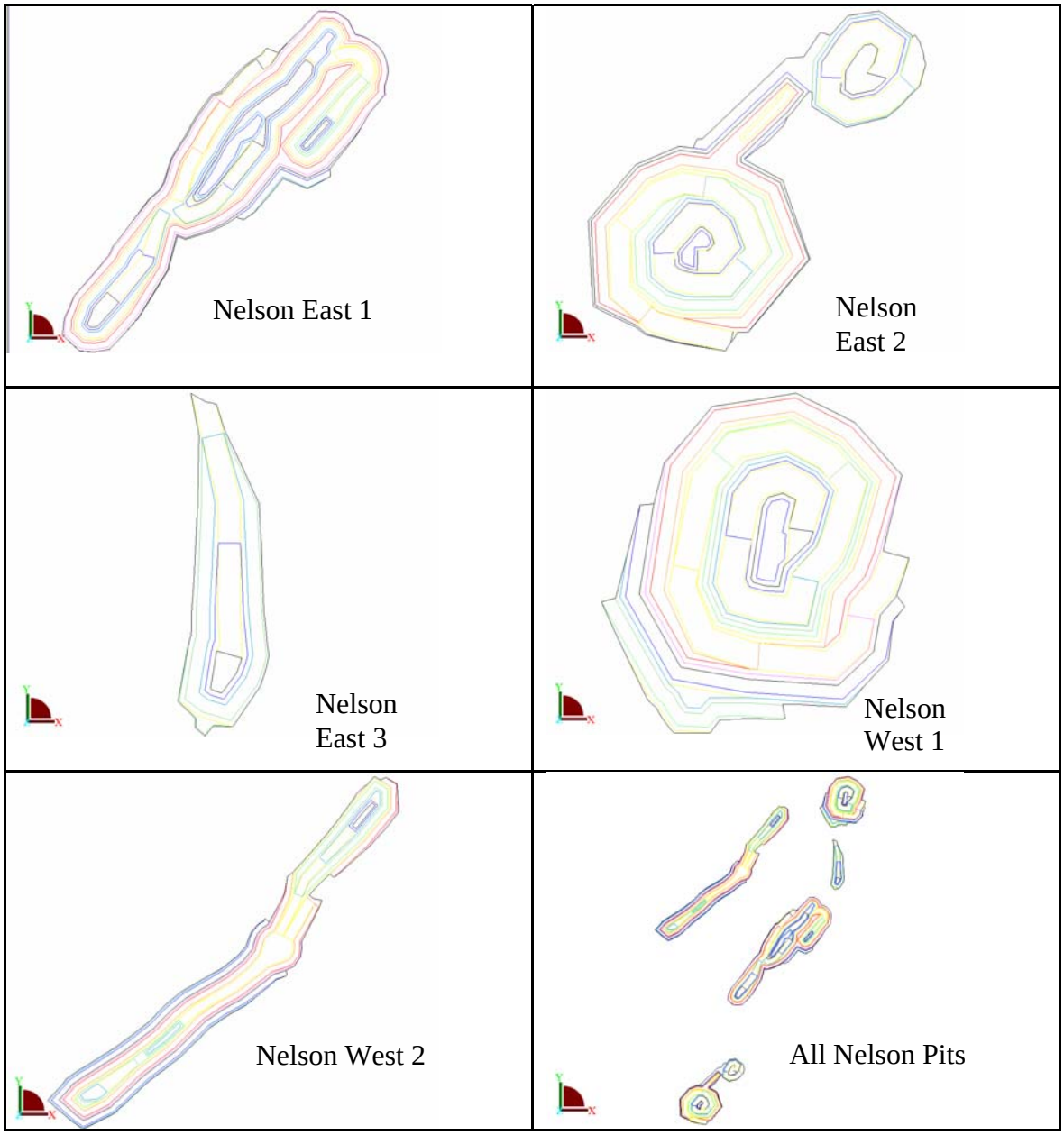


Nelson East and Nelson West pits

Five separate pits have been designed for the Nelson area. The Nelson East and Nelson West pits are designed to reduce haul distance to the ROM pad and their respective dump areas, and to reduce ramp length utilising the existing topography.

These pits are too small to justify push backs for separate mining stages. The Nelson pit designs are shown in Figure 9.4. Their vertical extents range from 1300 mRL to 1220 mRL, a depth of 80 m.

Figure 9.4 Nelson pit designs



To ensure the pit designs closely match the Whittle optimum pit shell, a comparison has been prepared considering volumes and metal contents in each. The comparison is shown in Table 9.2.

Table 9.2 Comparison of volumes and metal contents

Location	Pit design		Shell 25		Difference	
	Volume (bcm)	Metal Au (kg)	Volume (bcm)	Metal Au (kg)	Vol Diff %	Metal Diff %
Vein 49	10,144,019	15,313	9,971,875	15,744	1.6%	-2.8%
Nelson East 1	553,688	644	632,375	831	-14.2%	-29.1%
Nelson East 2	222,250	162	198,750	227	10.6%	-40.5%
Nelson East 3	25,375	24	32,000	32	-26.1%	-31.5%
Nelson West 1	267,000	267	204,563	279	23.4%	-4.7%
Nelson West 2	394,375	554	410,000	612	-4.0%	-10.5%
Total	11,606,707	16,963	11,449,563	17,725	1.3%	-4.5%

The overall metal discrepancy is less than 5% which Snowden considers to be satisfactory. The Nelson pits display, in general, a higher discrepancy. This is due to the very narrow and high grade characteristics of the mineralised material in the number of small pits identified by the Whittle optimisation. This impinges on the creation of practical pit designs which closely match the Whittle output.

Given the small proportion of total volume and ounces contained in the Nelson pits, Snowden considers the overall result to be acceptable for the purposes of this mine plan, and recognises that detailed planning during the operational phase may provide improved results.

9.6.2 Dilution evaluation

The pit designs and geological interpretations of the mineralised structures were assessed for possible dilution occurring during mining. This was done by examining horizontal sections bench by bench and estimating the impacts of practical mining widths achievable with the planned mining equipment and the geometry and nature of the mineralisation. The following properties were calculated:

- High grade mineralisation diluted by an additional 5% volume of adjacent material
- The high grade mineralisation dilution grade is at the low grade mineralisation average grades of 0.36 g/t Au and 4.93 g/t Ag, or at zero grade, depending on the provenance of the dilution
- Low grade mineralisation diluted by an additional 8% volume at zero grade
- Ore loss from the low grade mineralisation of 2.5% to the high grade mineralisation and 5% to waste material.

9.6.3 Waste dump design

The waste dumps are designed to accommodate the waste material reported by the cut-off grade interrogation of the pit designs. Provision has been made to backfill the Nelson East and West pits once they are completed. Two waste dumps are designed. The

Vein 49 dump takes the bulk of the Vein 49 waste material, and the southern dump takes the waste material from the Nelson East 1 and Nelson West 2 pits.

The topsoil dump is been designed to accommodate 0.5 m of topsoil to be stripped from all cleared areas. The topsoil dump is located close to the plant and tailings dam areas where it will be needed at the end of the mine life for rehabilitation purposes.

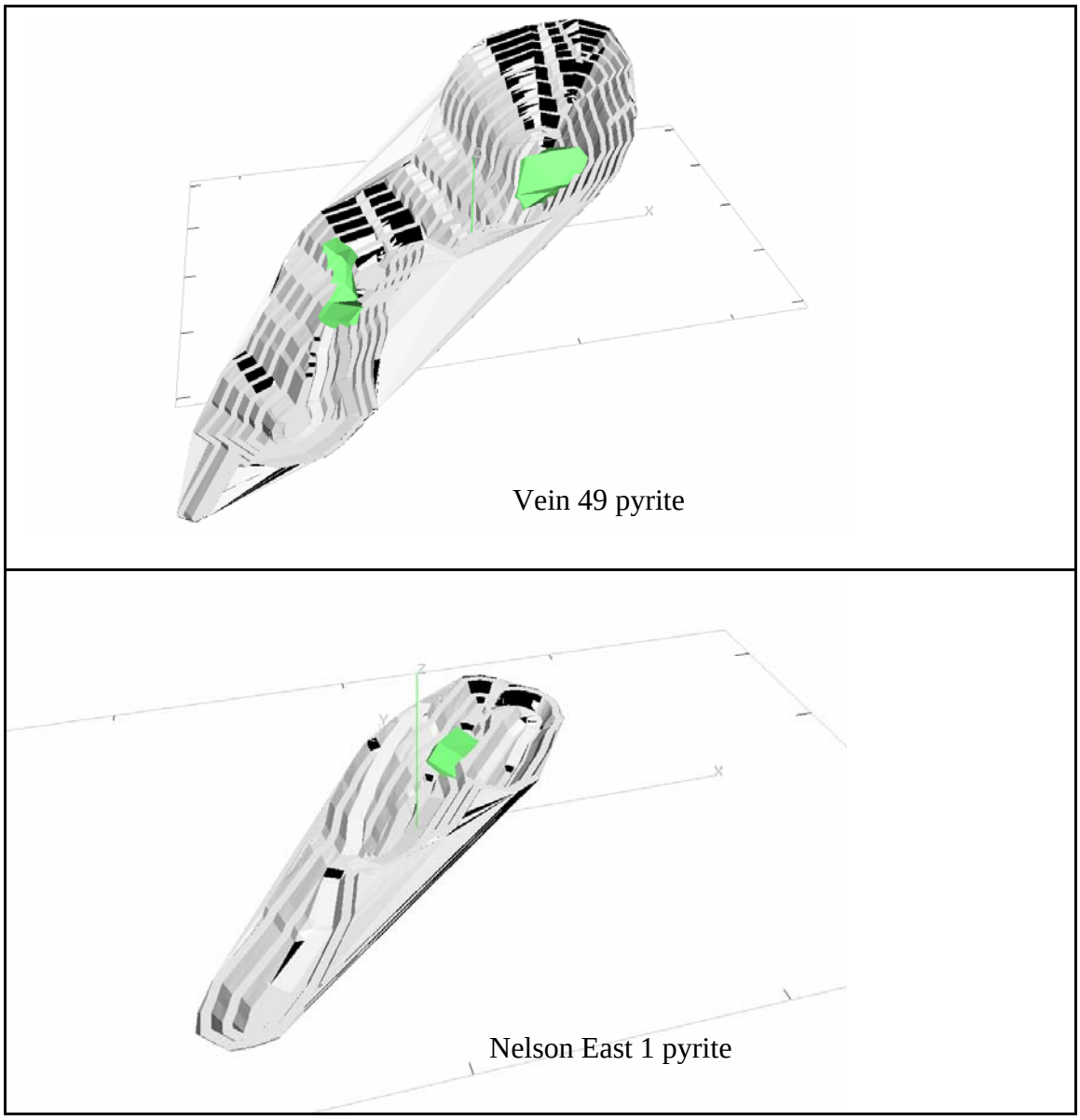
Table 9.3 lists the quantities for each dump and their respective sources.

Table 9.3 Waste dump volumes

Dump	Waste material source	Swell volume m^3
South	Nelson East 1 and Nelson West 2 waste	975,698
Vein 49	Vein 49 waste	10,119,370
Nelson pits	Nelson and Vein 49 pits	1,462,697
Topsoil	All pits	66,411

There is no differentiated dump design for oxide and fresh rock materials. AQI provided geological interpretations of waste character domaining which identify zones comprising medium (1% to 4%) and high (+4%) pyrite content in the Vein 49 and Nelson East 1 pits. Snowden developed wireframe solids around these interpretations for the purpose of scheduling the mining and dump deposition of the subject material. The quantity of pyritic material assumed to be potentially acid generating has been scheduled to be dumped either in mined out pits or encapsulated within one of the dumps. Encapsulation of the quantity of pyritic waste will be manageable through short term planning and operating practices. The interpreted pyritic waste solids are shown in Figure 9.5.

Figure 9.5 Interpreted pyritic waste solids



9.6.4 Dump Design method

The dump designs were created by first identifying a favourable location and defining the ultimate footprint required to accommodate the waste volume requirements. The design process was governed by the following guidelines:

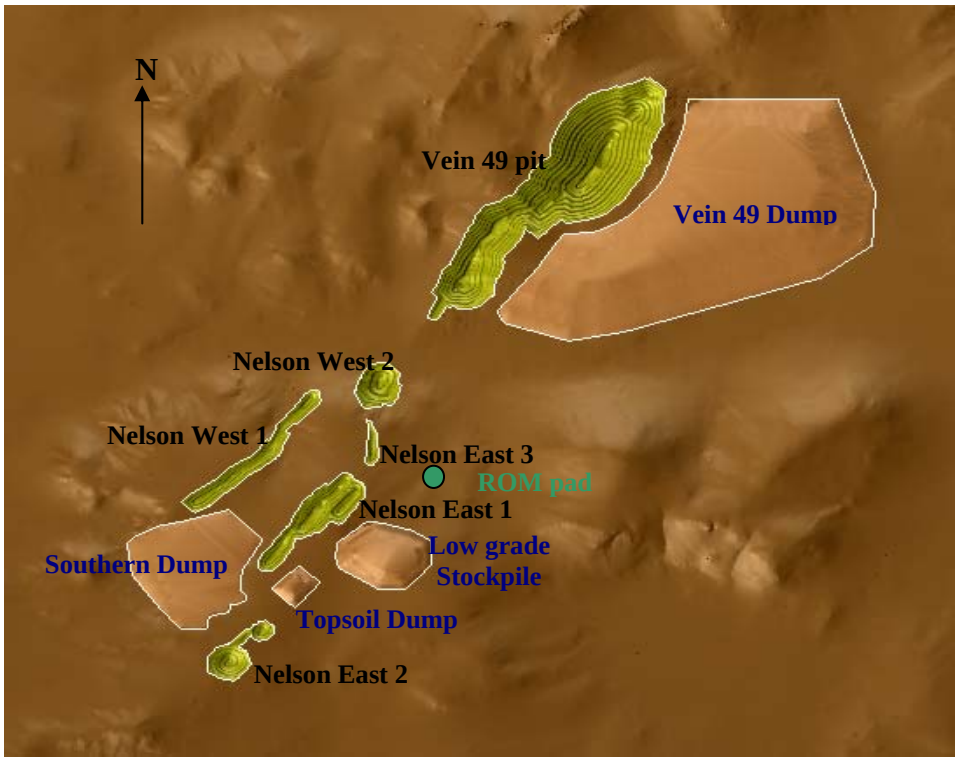
- Provision of an abandonment bund zone in accordance with the Western Australian Department of Industry and Resources Guideline "Safety bund walls around abandoned open pit mines" (December 97)
- Final slope angles of 18 degrees toe to crest, and 10 m berm widths at intervals of 20 m bench height, as recommended by Snowden Geotechnical Division to facilitate stable rehabilitation
- Volume requirement estimated by applying a swell factor of 30% to the broken material to be accommodated
- Minimum impact on the natural drainage of the region
- Match the dump profile with the existing topography as closely as possible
- Keep waste haul distances as short as possible
- Waste dump designs to be practical and manageable.

9.6.5 Backfill

The Nelson East and West pits are scheduled to be backfilled entirely. The waste material generated from the Nelson East 2, East 3, and West 1 pits is directly backfilled into the Nelson East 1 and West 2 pits. The remaining void is filled waste material generated from the Vein 49 pit at the later stage of its production. A high proportion of pyritic waste is scheduled to be backfilled into the Nelson pits.

Pit and dump locations are shown in Figure 9.6.

Figure 9.6 Pit design and dump locations



9.7 Mineral Reserve statement

No statement of a mineral reserve can be made for the Calcatreu Project in accordance with the requirements of NI 43-101 as a feasible processing method is yet to be identified. A technically successful method of processing has been identified which currently cannot be considered feasible due to the regulatory restrictions placed on the use of cyanide by the Rio Negro provincial government.

However, this study has used a mineral resource estimated in accordance with NI 43-101 to estimate the potential high grade mining inventory which may become available in the event the technically successful processing method can be implemented. Since the mineral resource estimate is classified as being Indicated, the estimate of potential mining inventory is at a medium level of confidence.

The estimated potential high grade mining inventory is:

- 3.50 Mt at 3.86 g/t Au and 33.22 g/t Ag for 435 koz Au and 3,739 koz Ag.

This is not a mineral reserve and should not be interpreted as such.

An additional potential low grade mining inventory has been estimated which is contingent on identifying a technically and economically feasible processing method. Due to the Indicated classification of the mineral resource and the uncertainty of the processing method, the potential low grade mining inventory is of a low level of confidence.

The potential low grade mining inventory is:

- 1.37 Mt at 0.64 g/t Au and 9.19 g/t Ag for 33 koz Au and 405 koz Ag.

This is not a mineral reserve and should not be interpreted or reported as such.

10 Other Relevant Data and Information (Item 20)

Relevant information regarding equipment selection, drilling and blasting and manning levels is presented below. Further details are provided in Appendix C.

Snowden is unaware of any other relevant data or information which has not been considered in the preparation of this report or in reports previously lodged in accordance with NI 43-101.

10.1 Production haulage fleet

10.1.1 Equipment options

The selection of production equipment is made based on productivity, operating and capital cost considerations. Haul cycle times and base cost models were generated for a range of possible combinations of production mining equipment. Inputs for this process were the estimated schedule from Whittle, haul routes as generated from the Whittle shells, and indicative costs which were drawn from a recent study with similar operational features, economic climate and background.

Combinations of the following items of equipment were assessed:

- Caterpillar Haul trucks 773D, 775E, and 777F
- Caterpillar Excavators 5110B ME, and 5130B ME
- Komatsu Haul trucks HD485, and HD785
- Komatsu Excavators PC1250, PC1800, and PC3000
- O&K Excavator RH90.

10.1.2 Haul cycle simulations

Haul cycles were simulated using Talpac software. Talpac software matches its internal library of equipment performance data to haul route profiles to determine cycle times for selected equipment at various stages of the mine development.

10.1.3 Equipment cost modelling

Each combination of equipment and its performance was input into an independent cost model to enable comparison of costs for the considered equipment options. Only load and haul activities and other activities directly associated with the load and haul process were included in the model to ensure a reliable comparison between each equipment type could be made.

10.1.4 Selection Criteria

Inputs

- Equipment specifications as per manufacturers' handbooks
- Manpower and maintenance costs estimated in similar past studies
- Schedule as provided from the Whittle output

- Haul cycle times generated by Talpac
- Capital requirements for each equipment type
- Practical mining considerations.

Results

The Cat 777F and the O&K RH90 units are the most suitable combination for this project, as shown in the Table 10.1. Key items influencing this choice are the start up capital expenditure, the equipment utilisation and the operating cost. The comparative operating cost was close to that of the Cat 777F and Cat 5130B ME combination but Cat has advised that the 5130B ME excavator will not be available for future purchase. The combination of truck size and number of excavator passes per truck load confirms the suitability of the selected combination of equipment. The suppliers advise availability of all major equipment to be less than 12 months ex-factory on placement of orders.

Table 10.1 Equipment selection comparison table

Truck		CAT 773D	CAT 775E	CAT 777F	CAT 777F	KOM HD465	KOM HD785	KOM HD785	KOM HD785	CAT 777F
Loader		CAT 5110B ME	CAT 5110B ME	CAT 5110B ME	CAT 5130B ME	KOM PC1250	KOM PC1250	KOM PC1800	KOM PC3000	O&K RH90
Truck capacity	t	49.4	62.1	91.2	91.2	55	99	99	99	91.2
Fill factor	%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Heaped volume	m3	35.2	41.2	60.1	60.1	33	55	55	55	60.1
Calc. vol. capacity	m3	24.8	31.2	45.8	45.8	28	50	50	50	45.8
Bucket capacity	t	20.36	20.36	20.36	28.5	13.0	13.0	20	30	28.5
Excav bucket vol.	m3	6.2	6.2	6.2	10.2	6.5	6.5	10.1	15.0	10.3
Fill factor	%	95	95	95	95	95	95	95	95	95
Calc. vol. capacity	m3	10.2	10.2	10.2	14.3	6.5	6.5	10.1	15.1	14.3
Passes	#	**4.2**	**5.3**	**7.8**	**4.7**	**4.5**	**8.1**	**5.2**	**3.5**	**4.7**
No. Excavators		**2**	**2**	**2**	**1**	**2**	**3**	**2**	**1**	**1**
Av. Utilisation	%	79	75	71	78	87	80	63	69	81
No. Trucks		**6**	**5**	**4**	**3**	**6**	**5**	**4**	**4**	**3**
Av. Utilisation	%	92	93	87	95	98	92	94	84	95
Capital expenditure	$M	8.41	7.86	7.14	5.30	8.91	10.39	7.30	8.00	5.10
Operating expenditure	$M	21.09	19.97	18.45	16.80	23.52	24.42	18.00	16.89	16.78
Total expenditure	$M	29.50	27.83	25.60	22.09	32.43	34.81	25.30	24.89	21.88
Operating cost	**$/t**	**0.63**	**0.60**	**0.55**	**0.50**	**0.70**	**0.73**	**0.54**	**0.51**	**0.50**
Net Present Cost	**$M**	**$26.06**	**$24.58**	**$22.61**	**$19.34**	**$28.60**	**$30.80**	**$22.38**	**$22.14**	**$19.13**
Ranking		**7**	**6**	**5**	**2**	**8**	**9**	**4**	**3**	**1**

10.2 Drill and blast

AQI's geotechnical expert report (Golder 2005) was reviewed and considered with an associate expert's advice and Snowden's experience with projects of a similar nature as the basis for determining drill and blast parameters appropriate to the Calcatreu project. Snowden's associate blasting expert's report is included as Appendix B.

The bench height in production material will be 5 m. This bench height is compatible with the requirement for selective mining of high grade vein ore systems such as at Calcatreu. It is also compatible with the selected mining equipment size which is matched to the required production rate.

The bench height in waste will be 10 m, which offers improved economy of scale over the lower production bench height and remains compatible with the selected mining equipment and production rate.

Trim blasting is selected as the means of controlling excavation of final walls. This offers a more cost effective approach than the sometimes used pre-split method though requires more effort in the mechanical cleaning of back-break from final walls.

Selected blasthole diameters are 102 mm for production and at the production/waste interface, and 127 mm in bulk waste. This ensures good geometric compatibility with the bench height, required selectivity of production material and fleet size.

The parameters adopted for drilling and blasting the different rock types at Calcatreu are shown in Table 10.2.

Table 10.2 Drill and blast parameters

Parameter	Unit	Pattern							
Pattern number		1	3	4	6	7	9	10	12
Ore/waste	O/W	O	W	O	W	O	W	O	W
Oxide/Fresh	Ox/F	Ox	Ox	Ox	Ox	F	F	F	F
Wet/Dry	Wet/D	D	D	Wet	Wet	D	D	Wet	Wet
Bench height	m	5	10	5	10	5	10	5	10
Diameter	mm	102	127	102	127	102	127	102	127
Drilled depth	m	5.6	10.8	5.6	10.8	5.7	10.9	5.7	10.9
Fallback	m	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.2
Effective depth	m	5.5	10.6	5.5	10.6	5.6	10.7	5.6	10.7
Effective subdrill	m	0.5	0.6	0.5	0.6	0.6	0.7	0.6	0.7
Stemming length	m	2.6	3.3	2.8	3.5	2.3	3.0	2.5	3.3
Charge length	m	2.9	7.3	2.7	7.1	3.3	7.7	3.1	7.4
Explosive type		Anfo	Anfo	Emul	Emul	Anfo	Anfo	Emul	Emul
Explosive density	g/cm^3	0.85	0.85	1.10	1.10	0.85	0.85	1.10	1.10
Charge weight	kg	20.1	78.6	24.3	98.9	22.9	82.9	27.9	103.1
Drilled burden	m	3.0	3.6	3.2	3.8	2.7	3.3	2.9	3.5
Drilled spacing	m	3.8	4.6	4.0	4.8	3.4	4.1	3.6	4.4
Powder factor	kg/m^3	0.35	0.47	0.38	0.54	0.50	0.61	0.53	0.67
Yield per hole @ 2.5 t/m^3	t	142.5	414.0	160.0	456.0	114.8	338.3	130.5	385.0
Specific drilling @ 2.5 t/m^3	t/m	25.4	38.3	28.6	42.2	20.1	31.0	22.9	35.3

The proportions of each rock type to which each drill and blast pattern is applied to are shown in Table 10.3.

Table 10.3 Proportional application of drill and blast patterns

Rock type	Pattern								
	1	3	4	6	7	9	10	12	Free dig
Quartz oxide	50%		20%						30%
Andesite oxide	50%		20%						30%
Waste oxide		50%		20%					30%
Quartz fresh					90%		10%		
Andesite fresh					90%		10%		
Waste fresh						90%		10%	

The requirement for drilling is shown in Table 10.4. This requirement has been determined to be achievable with 2 drill rigs at a moderate 65% utilisation. This provides adequate back-up capacity in the event of major breakdown.

Table 10.4 Drilling requirements

Material	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Production	M	47,438	41,538	47,114	43,445	27,377	**206,913**
Waste	M	155,135	194,925	134,691	142,166	36,250	**663,167**
Total	**M**	**202,573**	**236,463**	**181,805**	**185,611**	**63,628**	**870,080**

10.3 Manning

Snowden has estimated the operational manning requirement for the planned mining method and equipment fleet based on a four panel, three eight hour shift configuration.

Relief staffing is calculated as 6% of Operations and Maintenance personnel to allow for 15 vacation days per person per year. No relief staffing is provided for Mine Management or Engineering/Geology personnel. No additional personnel are provided for training. The manning schedule is shown in Table 10.5.

Table 10.5 Manning requirements

Position	Year 1	Year 2	Year 3	Year 4	Year 5
Mine Management	6	6	6	6	6
Operations	46	46	46	46	33
Maintenance	14	14	14	14	14
Maintenance labour	20	20	20	20	20
Engineering/Geology	13	13	13	13	13
Total	**99**	**99**	**99**	**99**	**86**

The salary levels supplied by AQI to use for cost estimation are shown in Table 10.6 and Table 10.7 in US$pa.

Table 10.6 Manning Base Salary Levels

Classification	Base Salary
Manager	140,400
Supervisor	114,075
Professional	49,140
Surveyor	38,610
Operator	21,060
Technician	17,550
Support/Clerical	10,530

Table 10.7 Manning Mainentance Salary Levels

Classification	Base Salary
Maintenance Manager	96,525
Superintendent	66,690
Professional/Supervisor	38,610
Tradesman	29,835
Technician	15,795
Support/Clerical	10,530

11 Interpretation and Conclusions (Item 21)

The conclusions reached by this study, which has prepared a mining plan for the Calcatreu Project based on a mineral resource reported in Micon 2004B, are:

- At the time of reporting, no economic or practical processing route for treating the Calcatreu mineralization is available due to legislative restrictions. Therefore no estimate of a mineral reserve can be made. However, an estimate has been made of the potential mining inventory which may become available in the event that the legislative restrictions are lifted.
- The potential mining inventory amenable to treatment by cyanide tank leaching has been estimated and is :
  - 3.50 Mt at 3.86 g/t Au and 33.22 g/t Ag for 435 koz Au and 3,739 koz Ag
- The potential mining inventory is of a medium level of confidence.
- The potential low grade mining inventory which may be amenable to treatment by a passive leach process has been estimated and is :
  - 1.37 Mt at 0.64 g/t Au and 9.19 g/t Ag for 33 koz Au and 405 koz Ag
- The potential low grade mining inventory is of a low level of confidence.
- A mine life of 4.7 years is achieved through mining and processing the potential mining inventory at the nominated throughput rate of 750 ktpa.
- Operating costs associated with the mining operation are $1.45 /t over the life of the project or $11.95 /t of the high grade production material mined.
- Capital costs associated with the mining operation are $12.7 M at start-up and $13.0 M over the mine life.
- Mining at Calcatreu will be simple with no especially challenging mining issues likely to be encountered.
- Mining will be relatively low cost with no especially high cost aspects evident which are atypical of projects of this type.
- Diligent grade control practices will need to be employed to derive the greatest value from the project. Employing such practices will result in mining the optimum combination of volume and grade with the minimum loss of economic material and the minimum inclusion of diluting waste.
- Control over the mining and dumping of pyritic waste will need to be exercised to ensure the potentially AMD generating material is effectively encapsulated in neutral waste, in accordance with expert design recommendations yet to be detailed.
- This study has used preliminary conclusions regarding:
  - Processing performance
  - Hydrogeology

  - Environmental matters.

  There are no findings in the preliminary conclusions which are especially unfavourable to the mine plan.

- This study has used a mineral resource estimate prepared in 2004 which has not been updated with the results of drilling campaigns conducted since that time.

12 Recommendations (Item 22)

Snowden recommends:

- The mineral resource estimate for Calcatreu should be updated using the results of the drilling campaigns conducted since 2004. A more accurate and higher confidence estimate of the potential mining inventory and potential low grade mining inventory could be made using the results of all drilling campaigns.
- Further work should be undertaken to confirm or modify and restate the preliminary conclusions referred to in Section 11 at a definitive level of confidence.
- When the preliminary conclusions referred to in Section 11 are confirmed or modified and restated, the mine plan should be reviewed to identify the impact of any changes, and modified to ensure its appropriateness.
- Processing investigations should be initiated to identify a technically and economically feasible processing method for the low grade potential mining inventory estimated in this study.

13 References (Item 23)

Atlas Copco Mining and Construction, 2006: Quotations and equipment specifications for Atlas Copco equipment, April 2006.

Ausenco Limited, 2006: Calcatreu Gold Project Metal Extraction Case Study, May 2006.

Finning Argentina, 2006: Quotations and equipment specifications for Caterpillar and O&K equipment, April 2006.

Golder 2005: "Pit Slope Design for Nelson and Vein 49 Open Pits Calcatreu Project Argentina"; Golder Associates Ltd, November 2005.

Hagan, T., 2006: Feasibility Study Of The Calcatreu Gold-Silver Project, Argentina – Recommended Drilling And Blasting Operations, Tim Hagan Blasting Pty Ltd, April 2006

Komatsu Latin America Corp., 2006: Quotations and equipment specifications for Komatsu equipment April 2006.

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Orica Argentina, 2006: Quotations and equipment specifications for Orica explosives supply, April 2006.

Puritch G., 2006: "Block Model Data.zip", "Drill Data.zip", "Topo Pit and Mineralized Domains.zip", "Whittle Data.zip"; AQI, January 2006.

Sandvik Argentina, 2006: Quotations and equipment specifications for Sandvik and Tamrock equipment, April 2006.

Secis, R., 2006: Calcatreu Pit Optimisation, Snowden Mining Industry Consultants, May 2006

Water Management Consultants Chile Ltd, 2005: “Preliminary dewatering analysis and recommendations, Calcatreu Project”, memorandum to AQI, February 2005.

14 Date and Signature Page (Item 24)

This report and the work contributing to Sections 9.1, 9.2, 9.3, 9.4, 9.5, 9.6, 10, 11, 12, 15, 16, Appendix A, Appendix B, Appendix C and Appendix D of it have been prepared or supervised by Peter Myers who is a Qualified Person in accordance with the definition contained within NI 43-101.

Peter A Myers BEng (Mining)(Hons), MAusIMM

Divisional Manager

Snowden Mining Industry Consultants

5th March 2007

15 Additional Requirements for Technical Reports on Development Properties and Production Properties (Item 25)

The additional requirements for technical reports on development properties and production properties required by NI 43-101 Item 25 not directly related to mining activities have been reported previously in Micon 2004B or are reported elsewhere as part of this Initial Feasibility Study.

The information from this study regarding mining operating and capital costs, mine labour requirements, the production and waste mining schedule, and the dump construction schedule is presented below.

15.1 Mining Costs

The mining cost estimate has been prepared by developing first principles estimates of activity, consumption and performance for input into a comprehensive cost estimation model. This ensures estimated costs are an accurate representation of actual operating costs. Further details are included as Appendix C.

15.1.1 Input Parameters

Equipment fleet

The primary equipment fleet consists of:

- 2 Tamrock Pantera 1500 drill rigs
- 3 Cat 777F haul trucks for all material movement
- 1 O&K RH90 C excavator for all material loading.

Ancillary equipment consists of:

- 1 Cat D9T Track Dozer
- 1 Cat 14H Grader
- 1 Komatsu HD465 watercart.

Other shop and support equipment for which costs have bee estimated includes:

- Utility lift truck
- Fuel and lubrication truck
- Mechanical field truck
- Crew bus
- Light vehicles
- Dewatering pump system
- Lighting plants
- Mobile radio units
- Surveying equipment

- Mining hardware and software.

Operating parameters

The operating parameters used for estimating the project costs are detailed in Appendix C. The main operational aspects considered are:

- Scheduled hours available
- Non-productive time
- Material types
- Equipment availability
- Use of equipment availability
- Grade control requirements
- Drill productivity
- Blast hole drilling requirements
- Blasting
- Loading performance
- Haul cycle times
- Shop and support fleet
- Technical and managerial staff

15.1.2 Operating cost inputs

Operating cost estimate inputs have been sourced from:

- Supplier estimates acquired for this study
- Information provided by AQI
- Snowden's database of cost estimates used in previous studies

Appendix C details the cost model inputs and cost estimate construction which is generated from activity based production inputs including:

- Equipment available hours
- Equipment units required
- Production rates for all equipment
- Maintenance cost and consumable consumption rates for equipment
- Explosives consumption rates
- Labour utilisation
- Assay sampling costs

The summary operating costs by operational activity are shown in Table 15.1. Where appropriate, contributory costs have been distributed over more than one operational activity.

The total unit cost is $1.45 for each tonne mined and averages $11.95 for each high grade production tonne mined.

Table 15.1 Mining operating costs

Item	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Grade Control	$'000s	942	999	880	866	405	4,091
Production Drilling	$'000s	885	1,101	1,038	1,146	509	4,678
Blasting	$'000s	1,389	1,577	1,279	1,305	682	6,232
Load & Haul	$'000s	2,532	2,918	2,607	2,699	1,481	12,237
Ancillary Equipment	$'000s	1,236	1,258	1,192	1,245	953	5,883
Mine Services	$'000s	1,119	1,119	1,178	1,178	935	5,528
Mine Supervision	$'000s	641	641	641	641	633	3,197
Total operating cost	$'000s	8,743	9,612	8,815	9,080	5,597	41,847
Total unit operating cost	$/dmt	1.32	1.21	1.49	1.40	2.85	1.45

15.1.3 Capital costs

Capital cost estimates have been sourced from:

- Supplier estimates acquired for this study
- Snowden's database of cost estimates used in other recent studies

Capital requirements

Updated equipment capacities, productivities and costs have been matched to the detailed mine design to validate the equipment selections described in section 10.1. Iterations of fleet make-up and the mining schedule were considered to develop the "best balanced" fleet in accordance with Snowden's experience.

Capital expenditure for main equipment is $11,624,020 in the first year. Due to the short project life, no replacement of main fleet components is required in following years.

A further $1,047,939 is required in the first project year for supporting equipment. An additional $278,875 is provided in the second year for expenditure for life of mine in-pit roads. There is no requirement for future replacement purchases of main or supporting equipment due to the short mine life.

These costs are exclusive of training, but include contingencies.

Details of the capital expenditure requirements are shown in Table 15.2.

Table 15.2Mining capital expenditure

	Unit cost	Number required					Total
		Year 1	Year 2	Year 3	Year 4	Year 5	
TAMROCK PANTERA 1500 Drill	$698,691	2	-	-	-	-	2
O&K RH90 C Excavator	$2,346,488	1	-	-	-	-	1
Cat 777F Haul Truck	$1,493,857	3	-	-	-	-	3
Cat D9T Dozer	$926,568	1	-	-	-	-	1
Cat 14H Grader	$524,047	1	-	-	-	-	1
Komatsu HD465 Water Truck	$790,000	1	-	-	-	-	1
Cat 980 ROM Loader	$500,000	1	-	-	-	-	1
Support equipment	$1,047,939	1					1
Infrastructure - roads	$278,875		1				1
Total Cost (includes 6% escalation on major items)		$12,671,959	$278,875	$0	$0	$0	$12,950,834

15.2 Mining Schedule

The mining schedule was created using Mine Works Planner software utilising the block model reported by the pit designs, by cut-off grade categorised material (primary ore, secondary ore, and waste materials), individual pits and pit stages, and in 20 m bench increments. These material units were scheduled by the following criteria:

- Replicate the optimal economic schedule as generated in Whittle
- Mine as a priority higher grade regions with the least stripping ratios first
- Simulate practical open pit mining practices
- Manage production rates to ensure the mill feed is consistently supplied with the nominated 750 ktpa of material
- Maintain realistic production rates that are appropriate for the selected mining fleet.

Table 15.3 summaries the produced mining schedule by year. The emphasis at the start of mining is on the Vein 49 Stage 2 with some production drawn from Vein 49 Stage 1. The key push back for the Vein 49 Stage 3 is scheduled to start in the middle of the third year of mining, using Vein 49 Stage 1 as the primary ore source during the push back

period. The smaller Nelson East and West pits are scheduled for full extraction during the fourth year. The Vein 49 Stage 3 is schedule to finish late in the fifth year. The detailed mining schedule is located in Appendix D.

Table 15.3 Mining Schedule for Life of Mine

		Year 1	Year 2	Year 3	Year 4	Year 5	Total
Primary production	bcm	306,065	307,774	304,562	302,801	192,762	1,413,964
Primary production	t	750,277	751,540	751,299	752,881	494,423	3,500,420
Au Grade	g/t	4.33	4.43	4.23	2.98	3.08	3.86
Ag Grade	g/t	25.92	35.03	44.72	26.33	34.59	33.22
Low grade production	bcm	137,368	80,466	135,814	136,288	63,165	553,101
Low grade production	t	338,244	195,843	335,961	339,361	161,982	1,371,391
Au Grade	g/t	0.63	0.62	0.63	0.66	0.65	0.64
Ag Grade	g/t	8.27	11.81	9.6	7.81	10.01	9.19
Waste	bcm	2,227,610	2,797,662	1,933,149	2,160,922	520,301	9,639,644
Waste	t	5,551,060	6,977,301	4,816,453	5,398,687	1,309,518	24,053,019
Total	bcm	2,671,042	3,185,901	2,373,525	2,600,011	776,228	11,606,707
Total	t	6,639,580	7,924,684	5,903,714	6,490,928	1,965,923	28,924,829

15.3 Primary ore material type schedule

Table 15.4 details the ore material types scheduled for delivery to processing.

Table 15.4 Scheduled primary ore catagorised by material type

Primary Ore Material		Year 1	Year 2	Year 3	Year 4	Year 5	Total
Split by Material type							
Quartz Oxide	T	551,262	166,316	31,282	255,231	-	1,004,091
	bcm	224,869	67,899	12,669	102,559	-	407,996
Quartz Fresh	T	44,245	485,816	557,931	333,658	419,437	1,841,087
	bcm	18,035	199,030	226,162	134,274	163,542	741,043
Andesite Oxide	T	141,473	22,398	17,316	75,995	-	257,182
	bcm	57,770	9,179	7,017	30,515	-	104,481
Andesite Fresh	T	13,297	77,011	144,769	87,998	74,985	398,060
	bcm	5,392	31,667	58,716	35,464	29,220	160,459
Total primary ore	T	750,277	751,541	751,298	752,882	494,422	3,500,420
	bcm	306,066	307,775	304,564	302,812	192,762	1,413,979

15.4 Dump construction schedule

The dump construction schedule is based on the various waste material types and the final dump destination for each. The schedule criteria were:

- Minimise haul distance from source to dump
- Utilise completed pits
- Return pyritic waste to pits.

The resultant dump construction schedule is shown in Table 15.5.

Table 15.5 Dump construction schedule (BCM)

Material source	Destination	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Low grade	Low grade	135,216	81,117	117,173	155,260	64,338	553,104
Non-pyritic waste	Vein 49 dump	2,225,627	2,797,635	1,800,866	511,853	0	7,335,981
	Nelson dump	0	0	0	961,021	0	961,021
	Nelson pits	0	0	0	481,750	277,383	759,133
Medium pyritic waste	Vein 49 dump	1,983	27	132,283	68,520	0	202,813
	Nelson dump	0	0	0	14,677	0	14,677
	Nelson pits	0	0	0	0	109,248	109,248
High pyritic waste	Vein 49 dump	0	0	0	0	0	0
	Nelson dump	0	0	0	0	0	0
	Nelson pits	0	0	0	123,101	133,670	256,771
Total	Low grade	135,216	81,117	117,173	155,260	64,338	553,104
	Vein 49 dump	2,227,610	2,797,662	1,933,149	580,373	0	7,538,794
	Nelson dump	0	0	0	975,698	0	975,698
	Nelson pits	0	0	0	604,851	520,301	1,125,152
Total	All dumps	2,362,826	2,878,779	2,050,322	2,316,182	584,639	10,192,748

CALCATREU GOLD PROJECT

Initial Feasibility Study
Volume 4: Processing

5 April 2007

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Executive Summary

Introduction

Aquiline Resources Incorporated (AQI) is evaluating the development of the Calcatreu Gold Project (Calcatreu) located in north central Patagonia, Argentina. Calcatreu is approximately 165 km southeast of San Carlos de Bariloche and about 1,300 km southwest of Buenos Aires. The project area straddles the southern boundary of Rio Negro Province and the northern boundary of Chubut Province. The mining and processing plant would be located in the Rio Negro Province.

Calcatreu was previously explored by Normandy Mining of Australia (Normandy) through its Argentine subsidiary Minera Normandy Argentina and later by Newmont Mining Corporation (Newmont), having purchased Normandy. Newmont subsequently sold Calcatreu to AQI during 2003 as the project did not meet Newmont's corporate size objectives.

In March 2005, AQI commissioned a Feasibility Study (FS) to evaluate the technical and commercial viability of the project. In February 2006, AQI reduced the scope of the study to an Initial Feasibility Study (IFS). The accuracy of the capital and operating cost estimates for the IFS are set at ± 25%.

The project entails mining ore containing gold and silver from an open pit mine at a nominal rate of 750,000 tonnes per year and treating it in a conventional crushing, grinding, and carbon-in-leach (CIL) process plant to produce gold/silver doré. The operational life of the project is estimated to be 4.7 years.

The preparation of the IFS has been managed by Ausenco Services Pty. Ltd. (Ausenco) from its Perth office in Western Australia.

Sub-consultant Vector Engineering (Vector), specialising in landfill design and closure, has completed the design of the tailings storage facility (TSF). The cost for construction of the TSF has been estimated by Ausenco using quantities provided by Vector, and Argentinean labour, fabrication and construction rates developed by sub-consultant engineering firm INDEC.

Ausenco developed the metallurgical testwork programme for the IFS. AMMTEC Pty. Ltd. (AMMTEC) of Perth, Western Australia, conducted the testwork under the supervision of Ausenco. The results of this testwork have been used to develop the process flowsheets, engineering design, and cost estimates for extracting gold and silver from the Calcatreu ore.

In light of the high concentration of silver relative to gold and the subsequent affect upon the carbon inventory in a CIL circuit, the IFS included a conceptual assessment of alternative gold and silver extraction processes. This took the form of a trade-off study comparing CIL with Merrill Crowe technology and a hybrid circuit, combining aspects from both technologies. Based on the short life of the project, potentially higher silver recovery and lower capital cost, the CIL process route is favoured, and this forms the basis of the IFS capital and operating cost estimate. However, this option entails a higher operating cost than the Merrill Crowe option and any change to the design criteria may impact on the selection.

The IFS also included a conceptual assessment of gold and silver recovery using heap leach extractive technology as an alternative process route for treating the Calcatreu ore.

The accuracy of capital and operating cost estimates for this study is considered to be ± 40%.

The IFS, at the request of AQI, also included a conceptual assessment of a non-cyanide gold and silver recovery process. This process incorporated gravity concentration followed by flotation of the tailing from the gravity circuit. Whilst this process route eliminates the use of cyanide, the recoveries of gold and silver were significantly lower than those achieved using a cyanide leach. The conceptual process flowsheet was developed based on testwork done at AMMTEC and Gekko Systems Pty Ltd (Gekko). The accuracy of capital and operating cost estimates for this study is considered to be ± 40%.

The Province of Rio Negro has imposed restrictions on the use of cyanide in mining operations and AQI will define the final ore treatment method at a later date, after negotiations with the Rio Negro government.

Project related areas such as geotechnical studies, resource and reserve estimations, mining schedule, water management, environmental and social impact assessment, and land acquisition are excluded from the Ausenco scope of work. These studies have been performed by independent consultants acting on behalf of, and managed by AQI. This work by others is reported separately.

Metallurgy

A comprehensive metallurgical testwork programme has been undertaken at AMMTEC using drill core samples selected by AQI. The testwork demonstrated that the ore is “free milling” and that high gold and silver recoveries can be achieved using a conventional grinding and cyanidation process. For the purpose of designing and costing the process plant, appropriate design criteria, recoveries and reagent consumption rates have been developed from the testwork results obtained.

Grade versus leach residue relationships were developed from the results of the testwork. These relationships equate to a gold recovery of 90% at the head grade of 3.86 g/t and a silver recovery of 74% at a head grade of 33.2 g/t.

Cyanide detoxification tests were conducted on samples of leach residue tails. The results show that the cyanide level in the leach residue tails can be readily detoxified to meet the International Cyanide Management Code (ICMC) standards. The ICMC standard for the discharge of tailings into a TSF is 50 mg/L weak acid dissociable cyanide (CNWAD), which is deemed safe for livestock and bird life.

Processing

Ausenco has developed a process flowsheet based upon the AMMTEC testwork results and a metal extraction case study. The run-of-mine (ROM) ore is crushed in a jaw crusher, and then ground in a SAG and ball mill circuit. The ground ore then reports to a CIL circuit consisting of eight tanks in series. Gold and silver values absorbed onto the activated carbon are recovered in the carbon elution circuit. The resultant concentrated strip liquor (“pregnant” eluate) is treated with zinc dust to precipitate gold and silver. The precipitate is subsequently smelted into doré bars.

The tailings from the CIL circuit are thickened, treated in a cyanide destruction circuit and discharged to the TSF. The cyanide concentration, after cyanide destruction, should be 50 mg/L or less.

Allowing for maintenance downtime, the process plant ore throughput capacity will be 750,000 tonnes per year, over an estimated project life of 4.7 years.

The process plant will be located approximately 600 metres southeast of the Vein 49 open pit while the TSF will be located in a lacustrine basin area, about 700 metres southeast of the process plant. The grinding area and gold room will be housed within a building for wind weather protection. This building will incorporate plant maintenance facilities, as well as plant offices, warehouse, ablution and change room facilities.

Infrastructure and Services

The Calcatreu site has no existing infrastructure except for an unsealed road between Ingeniero Jacobacci (Jacobacci) and Paso del Sapo towns, which passes close to the project site. The development of all necessary infrastructure to support the mine and plant operations is therefore required. On-site camp accommodation will be provided for the personnel, who would generally reside in Jacobacci.

No hydrogeological assessment has been undertaken in the area as part of this study. Ausenco has made an allowance for piping water to the plant.

Electrical power shall be provided by an owner-operated, power station using high-speed diesel generator sets. The decision to use site generate power was made on the basis that the Argentine grid supply is too far away from the project area and the costs to deliver power by overhead line would be prohibitive.

There will be a microwave link to Jacobacci for mobile phones and radios would be used in the mine operations with a repeater station on the road to Jacobacci.

The site buildings will comprise process plant, mine workshop, reagent storage, core storage and laboratory. These buildings will be of steel frame construction with galvanised cold-rolled sections for purlins and grits, zincalume or colorbond steel cladding and roofing, with concrete and bitumen floors.

Process Plant Operations and Administration

Personnel will be recruited locally to fill the majority of the available positions. Given the requisite skills, or the ability to quickly acquire such skills, Argentinean nationals living in the Río Negro province would be ideally placed to fill the available positions.

Those positions requiring experience in gold plant and related operations will be filled by nationals with the suitable experience, in preference to expatriate labour. Expatriate positions will account for a very small, but essential, proportion of the total personnel employed.

Operating Cost Estimates

The operating cost estimates are presented in United States dollars (USD) and use prices obtained in the second quarter of 2006 (2Q06). All references to dollars or $ are to USD. In broad terms the estimate includes all site related operating costs associated with processing ore to produce gold/silver doré bars. Details of operating cost estimates are given in Section 5.

The overall accuracy of the project operating cost estimate is considered to be ±25%.

The estimate excludes doré shipping, insurance and refining costs, mining, escalation, accuracy provisions, corporate overhead charges (such as legal, banking and insurances), financing costs, royalties, income taxes or similar imposts as well as expenditure classified as capital, sustaining capital, or rehabilitation and closure costs.

Operating cost estimates were prepared by Ausenco with input from AQI and other consultants for the various components of the Project. Major contributions to the estimate were made by:

- AQI participated in advice about and enquiries to local suppliers and advised on labour rates, labour loadings and manning schedules and security needs.
- Ausenco provided the operating cost components associated with the process plant.

Ausenco has combined these separate inputs to create an operating cost estimate for the processing plant and administration.

The average annual costs for processing Calcatreu ore at a rate of 750,000 tonnes per year over the life of the mine are summarised in Table 1.

Table 1 Operating Cost Summary

	Average Annual Costs USD M/Year	Unit Cost USD/t ore treated
Processing	11.16	14.89
Maintenance	1.30	1.74
General & Administration	2.19	2.92
Total	**14.65**	**19.55**

Capital Cost Estimates

The project initial capital cost estimate for the CIL option is presented in USD and has a base date of the second quarter 2006 (2Q06). All references to dollars or $ are to USD.

The overall accuracy of the project initial capital cost estimate is considered to be ±25%.

In broad terms, the estimate includes design and construction of the plant access road, power supply, mining infrastructure, process plant, TSF, water supply, on- and off-site infrastructure costs. Details of the initial capital cost estimate are given in Section 6 and the estimate is summarised in Table 2.

Table 2 Capital Cost Summary

Cost Centre	Estimated Cost (USD M)
Process Plant	26.6
On Site Infrastructure and Utilities	10.4
Off Site Infrastructure and Utilities	8.1
Mining Infrastructure, Haul Roads and ROM Pad Construction	2.8
Mobile Equipment, First Fill Consumables and Capital Spares	3.9
Indirects Temporary Construction Facilities, EPCM, Start-Up and Commissioning	14.4
Total Capital Cost	66.2

The initial capital cost estimates exclude escalation, duties, taxes, mining costs, working capital, Owner's costs, sustaining capital, financing costs, rehabilitation and closure costs and allowance for project growth.

Alternative Process Routes

In parallel with the IFS based on agitated cyanidation process, Ausenco has conducted two concept studies, at the request of AQI, covering two alternative process routes.

These alternative process options are:

- Gravity concentration followed by flotation to recover gold/silver to a concentrate for sale.
- Heap leach.

The findings of the conceptual studies covering these options are described in Section 8.

A summary showing operating cost comparison for these two options is given in Table 3.

These conceptual study operating cost estimates were done at ±40% accuracy and are presented in USD as at the 2Q06. Details of operating costs for the two conceptual options are given in Section 8.

Table 3 Summary of Operating Cost for Conceptual Studies

Average Annual Costs USD M/Year		Unit Cost USD/t ore treated
Gravity/Flotation Option	11.32	14.89
Heap leach Option	8.67	11.56

These estimates exclude doré shipping, insurance and refining costs, mining, escalation, accuracy provisions, corporate overhead charges, financing costs, royalties, income taxes or similar imposts as well as expenditures classified as capital, sustaining capital, or rehabilitation and closure costs.

A summary of initial capital cost estimates for the two conceptual studies considering heap leach and a non-cyanide recovery process is given in Table 4.

These estimates are presented in USD as at the 2Q06 and are considered to have an accuracy level of ±40%. Details of initial capital cost estimates for the two conceptual options are given in Section 8.

Table 4 Summary of Capital Cost for Conceptual Studies

		Gravity/Flotation Option	Heap leach Option
Project Capital Cost	USD M	60.5	56.1
Deferred capital	USD M	n/a	17.0

The initial capital cost estimates for all the options exclude escalation, duties, taxes, mining costs, working capital, owner's costs, sustaining capital, financing costs, rehabilitation and closure costs and allowance for project growth.

Conclusions and Recommendations

The viability of using a conventional CIL cyanidation process for the recovery of gold and silver from the Calcatreu ore has been demonstrated by a detailed testwork programme. The proposed circuit would be designed according to World's Best Practice, and comply with the guidelines stipulated in the ICMC.

During the next phase of the FS, AQI should obtain environmental and mine permitting licenses, and conduct hydrological studies to determine the source and quality of the raw water for the proposed processing facility.

Preliminary testwork results from alternative gravity/flotation and heap leaching flowsheets indicate significantly lower gold and silver recoveries from both options compared to agitated cyanidation. The testwork for these alternative flowsheets, however, was not sufficiently comprehensive to generate capital and operating cost estimates to the same accuracy as the base case CIL circuit. The capital and operating cost estimates derived for the alternate options are to an accuracy of ±40% and are intended to provide AQI with a tool on which to base possible future flowsheet development.

One of the predominant reasons for investigating the viability of a gravity/flotation circuit option is the absence of cyanide within this processing route. Currently the local Rio Negro government has banned the use of cyanide in mining operations within its jurisdiction. It is, therefore, imperative to establish whether this decision is likely to be reversed before progressing with the next Project phase.

Assuming the use of cyanide is sanctioned, there is need to review CIL versus Merrill Crowe with updated design criteria. If CIL is the preferred option, further testwork would be required to confirm the cyanide consumption in the leaching process and silver extraction. As the settling characteristics of many of the samples tested to date were poor, additional thickening testwork would be needed to provide greater definition for equipment sizing.

If, on the other hand, the "no-cyanide" policy is upheld, the alternative gravity/flotation circuit option will require further development. This would involve a comprehensive testwork programme to establish design criteria on which to base equipment sizing, metal recoveries, and reagent consumptions for accurate estimation of the capital and operating costs. In addition, the source of the samples sent to San Juan for gravity/flotation testwork would, firstly, need to be confirmed and then the reason(s) established for the higher gold

and silver recoveries than those obtained by Gekko and AMMTEC. The flotation circuit would require further development by conducting re-grinding, reagent and pH screening tests and, ultimately, locked cycle flotation tests. In addition to the above testwork on the gravity/flotation circuit, AQI must establish whether a low grade gold/silver flotation concentrate could be sold directly to a third party for downstream processing.

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1 Introduction

The Calcatreu Gold Project (Calcatreu) is owned by Minera Aquiline Argentina S.A., a fully owned subsidiary of Aquiline Resources Inc. (AQI). The project consists of a series of epithermal gold and silver rich vein systems which encompass the Vein 49 and Nelson vein systems, with approximately 3.36 Mt of ore grading 3.86 g/t of gold and 33.2 g/t of silver. The Calcatreu Project area covers more than 73,000 hectares, straddling the southern border of Rio Negro Province and the northern border of Chubut Province (Figure 1.1). The mine and processing plant will be located in the Province of Rio Negro.

Calcatreu was discovered by La Source Developpement Argentine in 1997 and subsequently acquired by Normandy Mining of Australia (Normandy), which explored the project through its Argentinean subsidiary Normandy S.A. Newmont Mining Corporation (Newmont) acquired Normandy and sold its interest in the Calcatreu Project to AQI, which gained control of the tenements in June 2003.

Since acquisition, AQI has continued the exploration programme in the area and engaged Micon International Limited (Micon) to provide a preliminary assessment and economic evaluation for the Calcatreu Project. A report outlining Micon's work was finalised in October 2004 and a copy of the report is available on the AQI website.

1.1 Location and Access

Calcatreu comprises a set of mining properties in the northern part of the Somuncura volcanic massif located in south-western Argentina, approximately 1,300 km southwest of Buenos Aires. The project exploration and mining tenement area straddles the Rio Negro and Chubut Province boundaries. The Vein 49 and Nelson vein system are located in the Rio Negro Province and it is proposed that the processing plant be constructed adjacent to these orebodies.

Major regional centres close to Calcatreu are San Carlos de Bariloche (Bariloche); which is about 160 km northwest (about 330 km by road), the city of Neuquén, about 290 km north, and the provincial capital of Rio Negro, Viedma, about 600 km to the east. The closest town is Ingeniero Jacobacci (Jacobacci), about 55 km to the north (Figure 1.1).

Jacobacci has about 6,500 inhabitants and is used by AQI as the logistical support base for the project. The town has a rural hospital, an airstrip, two hotels and a railway station. The town is crossed by the railway line between the tourist city of Bariloche and Viedma.

Jacobacci is accessed by unsealed, but well maintained provincial roads from Bariloche (about 3 hours drive), by train from Buenos Aires or Bariloche, or by light plane from Viedma or Bariloche. Both Viedma and Bariloche are serviced by commercial flights from Buenos Aires. Bariloche has an international airport and may be reached from Chile.

Provincial Road No. 76 transects the project area from north to south, linking the town of Jacobacci in Rio Negro with Paso del Sapo in Chubut. This road is wide and unsealed, but well maintained. Access around the project area is via a network of secondary public roads and farm tracks connecting with No. 76.

Figure 1.1 Location map, Calcatreu Project Area

1.2 Topography, Climate and Vegetation

The mine area is located on gentle hill slopes with typical rock exposures along major ridgelines and peaks, with elevation varying between 1,200 m and 1,500 m above sea level (ASL).

No defined surface water drainage geomorphic structures are formed in the mine area, with the exception of the small intermittent lagoons, which occur in closed topographical low areas. The valleys within the mine areas ultimately drain to the northwest of the site. Several areas of these valleys accumulate water and ponds are formed during the wetter months, although flows are low and slow moving, with no soil movement. Soil coverage is limited on the peaks and high relief slopes, within the mine area.

Shallow soil development occurs in areas where the slopes are gentle, typically less than a metre. In the valleys and in low topographical areas the soil development can occur to several metres.

The climate in this part of Patagonia is cold and dominated by strong westerly winds, high evaporation rates and scarce rainfall, with more precipitation in the winter months than in summer. Minor sporadic snowfall may occur in the coldest months of winter. The closest meteorological station is in the town of Maquinchao (888 m ASL), about 90 km to the

northeast, and it could be considered representative of the project area. The data measured is:

Extremes: Observed Maximum Temperature = 37°C (January)

Observed Minimum Temperature = - 25.6°C (June).

1961 to 1993 data average for day time highs:

Annual Temperature = 9.3°C

Maximum Temperature = 16.4°C

Minimum Temperature = 2.2°C

Principal Wind Direction = From NW/W to SE/E

Average Wind Velocity = 20 km/h

Maximum Wind Velocity = >100 km/h (spring)

Annual Rainfall/Snowfall = 200 mm.

AQI has set up a project weather station (1,343 m ASL), and although a complete data set is not yet available, temperatures recorded between December 2004 and March 2006 are in the range of the temperatures indicated above. However, the prevailing wind direction in the project area is from the southwest during summer, and for the rest of the year is from the northwest as in the case of Maquinchao. The wind speed is higher than in Maquinchao with average values of 29 km per hour and gusts of up to 125 km per hour have been recorded for an average period of 15 minutes.

Low rainfall, low relative humidity and strong winds produce a desert to a semi-desert like shrub steppe and shrub and grass steppe landscapes. The poor fertility of the soils, coupled with ashes blown from recent eruptions of volcanoes located in Chile to the west of the project area, makes the use of the local land confined to raising sheep and a few horses and cattle. There are no national parks or protected reserve areas within or near the project area.

1.3 Study Scope

In March 2005, AQI commissioned Ausenco Services Pty. Ltd. (Ausenco) to prepare a Feasibility Study (FS) to evaluate the technical and commercial viability of Calcatreu. As part of the scope, Ausenco designed a comprehensive metallurgical testwork programme that was carried out during 2005 at the AMMTEC laboratories, in Perth, Western Australia.

In February 2006, AQI reduced the scope of the study to an Initial Feasibility Study (IFS) primarily because of changes in the regulations regarding the use of cyanide in Rio Negro Province. The IFS includes capital and operating cost estimates produced to an accuracy of plus/minus twenty five per cent (±25%) for a process flowsheet with extraction of gold and silver by leaching in cyanide.

A technical and financial evaluation of three circuit types using cyanide for the recovery of precious metal from ore was performed by Ausenco in order to select the most appropriate for Calcatreu. The case study is included in Appendix 1.

The IFS also includes a conceptual assessment of recovery of gold and silver, by gravity concentration plus flotation, and by heap leaching. It must be noted that insufficient testwork has been performed at this time to develop either gravity-flotation or heap leach

flowsheets to an IFS standard; and it was not possible, therefore, to develop optimised flowsheets. However, conceptual flowsheets were developed for both process routes based on Ausenco's experience, and capital and operating costs were subsequently compiled to an accuracy of ± 40%. There is insufficient information available to predict, to any reasonable accuracy, the recovery of gold and silver by either gravity/flotation or heap leaching. It is envisaged that the conceptual assessment of these process routes will provide a tool for AQI to assess the value of developing either of these flowsheets further.

Ausenco's scope of services is as follows:

- Metallurgy:
  - Preparation of metallurgical testwork programme.
  - Metallurgical testwork, supervision and reporting.
- Process engineering:
  - Preparation of a case study to assess processing by CIL versus Merrill Crowe and make recommendations to AQI on the process route on which to base the IFS.
  - The development of preliminary design criteria and preparation of preliminary flow diagrams for the agitated leach flowsheet.
  - The development of conceptual design criteria, preparation of conceptual block flow diagram and conceptual equipment list for the gravity/flotation flowsheet.
  - The development of conceptual design criteria, preparation of conceptual block flow diagram and conceptual equipment list for the heap leach flowsheet.
- Engineering for agitated leach flowsheet
  - Preparation of a preliminary mechanical equipment list from the process flow diagrams.
  - Preparation of site layout drawings with a plant general arrangement plan and two general arrangement sections.
  - Preparation of preliminary earthworks drawings in sufficient detail to provide approximate quantities for estimations.
- Capital cost estimates:
  - Development of capital cost estimates in US dollars to an overall accuracy of ± 25% for the agitated leach flowsheet.
  - Development of capital cost estimates in US dollars to an overall accuracy of ± 40% for the conceptual heap leach and gravity plus flotation flowsheet.
- Operating cost estimates:
  - Development of operating cost estimates in US dollars to an overall accuracy of ± 25% for the agitated leach flowsheet.
  - Development of operating cost estimates in US dollars to an overall accuracy of ± 40% for the conceptual heap leach and gravity plus flotation flowsheet.

1.4 Exclusions

The following are excluded from the scope of work:

- Resource and reserve estimations.
- Mining schedule and planning.
- Tailings storage facilities (TSF).
- Waste stockpiles.
- Hydrological studies.
- Environmental compliance.
- Financial analysis.
- Legal and regulatory.
- Permitting.

1.5 Battery Limits

The battery limits for the Calcatreu IFS are as follows:

- Ore – discharge of truck or front end loader at ROM dump station.
- Product – production of gold/silver doré bars in the gold room.
- Power supply – includes the use of diesel generators owned by AQI.
- Raw water (plant area) – intake to plant.
- Tailings – discharge at the TSF.

1.6 Currency and Quantities

All currency amounts are stated in United States dollars (USD). Quantities are expressed in SI units, according to Australian and international practice, including metric tonnes (tonnes, t), kilograms (kg) and grams (g) for weight, kilometres (km) or metres (m) for distance, hectares (ha) or square metres (m2) for area and grams per metric tonne (g/t) for gold and silver values (Au g/t, Ag g/t).

1.7 Disclaimer

Site visits were made by an Ausenco representative in both May 2005 and July 2005. During the course of these visits the following activities occurred:

- AQI presented a selection of core and described the various types of material.
- The requirements for the samples to be used in the metallurgical testwork programme were discussed with AQI geological personnel.
- The locations were agreed with AQI personnel for plant and infrastructure items.

This report and its conclusions are based on material submitted by the professional staff of AQI or its consultants, a metallurgical testwork programme carried out in AMMTEC laboratories in Australia, which was supervised by Ausenco, and metallurgical testwork undertaken in Chile and Argentina that was not supervised by Ausenco.

The metallurgical plant design is based on the results of the metallurgical testwork programme, and geological information and mining production schedules supplied by AQI and its consultants. Capital and operating costs have been developed by Ausenco utilising quotations from major equipment suppliers, South American engineering firms, Ausenco's data base and experience.

The various agreements under which AQI holds title to the mineral lands for this project have not been investigated or confirmed by Ausenco and Ausenco offers no opinion as to the validity of the titled claimed.

While exercising all reasonable diligence in supervising the metallurgical testwork, Ausenco has relied upon data presented by AQI and its consultants in formulating the opinions in this report.

2 METALLURGY

2.1 Introduction

A testwork programme was developed with the specific objectives to verify technical feasibility and provide data for the preparation of the IFS. The programme is outlined in Appendix 2. This work was awarded to AMMTEC in Perth, Western Australia and commenced in 2005. The programme included testwork to establish:

- Optimum leach conditions for both oxidised vein and unoxidised vein composite samples.
- Potential for recovery of gold and silver using a flotation process.
- Engineering data, including testing of oxygen uptake, carbon kinetics and slurry viscosity properties of the oxidised vein and unoxidised vein composite samples.
- Performance of the SO2/air process for cyanide detoxification.
- Comminution characteristics of oxidised vein, unoxidised vein, oxidised mineralised andesite and unoxidised mineralised andesite samples.
- Work indices and leach performance of 14 variability samples.
- Leach performance of 24 variability samples.
- Heap leaching performance of oxidised mineralised andesite and unoxidised mineralised andesite samples.

In addition, AMMTEC prepared samples for dispatch to:

- Gekko Systems Pty Ltd (Gekko) for gravity plus flotation testing.
- Outokumpu Technology Pty Ltd (Outokumpu) for thickening testwork on the oxidised vein and unoxidised vein composite samples and on composite samples of oxidised mineralised andesite and unoxidised mineralised andesite.
- Delkor Pty Ltd (Delkor) for filtration testwork on the oxidised vein and unoxidised vein composite samples.
- Coffey Geosciences Pty Ltd (Coffey) to establish physical and geochemical characteristics of plant tailings.
- TUNRA Bulk Solids Handling Research Associates (TUNRA), to establish materials handling characteristics of oxidised mineralised andesite.

AMMTEC's work is reported in a document titled, "Metallurgical Testwork Programme on Samples from Calcatreu, Argentina for Aquiline Resources, Report No. A9876, April 2006."

Gravity concentration and flotation testwork is described in Gekko's report, "Aquiline Resources Gravity and Flotation Recovery Testwork," dated 7th February 2006.

Thickening testwork is discussed in Outokumpu's reports S748TA and S748TA_B and the filtration testwork in Delkor's report dated 1st February 2006.

TUNRA's report "Flow Properties of Calcatreu Ore," dated October 2005, contains its findings on the material handling characteristics of the ores.

Further samples of oxidised vein and unoxidised vein were collected and dispatched to Universidad Nacional de San Juan (UNSJ) for gravity plus flotation testing. This work is reported in a document titled, "Ensayos Metalúrgicos de Muestras Auríferas del Proyecto Calcatreu."

Copies of the above reports are in Appendix 2.

The testing of plant tailings conducted by Coffey, was supervised by Vector, and is discussed in a separate report by Vector (Appendix 10).

2.2 Previous Testwork

Limited metallurgical testwork was undertaken in 2004 at SGS Lakefield Research, Chile

S.A. (SGS). Results from this work are contained in a report titled "Metallurgical testing for Calcatreu Gold Deposit," dated May 2004. The main objective of this programme was to assess the amenability of the samples to gold and silver recovery through gravity concentration and cyanidation.

In addition, SGS performed bottle roll tests on crushed samples to investigate the potential for heap leaching Calcatreu ores. While no formal report has been issued on these tests, preliminary results sheets have been made available (attached in Appendix 2). The results of this work were considered when developing the programme for this phase of the study.

2.3 Samples for AMMTEC (2005/6)

Various samples were obtained specifically for the different elements of the testwork programme. Ausenco identified the requirements for each sample and AQI geologists selected the individual components to match Ausenco's requirements. Ausenco was not involved in the selection and composition of the metallurgical samples. The detailed composition of each sample is listed in Appendix 2.

- Leach Optimisation Samples
  - Approximately 200 kg of oxidised vein of average grade and typical mineralisation, including 15% dilution obtained equally from the hanging and foot walls, was composited. These samples were taken from existing split core or whole core freshly drilled for the metallurgical samples. This oxidised vein sample was all from Vein 49 and was designated Cal-1.
  - The unoxidised vein sample (Cal-2), also entirely from Vein 49, was composited in a similar fashion.
- Variability Leach Samples
  - Approximately 6 kg of existing split core was used for each of the 24 samples, in accordance with Table 2.1.

Table 2.1 Variability leach samples

Sample No.	Classification	Target Grade, g/t	Deposit
Cal-3	Oxidised vein	±1.5	Vein 49
Cal-4	Oxidised vein	±1.5	Vein 49
Cal-5	Oxidised vein	±3.0	Vein 49
Cal-6	Oxidised vein	±3.0	Vein 49
Cal-7	Oxidised vein	±4.5	Vein 49
Cal-8	Oxidised vein	±4.5	Vein 49
Cal-9	Oxidised vein	6.0 – 8.0	Vein 49
Cal-10	Oxidised vein	6.0 – 8.0	Vein 49
Cal-11	Unoxidised vein	±1.5	Vein 49
Cal-12	Unoxidised vein	±1.5	Nelson
Cal-13	Unoxidised vein	±3.0	Vein 49
Cal-14	Unoxidised vein	±3.0	Nelson
Cal-15	Unoxidised vein	±4.5	Vein 49
Cal-16	Unoxidised vein	±4.5	Vein 49
Cal-17	Unoxidised vein	6.0 – 8.0	Vein 49
Cal-18	Unoxidised vein	6.0 – 8.0	Vein 49
Cal-19	Oxidised mineralised andesite	±0.9	Vein 49
Cal-20	Oxidised mineralised andesite	±1.2	Vein 49
Cal-21	Oxidised mineralised andesite	±1.5	Nelson
Cal-22	Oxidised mineralised andesite	±1.5	Vein 49
Cal-23	Unoxidised mineralised andesite	±0.9	Vein 49
Cal-24	Unoxidised mineralised andesite	±1.2	Vein 49
Cal-25	Unoxidised mineralised andesite	±1.5	Nelson
Cal-26	Unoxidised mineralised andesite	±1.5	Vein 49

- Variability Leach and Work Index Samples
  - Approximately 50 kg of core was used for each of the 14 samples, in accordance with Table 2.2. These samples comprised freshly drilled whole core, except Cal-37 and Cal-38, which contained a mixture of existing split and fresh whole core.

Table 2.2 Variability leach samples

Sample No.	Classification	Target Grade, g/t	Deposit
Cal-27	Oxidised vein	±1.5	Vein 49
Cal-28	Oxidised vein	±3.0	Vein 49
Cal-29	Oxidised vein	±4.5	Vein 49
Cal-30	Oxidised vein	6.0 – 8.0	Vein 49
Cal-31	Unoxidised vein	±1.5	Nelson
Cal-32	Unoxidised vein	±3.0	Vein 49
Cal-33	Unoxidised vein	±4.5	Vein 49
Cal-34	Unoxidised vein	6.0 – 8.0	Vein 49
Cal-35	Oxidised mineralised andesite	±0.9	Vein 49
Cal-36	Oxidised mineralised andesite	±1.2	Vein 49
Cal-37	Oxidised mineralised andesite	±1.5	Vein 49
Cal-38	Unoxidised mineralised andesite	±0.9	Vein 49
Cal-39	Unoxidised mineralised andesite	±1.2	Vein 49
Cal-40	Unoxidised mineralised andesite	±1.5	Vein 49

- Comminution Samples

  A sample of oxidised vein (Cal-41) was collected from an outcrop on Vein 49. Samples of unoxidised vein (Cal-42), oxidised mineralised andesite (Cal-43) and unoxidised mineralised andesite (Cal-44) were prepared from whole core from Vein 49. These samples had masses in the range of 55 to 60 kg.

- Materials Handling Sample

  A 78 kg sample of oxidised mineralised andesite was collected from material exposed at the edge of drill pads on Vein 49, which was expected to display some of the worst materials handling characteristics of the deposit.

- Advanced Media Competency Sample

  A 209 kg sample of Vein 49 unoxidised vein was collected from freshly drilled PQ core.

- Heap Leach Samples

  Approximately 100 kg each of oxidised mineralised andesite and unoxidised mineralised andesite was collected from whole core from Vein 49.

- Water Sample

  500 L of water were collected from a bore at the project site.

2.4 Ore and Water Characterisation

2.4.1 Head Assays – Leach Optimisation Samples At AMMTEC

The leach optimisation samples were submitted for comprehensive analysis. Results are listed in Table 2.3.

Table 2.3 Leach optimisation samples analysis

Element	Units	Oxidised Vein: (Cal-1)	Unoxidised Vein: (Cal-2)
Au-1	g/t	2.86	2.92
Au-2	g/t	3.56	3.12, 2.86
Au-3	g/t	3.14	3.48
Au-4	g/t	3.00	3.44, 3.56
As	%	10, 10	<10
Ag	ppm	33, 32	35
Al	%	3.49, 3.46	1.45
Ba	ppm	519, 511	158
Bi	ppm	<10, <10	<10
C_{ORG}	%	0.04, 0.04	0.03
C_{TOTAL}	%	0.40, 0.39	2.85
Ca	%	1.61, 1.58	8.06
Cd	ppm	6, 7	7
Co	ppm	10, 11	6
Cr	ppm	210, 210	200
Cu	ppm	101, 96	35
Fe	%	1.81, 1.85	1.12
Hg	ppb	50, 60	<10, <10
K	%	0.76, 0.73	0.42
Li	ppm	57, 50	35
Mg	ppm	1524, 1580	2168
Mn	ppm	304, 308	690
Mo	ppm	<5, <5	<5
Na	ppm	400, 350	180
Ni	ppm	72, 69	72
P	ppm	400, 300	200
Pb	ppm	465, 429	116
Sb	ppm	3.7, 3.6	2.9, 2.9
Sr	ppm	182, 176	101
S_{TOTAL}	%	0.87, 0.85	0.56
Te	ppm	0.2, 0.2	0.4, 0.2
Ti	ppm	1440, 1362	619
V	ppm	62, 58	20
Y	ppm	7, 7	6
Zn	ppm	400, 399	310
Zr	ppm	21, 19	10

The specific gravity of Cal-1 was measured at 2.64 and Cal-2 at 2.65.

Head assays for other samples are listed in Table 2.4.

Table 2.4 Head grade analysis for Cal-3 to Cal-40, Cal-49 and Cal-50.

Sample	Au-1	Au-2
Cal-3	1.14	1.15, 1.35
Cal-4	1.87, 1.83	1.98
Cal-5	2.36	2.42
Cal-6	2.89	2.92
Cal-7	1.98	1.90
Cal-8	5.03	4.88
Cal-9	7.41	7.08
Cal-10	6.73	6.74
Cal-11	1.33	1.12
Cal-12	1.20	1.21
Cal-13	2.75	2.35
Cal-14	2.35	2.54
Cal-15	6.18	6.47
Cal-16	5.38	4.77
Cal-17	6.18	5.65, 5.82
Cal-18	5.33	3.80
Cal-19	1.07, 1.18	1.00
Cal-20	1.41	1.27
Cal-21	2.96	2.71
Cal-22	1.06	1.05
Cal-23	1.00	0.96
Cal-24	1.16	0.94
Cal-25	0.151	0.25
Cal-26	3.38	3.28
Cal-27	2.74	2.82
Cal-28	4.09	3.64
Cal-29	4.19	4.62, 4.36
Cal-30	3.42	3.53
Cal-31	4.48	4.59
Cal-32	5.32	5.77
Cal-33	5.21	5.08
Cal-34	8.09	7.32
Cal-35	0.39	0.50
Cal-36	0.65	0.59
Cal-37	0.95	0.88
Cal-38	0.09	0.14
Cal-39	0.44	0.42
Cal-40	1.62	1.75, 2.07
Cal-49	0.44	0.43
Cal-50	0.26	0.27, 0.30

2.4.2 Mineralogy

Portions of Cal-1 and Cal-2 were screened at 0.1 mm and heavy minerals were separated using heavy media liquid separation techniques with the "sinks" fractions mineralogically examined.

Cal-1 Sample Sinks (+0.1 mm)

This fraction was observed to contain pyrite and goethite. Only one occurrence of gold was detected, an irregular mass of liberated electrum (55 x 20 µm). There were a number of examples of native silver, as irregular liberated grains, but none of these showed any gold content. These had long dimensions of 170, 50 30 and 40 µm. The silver also occurred as acanthite and iodargyrite, with the acanthite occurring as a rim to pyrite and iodargyrite as fines in goethite.

Cal-1 Sample Sinks (-0.1 mm)

The finer Cal-1 fraction was observed to contain pyrite and iron "oxides." No gold was detected. Silver was present as native silver, acanthite and as an unidentified Cu/Pb/Ag sulphide containing about 6% Se. The native silver occurred as discrete, irregular grains, less than 0.1 mm in size. The acanthite occurred as an 80 µm discrete phase and as fines in pyrite.

Cal-2 Sample Sinks (+0.1 mm)

This fraction of unoxidised vein was observed to contain more sulphides than oxides. The sulphides were dominated by pyrite, occurring mostly as clusters of medium to fine grains in quartz. Four examples of electrum were detected:

- A 4 µm electrum in goethite also containing pyrite.
- Two electrums of 10 and 8 µm in a 120 µm acanthite grain.
- A 15 µm electrum composite with sphalerite and chalcopyrite.
- A pair of electrums in a 50 µm pyrite also containing a rim of digenite.

Silver was also found to occur as native silver and acanthite. The native silver occurred as fines in goethite and as a 50 µm grain attached to goethite in quartz. The acanthite was composite with sphalerite and galena.

Cal-2 Sample Sinks (-0.1 mm)

Again, more sulphides were observed than oxides. The sulphides were dominated by fresh discrete pyrite. One occurrence of electrum was detected, in jalpaite. Silver also occurred as native silver, acanthite, and jalpaite. Native silver was present at 55 µm and as a core in a 120 µm strip of acanthite. The jalpaite was a 140 µm grain hosting a 20 µm electrum grain.

2.4.3 Site Water

The analysis of site water is presented in Table 2.5.

Table 2.5 Site Water Analysis

Analyte	Unit	Value
Ag	mg/L	<0.02
Al	mg/L	<0.20
Ba	mg/L	<0.05
Bi	mg/L	<0.10
Ca	mg/L	740
Cd	mg/L	<0.05
Co	mg/L	<0.05
Cr	mg/L	<0.10
Cu	mg/L	<0.02
Fe	mg/L	<0.10
K	mg/L	2.6
Li	mg/L	0.10
Mg	mg/L	70
Mn	mg/L	0.31
Mo	mg/L	<0.05
Na	mg/L	160
Ni	mg/L	<0.05
P	mg/L	<1.0
Pb	mg/L	<0.05
Sr	mg/L	3.4
Ti	mg/L	<0.10
V	mg/L	<0.02
Y	mg/L	<0.01
Zn	mg/L	0.02
Zr	mg/L	<0.05
HCO_3	mg/L as $CaCO_3$	115
CO_3	mg/lL as $CaCO_3$	0.0
Cl	mg/L	<100
SO_4	mg/L	1660
TDS	mg/L	2590
pH		8.13
Conductivity	(ms/cm)	2.70

2.5 Leach Optimisation (2005)

The leach optimisation at AMMTEC was approached in four rounds of tests:

- Optimisation of grind size.
- Optimisation of general conditions.
- Evaluation of effect of pulp density.
- Optimisation of cyanide concentration.

Prior to commencing a round of tests, the results from the previous round were assessed and conditions selected for the subsequent work.

2.5.1 Grind Size

Oxidised Vein

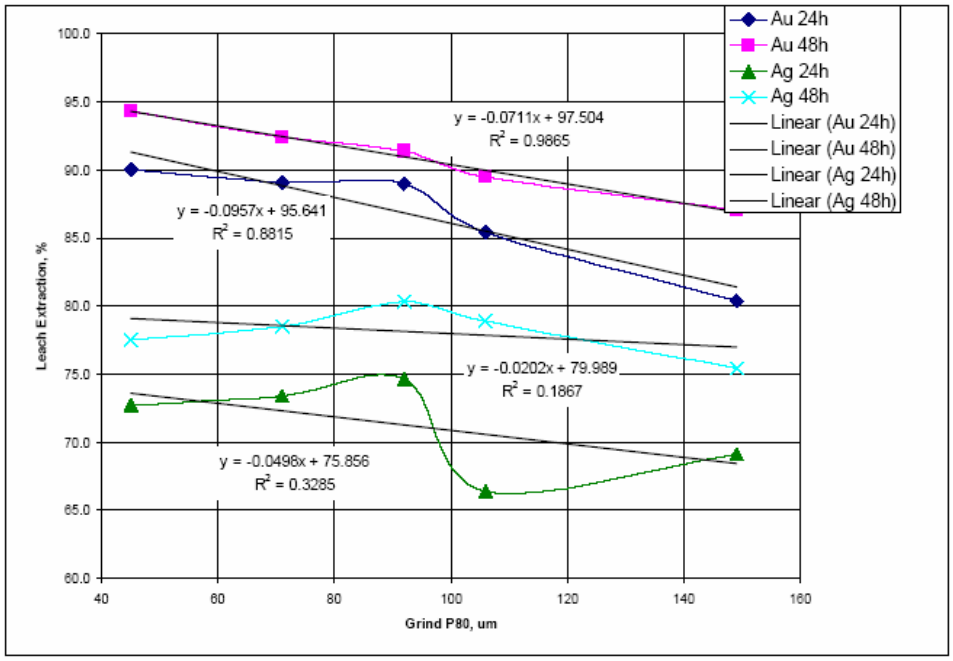
Portions of the oxidised leach optimisation composite sample were leached at various grind sizes. All tests were conducted at 40% pulp density, pH 10.5 and an initial cyanide concentration of 0.05%, with cyanide concentration maintained above 0.025%. Table 2.6 summarises the results for extractions after 24 hours and 48 hours.

Table 2.6 Oxidised Vein Leach Results at Various Grinds

Target Grind	µm	38	53	75	106	106	150
Actual Grind	µm	-	45	71	92	106	149
24 h Au extraction	%	93.4	90.0	89.1	89.0	85.4	80.4
48 h Au extraction	%	96.3	94.3	92.4	91.4	89.5	87.0
24 h Ag extraction	%	76.2	72.7	73.4	74.6	66.4	69.1
48 h Ag extraction	%	81.3	77.5	78.5	80.3	78.9	75.4

There is a trend of increasing extraction with decreasing grind size. The actual sizing of the material with a target grind of 38 µm was suspect and as this size is below any likely selection for the project, the results have not been included in the assessment. Results from the remaining tests are plotted in Figure 2.1.

Figure 2.1 Oxidised Vein Leach Results at Various Grinds



Linear trendlines were fitted to the results and leach extraction versus grind relationships established for use in the grind assessment.

The criteria shown in Table 2.7 were used to compare the value of increased extraction of gold and silver with the cost of grinding finer.

While some of these criteria have since been superseded, they represented the best information available at the time of the assessment.

Table 2.7 Criteria Used for Grind Size Assessment

Criterion	Unit	Value	Source
Project throughput	tpa	730 000	Preliminary assessment: process design basis
Gold head grade	g/t	3.4	Preliminary assessment: mineral resources
Silver head grade	g/t	30	Preliminary assessment: mineral resources
Gold price	$/oz	500	Spot price at time of assessment
Silver price	$/oz	8.50	Spot price at time of assessment
Bond ball mill work index – oxidised vein	kWh/t	19.4	Preliminary AMMTEC results
Bond ball mill work index – unoxidised vein	kWh/t	18.5	Preliminary AMMTEC results
Abrasion index – oxidised and unoxidised vein		0.6	Preliminary AMMTEC results
Power cost	$/kWh	0.15	Preliminary assessment, adjusted for fuel price increase
Grinding media cost	$/t	700	Ausenco database
Mill liner cost	$/t	5000	Ausenco database
Mill capital cost	$/kW	1000	Rule-of-thumb

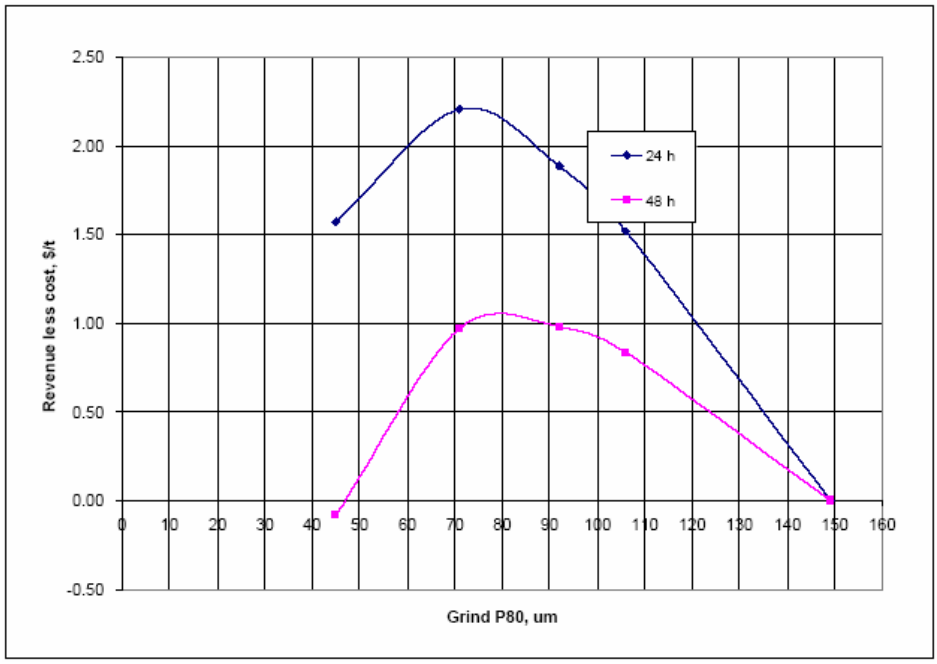
Application of the leach extraction versus grind relationships to these criteria is presented in Table 2.8. All costs and revenues are calculated on an incremental basis for grinding finer than 149 µm. As a means of introducing the mill capital cost to the assessment, which is otherwise incremental revenue less incremental operating cost, an approximate incremental mill capital cost is depreciated over a period of two years and converted to a "$/t" basis.

Table 2.8 Oxidised Vein Size Optimisation

Actual Grind	**µm**	**45**	**71**	**92**	**106**	**149**
Costs						
Incremental power	kWh/t	13.98	7.13	4.33	2.95	-
Incremental power cost	US$/t	2.10	1.07	0.65	0.44	-
Incremental media cost	US$/t	0.77	0.39	0.24	0.16	-
Incremental liner cost	US$/t	0.55	0.28	0.17	0.12	-
Incremental mill size	kW	1276	651	395	269	-
Incremental mill capital cost	$/t	0.87	0.45	0.27	0.18	-
Overall incremental cost	$/t	4.30	2.19	1.33	0.91	-
Revenue - 24 h Leach Basis						
Incremental gold extraction	%	10.0	7.5	5.5	4.1	-
Incremental silver extraction	%	5.2	3.9	2.8	2.1	-
Incremental revenue	$/t	5.87	4.40	3.21	2.42	-
Incremental revenue less cost	$/t	1.57	2.21	1.88	1.52	-
Revenue - 48 h Leach Basis						
Incremental gold extraction	%	7.4	5.5	4.1	3.1	-
Incremental silver extraction	%	2.1	1.6	1.2	0.9	-
Incremental revenue	$/t	4.21	3.16	2.31	1.74	-
Incremental revenue less cost	$/t	-0.08	0.97	0.98	0.84	-

The incremental revenue less cost figures are plotted in Figure 2.2. It should be noted that the graph only compares incremental revenue and cost between the grind size and 149 µm and does not provide any comparison of leach residence times.

Figure 2.2 Oxidised Vein Size Optimisation



Unoxidised Vein

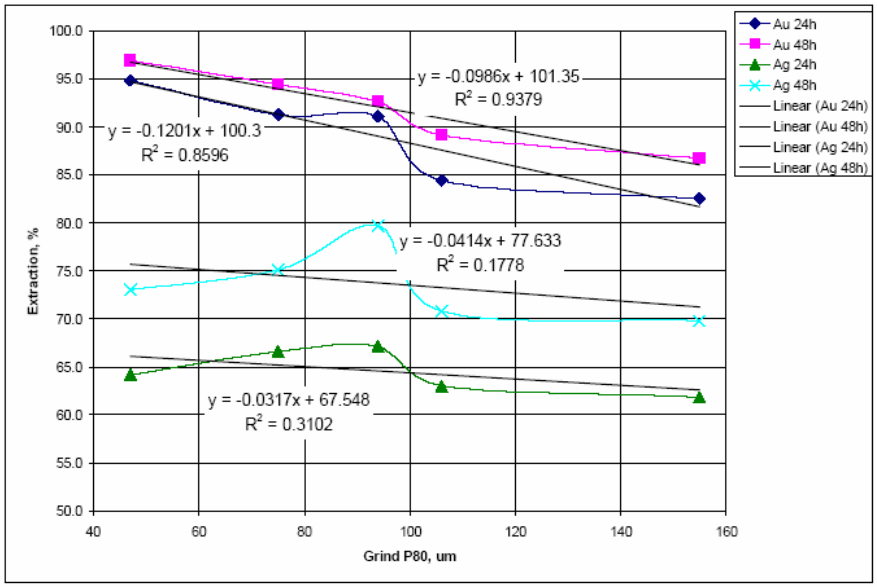
Portions of the unoxidised leach optimisation composite sample were leached at various grind sizes. All tests were conducted at 40% pulp density, pH 10.5 and an initial cyanide concentration of 0.05%, with cyanide concentration maintained above 0.025%. Table 2.9 summarises the results for extractions after 24 hours and 48 hours.

Table 2.9 Unoxidised Vein Leach Results at Various Grinds

Target Grind Actual Grind	µm µm	38 -	53 47	75 75	106 94	106 106	150 155
24 h Au extraction	%	96.0	94.9	91.3	91.1	84.4	82.5
48 h Au extraction	%	97.3	96.9	94.4	92.6	89.1	86.7
24 h Ag extraction	%	70.2	64.1	66.6	67.1	63.0	61.8
48 h Ag extraction	%	76.8	73.0	75.1	79.7	70.8	69.8

As for the oxidised vein sample, there is a trend of increasing extraction with decreasing grind size. The actual sizing of the material with a target grind of 38 µm was again suspect and the results from this test were omitted from the grind size assessment. Results from the remaining tests are plotted in Figure 2.3 with linear trendlines fitted to the results to establish leach extraction versus grind relationships for use in the grind assessment.

Figure 2.3 Unoxidised Vein Leach Results at Various Grinds



The criteria in Table 2.7 were used again to compare the value of increased gold and silver recovery with the cost of grinding finer for unoxidised vein.

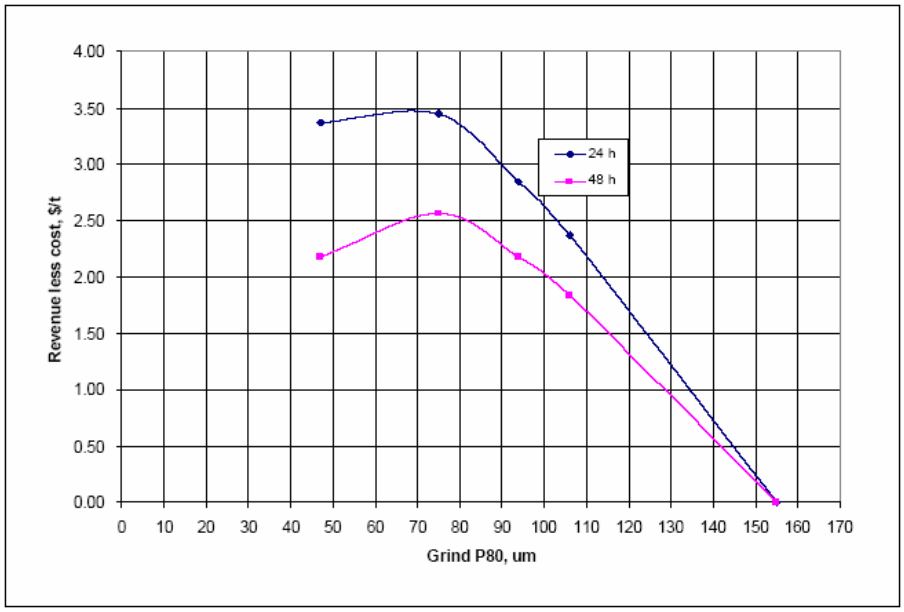
Application of the results of the leach tests to these criteria is presented in Table 2.5.5. All costs and revenues are calculated on an incremental basis for grinding finer than 155 µm.

Table 2.10 Unoxidised Vein Size Optimisation

Actual Grind	µm	47	75	94	106	155
Costs						
Incremental power	kWh/t	12.91	6.46	4.22	3.11	-
Incremental power cost	US$/t	1.94	0.97	0.63	0.47	-
Incremental media cost	US$/t	0.71	0.36	0.23	0.17	-
Incremental liner cost	US$/t	0.51	0.26	0.17	0.12	-
Incremental mill size	kW	1178	589	385	284	-
Incremental mill capital cost	$/t	0.81	0.40	0.26	0.19	-
Overall incremental cost	$/t	3.97	1.99	1.30	0.96	-
Revenue - 24 h Leach Basis						
Incremental gold extraction	%	13.0	9.6	7.3	5.9	-
Incremental silver extraction	%	3.4	2.5	1.9	1.6	-
Incremental revenue	$/t	7.34	5.44	4.14	3.33	-
Incremental revenue less cost	$/t	3.37	3.45	2.85	2.37	-
Revenue - 48 h Leach Basis						
Incremental gold extraction	%	10.6	7.9	6.0	4.8	-
Incremental silver extraction	%	4.5	3.3	2.5	2.0	-
Incremental revenue	$/t	6.14	4.55	3.47	2.79	-
Incremental revenue less cost	$/t	2.18	2.57	2.17	1.83	-

The incremental revenue less cost figures are plotted in Figure 2.4.

Figure 2.4 Unoxidised Vein Size Optimisation



Grind Size Selection

A preliminary assessment of the value of increased gold and silver recovery with the cost of grinding finer was made as the results from the above tests became available, to form the basis for the following leach optimisation tests. At this stage, a grind of 80% passing 90 µm was selected for both oxidised and unoxidised vein samples.

The preliminary assessment was reviewed prior to commencing variability leach testing, in light of significant increases in the spot prices of gold and silver and fresh Bond ball mill work index results. The criteria shown in Table 2.7 were used at this time. These criteria, when applied to the results from the leach optimisation tests provided the incremental revenue less incremental cost data shown in Figure 2.2 and Figure 2.4. The graphs indicate that a grind of 80% passing 75 µm offers the optimal balance between increased revenue and increased cost from grinding finer for the oxidised and unoxidised vein samples, when leaching for 24 hours or 48 hours. (Note that 75 µm is the standard sieve size to which the peak of these curves are closest. Under normal circumstances, standard sieve sizes are preferred selections, due to ready availability of testing equipment.)

Based on the results from leaching the oxidised and unoxidised vein samples and the above analysis, the grind size selection was modified to 80% passing 75 µm.

2.5.2 CIL

CIL tests at a nominal grind of 80% passing 75 µm were conducted in parallel with the grind optimisation tests to assess whether the material has any preg-robbing characteristics. The tests were conducted at 40% pulp density, pH 10.5 and an initial cyanide concentration of 0.05%, with cyanide concentration maintained above 0.025%.

Table 2.11 summarises the results of the carbon-in-leach tests and provides a comparison with the direct leach tests under the otherwise same conditions.

Table 2.11 Carbon-in-Leach Test Results Summary

Sample	Test	Leach Extraction, %			
		24 h Au	24 h Ag	48 h Au	48 h Ag
Oxidised Vein	CIL	89.5	91.2	92.4	90.4
	Leach	89.1	73.4	92.4	78.5
Unoxidised Vein	CIL	89.3	81.4	92.0	86.1
	Leach	91.3	66.6	94.4	75.1

The results do not indicate significant preg-robbing and there is no clear improvement in gold recovery in the presence of carbon. Silver extraction rates appear to improve in the presence of carbon.

Ausenco's experience of projects with silver:gold ratios in the order of 10:1 has been that a Merrill Crowe circuit provides a more cost effective flowsheet than CIL. As there appeared to be no advantage, in terms of gold recovery, to leaching in the presence of carbon, it was decided to proceed with testwork, using leach tests without carbon addition, which are applicable to the Merrill Crowe process. Such leach tests are also applicable to the CIL process, but could be expected to provide a conservative result in terms of silver extraction, in this case. Proceeding with testwork, using CIL tests, would

have otherwise added significantly to the complexity and cost of the testwork programme and the results would not have been applicable to the Merrill Crowe process.

2.5.3 General Leach Conditions

The effect of 4 hours of pre-aeration, dosing with 100 g/t lead nitrate, sparging with oxygen and combining gravity recovery with leaching were assessed. All tests were conducted at a grind of 80% passing 90 µm, 40% pulp density, pH 10.5 and an initial cyanide concentration of 0.05%, with cyanide concentration maintained above 0.025%.

The material for gravity tests was ground to 80% passing 150 µm for gravity separation using a laboratory scale, Knelson 3" separator. The concentrate was amalgamated and amalgam tails combined with gravity tails before grinding to 80% passing 90 µm and subsequently leaching. The gold recovery to amalgam was 7.7% from the oxidised vein sample and 10.4% for the unoxidised vein.

The test results are summarised in Table 2.12 and compared with the results from the test at the same grind size from the grind optimisation, the "control" test. Note that the reported leach extractions include recovery of gold and silver to amalgam, in the case of the gravity tests.

Table 2.12 General Leach Conditions Test Results Summary

Sample	Test	Leach Extraction, % (recovery to amalgam included for gravity tests)			
		24 h Au	24 h Ag	48 h Au	48 h Ag
Oxidised Vein	Control	89.0	74.6	91.4	80.3
	Gravity Lead	87.4	74.2	91.5	82.5
	Nitrate	90.6	79.8	91.8	83.8
	Preaeration	87.1	75.3	91.3	79.1
	Oxygen	92.8	77.1	92.3	82.1
Unoxidised Vein	Control	91.1	67.1	92.6	79.7
	Gravity Lead	91.2	65.2	92.4	75.9
	Nitrate	90.3	66.9	92.5	75.7
	Preaeration	87.2	62.3	91.9	71.7
	Oxygen	93.6	68.3	93.4	79.8

The results summarised in Table 2.12 indicate no benefit from combining gravity with leaching or from pre-aerating. Lead nitrate is shown to offer an improvement, compared to the control test, for oxidised vein, but not for unoxidised vein. The results indicate that oxygen sparging offers the best gold extraction in all cases and the best silver extraction, with the exception of lead nitrate of oxidised vein.

On the basis of the results summarised in Table 2.12, the value of increased extraction with oxygen sparging justifies the capital and operating costs associated with its inclusion. Consequently, oxygen sparging has been used in subsequent testwork.

2.5.4 Pulp Density Evaluation

In the event that Merrill Crowe was to be the selected process, it is considered likely that a pre-leach thickener would be justified and an evaluation was conducted of the effect of pulp density on leach performance. Tests were conducted at pulp densities of 45% and

50% solids and compared with results from the previous test at 40% solids. All tests were conducted at a grind of 80% passing 90 µm, pH 10.5, a dissolved oxygen level of 20 – 25 ppm with oxygen sparging and an initial cyanide concentration of 0.05%, with cyanide concentration maintained above 0.025%. Table 2.13 summarises the results of tests to evaluate effect of pulp density.

Table 2.13 Pulp Density Evaluation Result Summary

Sample	Pulp Density	Leach Extraction, %			
		24 h Au	24 h Ag	48 h Au	48 h Ag
Oxidised Vein	40% solids	89.0	74.6	91.4	80.3
	45% solids	90.5	76.3	90.5	84.5
	50% solids	89.7	70.5	91.1	82.9
Unoxidised Vein	40% solids	91.1	67.1	92.6	79.7
	45% solids	92.4	64.0	92.4	74.9
	50% solids	93.5	64.6	92.9	79.3

Table 2.13 indicates that use of higher pulp densities, up to 50% solids, has no discernable affect upon leach performance. Consequently, a pulp density of 50% solids was used in subsequent testwork.

2.5.5 Cyanide Concentration Evaluation

The sensitivity of leach performance to cyanide concentration was examined to provide an indication of optimum cyanide consumption and to provide design criteria for cyanide detoxification. All tests were conducted at a grind of 80% passing 90 µm, pH 10.5, a dissolved oxygen level of 20 – 25 ppm and 50% solids. The following cyanide concentrations were compared:

- Initial concentration of 0.025% and maintained above 0.015%.
- Initial concentration of 0.035% and maintained above 0.02%.
- Initial concentration of 0.05% and maintained above 0.025% (as used in all prior tests).
- Initial concentration of 0.075% and maintained above 0.04%.

Table 2.14 summarises the results of these tests.

Table 2.14 Cyanide Concentration Evaluation Result Summary

Sample	Cyanide Concentration, % Initial/maintain	Leach Extraction, %			
		24 h Au	24 h Ag	48 h Au	48 h Ag
Oxidised Vein	0.025/0.015	84.2	69.0	86.9	79.8
	0.035/0.2	87.0	73.1	93.1	84.8
	0.05/0.025	89.7	70.5	91.1	82.9
	0.075/.004	92.0	79.7	91.6	87.9
Unoxidised Vein	0.025/0.015	87.5	60.7	92.1	75.5
	0.035/0.2	89.9	67.0	93.1	79.5
	0.05/0.025	93.5	64.6	92.9	79.3
	0.075/0.04	88.7	71.2	93.0	85.7

The results in Table 2.14 suggest a trend of increasing extraction with increasing cyanide concentration. Evaluation of the value of increased recovery with cost of increased cyanide dosage led to selection of an initial cyanide concentration of 0.05%, maintained above 0.025%, for ongoing work.

2.6 Variability Leach Testing

A total 38 variability samples were subjected to vat leaching using oxygen injection, an initial cyanide concentration of 0.05%, maintained above 0.025% and a grind size of 80% passing 75 µm. The results are summarised in Table 2.15.

Table 2.15 Variability Leach Test Results

Sample No.	Ore Type	Gold				Silver			
		Calc'd Head g/t	24 h Extr'n %	48 h Extr'n %	Residue g/t	Calc'd Head g/t	24 h Extr'n %	48 h Extr'n %	Residue g/t
Cal-3	Oxidised	1.21	99.9	96.8	0.04	5.2	90.1	86.6	0.7
Cal-4	vein	1.65	86.1	85.7	0.24	23.4	80.4	81.2	4.4
Cal-5		2.03	87.1	89.8	0.21	7.3	78.1	83.6	1.2
Cal-6		2.67	90.0	88.8	0.30	11.6	97.4	89.6	1.2
Cal-7		2.03	83.0	87.1	0.26	34.1	89.8	95.0	1.7
Cal-8		4.42	99.0	95.5	0.20	50.2	81.5	79.7	10.2
Cal-9		6.52	95.3	93.9	0.40	47.2	80.9	78.2	10.3
Cal-10		5.62	92.7	87.8	0.69	96.8	51.5	54.5	44.0
Cal-11	Unoxidised	1.15	93.9	92.0	0.09	12.4	83.0	84.6	1.9
Cal-12	vein	1.12	90.2	96.2	0.04	9.2	72.4	78.3	2.0
Cal-13		2.49	92.1	92.6	0.18	44.1	75.3	83.4	7.3
Cal-14		2.15	95.1	96.0	0.09	48.3	62.2	78.1	10.6
Cal-15		6.05	69.7	92.7	0.45	72.0	41.9	62.5	27.0
Cal-16		5.18	85.5	85.2	0.77	18.4	81.6	84.8	2.8
Cal-17		5.30	88.6	86.2	0.32	16.7	88.6	86.3	2.3
Cal-18		4.88	95.9	92.7	0.35	50.0	78.5	87.4	6.3
Cal-19	Oxidised	0.98	85.1	86.2	0.14	2.1	94.2	95.2	0.1
Cal-20	mineralised	1.16	96.2	91.9	0.09	8.7	78.3	81.7	1.6
Cal-21	andesite	2.81	98.1	93.9	0.17	174.5	49.7	76.5	41.0
Cal-22		0.95	87.1	88.6	0.11	26.0	91.5	95.0	1.3
Cal-23	Unoxidised	0.97	95.1	91.0	0.09	4.7	63.2	62.9	1.7
Cal-24	mineralised	1.16	95.2	93.1	0.06	9.0	66.1	71.0	2.6
Cal-25	andesite	0.21	95.1	89.3	0.02	3.6	60.1	58.5	1.5
Cal-26		3.62	94.8	91.9	0.30	31.7	59.7	61.9	12.1
Cal-27	Oxidised	2.80	97.9	94.6	0.15	29.2	80.8	82.6	5.1
Cal-28	vein	4.04	88.5	89.3	0.43	9.9	73.4	75.8	2.4
Cal-29		4.67	90.0	91.5	0.40	43.5	78.4	83.9	7.0
Cal-30		3.98	89.4	92.3	0.31	25.6	68.0	83.6	4.2
Cal-31	Unoxidised	4.46	81.5	91.5	0.38	10.7	65.4	78.5	2.3
Cal-32	vein	5.59	91.3	92.6	0.41	49.7	78.2	88.5	5.7
Cal-33		5.40	89.0	90.3	0.52	26.5	67.1	79.6	5.4
Cal-34		7.02	78.9	86.2	0.97	40.3	54.6	60.3	16.0
Cal-35	Oxidised	0.37	85.2	85.7	0.05	11.9	90.9	92.5	0.9
Cal-36	mineralised	0.65	91.3	93.2	0.04	10.2	71.8	74.4	2.6
Cal-37	andesite	0.97	91.6	94.4	0.05	16.3	76.6	82.8	2.8
Cal-38	Unoxidised	0.12	82.2	82.2	0.02	2.6	68.1	69.4	0.8
Cal-39	mineralised	0.41	92.4	88.5	0.05	7.1	79.1	77.5	1.6
Cal-40	andesite	1.42	91.5	92.6	0.11	16.4	75.9	79.9	3.3

The laboratory noted high slurry viscosity when testing the following samples:

- Oxidised vein: Cal-27, Cal-29 and Cal-30.
- Unoxidised vein: Cal-31 and Cal-34.
- Oxidised mineralised andesite: Cal-19, Cal-22, Cal-35, Cal-36 and Cal-37.
- Unoxidised mineralised andesite: Cal-38, Cal-39 and Cal-40.

It was necessary to reduce the pulp density used in the leaching tests for Cal-22, Cal-37 and Cal-38 to 40% solids, because the viscosity of the pulp at 50% solids was too high for effective agitation.

The silver extraction kinetics were found to range from comparatively fast to quite slow throughout the variability testwork programme. This may be the result of the presence of silver in different forms, which perform quite differently with cyanidation.

2.7 Diagnostic Leaching

Diagnostic analysis was performed on the leach residues obtained from some of the tests performed to evaluate the effect of cyanide concentration. Diagnostic analysis involves leaching the residue for 48 hours using a relatively high concentration of cyanide to indicate “free gold” content, followed by aqua regia analysis of the residue to determine “sulphidic gold” and finally fire assay of the aqua regia residue to measure “silicate gold.”

Details of the test results are contained in AMMTEC’s report. They indicate significant proportions of the leach residue gold present in the sulphidic and silicate components. As the concentration of cyanide used in the leaching tests decreased, the residual “free gold” tended to increase, perhaps indicating incomplete gold dissolution during the leaching process.

2.8 Flotation Testwork

The two main composite samples (Cal-1 and Cal-2) were tested for their response to flotation. The tests were simple sighter tests to evaluate the potential for using flotation. While the recovery of gold and silver could be improved with optimisation, the recoveries were poor and not considered encouraging.

Summaries of the results of the flotation tests on Cal-1 and Cal-2 are shown in Table 2.16 and Table 2.17, respectively.

Table 2.16 Flotation Results for Cal-1

Product	Mass Distribution	Gold		Silver		Sulphur	
		Grade	Dist'n	Grade	Dist'n	Grade	Dist'n
	%	g/t	%	g/t	%	%	%
Con-1	4.9	26.9	43.9	359	51.9	8.49	43.1
Con-2	1.6	9.8	5.1	129	6.0	7.14	11.6
Con-3	2.5	4.0	3.4	59	4.4	3.48	9.2
Tails	91.0	1.6	47.6	14	37.8	0.38	36.0
Calc'd Head	100.0	3.0	100.0	34	100.0	0.96	100.0

Table 2.17 Flotation Results for Cal-2

Product	Mass Distribution	Gold		Silver		Sulphur	
		Grade	Dist'n	Grade	Dist'n	Grade	Dist'n
	%	g/t	%	g/t	%	%	%
Con-1	2.7	48.7	43.4	671	49.7	13.90	61.0
Con-2	1.5	12.9	6.6	178	7.6	4.42	11.1
Con-3	1.2	7.2	2.8	106	3.4	2.40	4.5
Tails	94.6	1.5	47.3	15	39.3	0.15	23.3
Calc'd Head	100.0	3.0	100.0	36	100.0	0.61	100.0

Diagnostic leach tests were performed on the flotation tails fractions from both samples to assess the distribution of the remaining gold. Most of the gold was indicated to be "free," 73% in the case of Cal-1 and 69% with Cal-2.

2.9 Comminution Testing

2.9.1 Unconfined Compressive Strength

A number of pieces of core were selected for Unconfined Compressive Strength (UCS) Tests. Samples Cal-28, Cal-29 (both oxidised vein) and Cal-38 (unoxidised mineralised andesite) were too friable to test. Table 2.18 shows the results, with a descriptive classification.

Table 2.18 UCS Test Results

Sample No.	Material	UCS MPa	Classification
Cal-27	Oxidised vein	22.7	Medium strong
Cal-30	Oxidised vein	>170	Too hard to measure
Cal-31	Unoxidised vein	33.5	Medium strong
Cal-32	Unoxidised vein	48.6	Medium strong
Cal-33	Unoxidised vein	54.1	Medium strong
Cal-34	Unoxidised vein	75.0	Strong
Cal-35	Oxidised mineralised andesite	38.0	Medium strong
Cal-36	Oxidised mineralised andesite	19.7	Weak
Cal-37	Oxidised mineralised andesite	17.5	Weak
Cal-39	Unoxidised mineralised andesite	12.8	Weak
Cal-40	Unoxidised mineralised andesite	25.2	Medium strong
Cal-46-a	Unoxidised vein	59.1	Medium strong
Cal-46-b	Unoxidised vein	64.1	Strong
Cal-46-c	Unoxidised vein	84.2	Strong
Cal-46-d	Unoxidised vein	79.2	Strong
Cal-46-e	Unoxidised vein	44.0	Medium strong

2.9.2 Bond Rod and Ball Mill Work Indices and Abrasion Indices

A number of samples were tested to provide Bond rod mill work indices, Bond ball mill work indices and abrasion indices, as shown in Table 2.19.

Table 2.19 Bond Rod and Ball Mill Indices and Abrasion Indices

Sample No.	Ore Type	Rod Mill WI kWh/t	Ball Mill WI kWh/t	Abrasion Index
Cal-27	Oxidised vein	18.9	17.1	0.574
Cal-28	Oxidised vein	21.4	21.4	0.559
Cal-29	Oxidised vein	22.9	19.6	0.675
Cal-30	Oxidised vein	22.2	19.5	0.600
Cal-31	Unoxidised vein	13.4	13.5	0.275
Cal-32	Unoxidised vein	20.5	20.0	0.561
Cal-33	Unoxidised vein	22.6	18.4	0.650
Cal-34	Unoxidised vein	25.8	22.3	0.826
Cal-35	Oxidised mineralised andesite	16.8	15.7	0.161
Cal-36	Oxidised mineralised andesite	17.2	16.3	0.165
Cal-37	Oxidised mineralised andesite	12.8	9.4	0.072
Cal-38	Unoxidised mineralised andesite	12.3	9.2	0.111
Cal-39	Unoxidised mineralised andesite	14.1	12.6	0.222
Cal-40	Unoxidised mineralised andesite	18.5	16.3	0.219
Cal-41	Oxidised vein	23.6	20.6	0.593
Cal-42	Unoxidised vein	21.9	20.9	0.613
Cal-43	Oxidised mineralised andesite	13.5	12.1	0.129
Cal-44	Unoxidised mineralised andesite	17.7	16.6	0.202
Cal-46	Unoxidised vein	17.8	17.2	0.519

The rod and ball mill indices suggest that vein material is hard and competent, while the mineralised andesite has medium hardness and competency.

The abrasion indices indicate the following:

- Oxidised vein: abrasive.
- Unoxidised vein: generally abrasive with one very abrasive and one slightly abrasive result.
- Oxidised mineralised andesite: slightly abrasive.
- Unoxidised mineralised andesite : slightly abrasive.

2.9.3 Autogenous Media Competency Test

A sample of unoxidised vein (Cal-46) was subjected to an Autogenous Media Competency (AMC) Test. A summary of the test results is shown in Table 2.20.

Table 2.20 AMC Test Results Summary

Energy Consumption kWh/t	CWi -101+76 mm kWh/t		CWi -76+51 mm kWh/t		CWi -51+38 mm kWh/t		CWi -38+25 mm kWh/t		CWi -25+19 mm kWh/t	
	Av.	Max.	Av.	Max.	Av.	Max.	Av.	Max.	Av.	Max.
3.62	19.7	24.8	13.8	23.7	11.6	20.9	9.2	20.5	8.5	14.8

The AMC test is used to assess the suitability of a SAG mill to grind the material. None of the results in Table 2.20 raise any concern with the application of SAG milling.

2.9.4 SMC Tests

Samples of each of the main ore types were subjected to the SAG Mill Comminution (SMC) test to assist with evaluating the comminution characteristics for the different materials. A summary of the results is given in Table 2.21.

Table 2.21 SMC Test Results Summary

Sample	DWi	A x 6	t_{10} @ 1 kWh/t
Oxidised vein	4.8	51.5	37.1
Unoxidised vein	6.4	40.0	32.2
Oxidised mineralised andesite	2.6	89.2	45.1
Unoxidised mineralised andesite	4.9	51.5	36.4

The Drop Weight Index (DWi) measurements typically range from 2 to 12, with softer ores being at the low end of the scale. The unoxidised vein is seen to be in the middle of the range and the other samples are in the lower half of the range.

The Ax6 value is a measure of resistance to breakage, with higher values indicating softer material. The results indicate the following:

- Oxidised vein sample: medium hardness.
- Unoxidised vein sample: moderately hard.
- Oxidised mineralised andesite sample: soft.
- Uunoxidised mineralised andesite sample: medium hardness.

The t10 at 1 kWh/t values indicate the following:

- Oxidised vein sample: moderately soft.
- Unoxidised vein sample: medium hardness.
- Oxidised mineralised andesite sample: soft.
- Unoxidised mineralised andesite sample: moderately soft.

2.10 Testing for Engineering Data

2.10.1 Oxygen Uptake

Splits of the main composite samples (Cal-1 and Cal-2) were milled to a P80 of 75 µm and tested for oxygen uptake. The results are summarised in Table 2.22 and indicate low oxygen consumption for both samples.

Table 2.22 Oxygen Uptake Results

Sample	Oxygen Consumption mg/L						
	0 h	1 h	2 h	3 h	4 h	5 h	6 h
Cal-1	-0.045	0.007	-0.009	-0.003	0.006	-0.004	0.008
Cal-2	-0.030	0.023	0.015	0.014	-0.008	-0.001	-0.009

2.10.2 Carbon Sequential Loading Test

Splits of the main composite samples were milled to a P80 of 75 µm, leached for 48 hours and subjected to carbon sequential loading tests. The results are summarised in Table 2.23 and are considered typical for gold-silver ores.

Table 2.23 Carbon Loading Kinetics

Sample	Gold		Silver	
	Fleming k h^{-1}	Fleming n	Fleming k h^{-1}	Fleming n
Cal-1	151.5	0.714	96.9	0.703
Cal-2	183.8	0.630	84.1	0.656

2.10.3 Viscosity Testing

Splits of the main composite samples were leached for 48 hours at a P80 of 75 µm and slurry viscosity was measured over a range of pulp densities, using a Bohlin Visco 88 viscometer. A summary is shown in Table 2.24.

Table 2.24 Viscosity Measurements

Sample	Shear rate	4.2 s^{-1}	7.3 s^{-1}	12.7 s^{-1}	22.4 s^{-1}	39.1 s^{-1}	68.3 s^{-1}	119.5 s^{-1}	209 s^{-1}
	% solids	Viscosity (cP)							
Cal-1	60	10 680	6065	3650	2380	1465	949	598	371
	50	1938	1162	713	456	284	177	128	116
	40	458	320	183	119	73	59	53	63
Cal-2	60	3047	1803	1106	728	454	286	182	175
	50	479	294	194	125	80	68	64	78
	40	n/a	n/a	n/a	n/a	n/a	31	38	48

It was noted by the laboratory that some of the variability samples, when undergoing leach tests, demonstrated excessively high viscosity (refer to Section 2.6).

2.10.4 Settling Tests

A number of samples were subjected to settling tests in a measuring cylinder to ascertain the settling rates. A dose of 40 g/t Magnafloc 5250 flocculant was used in each test. Composites of each ore type were tested at grinds of both 80% passing 75 µm and 90 µm, in an attempt to correlate the results with those from the thickening testwork, which was performed at a grind of 80% passing 90 µm. The settling test results are shown in Table 2.25.

Table 2.25 Settling Test Results

Sample	Ore Type	Grind size, P_{80} µm	Initial % Solids	Final % Solids	Free Settling Rate m/h
Cal-1	Oxidised vein	90	10.3	44.1	7.3
		75	10.1	43.1	7.1
Cal-2	Unoxidised vein	90	10.4	50.8	8.2
		75	10.5	46.7	4.1
Oxidised mineralised andesite		90	10.1	41.1	6.5
		75	10.6	43.5	6.6
Unoxidised mineralised andesite		90	10.0	38.9	4.5
		75	11.0	41.9	2.2
Cal-27	Oxidised vein	75	9.9	45.6	8.3
Cal-28			9.7	48.0	7.8
Cal-29			9.9	39.7	7.2
Cal-30			9.4	43.0	7.1
Cal-31	Unoxidised vein	75	9.9	44.8	7.4
Cal-32			9.9	46.9	7.4
Cal-33			10.0	47.1	6.6
Cal-34			10.2	49.9	7.6
Cal-35	Oxidised mineralised andesite	75	9.7	38.2	3.1
Cal-36			9.9	47.4	6.7
Cal-37			9.2	34.4	1.7
Cal-38	Unoxidised mineralised andesite	75	10.4	41.1	1.9
Cal-39			9.8	40.0	2.6
Cal-40			10.6	45.3	4.3

2.11 Cyanide Detoxification

Splits of the main composite samples were milled to a P80 of 75 µm, leached for 48 hours in preparation for air-SO2 cyanide detoxification tests. Laboratory scale continuous cyanide detoxification tests were conducted on the slurries at a pulp density of 50% solids.

At the time that the testwork was undertaken, precious metal recovery was thought more likely to involve Merrill Crowe in the final design rather than CIP processing. 40 ppm zinc was added to portions of the leach residue slurries to simulate the recirculation of barren solution from a zinc precipitation circuit into the leach circuit.

The results from testing the Cal-1 and Cal-2 composite samples are summarised in Table 2.26 and Table 2.27, respectively.

Table 2.26 Cyanide Detoxification Tests on Cal-1

Test Conditions						Solution Assays	
Slurry Type	pH	Retention Time min	SO_2 g/g CN_{WAD}	$CuSO_4.5H_2O$ mg/L	Lime g/g SO_2	Feed CN_{WAD} mg/L	Treated CN_P mg/L
Merrill Crowe	9.13	57	4.04	0	0.71	153	0.10
	9.33	58	3.34	0	0.37	153	1.93
	9.59	59	2.67	0	0.00	153	9.38
CIP	9.19	58	3.64	0	0.19	153	0.65

Note that CN_P is equivalent to weak acid dissociable cyanide(CN_{WAD}).

Table 2.27 Cyanide Detoxification Tests on Cal-2

Test Conditions						Solution Assays	
Slurry Type	pH	Retention Time min	SO_2 g/g CN_{WAD}	$CuSO_4.5H_2O$ mg/L	Lime g/g SO_2	Feed CN_{WAD} mg/L	Treated CN_P mg/L
Merrill Crowe	9.12	57 58	3.99	0 27	0.82	144	24.7
	9.00		4.28		n/a	144	0.10
	9.21	56	3.15	26	0.97	144	71.1
CIP	9.07	56	4.18	25	0.95	139	0.81

For Cal-1, a dose of 4.04 g SO_2 per g CN_{WAD} reduced the CN_{WAD} to <0.5 ppm in all cases, while a lower dose of 3.34 g SO_2 per g CN_{WAD} resulted in an effluent containing generally <2 ppm CN_{WAD} with one measurement at 8.2 ppm CN_{WAD}. Further reduction of the SO_2 dosage, resulted in the system becoming unstable with a progressive worsening of detoxification performance.

For Cal-2, a dose of 4.28 g SO_2 per g CN_{WAD} reduced the CN_{WAD} to <0.5 ppm in all cases, while a lower dose of 3.15 g SO_2 per g CN_{WAD} resulted in the system becoming unstable and a progressive worsening of detoxification performance.

The target retention time of 60 minutes was adequate to produce treated effluents containing relatively low levels of CN_{WAD}.

The Cal-2 samples contained insufficient copper in the leach solution to catalyse the detoxification reaction and addition of copper in the form of copper sulphate was found to be necessary.

The effect of zinc in the leach solutions on cyanide detoxification performance was found to be minimal.

2.12 Heap Leach Testing

Samples of oxidised mineralised andesite and unoxidised mineralised andesite were leached for five days at coarse crush sizes to assess amenability for treating by heap leach. The results are summarised in Table 2.28. The head grades of the samples were much lower than intended. A trend of increasing gold extraction with finer crush size is evident; although poor leach extraction was found in all tests.

Table 2.28 Coarse Bottle Roll Leaches

Sample	Crush Size mm	Calculated Head g/t	120 h Gold Extraction %	Leach Residue g/t
Oxidised	25	0.35	27.6	0.26
mineralised	12.5	0.33	35.2	0.22
andesite	6.3	0.29	55.1	0.13
Unoxidised	25	0.17	27.8	0.13
mineralised	12.5	0.19	5.4	0.18
andesite	6.3	0.12	48.4	0.06

The testwork programme had allowed for a column leach test on each sample at a selected crush size. However, it was decided not to proceed with this work, as the sample grades were below that of a likely feed to a heap leach operation.

The testwork programme had also provided for heap leaching on oxidised and unoxidised vein material, but samples were not available for this work.

2.13 Material Handling Characterisation

A sample of oxidised mineralised andesite was selected, as this was expected to have the worst physical handling characteristics. The sample was sent to TUNRA at the University of Newcastle in NSW for testing to obtain information to assist with design of stockpiles, bins and chutes.

The sample was classified by TUNRA as a difficult handling material.

2.14 Vendor Testwork

Samples were prepared and sent by AMMTEC to the following equipment vendors:

- Outokumpu for high rate thickening testwork.
- Delkor for belt filtration testwork.
- Gekko for gravity and flotation testwork.

Testwork reports by each of these vendors are contained in Appendix 2.

2.14.1 Outokumpu

Four samples of leach residue were prepared for testwork at Outokumpu to assess their likely performance in a high rate thickener:

- Oxidised vein: a portion of Cal-1.

- Unoxidised vein: a portion of Cal-2.
- Oxidised mineralised andesite: a composite made from portions of Cal-35, Cal-36 and Cal-37.
- Unoxidised mineralised andesite: a composite made from portions of Cal-38, Cal-39 and Cal-40.

The samples were intended to have been ground to 80% passing 90 µm, the preliminary grind size selection. However, laser sizing by the CSIRO Malvern® laser indicated the P80 sizes shown in Table 2.29.

Table 2.29 Outokumpu Samples Laser Sizing Results

Material	80% passing, µm
Oxidised vein	100
Unoxidised vein	140
Oxidised mineralised andesite	49
Unoxidised mineralised andesite	35

Flocculant screening indicated that, of the flocculants tested, Magnafloc 5250 was suited to the samples. A feed density of 10% solids was selected with the Magnafloc 5250.

Bench scale dynamic thickening testwork was conducted on the four samples using Outokumpu's 94 mm diameter unit. The results are shown in Table 2.30.

Table 2.30 Outokumpu Dynamic Thickening Test Results

Material	Solids Loading t/m2h	Rise Rate m/h	Flocculant Dose g/t	Underflow Density % solids	Overflow Clarity ppm
	0.81	7.5	21	56	60
Oxidised vein	0.81	7.5	32	57	40
	1.08	9.4	30	55	50
	1.01	9.3	43	56	50
	0.85	7.5	31	65	100
Unoxidised	0.81	7.5	43	64	120
vein	1.00	9.0	33	62	130
	1.01	8.8	43	63	170
	0.81	7.6	32	52	100
Oxidised	0.77	7.4	44	53	60
mineralised	0.61	5.7	32	53	90
andesite	0.61	5.7	43	53	120
	0.40	3.8	44	54	90
	0.80	7.5	33	51	130
Unoxidised	0.82	7.6	42	51	100
mineralised	0.60	5.8	32	51	100
andesite	0.58	5.7	44	52	60
	0.40	3.6	32	51	60

Except for the unoxidised vein sample, the underflow densities achieved in these tests were relatively poor. It should be noted that both oxidised and unoxidised mineralised andesite samples were ground finer than intended which was likely to have had an adverse effect on the thickening characteristics of the samples. Conversely, the oxidised and unoxidised vein samples were both coarser than the optimum grind, now considered to be 75 µm, and this could be expected to result in overly optimistic test results.

In the Merrill Crowe process, solid liquid separation is performed in the counter current decantation circuit, commonly in thickeners. Relatively low underflow densities result in low stage washing efficiencies in such a circuit, leading to high soluble gold and silver losses.

2.14.2 Delkor

Two samples of leach residue from portions of Cal-1 and Cal-2 were prepared for testwork at Delkor to assess their likely performance in a horizontal belt filter. Belt filters may be used for solid liquid separation in a Merrill Crowe circuit, as an alternative to thickeners. The target particle size for these samples was 90 µm.

Cal-1 was found to filter poorly under vacuum and Cal-2 to filter reasonably well.

It is unlikely that mineralised andesite samples would filter satisfactorily and in light of the performance of the Cal-1 sample, filtration testwork was not pursued.

2.14.3 Gekko

A composite sample, containing similar proportions of oxidised vein (Cal-1) and unoxidised vein (Cal-2) was prepared by AMMTEC and sent to Gekko in Victoria. Gekko undertook a programme of testing to evaluate the potential for gold and silver recovery by means of gravity and flotation.

Gekko conducted a number of tests using a shaking table:

- In the first test, the sample was crushed to 1.0 mm and tabled. Table tails were reduced to 500 µm and tabled. The resulting tails were ground to 106 µm and then tabled again. The concentrates (1.0 mm, 500 µm, 106 µm) were combined and passed over the table five times, the results of which are shown in Table 2.31.

  In the second test, the sample was crushed to 1.0 mm and passed over the table four times. The results are shown in Table 2.32.

- The third test was the same as the second test, but the sample was finer (0.5 mm). The fourth test was a repeat of the third test. The results are shown in Table 2.33and Table 2.34.

Table 2.31 Gekko Progressive Grind Table Test

	Mass	Cum. Mass	Grade	Cum. Grade	Distribution	Cum. Distr
Fraction	**%**	**%**	**g Au/t**	**g Au/t**	**%**	**%**
Con 1	0.5	0.5	32.9	32.9	4.7	4.7
Con 2	0.9	1.4	4.5	14.1	1.3	6.0
Con 3	2.2	3.6	5.3	8.8	3.4	9.4
Con 4	3.0	6.6	5.1	7.1	4.6	14.0
Con 5	16.1	22.8	3.2	4.4	15.4	29.4
Table Tails	77.2	100.0	3.08	3.4	70.6	100.0
Feed Assay				3.28		

Table 2.32 Gekko Single Pass Table Test at 1.0 mm

	Mass	Cum. Mass	Grade	Cum. Grade	Distribution	Cum. Distr.
Fraction	**%**	**%**	**g Au/t**	**g Au/t**	**%**	**%**
Con 1	1.9	1.9	5.37	5.4	3.6	3.6
Con 2	1.5	3.5	3.31	4.5	1.8	5.4
Con 3	3.3	6.8	2.84	3.7	3.2	8.6
Con 4	11.5	18.3	3.46	3.5	13.8	22.3
Table Tails	81.7	100.0	2.75	2.9	77.7	100.0
Feed Assay				3.28		

Table 2.33 Gekko Single Pass Table Test at 0.5 mm

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution %	Cum. Distr. %
Con 1	0.9	0.9	13.70	13.7	3.7	3.7
Con 2	2.1	3.0	3.81	6.7	2.5	6.2
Con 3	3.6	6.6	3.98	5.2	4.5	10.7
Con 4	10.5	17.1	3.96	4.4	13.0	23.7
Table Tails	82.9	100.0	2.94	3.2	76.3	100
Feed Assay				3.28		

Table 2.34 Gekko Repeat Single Pass Table Test at 0.5 mm

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution %	Cum. Distr. %
Con 1	1.6	1.6	21.90	21.9	10.1	10.1
Con 2	5.0	6.5	6.08	9.9	8.7	18.8
Con 3	6.0	12.5	3.46	6.8	6.0	24.8
Con 4	5.9	18.4	3.07	5.6	5.2	30.0
Table Tails	81.6	100.0	2.96	3.5	70.0	100.0
Feed Assay				3.28		

In the four tests, only 22 to 30% of the gold reported to 17 to 22% of the mass. Only 3.6 to 10.1% of the gold reported to lower mass yields of 0.5 to 1.9%. This suggests that little upgrading of the gold into low mass fractions can be achieved at these particle sizes.

The concentrate 2, 3, 4 and table tails fractions from the repeat single pass table test at 0.5 mm were combined and ground to 75 µm. The material was then passed through a 3" Falcon concentrator. Results are shown in Table 2.35.

Table 2.35 Gekko Falcon Test at 75 µm

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution %	Cum. Distr. %
Concentrate	0.6	0.6	104.0	104.0	24.3	24.3
Falcon Tails	99.4	100.0	2.08	2.7	75.7	100.0
Feed Assay				3.15		

Only 24% of the remaining gold reported to concentrate, providing an overall gold recovery of 34.4% by gravity to a concentrate comprising 2.2% of sample mass.

A flotation test was performed on the Falcon tails. Results are shown in Table 2.36.

Table 2.36 Gekko Flotation Test at 75 µm

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution %	Cum. Distr. %
Con 1	22.6	22.6	3.66	3.66	40.4	40.4
Con 2	8.9	31.6	4.94	4.02	21.5	61.9
Con 3	0.4	32.0	23.6	4.29	5.03	66.9
Flot'n Tails	68.0	100.0	1.00	2.05	33.1	100.0
Feed Assay				2.08		

Gekko combined the results from Table 2.34, Table 2.35 and Table 2.36 to provide the results shown in Table 2.37.

Table 2.37 Gekko Combined Results

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution %	Cum. Distr. %
Table Con 1	1.7	1.7	21.90	21.9	12.2	12.2
Falcon Con	0.6	2.3	104.0	44.2	21.5	33.7
Flot'n Con	31.3	33.6	4.29	7.0	44.4	78.0
Flot'n Tails	66.4	100.0	1.00	3.0	22.0	100.0
Feed Assay				3.28		

The combined results show a gold recovery of 78% into 33.6% of the original sample mass at a grade of 7.0 g/t. While flotation increased the recovery of gold compared with that obtained by gravity alone, it was at the expense of a high mass pull. The grade of the combined concentrate, at 7.0 g/t, is only 2.3 times the calculated head grade.

2.15 Testwork at Universidad Nacional de San Juan

Two 30 kg samples were collected by AQI personnel to represent typical oxidised vein and unoxidised vein deposits and dispatched to UNSJ for evaluation of gold and silver recovery by gravity means and flotation. UNSJ's report is contained in Appendix 2.

According to Table 1 in the annexure of UNSJ's report:

- The unoxidised vein sample was obtained from two bore holes and had a total mass of 26.6 kg.
- The oxidised vein sample weighed 36.6 kg, but its origin is unclear.

Both samples were processed through a Falcon concentrator at grind sizes of approximately 90% passing 400 µm and 150 µm. In all cases the primary Falcon concentrate was upgraded by panning. The concentrator and pan tails from the 150 µm tests were combined to produce oxidised and unoxidised vein gravity tails samples for processing by flotation.

The gravity concentration results at 400 µm obtained from the two samples are shown in Table 2.38 and Table 2.39, while those at 150 µm are shown in Table 2.40 and Table 2.41.

Table 2.38 UNSJ Gravity Concentration of Oxidised Vein at 400 µm

	Mass	Cum. Mass	Grade	Cum. Grade	Distribution	Cum. Distr.
Fraction	**%**	**%**	**g Au/t**	**g Au/t**	**% Au**	**% Au**
Pan con	0.5	0.5	499.09	499.09	36.2	36.2
Pan tail	0.5	0.9	63.8	290.28	4.3	40.4
Falcon tail	99.1	100.0	4.08	6.78	59.6	100.0
Feed Assay				6.48		

Table 2.39 UNSJ Gravity Concentration of Unoxidised Vein at 400 µm

	Mass	Cum. Mass	Grade	Cum. Grade	Distribution	Cum. Distr.
Fraction	**%**	**%**	**g Au/t**	**g Au/t**	**% Au**	**% Au**
Pan con	0.3	0.3	971.12	971.12	62.7	62.7
Pan tail	0.9	1.2	12.41	260.70	2.3	65.0
Falcon tail	98.8	100.0	1.73	4.88	35.0	100.0
Feed Assay				4.06		

Table 2.40 UNSJ Gravity Concentration of Oxidised Vein at 150 µm

	Mass	Cum. Mass	Grade	Cum. Grade	Distribution	Cum. Distr.
Fraction	**%**	**%**	**g Au/t**	**g Au/t**	**% Au**	**% Au**
Pan con	0.6	0.6	248.3	248.30	27.0	27.0
Pan tail	0.4	1.0	229.6	241.10	15.6	42.6
Falcon tail	99.0	100.0	3.43	5.91	57.4	100.0
Feed Assay				6.11		

Table 2.41 UNSJ Gravity Concentration of Unoxidised Vein at 150 µm

	Mass	Cum. Mass	Grade	Cum. Grade	Distribution	Cum. Distr.
Fraction	**%**	**%**	**g Au/t**	**g Au/t**	**% Au**	**% Au**
Pan con	0.7	0.7	530.75	530.75	61.7	61.7
Pan tail	0.4	1.0	50.4	364.72	3.1	64.8
Falcon tail	99.0	100.0	2.06	5.79	35.2	100.0
Feed Assay				4.57		

High recoveries of gold to gravity concentrate, much higher than those obtained in testing by AMMTEC and Gekko, were achieved. It should be noted that AMMTEC and Gekko work was conducted on portions of the same sample composites, whereas different samples were sent to UNSJ. It is recommended, therefore, that the source of the samples sent to UNSJ is established.

The flotation results from gravity tails originating from the two samples are shown in Table 2.42 and Table 2.43 at natural pH and in Table 2.44 and Table 2.45 at pH 10.5.

Table 2.42 UNSJ Flotation of Oxidised Vein at Natural pH

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution % Au	Cum. Distr. % Au
Con 1	2.7	2.7	29.86	29.86	18.3	18.3
Con 2	1.8	4.4	22.54	26.91	9.3	27.6
Con 3	3.9	8.3	9.35	18.74	8.3	35.9
Tail	91.7	100.0	3.04	4.35	64.1	100.0
Feed Assay				4.21		

Table 2.43 UNSJ Flotation of Unoxidised Vein at Natural pH

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution % Au	Cum. Distr. % Au
Con 1	0.5	0.5	18.22	18.22	4.0	4.0
Con 2	3.3	3.8	13.78	14.38	19.6	23.6
Con 3	2.5	6.3	24.34	18.33	26.3	49.9
Tail	93.7	100.0	1.24	2.32	50.1	100.0
Feed Assay				2.40		

Table 2.44 UNSJ Flotation of Oxidised Vein at pH 10.5

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution % Au	Cum. Distr. % Au
Con 1	6.6	6.6	11.99	11.99	23.0	23.0
Con 2	4.1	10.6	10.62	11.47	12.6	35.6
Con 3	3.7	14.3	8.01	10.57	8.7	44.3
Tail	85.7	100.0	2.22	3.42	55.7	100.0
Feed Assay				3.87		

Table 2.45 UNSJ Flotation of Unoxidised Vein at pH 10.5

Fraction	Mass %	Cum. Mass %	Grade g Au/t	Cum. Grade g Au/t	Distribution % Au	Cum. Distr. % Au
Con 1	3.1	3.1	7.27	7.27	12.6	12.6
Con 2	8.0	11.1	5.48	5.98	24.4	37.0
Con 3	10.5	21.6	2.93	4.50	17.1	54.1
Tail	78.4	100.0	1.05	1.79	45.9	100.0
Feed Assay				2.10		

The flotation and gravity concentration results may be combined to demonstrate the overall effect of gravity followed by flotation, as shown in Table 2.46. The "Recovery to Overall Concentrate" figures are obtained from Table 4 in the UNSJ report.

Table 2.46 UNSJ Combined Results

Sample	Flotation pH	Overall Concentrate Mass %	Recovery to Overall Concentrate % Au	Flotation Tails Grade g Au/t
Oxidised	Natural	8.8	53.2	3.04
Unoxidised	Natural	6.6	80.8	1.24
Oxidised	10.5	14.9	59.4	2.22
Unoxidised	10.5	22.3	82.4	1.05

As would be expected, the unoxidised material performed better in flotation than the oxidised, with approximately 40% of the gold from oxidised vein flotation feed reporting to concentrate compared to approximately 50% from the unoxidised vein material.

The flotation tests at pH 10.5 had much higher mass pulls than those conducted at natural pH. Despite the considerably lower concentrate mass produced at natural pH, (6.6% and 8.8%), these are still too high for direct smelting and would require additional upgrading.

The gold grade of the flotation tails ranges from 1.05 to 3.04 g/t, demonstrating the losses that could be expected from a gravity and flotation circuit.

2.16 Cyanide and Lime Consumption Estimates

Cyanide and lime consumed in the variability tests are listed in Table 2.47, with mean consumptions presented for each ore type. These tests are all at the conditions selected as optimum, whereas the leach optimisation tests performed on the Cal-1 and Cal-2 samples used various leach conditions.

Table 2.47 Cyanide and Lime Consumptions in Variability Tests

Sample No.	Ore Type	Test	Consumption	
			Cyanide kg/t	Lime kg/t
Cal-3	Oxidised vein	Variability	0.53	1.81
Cal-4		Variability	0.60	0.53
Cal-5		Variability	0.58	0.85
Cal-6		Variability	0.60	0.75
Cal-7		Variability	0.60	0.76
Cal-8		Variability	0.92	0.81
Cal-9		Variability	0.60	1.27
Cal-10		Variability	0.60	1.49
Cal-11	Unoxidised vein	Variability	0.38	1.04
Cal-12		Variability	0.18	0.96
Cal-13		Variability	0.88	0.75
Cal-14		Variability	0.35	0.98
Cal-15		Variability	0.70	1.04
Cal-16		Variability	0.66	0.68
Cal-17		Variability	0.20	0.98
Cal-18		Variability	0.84	0.69
Cal-19	Oxidised mineralised andesite	Variability	0.56	1.17
Cal-20		Variability	0.89	1.53
Cal-21		Variability	0.98	1.10
Cal-22		Variability	1.44	2.26
Cal-23	Unoxidised mineralised andesite	Variability	0.69	1.30
Cal-24		Variability	0.74	1.28
Cal-25		Variability	0.59	2.49
Cal-26		Variability	0.62	0.99
Cal-27	Oxidised vein	Variability	0.59	0.83
Cal-28		Variability	0.37	1.08
Cal-29		Variability	0.61	1.07
Cal-30		Variability	0.50	1.17
Cal-31	Unoxidised vein	Variability	0.47	0.53
Cal-32		Variability	0.53	0.65
Cal-33		Variability	0.56	1.05
Cal-34		Variability	0.24	0.53
Cal-35	Oxidised mineralised andesite	Variability	0.83	1.56
Cal-36		Variability	0.63	1.87
Cal-37		Variability	0.62	2.14
Cal-38	Unoxidised mineralised andesite	Variability	0.56	0.86
Cal-39		Variability	0.58	1.49
Cal-40		Variability	0.63	1.13
Oxidised vein average			0.59	1.04
Unoxidised vein average			0.50	0.82
Oxidised mineralised andesite average			0.85	1.66
Unoxidised mineralised andesite average			0.63	1.36

Cyanide consumption in the CIL tests, at 1.80 kg/t and 1.69 kg/t for the Cal-1 and Cal-2 samples respectively, is significantly higher than in all the other tests. This may result from a combination of the high activated carbon concentration and the adopted test procedure. The high carbon concentration was used to capture the anticipated high silver solution values and to provide sufficient carbon mass for the intermediate silver assays. It is also possible that excess air was sparged into the vat during the test, thereby increasing cyanide consumption.

At the time the testwork was conducted, the Merrill Crowe process was expected to be the adopted flowsheet and it was not considered necessary to investigate the cyanide consumption in the CIL tests. Should the CIL process continue to be attractive, it is recommended that further testing be conducted to gain a better understanding of cyanide consumption in the presence of activated carbon.

For the purpose of this IFS Ausenco has assumed that cyanide consumption in the CIL plant is equivalent to the averages shown in Table 2.47, plus 20% for scale up and the presence of activated carbon. In addition to the cyanide consumed, a residual cyanide concentration of 150 ppm has been assumed in the tailings thickener underflow solution. The predicted cyanide consumption for the main types of material is shown in Table 2.48.

Throughout the leach variability tests, AMMTEC used metallurgical grade lime having 60% CaO. The predicted lime consumption (60% CaO) is equal to the average consumptions also shown in Table 2.48.

Table 2.48 Predicted Cyanide and Lime Consumptions

	Predicted Cyanide Consumption kg/t	Predicted Lime Consumption (60% CaO) kg/t
Oxidised vein	0.8	1.04
Unoxidised vein	0.7	0.82
Oxidised mineralised andesite	1.1	1.66
Unoxidised mineralised andesite	0.9	1.36

2.17 Gold and Silver Recoveries

The leach extractions obtained in the variability tests, which were all conducted under the selected leach conditions, were used to predict gold and silver recoveries. The leach optimisation phase of the testwork programme, conducted on samples Cal-1 and Cal-2, assessed differing leach conditions, rather than optimised conditions, and results from this phase were not used in the development of the grade recovery relationships.

The following relationships were examined:

- Grade and 48 hour leach extraction for gold by ore type.
- Grade and 48 hour leach extraction for silver by ore type.
- Grade and 48 hour leach extraction for gold for all variability samples.
- Grade and 48 hour leach extraction for silver for all variability samples.
- Grade and 48 hour leach residue for gold by ore type.

- Grade and 48 hour leach residue for silver by ore type.
- Grade and 48 hour leach residue for gold for all variability samples.
- Grade and 48 hour leach residue for silver for all variability samples.

All grade versus leach extraction relationships showed relatively poor correlation. The grade versus leach residue for gold for all variability samples showed better correlation than grade versus leach residue for gold by ore type, perhaps because of a larger data population. The grade versus leach residue for silver by ore type and grade versus leach residue for silver for all variability samples show similar correlations. It was decided to use the grade versus leach residue for all variability samples for both gold and silver for the purpose of predicting gold and silver recoveries. Figure 2.5 and Figure 2.6 illustrate the grade versus leach residue relationships for gold and silver, respectively.

Figure 2.5 Grade vs Leach Residue Relationship for Gold

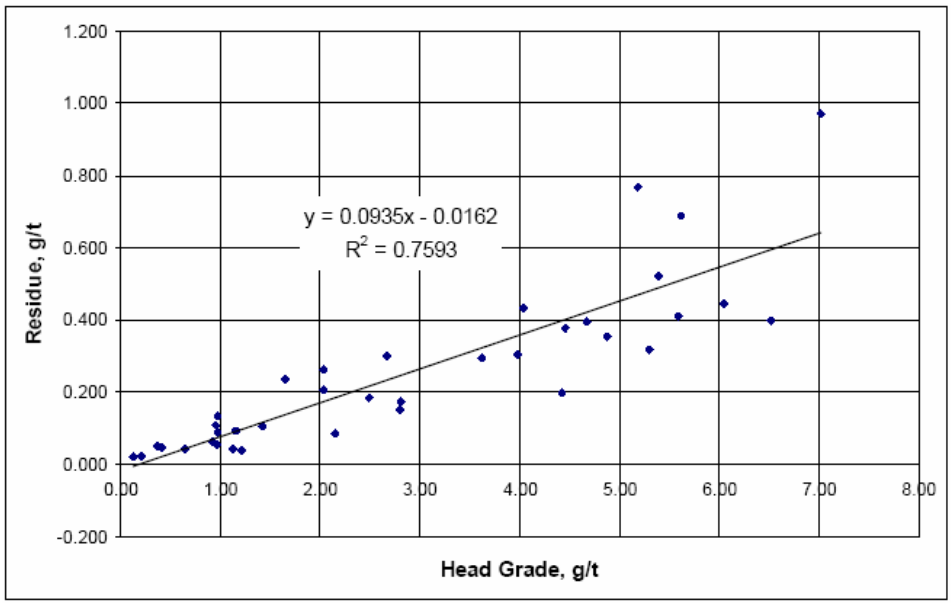
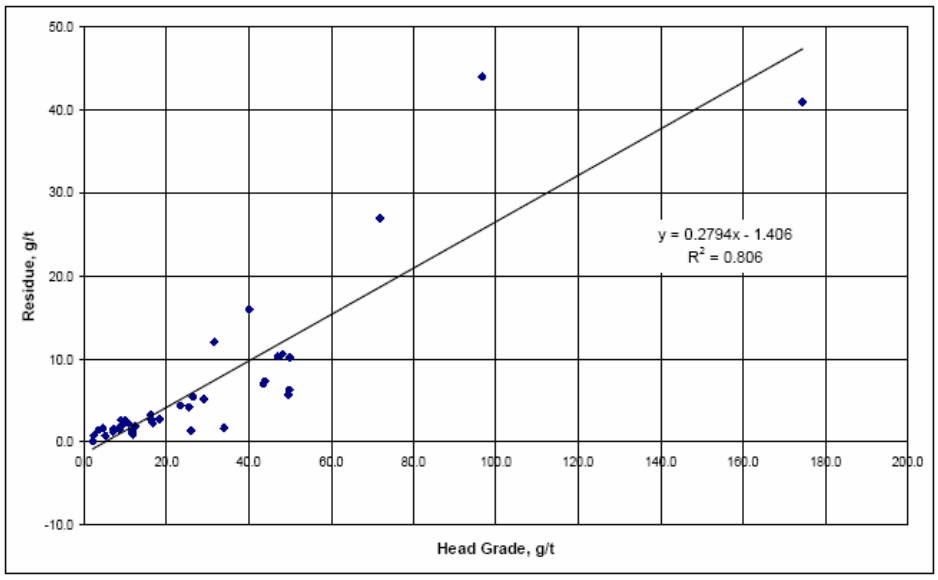


Figure 2.6 Grade vs Leach Residue Relationship for Silver



The following relationships have been developed from the variability tests results using linear regression:

Au leach residue = (0.0935 x Au head grade) – 0.0162

Ag leach residue = (0.2794 x Ag head grade) – 1.406

Recoveries predicted using the above grade versus leach residue relationships should be discounted by 1% for gold and 2% for silver to cater for scale-up and solution losses to tailings. Based on the test results from leaching the variability samples, the 48 hour leach residues for the plant may therefore be predicted by the following relationships:

Au leach residue = (0.1035 x Au head grade) – 0.0162

Ag leach residue = (0.2994 x Ag head grade) – 1.406

2.18 Recommendations for Further Work

If the CIL process remains attractive for the project, then further testwork is required to investigate the reasons for the high cyanide consumption in the AMMTEC CIL tests. An examination of the silver kinetics, in the presence of activated carbon, may indicate an opportunity for leach residence time reduction. Predictions of gold and silver recovery will require review following this work.

Settling characteristics of many of the samples tested were poor and additional thickening testwork is required at the selected grind.

The source of the samples sent to San Juan needs to be identified and the reason for higher gravity recoveries than those obtained at AMMTEC and Gekko needs to be investigated.

3 PROCESS PLANT

3.1 General

The Calcatreu process plant and associated service facilities will receive run-of-mine (ROM) ore and, through a series of processing steps, produce gold and silver bullion doré. The proposed process encompasses crushing and grinding of the ROM ore, followed by leaching of precious metals into a dilute cyanide solution, adsorbing the precious metals onto activated carbon and then stripping and recovery of precious metals from the carbon to produce doré. The barren tails will be further treated in a SO_2/air process to destroy the weak acid dissociable cyanide (CN_{WAD}) before placement in the tailings storage facility (TSF).

3.2 Flowsheet Design Basis

The proposed process plant design is based on a metallurgical flowsheet with unit operations that are well proven in the gold industry.

The key criteria for equipment selection have been the suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a compact footprint. The key project and ore specific criteria for the plant design are:

- Treatment of 750,000 dry tonnes per year of the unoxidised vein ore type.
- Operation of the plant on a 24 hours per day, 7 days per week basis.
- Design availability of 91.3%, (8,000 operating hours per year).
- Inclusion of standby equipment in critical areas within the circuit.
- Sufficiently automated plant control to minimise the need for operator interface on a continuous basis but allow manual override and control if required.
- Enclosure of the grinding mills, elution circuit, reagent mixing and associated equipment in a building to facilitate operation and maintenance during the prolonged periods of high winds that characterise the local climate.

Sections 3.3, 3.4 and 3.5 outline the flowsheet development and describe the plant processes and plant layout. These sections should be read in conjunction with the following documents:

- Simplified Block Flow Diagram 1497-F-100 (Appendix 3).
- Process Flow Diagrams (Flowsheets) and General Arrangement Drawings provided in Appendix 9.
- Process Design Criteria (Appendix 3).
- Mechanical Equipment List (Appendix 3).

3.3 Flowsheet Development

The process plant design incorporates the following unit process operations:

- Single stage crushing using a jaw crusher to produce a crushed product of a size 80% passing (P_{80}) 120 mm.
- Semi-autogenous grinding (SAG) mill/ball mill circuit (SAB circuit). configured as an open circuit SAG mill and closed circuit ball mill in series to produce a milled product of a size 80% passing (P_{80}) of 75 µm.
- Carbon-in-leach (CIL) circuit consisting of two leach tanks followed by six adsorption tanks in series.
- Anglo American Research Laboratories (AARL) type elution circuit (excluding electrowinning).
- Precipitation of gold and silver from pregnant elution solution using zinc, followed with gold and silver recovery to doré.
- Thickening of leach residue and cyanide destruction of the thickened slurry using the INCO SO_2/air process.

3.3.1 Comminution Circuit

The key A x b parameters, obtained from SMC tests and shown in Table 3.1 indicate the oxidised vein ore as being “moderately competent” and the unoxidised vein ore as “competent”. Both ores are suited to SAG milling, with the unoxidised vein ore more likely to result in a build-up of critical size material in the SAG mill. The possible build-up of this critical size material may warrant the inclusion of recycle crushing. The suitability of the milling circuit with recycle pebble crushing (SABC) and without recycle pebble crushing (SAB) was reviewed, with the report included in Appendix 3. The report concluded that an SAB circuit, with provision to recycle up to 25% of cyclone underflow to SAG mill to balance power, was the preferred circuit. The need for the recycle crusher is negated by a marginal increase in diameter and length of the SAG mill and allows commonality of motor and gearbox capital spares. If ores harder than those tested are encountered in significant quantity, a recycle crusher could be retrofitted to maintain throughput.

The crushing and milling circuits will be designed to operate continuously at a throughput of 94 t/h of unoxidised vein ore, giving an annual capacity of 750,000 tonnes at the design availability of 91.3%.

Grind optimisation testwork was carried out on the oxidised vein and unoxidised vein composites. The samples were ground to five different sizes, P_{80} of 38, 58, 75, 106, and 150 µm, and leached for 48 hours. Solution samples were taken at 24 hours and 48 hours to obtain a leach curve for each grind size. The tailings from each test were sized to obtain a true sizing. The testwork indicated that gold and silver recoveries are related to grind size, with recoveries increasing with fineness of grind size. An economic analysis indicated that the optimum grind sizes for the oxidised vein and unoxidised vein ore samples were the same at a P_{80} of 75 µm. This grind size was selected for design.

ROM Ore Stockpile

The ROM bin has been designed to be fed by both front end loader and by direct tipping from haul truck. However, the operating cost estimate has been based on 100 % rehandle of ROM ore by front end loader. The ROM stockpile will enable a considerable degree of blending using multiple fingers.

Crusher Sizing

A 950 x 1,250 mm single toggle jaw crusher has been selected based on the maximum lump size passing the 550 mm aperture parallel bar grizzly. It is proposed that the crusher will operate with a closed side setting (CSS) of 100 mm to produce a crushed product with a P_{100} of 150 mm and a P80 of approximately 120 mm.

Crushed Ore Stockpile

The prevailing wind conditions encountered at site will cause considerable dust problems around a crushed ore stockpile. A study of seventeen single stage crushing/milling plants showed no significant difference in plant availability between circuits with or without a primary crushed ore stockpile, and thus no primary crushed ore stockpile has been included.

Mill Sizing

The key comminution parameters determined by testwork and used for mill sizing are summarised in Table 3.1.

Table 3.1 Mill Design Criteria

Design Criteria	Units	Oxidised Vein	Unoxidised Vein	Design Value	Source
Mill Capacity	t/h			93.8	Aquiline
Feed Size – 80% passing (F80)	mm			117	Ausenco
Product Size – 80% passing (P_{80})	µm			75	Testwork
Max. Crushing Work Index (CWI)	kWh/t	n/a	24.8	24.8	Testwork
Bond Rod Work Index (RWI)	kWh/t	18.9-23.6	13.4-25.8	22.9	Testwork
Bond Ball Work Index (BWI)	kWh/t	17.1-21.4	13.5-22.3	20.8	Testwork
JK – A		73.6	88.8	88.8	Testwork
JK – b		0.70	0.45	0.45	Testwork
JK – A x b		51.5	40	40	Testwork
Abrasion Index - Ai		0.56-0.67	0.28-0.83	0.64	Testwork
Ore SG		2.64	2.65	2.65	Testwork

The design values selected for the Bond rod mill, ball mill and abrasion indices, represent the 80th percentile of the values recorded during the testwork for the combined oxidised vein and unoxidised vein samples. The unoxidised vein A and b parameters were selected for design.

The grinding power demand using the criteria shown in Table 3.1 was determined using Ausenco's in house grinding circuit design program. The results are summarised in Table 3.2.

Table 3.2 SAB Mill Grinding Power Demand

Criteria		Units	Design Value
Pinion Power	SAG Mill	kW	1,381
	Ball Mill	kW	1,542
Specific Energy	SAG Mill	kWh/t	10.2
	Ball Mill	kWh/t	20.0
Total Corrected Circuit Power		kWh/t	30.2

The dimensions and specifications of the selected SAG and ball mills are summarised in Table 3.3.

Table 3.3 SAG and Ball Mill Sizing

Criteria	Units	SAG Mill	Ball Mill
Diameter -Inside Shell	m	5.94	4.11
Length -Effective Grinding Length	m	3.67	6.17
L:D Ratio	m	0.65	1.5
Discharge Arrangement		Grate	Overflow
Design Mill Speed	% Nc	60 to 80	75
Liner Thickness -New	mm	105	75
-Design NominalBall Charge	% vol	8	35
-Maximum	% vol	15	40
Total Load -Design	% vol	26	N/A
Pinion Power -Design	kW	1,381	1,582
Installed Power	kW	1,750	1,750

The SAG mill sizing has been based on a normal ball charge of 8% and a total charge of 26% of mill volume.

Classification

A mill mounted trommel screen will provide coarse classification around the SAG mill. In line with conventional practice, hydrocyclones have been selected for the fine classification duty. The pulps displayed normal viscosity characteristics, thus the hydrocyclones have been sized to operate at an overflow density of 40% w/w solids for direct feed to the leach circuit.

3.3.2 Metal Extraction Circuit

A technical and financial evaluation of three circuit types for the recovery of precious metal from ore was performed by Ausenco. The case study is included in Appendix 1. The circuits considered were:

- CIL followed by zinc precipitation for recovery of precious metals from the pregnant eluate.
- Merrill Crowe.
- A hybrid CIL – Merrill Crowe Circuit.

The conclusions drawn from the case study review are summarised below.

Option 1 – CIL with Zinc Precipitation

The CIL circuit considered in the evaluation consisted of two leach and six adsorption tanks, an AARL elution circuit, and zinc precipitation of precious metals from pregnant eluate.

The CIL circuit has the lowest capital cost of the options considered, is not sensitive to feed rate fluctuations (up to the maximum inter-stage screen flow rate) and offers potentially higher silver extractions. However, the CIL circuit has the highest operating cost of the options considered and it has the largest inventory of precious metals held in circuit due to the large carbon inventories required to ensure adequate adsorption kinetics.

Key process risks for the CIL circuit are the ability of the circuit to achieve the design carbon loadings, and insufficient carbon inventory to recover gold/silver if higher leach extractions and/or lower than design adsorption rates are encountered.

Option 2 – Merrill Crowe

The Merrill Crowe circuit considered in the evaluation consisted of a pre-leach thickener, six leach tanks, a four stage counter current decantation (CCD) circuit, clarification, de-aeration and recovery of gold and silver from pregnant leach solution by precipitation with zinc dust.

The Merrill Crowe circuit is best suited to treating ores with spikes of high silver and has the lowest inventory of precious metals in circuit. It has the highest capital cost and lowest operating cost of the three circuit options considered.

The main risk identified with this circuit is the sensitivity of CCD performance to fluctuations in plant feed rate and to the processing of ores with poor settling characteristics. Surging feed or reduced settling rates may result in poor CCD circuit performance and a “dirty” overflow, which may overload the clarifier filters. The risk of this occurring is real because of the direct mill feed from crusher and the observation in variability testwork of ores with poor settling characteristics.

Option 3 – Hybrid CIL-Merrill Crowe

The hybrid circuit consists of two leach tanks, a leach discharge thickener, six adsorption tanks, an AARL elution circuit and a clarification, de-aeration and zinc precipitation circuit for the combined leach discharge thickener overflow and pregnant eluate solutions.

The hybrid circuit is complex with more unit operations than the other two options and combines the advantages, disadvantages, and process risks of both circuits.

Circuit Type and Size

A CIL circuit was selected on the basis that it has a lower risk when processing ores with poor settling characteristics, has potential for higher silver extractions, is less susceptible to variations in feed rate, and has the lowest capital cost. The net present cost (NPC), a measure of cash flow over the project life (less than 5 years) is not significantly different to that of the Merrill Crowe circuit which had the lowest total cost. Any change to the mine plan may alter the economics of the various circuit options necessitating a re¬validation of the optimum processing route.

The CIL circuit will have two leach tanks and six adsorption tanks, with the leach tanks included to provide additional leach residence time and higher solution tenors during initial contact with the activated carbon in the adsorption circuit.

The six adsorption tanks are considered the optimum number given the carbon adsorption kinetics determined from the testwork and the need to maintain acceptable performance in the event of an adsorption tank being bypassed. Average carbon concentrations of between 20 g/L and 25 g/L are required in all tanks, particularly the first and last, to ensure high gold and silver adsorption efficiencies and a low solution tail.

All tanks will be identical in size, with their individual volumes of 1,100 m^3 giving a total circuit plug flow residence time of 48 hrs at 40% w/w solids pulp density.

Leach Oxygenation and Agitation

The ores displayed a low oxygen uptake rate. Testwork showed that oxygen injection enhanced both leach kinetics and total extraction over those achieved by straightforward aeration.

High shear oxygenation devices utilising gaseous oxygen, such as the Filblast or Multi Mix Systems, have proven to be an economically attractive means of achieving efficient oxygen injection, but are only available on a lease basis. A device of this type has been included, with slurry recycle from the base of the leach tank, and an oxygen supply from a 1 t/d pressure swing adsorption (PSA) oxygen plant.

The normal viscosity characteristics of the pulps allow use of conventional dual impeller agitators of moderate power to achieve even suspension of solids and sufficient shear action to achieve satisfactory carbon adsorption kinetics.

Elution and Precipitation

A split AARL elution circuit has been selected on the basis that: (a) the raw water available has too high total dissolved solids content necessitating the use of potable water; and (b) there may be a requirement to increase the number of elution cycles to 14 per week for “catch-up”.

An 8 tonne batch size has been nominated and, based on the calculated carbon movements, a total of 10 to 11 elution cycles are required each week. To enable continued processing at peak head grades, the circuit will be designed for up to 14 strips per week.

The goldroom will be operated 7 days a week.

Pregnant eluate will be pumped to the barren eluate tank via one of two precipitation plate filters (one filter in use while the other is cleaned). Lead nitrate and zinc dust will be added to the pregnant solution prior to the filters. The zinc dust precipitates gold and silver from solution which is then retained by the filter. Diatomaceous earth is added to

enhance the filter rate of the precipitate. The precipitate filters will be designed to hold 36 hours of precipitate at peak gold and silver production.

When a filter is full of precipitate it will be taken offline, and the precipitate recovered. The filter cake is then oven dried prior to smelting.

Cyanide Destruction and Tails Disposal

A 16 m diameter high rate thickener has been included in the process flowsheet ahead of the cyanide destruction circuit to reduce the volume of slurry requiring detoxification and to recycle cyanide water back to the mill. This reduces both the mass of cyanide requiring detoxification and the overall cyanide consumption. The tails thickener has been sized on the design settling rate of 0.60 t/m^2h and solids feed rate of 94 t/h. A further 28% is added to the settling area to allow for surges in solids flow, to ensure high underflow slurry density and minimum flocculant use.

The design of the cyanide destruction circuit has been based on the INCO process (oxidation of cyanide to cyanate by SO_2) a relatively simple and inexpensive process with a proven track record at over 80 mining sites worldwide. A free cyanide concentration of 150 mg/L in the CIL residue slurry discharging from the final adsorption tank has been assumed, with a target average CN_{WAD} concentration of 50 mg/L in the slurry discharging from the circuit. This final cyanide concentration complies with the International Cyanide Management Code recommendations (CN_{WAD} <50 mg/L).

The circuit design comprises two tanks in series, each with a nominal 45 minutes residence time. This provides sufficient redundancy such that the plant can operate at near design capacity with one tank offline for maintenance and still achieve the target cyanide concentration in the final tailings slurry.

The tank residence time, aeration rates and reagent consumptions have been calculated using industry standard values and ratios, which have been confirmed by testwork.

3.4 Process Design Criteria Summary

The Process Design Criteria report is included in Appendix 3. The report should be read in conjunction with the Process Flow Diagrams and the Mass Balance data. A summary of the design criteria underlying the equipment sizing and operating costs is provided in Table 3.4.

Table 3.4 Summary of the Process Plant Design Criteria

Criteria		Units	Value
Ore Throughput		t/y	750,000
		t/h	93.8
Plant Availability	Design	%	91.3
Gold Grade	Design	g Au/t	3.86
Silver grade	Design	g Ag/t	33.2
Physical Characteristics	BWI – vein	kWh/t	20.8
	RWI – vein	kWh/t	22.9
	AI		0.64
Grind Size	P_{80}	µm	75
Plant Recovery	Gold	%	90
	Silver	%	74
Leach/CIL Residence Time		h	48
Residual Free Cyanide Level Concentration		mg/L	150
CN Destruction Residence Time		minutes	60
Aeration Rate		Nm^3/t	1
Residual CN_{WAD} Level Concentration		mg/L	< 50
Oxygen Requirement (100% O_2)	Total	kg/t	0.11
Flocculant Consumption		kg/t	0.040
Cyanide Consumption (100% NaCN)		kg/t	1.1
CIL Lime Consumption (90% avail. CaO)		kg/t	1.66
CN Destruction Lime Consumption (90% avail. CaO)		kg/t	0.69
Sodium Metabisulphite Consumption (100% SMBS)		kg/t	0.78
Copper Sulphate (as pentahydrate)		kg/t	0.03

The selected design reagent consumptions reflect the testwork data for the highest consuming ore types. These values were chosen for design of reagent handling systems. The plant actual consumptions will be dependent of ratio of the various ores being fed to the plant.

3.5 Process Description

3.5.1 Crushing and Mill Feed

Refer to Flowsheet 1497-F-101

Stockpiled ROM ore will all be reclaimed from the ROM pad by a Cat 980 or equivalent front-end loader (FEL) and fed to the 80 tonne capacity ROM bin, which will be fitted with substantial “hungry boards”. A standby FEL, Cat 980 or equivalent, will ensure constant feed availability. The ROM bin will be equipped with a 550 mm aperture parallel bar static grizzly sloped to bring oversize back to the loading side for immediate removal by the

same FEL. Oversize may be broken on the ROM pad area by a rock breaker. The design of the ROM bin will permit direct dumping of ROM ore by the mine haul trucks.

A 300 mm static grizzly will be located on the ROM pad, for the purpose of generating an emergency feed stockpile. ROM ore containing a higher fines fraction will be selectively fed to the static grizzly by FEL. The grizzly oversize will be returned to the ROM stockpile, while the grizzly undersize will be trammed to a stockpile adjacent to the SAG mill emergency feed hopper.

Ore will be drawn from the ROM bin via a 1,524 mm x 9,000 mm variable speed apron feeder into a 950 mm x 1,250 mm single toggle jaw crusher operating with a closed side setting (CSS) of 100 mm.

The ROM bin and apron feeder will be mounted on an elevated, hollow concrete foundation also serving as a retaining wall. An elevated, composite steel/concrete bridge, supported at one end on a deadman anchored concrete foundation set at the edge of the ROM pad and at the other end on the ROM bin itself, will provide access for the FEL. The concrete primary crusher foundations will be an integral extension of the bin and feeder support structure with columns along the outer edge to provide generous access to the area beneath the crusher.

The primary crusher product will discharge onto the 900 mm wide primary crusher sacrificial conveyor, which will discharge to a 900 mm wide mill feed conveyor belt, fitted with a weightometer, which in turn will discharge into the SAG mill. In common with all other conveyors on the project, the conveyor will include:

- Proprietary idlers, including rubber impact at 300 mm centres at load points, carry and return idlers and at least two sets of steering idlers on carry and return strands.
- Crowned and rubber lagged drive, tail, take-up and idler pulleys.
- Fabric reinforced belts.
- Shaft mounted drive gearboxes, run-backs, fluid couplings, vee belt drives and gearbox mounted motors, all with appropriate guards.
- Hooded load areas with proprietary adjustable skirt systems insertable filters where specified.
- Shedder plates at load points.
- Head pulley scrapers (belle banne or equivalent).
- Return belt scrapers at take-ups and tail pulleys.
- Head, tail end and pinch point guards as per regulations.
- Maintenance access as per regulations.
- Under speed, belt drift, chute blockage and pull wire switches.
- Feed and discharge chutes and skirts with plug welded wear plates of QT360 or equivalent.

Dust suppression sprays will be installed at the ROM bin, and on the SAG mill feed conveyor.

No surge capacity between the crusher and SAG mill has been provided. An emergency feed hopper located on the SAG mill feed conveyor will allow for continuity of feed to the SAG mill during periods of crusher downtime. The hopper will be fed by a FEL using ore from the emergency feed stockpile.

3.5.2 Grinding Circuit

Refer to Flowsheet 1497-F-102

The grinding circuit comprises a single open circuit SAG mill and a ball mill in closed circuit with cyclones to produce a final product. The SAG mill will be a 5.94 m diameter x

3.67 m effective grinding length, grate discharge, steel lined mill driven through a gearbox by a 1750 kW wound rotor induction motor and will operate with a nominal ball charge of 8% of mill volume and a total charge of 26% of mill volume. The mill speed will be variable over a range of 60% to 80% critical speed (Nc) by varying the rotor resistance with the liquid resistance starter.

The SAG mill will discharge over a 12 mm aperture trommel screen, and the trommel oversize (pebbles and steel scats) will be discharged onto a series of two conveyors that will recycle the pebbles back to the SAG mill feed conveyor belt. The pebble recycle conveyors will be equipped with a weightometer to indicate the mass of pebbles being recycled. The pebbles will discharge back onto the SAG mill feed conveyor belt.

The SAG mill trommel undersize slurry will gravitate to the mill discharge hopper where it is diluted with process water and pumped (1 duty/1 standby) to a cluster of hydrocyclones for classification. The cluster will consist of 5 x 380 mm hydrocyclones inclusive of one spare hydrocyclone. The overflow from the hydrocyclones (ground product) will gravitate to a trash removal screen prior to the CIL circuit while the hydrocyclone underflow will be split with the majority gravitating to the ball mill and the remainder recycled back to the SAG mill for further grinding.

The ball mill, will be a 4.11 m diameter x 6.17 m effective grinding length, overflow discharge, rubber lined mill driven through a gearbox by a 1,750 kW fixed speed wound rotor induction motor and will operate with a nominal ball charge of 35% of mill volume. The ball mill will discharge over a 10 mm aperture trommel screen with the oversize discharging into a bunker for regular collection by a FEL or skid steer loader. The undersize slurry will gravitate to the combined mill discharge hopper.

The mills will be supported on a common foundation consisting of a large concrete plinth with integral pedestals to support the mill bearings and drive trains. The mills will be mounted to give the same longitudinal and mill discharge bearing centre lines allowing a chute arrangement for the mill discharge slurry to easily flow to the common discharge hopper. The mill discharge pumps (1 duty/1 standby) are located centrally between the mills and in towards the centre of the main process building to allow easy access for the overhead crane to be used for maintenance.

Process water will be added to the mill feed chutes to achieve the required pulp density in the mills for maximum grinding efficiency. Steel balls will be added to the mills via the ball loading chutes which are integral to the mill feed chutes and allow balls to be directly dumped into the mill feed chute to maintain the desired ball loads and mill power draw.

Milk of lime slurry will be added to the SAG mill feed chute through a controlled valve fed by the milk of lime ring main. Lime is added to increase slurry pH ahead of the CIL circuit.

The complete area beneath the mills, incorporating the mill discharge pump area, will have a concrete slab and be bunded to contain spillage. A spillage pump will be provided in the mill bunded area to reclaim any spillage and return it back to the mill discharge hopper. The spillage pump will be located in a sump at the low point in the mill bund.

A 3 axis, 500 kg mill liner handler has been included in the plant to facilitate removal and installation of SAG mill liner plates and lifter bars during planned mill relines. The liner handler will be stored on the concrete work platform that forms the roof of the MCC and be relocated into working position when required for a reline by the building overhead crane. The liners will be positioned on the concrete work platform by the overhead crane so that the liner handler can lift them from the transport pallet and position them into the mill.

3.5.3 CIL Circuit

Refer to Flowsheet 1497-F-103.

The cyclone overflow pulp with a nominal pulp density of 40% w/w solids will gravitate to the vibrating trash removal screen to remove any debris from the slurry ahead of the hybrid CIL circuit. The cyclone cluster is situated adjacent to the first tank at a height sufficient to allow gravity flow to the trash screen and subsequently to the leach tank via a launder. The trash that is removed will gravitate to a trash hopper situated at ground level near to the first CIL tank.

The hybrid CIL circuit will consist of two 1,100 m^3 (live) leach tanks and six (6) 1,100 m^3 (live) adsorption tanks in series interconnected with launders to allow the slurry to flow by gravity through the tank train. Each tank will be fitted with a mechanical agitator to ensure uniform mixing. The tanks will be located on circular concrete ring beams which will be in filled with a sand/oil mixture to prevent moisture from contacting the underside of the tank base. A layer of bitumastic board will be placed between the tank floor and the concrete ring beam along with a bead of silastic at the floor edge to ensure that ingress of slurry or water to the underside of the tank floor area is prevented.

Oxygen will be injected into the leach slurry by drawing slurry from the base of the leach tank, pumping it through a high shear reactor and returning the slurry back to the top of the leach tank.

Each adsorption tank will be fitted with a mechanically swept, woven wire inter-tank screen to retain carbon. A traveling gantry hoist will be used to remove the screens for maintenance and routine cleaning. All tanks will be fitted with bypass launders to allow any tank to be removed from service for agitator or screen maintenance. The top of tanks will have grid mesh covering a substantial portion of the tank and intermediate tank areas to allow a large platform for access to the mechanical items for operations and routine and breakdown maintenance. Removable sections of the grid mesh will allow access to the launders for insertion and removal of the blanking plates used to bypass a tank that may need to be temporarily removed from service. A drop down area between the structural steelwork at the tank train will allow an inter-tank screen to be lowered for cleaning and maintenance. A spare inter-tank screen will be provided to allow this screen cleaning and maintenance to be carried out on a regular basis.

Barren carbon will enter the adsorption circuit at the last tank (CIL tank 6) and will be advanced counter-current to the slurry flow by moving slurry and carbon from CIL tank 6 to CIL tank 5 using recessed impeller carbon transfer pumps. The inter-tank screen in CIL tank 5 will retain the carbon and the slurry will flow by gravity back to CIL tank 6. This counter-current process will be repeated until the carbon reaches CIL tank 1. A recessed

impeller pump will be used to transfer slurry to a loaded carbon recovery screen mounted above the acid wash column in the stripping plant. The carbon reporting as screen oversize will gravitate to the acid wash column and the slurry will return to CIL tank 1.

Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened over a vibrating screen to remove fine carbon, which is discarded to the CIL tailings hopper. The sized and regenerated carbon will report directly to CIL tank 6.

Sodium cyanide solution will be metered into the leach tanks and CIL tanks 1 and 2 via a ring main system.

Provision has been made in the layout and piping runs to allow milk of lime slurry to be added to the leach tanks for pH control.

The CIL area, including the areas for inter-tank screen maintenance, under the cyclones and trash screen, and under the carbon safety screen will have a concrete floor and bunds to retain any overflow or spillage. The included volume of the bunded area will be sufficient to hold a minimum of one tank volume in case of tank rupture. The area will be provided with two spillage pumps which will deliver spillage back to the first leach tank. If the leach tank is off-line, the spillage pump delivery will be diverted to an adjacent tank using a hose.

Tailings slurry from the last CIL tank will gravitate to the vibrating carbon safety screen to recover any carbon which may be present following damage, wear or incorrect installation of the final stage inter-stage screen. Carbon recovered on the screen will be delivered to a bulk bag for reuse. Tailings discharging from the tailings screen will gravitate to the tailings thickener.

3.5.4 Elution and Precipitation

Refer to Flowsheets 1497-F-104 to F-105

The following operations will be carried out in the elution and goldroom areas:

- Acid washing of carbon.
- Stripping of gold and silver from loaded carbon using a split AARL elution circuit.
- Precipitation of gold and silver from pregnant solution using zinc.
- Filtering of precious metals precipitate.
- Smelting of precipitation products.

The elution and goldroom areas will operate up to 7 days a week, with the loaded carbon recovery on night shift and the majority of the elution occurring during day shift. The 8 tonne AARL elution circuit will consist of separate rubber lined acid wash column and stainless steel elution column and will accommodate a maximum throughput of 112 tonne/week of loaded carbon.

Acid Wash

Loaded carbon will be recovered on the loaded carbon recovery screen and directed to the acid wash column. Transfer and fill operations will be controlled manually. All other aspects of the acid wash and the pumping sequence will be automated.

Acid washing of the carbon will commence after carbon transfer is complete.

Concentrated hydrochloric acid (33% w/v HCl) will be pumped from the storage tank and injected into the fresh water line at the base of the acid wash column to produce a dilute acid wash solution approximating, 3% w/w hydrochloric acid (HCl) in water.

During acid washing the dilute HCl solution will be circulated through the column in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. This process improves the elution efficiency and has the beneficial effect of reducing the risk of calcium-magnesium 'slagging' within the carbon during the regeneration process.

Following the acid wash cycle the carbon bed will be rinsed with fresh water. Four bed volumes of fresh water will be pumped through the column to displace any residual acid from the carbon. The residual spent acid and rinse water will be disposed of directly to the tails hopper. The acid-washed carbon will then be hydraulically transferred to the elution column using high pressure raw water.

The acid wash column will be located between the main process building and the CIL tank train in a steel structure that will provide access for operations and maintenance. An acid proofed concrete floor and bund will be provided under the acid wash column area to ensure all spillage is captured. This area will flow into the CIL bunded area and spillage will be dealt with by the CIL sump pump which will deliver any spillage back to the first leach tank.

Elution

Lean solution from the lean eluate tank will be pumped through a recuperative heat exchanger and indirect heater in series to heat it to 130°C before entering the base of the elution column. Sodium hydroxide and sodium cyanide solutions will be dosed into the discharge line of the strip solution pump to achieve a 2% w/v concentration of both reagents.

A recuperative heat exchanger will recover a proportion of the heat from the return eluate thereby preheating the incoming eluant. A diesel-fired heater will heat a thermal fluid which is then circulated through a second heat exchanger to raise the temperature of the incoming barren solution to a nominal 130°C.

The heated solution will then flow into the base of the column filling the interstitial voids in the carbon bed. The hot solution will be circulated around the column for a period of 30 to 40 minutes during which time the gold and silver will be eluted from the loaded carbon. After this period has expired, heated solution from the lean eluate tank will continue to be pumped into the base of the column, in a plug flow regime, to flush the pregnant solution from the carbon bed. The pregnant solution exiting the top of the column will be cooled through the recuperative heat exchanger, used to pre-heat the incoming solution, before gravitating to the eluate tank. After four bed volumes have been removed from the column, heated potable water is pumped from the break tank, into the base of the elution column and continues for a further 4 bed volumes. During this cycle the solution discharging the column is diverted to the lean eluate tank for use in the next strip.

The elution column is situated adjacent to the acid wash column in the same structure between the main process building and the CIL tanks. As for the acid wash column, a separate bunded area will be provided to contain spillage. The pregnant eluate tank will be situated outside the main building adjacent to the reagents annex. The pregnant eluate tank will have a solid concrete plinth surrounded by a concrete slab and separate bund to contain spillage. A sump will be contained within the bunded area so that a portable sump pump can be used to remove spillage and/or rain water and depending on

the contents of the bunded area it will be directed to different process areas. The break tank, lean eluate tank, elution heater, and heat exchangers are located on the ground floor in the area adjacent to the mills and the gold room. A separate bunded area with concrete floor is provided to contain spillage and allow easy cleanup. All spillage and cleanup liquids are pumped by a sump pump back to the leach circuit. Pipe supports for the required piping are provided by simple attachments to adjacent steelwork.

Precipitation and Goldroom

Precious metals will be recovered from the pregnant solution using zinc cementation.

Zinc dust will be metered into the gold and silver rich solution as it is pumped from the pregnant eluate tank through a precipitation filter. Gold and silver will precipitate on contact with the zinc and collect in the precipitation filter. Lead nitrate will be added during precipitation to act as a catalyst for precipitation. Diatomaceous earth will be added to pre-coat the filters before every batch and act as a body aid, to maintain filter cake porosity. Filtrate solution, barren of gold and silver, will discharge into the barren solution tank, from where it will be used for mixing with zinc dust, lead nitrate and diatomaceous earth, or pumped back to the CIL circuit for disposal.

Diatomaceous earth will be added as a solid into the pre-coat mixing tank and mixed with barren solution to obtain a preset slurry density. The slurry will then be circulated through the filter press, with any displaced solution being returned to the pre-coat mixing tank, in order to pre-coat the filter before the introduction of pregnant solution.

Zinc dust, lead nitrate and diatomaceous earth will be metered into a zinc cone, and wetted with barren solution prior to dosing into the pregnant solution close to the filter feed pump suction. Flow from the zinc cone will be distributed between the operating filters by means of valves.

The precipitation filter feed pumps will have submerged seals to prevent air ingress into solution, which adversely affects the precipitation process.

Two recessed plate filter presses will be used for the precipitation filtration duty, one operating, while the other is off line for harvesting and preparation for the next batch. Each filter will be designed for 36 hours production at design maximum grades.

Filter cake harvested from the precipitate filters will be collected in trays. The trays will be provided with castors to assist with their withdrawal from beneath the filters and with lifting points for a fork-lift. The filter cake will be dried in a drying oven before direct smelting with fluxes in a diesel-fired furnace. The smelted precious metals will be poured into a cascade of five 600 oz doré moulds. The drying oven and the furnace will be located in the ground floor area of the secure gold room and designed for a maximum of two smelts per day, seven days per week.

Separate fume extraction equipment will be provided to remove gases from the drying oven and barring furnace and exhaust to atmosphere.

Doré bars will be cleaned, weighed, stamped, sampled and then stored in a safe while awaiting dispatch.

The gold room will have a concrete floor complete with a sump trap overflowing to a sump. A sump pump will pump all wash down and spillage liquor back to the leach circuit. All heavy materials collected in the sump trap will be returned to the drying oven and furnace if required.

Carbon Regeneration

After completion of the elution process, the barren carbon will be hydraulically transferred from the elution column over the carbon dewatering screen to the feed hopper of the horizontal carbon regeneration kiln. In the kiln feed hopper any residual and interstitial water will be drained from the carbon before it enters the kiln.

The carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched in water and then pumped, using a recessed impeller pump, to the carbon sizing screen. The screen oversize (regenerated, sized carbon) will return to CIL tank 6 while the quench water and fine carbon will report to the tails hopper via the carbon safety screen. The regeneration kiln will be located on the upper floor within same area as the elution boiler, adjacent to the ball mill and the gold room. The quench tank will be located beneath the kiln in the same area. The concrete floor and bunds for the regeneration equipment will be the same as those for the elution boiler as will the associated sump pump.

3.5.5 Cyanide Destruction and Tails Disposal

Refer to Flowsheet 1497-F-106

CIL residue slurry discharging from the final adsorption tank (CIL Tank 6) will gravitate to the vibrating carbon safety screen to recover any carbon leaking from worn or damaged inter-tank screens. The carbon safety screen undersize slurry will gravitate to a 16 m diameter high rate thickener. The thickened tailings slurry will be drawn from the thickener underflow by variable speed pump (1 duty/1 standby) to control thickener bed mass and pumped to a distribution box ahead of the cyanide destruction tanks.

The supernatant will overflow to the adjacent process water tank for subsequent reuse in the milling circuit.

The cyanide destruction circuit will consist of two (2) 110 m^3 (live) tanks arranged in series and connected by a launder. The tanks will be agitated and aerated to provide sufficient dissolved oxygen in the slurry to meet the oxygen demand of the CN_{WAD} destruction reaction.

Sodium metabisulphite and copper sulphate solutions will be dosed into the feed boil box providing the source of sulphur dioxide (SO_2) and catalyst respectively. The reaction between the CN_{WAD} and SO_2 generates sulphuric acid (H_2SO_4). Milk of lime, from the ring main, will therefore be added to the feed distribution box to maintain the pH of the slurry at between 8 and 9 in the final tank.

The detoxified effluent will gravitate to the tails hopper from where it will be pumped to the TSF by variable speed tails pumps (2 duty/2 standby) and discharged by spigots from selected outlet points around the periphery of the facility. Supernatant and run-off from rainfall collected in the decant pond will be pumped to the decant water tank located on at the plant.

The tails thickener and cyanide destruction area will have a separate bunded concrete area for collecting spillage and clean up water. A spillage pump will be installed in the thickener bunded area to reclaim and return any spillage to the carbon safety screen. The tails thickening and cyanide destruction equipment will be located at the end of and adjacent to the CIL tank train.

3.5.6 Reagents

Refer to Flowsheet 1497-F-107/108

Lime

Quicklime will be delivered to site in bulk 25 t road tankers and pneumatically unloaded directly to a 40 m^3 silo. A dust collector will mitigate dust emissions during pneumatic transfer. Quicklime will be withdrawn from the silo by a variable speed screw feeder to provide feed to a slaking mill, where it will be ground in decant water, slurried to 15% w/w solids and stored in a mechanically agitated tank with 24 hours capacity.

Lime slurry will be pumped to the process through a circulating main with off-takes at the SAG mill feed, at the CIL circuit (emergency pH control) and at the cyanide destruction circuit for pH control. Solenoid valves with adjustable timers will control the rate of lime addition from the circulating main.

Currently only quicklime is confirmed as being available for use on site. Price, packaging and availability of hydrated lime is unknown. Given the expected mine life, the use of hydrated lime may provide an avenue for total project cost reduction by the elimination of the lime slaking mill package. This should be investigated in future studies.

Sodium Cyanide (NaCN)

Sodium cyanide will be received as briquettes in 1 tonne bulk bags. A hoist will be used for lifting the bags to the enclosed feed chute with a bag breaker above the 5 m^3 cyanide mixing tank. The sodium cyanide will be dissolved with filtered raw water to make up a 20% w/v sodium cyanide solution and then transferred to a 7 m^3 storage tank.

Cyanide solution will be distributed through a ring main with off-takes at the leach tanks and CIL tanks 1 and 2 for cyanide profile control. A variable speed progressive cavity pump will be used to meter the cyanide solution into the suction line of the elution water pump during the elution cycle.

Sodium Hydroxide (Caustic Soda, NaOH)

Caustic soda will be delivered to site as a solid in 1.25 t bulk bags. A hoist will be used for lifting the bags to the enclosed feed chute with a bag breaker above an 18 m^3 storage tank located in a concrete bunded area. Solid sodium hydroxide will be dissolved in filtered raw water to generate a 20% w/v caustic solution.

A variable speed progressive cavity pump will be used to meter the caustic solution into the suction line of the elution water pump during the elution cycle.

Hydrochloric Acid (HCl)

Concentrated hydrochloric acid (33% w/v HCl) will be delivered to site in bulk 1 tonne containers. The concentrated acid will be pumped from the containers to the 14 m^3 storage tank located in a remote acid-proofed bunded area. The concentrated hydrochloric acid will be pumped from the storage tank by a positive displacement, peristaltic pump and injected into the fresh water line at the base of the acid wash column.

Grinding Media

SAG mill grinding balls of 125 mm diameter will be delivered to site in 200 L drums and transferred from the drums to a 2 tonne media kibble. The kibble will discharge into the SAG mill ball loading chute located above the SAG mill feed chute.

Ball mill grinding balls of 50 mm diameter will be delivered to site in 200 L drums and similarly, transferred from the drums to a 2 tonne media kibble. The kibble will discharge into the ball mill ball loading chute located above the ball mill feed chute.

Lime slaking mill grinding balls of 50 mm will be delivered to site in 200 L drums and transferred from the drums to a 1 tonne media kibble. The kibble will discharge into the slaking mill ball loading chute located above the slaking mill feed chute.

Copper Sulphate ($CuSO_4 . 5H_2O$)

Copper sulphate (as pentahydrate) will be received as a crystalline solid in 25 kg bags. The bags will be emptied into the copper sulphate mixing tank and dissolved with raw water to make up a 20% w/v copper sulphate solution, and then transferred to a 9 m^3 storage tank.

A variable speed diaphragm pump (1 duty/1 standby) will be used to meter the copper sulphate solution into the distribution box ahead of the cyanide destruction circuit where copper sulphate acts as a catalyst for destruction of cyanides.

Flocculant

Anionic flocculant will be mixed in a flocculant mixing package to provide diluted flocculant to the tails thickener. The flocculant will be received on site in powder form in 25 kg bags, opened and emptied by hand into the flocculant storage hopper.

A variable speed progressive cavity pump will be used to meter the dilute flocculant solution into the feed box at the thickener.

Sodium Metabisulphite (SMBS, $Na_2S_2O_5$)

Sodium metabisulphite (SMBS) will be received as a powder in 1 t bulk bags. A hoist will be used for lifting the bags to the enclosed feed chute with bag breaker above a 13 m^3 sodium metabisulphite mixing tank. The SMBS will then be dissolved with raw water to make up a 20% w/v SMBS solution and then transferred to the 19 m^3 storage tank.

A variable speed progressive cavity pump (1 duty, 1 standby) will be used to meter the SMBS solution into the distribution box ahead of the cyanide destruction circuit.

Activated Carbon

Activated carbon will be delivered in 500 kg bulk bags and added to the carbon quench tank using the reagent area hoist. The activated carbon inventory will be replenished on a demand basis, with fines removal from the fresh carbon being performed on the barren carbon sizing screen.

Zinc Dust

Zinc dust will be delivered to site in 25 kg bags. Zinc dust is used to precipitate precious metals from solution ahead of the precipitate filters. Pallets are transferred to the precipitate area, as required, and bags emptied into a hopper. A screw feeder at the base of the hopper meters zinc dust into a cone, where the dust is wetted, prior to its distribution between the suctions of the precipitate filter feed pumps.

Lead Nitrate

Lead nitrate will be delivered to site in 25 kg bags on a pallet.

Pallets will be transferred to the precipitate area, as required, and the bags manually emptied into a hopper. A screw feeder at the base of the hopper will meter lead nitrate into the zinc cone, where it is wetted with barren solution prior to distribution between the suctions of the precipitate filter feed pumps.

Diatomaceous Earth

Diatomaceous earth will be delivered to site in 35 kg bags of a pallet. Diatomaceous earth will be used for pre-coating and body feed duties and thus the bags will be manually emptied into both the pre-coat hopper and body feed hopper. The contents of the pre-coat hopper will discharge into a 0.5 m^3 pre-coat mixing tank, where it will be mixed with barren solution to generate a 10% w/w slurry and pumped to the duty precipitation filter.

A screw feeder at the base of the body feed hopper, meters diatomaceous earth into the zinc cone where, it is wetted prior to its distribution between the suctions of the precipitate filter feed pumps.

3.5.7 Plant Services – Water

Refer to Flowsheet 1497-F-009

Process Water Services

The process water system provides high volume low pressure water for the grinding circuit. The system consists of a storage tank, duty / standby centrifugal pumps and a water main. The storage tank is sized to provide two hours of residence time at design flow rates. Water supply to the tank is primarily from tailings thickener overflow, with additional makeup from the decant water tank or raw water tank as required.

Process water will be delivered to:

- SAG and ball mill inlets.
- Cyclone feed hopper.
- SAG and ball mill trommels screens.

A separate process water system, decant water, delivers water to the circuit where low residual cyanide water is required. The decant water system consists of a storage tank with six hours residence time at design rates, duty / standby centrifugal pumps and a water main. The main water supply to the tank is decant water reclaimed from the tailings management facility. Raw water can be added to the tank to maintain water supply in the event of decant return water pump or line failure.

Decant water will be delivered to:

- The tailings thickener for flocculant dilution.
- Cyanide detoxification for dilution water.
- The CIL circuit as spray water.
- The lime slaking facility.
- The process water tank.

Raw Water Services

Raw water will be supplied from a local borefield by two pumps to the site raw water storage tank. The raw water storage tank will supply water to the high pressure raw water, low pressure raw water, filtered water and fire water circuits.

The high pressure raw water circuit will consist of duty / standby pumps and provides water to the:

- Crushing area – spray water.
- Desorption area – for eduction water.
- Tailings area – spray water for carbon safety screen.
- Gland water circuit.

The low pressure raw water circuit will consist of duty / standby pumps and provide water to the:

- Mill area – water services.
- Desorption area – carbon transfer water.
- Reagents area – copper sulphate mixing.
- Raw water filter plant.

The filtered raw water circuit will be a vendor package and supply water to the:

- Potable water plant.
- Desorption area – acid wash water.
- Reagent area – cyanide, flocculant, SMBS and caustic soda mixing.

The fire water circuit will consist of a dedicated pump package and ring main. It will be fed from the raw water tank from an outlet located below the high and low pressure pump outlets. The position of this outlet will be calculated to ensure that the volume of water accessible by the fire water circuit only, is sufficient for statutory fire fighting requirements. The fire water pump package will consist of an electric jockey pump, electric booster pump and diesel booster pump.

Potable Water Services

The water treatment circuit will provide potable water for use in the plant safety shower circuit, workshop/store, crib room, ablution building, site administration, guard house, the accommodation village and the elution circuit.

The circuit will consist of a reverse osmosis plant, storage tank with twenty four hour residence time at design flow rates, and duty / standby distribution pumps.

3.5.8 Plant Services – Air

Refer to Flowsheet 1497-F-110

Plant and Instrument Air Services

Two screw type compressors will supply compressed air to the plant air receiver. The compressors will also supply air to the instrument air receiver via a refrigerant air dryer. Plant air and instrument air at 700 kPag will be reticulated throughout the process plant.

Low Pressure Air Services (LP Air)

Two screw compressors (1 duty/1 standby) will supply low pressure air at 120 kPag to the cyanide destruction and the reagent unloading areas. After-coolers will be installed on each blower to reduce the discharge air temperature. Low pressure air will be reticulated to the points of usage.

Oxygen (PSA Plant)

A pressure swing adsorption (PSA) plant will supply oxygen at 700 kPag to an oxygen receiver for subsequent use in the CIL plant. Oxygen will be injected in to a circulating stream of leach slurry via a high shear device. The flow of oxygen will be controlled to maintain a dissolved oxygen concentration according to a set point, as measured by a dissolved oxygen meter located in leach tank 1.

3.6 Control Philosophy

3.6.1 General

The basic design philosophy is that the plant be appropriately automated to reduce the need for operator intervention on a continuous basis. Moderate levels of process and engineering data collection and equipment monitoring will be provided via a programmable logic controller (PLC) based system and operator interface terminal (OIT). A fully redundant PLC system will be installed to provide online backup of plant process control in the event of PLC component failure. Critical safety and equipment protection interlocks will be hardwired. Control of process variables will be via the OIT or discrete controllers in the field.

The plant will be provided with a central control room from which the status of major electrical and mechanical equipment can be monitored and major regulatory control loops can be monitored and adjusted via the OIT.

The proposed control system will allow the starting/stopping of the following drives from the OIT in the main control room:

Table 3.5 Drives to be controlled from Main Control Room

Comminution Circuit	Apron feeder Jaw crusher Conveyor CV 01 Conveyor CV 02 Conveyor CV 03 Conveyor CV 04 Mill motors Mill discharge pumps Trash screen
CIL Circuit	Leach and CIL tank agitators Inter-tank screens Carbon recovery pump Carbon transfer pumps Carbon recovery screen Carbon safety screen
Cyanide Destruction/Tails	Thickener underflow pumps Cyanide destruction tank agitators Tails pumps
Reagents	Milk of lime circulation pumps Milk of lime storage tank agitators Cyanide circulation pumps Thickener flocculant pumps Cyanide destruction reagent dosing pumps
Services	Cyanide reduction blowers PSA plant
	Plant/instrument air compressors Raw water pumps Potable water pumps Decant water pumps Process water pumps

The following drives will be started from control panels in the field:

Table 3.6 Drives to be controlled from field-based panels

Crushing Control Panel	Feeder Primary crusher Conveyor CV-01 Conveyor CV-02 Conveyor CV-03 Conveyor CV-04
Mill Control Panel	Mill auxiliary drives and mill motor – SAG and Ball mill
Thickener Control Panel	Thickener auxiliary drives and rake motor
Elution Circuit Panel	Acid wash pump Elution water pump Pregnant eluate pump Heater auxiliary drives Kiln auxiliary drives Carbon de-watering screen
Precipitation Filter Panel	Precipitation filters Precipitation filter feed pumps Carbon Regeneration Kiln Calcine Oven Barring Furnace

All other drives will be started in the field at the local stop/start control station located adjacent to the drive.

The PLC will be utilised to accept status signals from the electrical switchgear for monitoring drive status conditions on an OIT.

Two OITs will be installed in the main control room comprising two computer screens, keyboard, mouse and printer with an uninterruptible power supply (UPS) system.

3.6.2 Drive Controls

Each drive will be supplied from a motor control centre (MCC) switchboard. All drive control circuits will be hardwired and generally operate at 110V AC. Drive safety interlocks will be hardwired.

In the field, each drive will be provided with a stop/start push-button control station. The stop button will be of an Off-Stop style.

The SAG mill speed will be varied by the operation of the liquid resistance starter.

Other variable speed control units will use variable voltage variable frequency (VVVF), utilising pulse width modulated (PWM) technology. The drives will be mounted in free-standing cubicles, which will be provided with integral control panels for programming and operation at the VVVF unit, for commissioning and emergency running.

3.6.3 Control Loops

The regulatory control loops controlled from the main control room will appear as a dedicated faceplate on the OIT and will appear and function in a similar manner to traditional field mounted proportional integral derivative (PID) controllers.

The regulatory control loops not controlled from the OIT will be performed with single loop PID controllers mounted on the field control panel (i.e. crusher, mill thickener, elution, kiln, calcine oven, and barring furnace control panels). The controllers will be capable of standard PID control and ratio control and come with standard operator interface facilities.

3.6.4 Crushing Circuit

Operation of the crushing circuit will have essential hardwired interlocks to protect equipment. The only control point in the crushing circuit will be the speed of the primary feeder which will be controlled to achieve the desired feed rate to the SAG mill.

A small crusher control panel will be provided in a cubicle above and adjacent to the primary crusher. The crusher operator will have the facility to indicate a dump/no dump ROM bin condition to the FEL driver.

3.6.5 Milling Circuit

Ore feed to the SAG mill will be measured with a belt weightometer on the SAG mill feed conveyor, located upstream of the crushed recycled pebbles discharge point. The speed of the apron feeder will be modulated to achieve the desired set point on the SAG mill feed PID controller on the OIT. SAG mill power and weight will also be measured.

Process water addition to the SAG mill feed chute will be measured and controlled to a m^3 per tonne ratio with the ore feed, in order to maintain a desired slurry density in the SAG mill. A flow meter will be provided on the total process water addition to the milling circuit. A nuclear density gauge located on the cyclone feed line will indicate the cyclone feed slurry density. The water addition to the SAG mill discharge hopper will be controlled to maintain a set point on the cyclone feed density PID controller on the OIT.

An ultrasonic level element will measure the slurry level in the mill discharge hopper. A PID controller on the OIT will be used to control the speed of the mill discharge pumps to maintain the required hopper level.

The SAG mill feed tonnage, SAG mill weight, SAG mill power draw, SAG mill speed, ball mill power draw, cyclone feed density, mill discharge hopper level, mill discharge pump speeds (output), and hydrocyclone feed flow will be trended and recorded through the OIT.

3.6.6 Hybrid CIL Circuit

The pH of the slurry in leach tanks ahead of the CIL circuit will be continuously measured by a submerged pH probe, which will control the set point on the lime addition PID controller on the OIT.

All process parameters within the CIL circuit, with the exception of the pH control loop, will be manually controlled or measured. The following is a list of the main metallurgical monitoring functions that the CIL shift operators will be required to perform:

- pH measurements on pulp in the leach tanks and CIL tank 1 by hand-held pH meter - used to cross-check the automatic pH probe.

- Density measurements of the pulp in the leach tanks and CIL tank 1 by Marcy Scale - used to assess performance of the classification circuit.
- Cyanide concentration measurement in leach solutions - measured on filtrate by titration.
- Carbon concentration measurement - by volume from dip samples.
- Composite head and tail sample collection for assay.
- Slurry level observation in tanks and observation of inter-tank screen condition.
- Carbon recovery screen performance monitoring.
- Mechanical equipment monitoring - temperature, vibration, etc.

3.6.7 Elution Circuit

All steps in the AARL elution cycle will be controlled automatically with the exception of the carbon transfer steps. The sequence control of the AARL elution cycle will be monitored and controlled by a dedicated PLC mounted in the elution circuit control panel. The following sequence control will be performed by the elution circuit PLC:

- The positive displacement hydrochloric acid metering pump will operate on a timer to ensure the correct dosage of concentrated acid to the water line during the acid wash cycle.
- The positive displacement elution water pump will operate on a timer to ensure sufficient water is used to rinse the acid from the acid wash column.
- The positive displacement elution water pump will operate on a timer to ensure sufficient water is used during the pre-soak cycle.
- The positive displacement caustic and cyanide metering pumps will operate on timers to ensure the correct dosage of reagents to the suction of the stripping water pump during the pre-soak cycle.
- The eluate temperature will be measured by a thermocouple located in the eluate line to the elution column. A PID temperature controller mounted on the elution circuit control panel will modulate the input of energy from the heater to maintain the desired eluate temperature.

The stripping plant will strip up to fourteen batches of carbon per week.

The carbon regeneration kiln temperature will be controlled by modulating the speed of the screw feeder to maintain the temperature set point on the PID controller mounted in the local kiln control panel.

The critical process parameters (elution temperature and pressure, and kiln temperature) will be trended on the OIT to assist the operator with process monitoring.

3.6.8 Precipitation and Gold Room

The start-up and shutdown of precipitation filters will be automated and controlled by a PLC supplied as part of the filter package. Inlet and outlet valves of the precipitation filters will be automated and controlled from a local control station with their positions indicated in the central control room.

Calcine oven and barring furnace temperatures will each be controlled by their respective temperature controllers mounted in the local control panels. All other aspects of the gold room operation will be manually controlled.

3.6.9 Cyanide Destruction and Tails Disposal

The pH of the slurry in the first cyanide destruction tank will be continuously measured by a submerged pH probe which will control the set point on the lime slurry addition PID controller on the OIT.

The Eh of the slurry in the first detoxification tank will be continuously measured by a submerged Eh probe which will control the speed of the positive displacement SMBS solution metering pump to maintain the set point on the PID controller on the OIT.

A copper sulphate solution will be added to the distribution box above the cyanide destruction reactors using a variable speed, positive displacement pump with the speed manually adjusted on the OIT.

Air flow to the detoxification tanks will be controlled manually.

An ultrasonic level element will measure the slurry level in the detoxification tails hopper. A single PID controller on the OIT will be used to control the speed of the detoxification tails pumps to maintain the required hopper level.

3.6.10 Reagents

Level indicator transmitters will be installed on all reagent mixing tanks and most reagent storage tanks with level indication on the OIT. Provision will be made for On/Off control of water addition to reagent mixing tanks via the OIT.

3.6.11 Services

A level indicator will indicate the water level in the decant water tank, raw water tank, potable water tank and process water tank. The tank levels and high/low level alarms will register in the control room on the OIT.

Pressure indicator transmitters will be provided for the plant, instrument and LP air systems with signals monitored and alarmed via the OIT.

Individual magnetic flow meters will measure the bore field water production, and the raw and process water flows. Water flow monitoring will be via the OIT.

3.6.12 Control Interfaces

Crusher Control Panel

Start/stop facilities will be provided at the crushing control panel and field stop/start stations.

A remote/local/maintenance switch will be provided at the crushing control panel to allow the operator to select one of the following options.

- Remote Selected - crushing circuit will be started from the crushing panel.
- Local Selected - crushing circuit will be started from the field stop/start stations with the sequence interlocks maintained.

- Maintenance Selected - allows sequence interlocks to be disabled. Starting will be possible locally or remotely.

The crusher control panel will also house the following:

- Load current indication for the crusher and conveyor drive motors.
- An audible alarm with a panel mounted accept button.
- Panel lamps will indicate Ready, Run and Fault status.

Main Control Room (MCC)

The MCC will be adjoining the central MCC and substation and will contain two OITs on an operator desk.

The OIT screen indication will include plant mimic pages with the following:

- Ready, Run and Fault status of all drives monitored directly by the OIT.
- Drive electrical parameters and running hours monitored and displayed for all critical drives to assist in maintenance planning.
- Indication for each field instrument monitoring a process state with alarm status.
- Access to faceplates for associated single loop PID controllers.
- Drive start and stop controls for associated drives configured to start remotely via the OIT.
- Remote/Local/Maintenance selector for associated drives configured to start remotely via the OIT.

Audible alarms will be provided on the OIT and around the plant to advise when abnormal conditions occur. The alarm will be acknowledged on the OIT and the status indicated on alarm pages.

Mill Control Panel

The SAG mill will be provided with a local control panel adjacent to the mill. The control panel will house the following:

- Individual mill alarm faults.
- Local mimic panel to assist fault diagnostic information.
- Mill bearing temperatures.
- Mill load cell readings, power draw and speed.

The SAG mill will be started via a liquid resistance starter, which will also be used to adjust the mill speed within a limited range. The mill speed will be adjusted via the OIT.

The ball mill will be provided with a local control panel adjacent to the mill. The control panel will house the following:

- Individual mill alarm faults.
- Local mimic panel to assist fault diagnostic information.
- Mill bearing temperatures.

Elution Circuit Panel

The AARL elution circuit control panel will house the following:

- A PLC that will control the sequencing, control and alarming functions.
- Ready, Run and Fault status of stages of the elution cycle.
- Remote temperature set point for the elution heater.

The elution heater will be a vendor package controlled locally with standard control facilities. Remote start and temperature set point signals will be initiated from the elution control panel.

Precipitation Filter Panel

The precipitation circuit control panel will house the following:

- A PLC that will control the sequencing, control and alarm functions.
- Ready, Run and Fault status of the filter throughout its cycle.

The precipitation filters will be a stand alone vendor package located inside the secure area, adjacent to the goldroom.

Operator Interface Terminal

Operator interface terminals (OIT) will be provided in the main plant control room. The OIT will consist of the following:

- Two Pentium® IV central processing units (CPU) complete with CD ROM. The computers will run in a master/slave operation.
- Two 430 mm (17 inch) colour Video Display Unit (VDU).
- Ethernet Cards.
- Standard industrial keyboard.
- Standard two button, serial mouse and mouse pad.
- Windows® and Disk Operating Software (DOS).
- Windows® based Data Logging Software for recording, trending, and alarming.
- Colour printer.
- A 6 kVA uninterruptible power supply (UPS) unit.

The OIT will be interfaced to the control system which collects all the digital status and analogue measurement signals from the MCC PLC and other PLC in the plant. The interface between the data logger and the PLC will be via an Ethernet link.

The OIT will read all digital and analogue information within the PLC and perform the following functions:

- Display analogue process values on the VDU as a bar graph display. Each bar will display the process value and alarm set points. A facility for the operator to change the alarm values will also be provided for each bar.

- Display faceplates for PID controllers configured in the OIT. Each faceplate will allow the operator to change the process set point, Manual/Auto/Remote selector and alarm values.
- Store all major analogue process values on the hard disk. The stored data can be archived off the disk or will be overridden on a regular basis as required.
- Display analogue process values on the VDU as a trend display. The trend displays will enable the display of data from the past up to present time. The operator will be able to change the time scale and magnitude axis for each trend.
- Display digital alarms and analogue alarms on an alarm display. This display will show the date and time the alarm was generated and the alarm condition. Acknowledged alarms which are still present will be displayed in a subdued colour until they are cleared. Alarms will be printed as they occur.

3.7 Metallurgical Accounting

A weightometer mounted on the SAG mill feed conveyor will measure crushed ore feed and will determine the SAG mill feed tonnage. Each weightometer will provide an instantaneous tonnage output, as well as an integrated tonnage value for metal accounting purposes.

The density and flow rate signals from the nuclear density gauges and magnetic flow meters located on the thickener underflow and CIL tail lines will be used to calculate the mass flow at these points in the circuit.

Regular sampling by auto-samplers on the hydrocyclone overflow and the CIL residue streams will provide reliable samples for head grade and final solution and residue grades. These analyses will be performed in the site laboratory and the results used for overall recovery estimates.

Regular gold and silver in circuit surveys of the CIL circuit will allow reconciliation of precious metals in feed compared to doré production.

4 INFRASTRUCTURE AND SERVICES

4.1.1 General

Both on-site and off-site infrastructure requirements have been addressed in this study. This section provides a description of the infrastructure. It addresses roads, water, fuel and power supplies, buildings and control room, communications, security, fire protection and accommodation camp, which will be provided to support the operation of the process plant.

4.2 In-country Telecom/In-plant Communications

On site communications will comprise inter-connected mobile and fixed systems. The mobile system will consist of a base and a repeater radio station with 20 hand-held and 14 vehicle mounted slave sets. The fixed system will include a master "communications manager" controlling a network of up to 64 fixed handsets and an Ethernet data network.

A microwave station with a capacity of 45 Mb/sec will provide a voice and data link from the mine site "communications manager" between the existing AQI Exploration Office in Jacobacci, and thence by ISDN to the Buenos Aires offices.

4.3 Access Roads and On-Site Road Constructions

4.3.1 Project Access and Bypass Roads

The existing access road into the project site will be widened and upgraded to meet the project requirements. This road is some 23 km long and ties in to Provincial Route No. 76 near Ingeniero Jacobacci and ends at the project camp.

Two new bypass roads will be constructed to divert traffic from the project access road around the site. The northern bypass road will be 3.6 km long. The eastern bypass road will be 3.3 km long.

A total of 6 km of new roads will be constructed from the end of the project access road providing access to the mine haul roads, plant site, infrastructure and the main camp. This includes a spur road to the explosives magazine. These roads will not be normally open for public access.

The project access and bypass roads will have the following design features:

- Design speed 60 km/h, but restricted to 50 km/h at some corners.
- Two lane road, total carriageway width 6 m.
- 1 m shoulders both sides of road.
- Provision of top side table drain.
- Corrugated pipe culverts.
- In general; fill batter slopes 1:1.5 (vertical:horizontal), Cut batter slopes 1:0.5 (vertical:horizontal).
- A granular sub-base and pavement comprising compacted crushed rock (i.e. limestone gravels).

- Appropriate road markings, guide posts and signs.

4.3.2 Mine Haul Roads

- Approximately 3.9 km of haul roads will be constructed initially prior to development of the open pit. These will connect the Run-of-Mine (ROM) ore storage pad with an exit of the first two deposits that will be exploited, the northern pit and the pit just to the west of the plant site. A spur haul road allowing trucks to access the refuelling area is included.
- Cut to fill construction will be used to the maximum possible extent. The main section of this road will comprise a 20 m wide running surface, 1 m shoulders and a 3 m upside drain.
- Maximum allowable grade will be 10%, which will reduce on sharp curves. Embankment slopes will be 1:1.5 in fill and 1:0.5 in cut. The road foundation will be substantial, as is required for the passage of mine trucks. It will be sheeted with gravel maintained by frequent grading and water sprinkling.
- The upper end of this road will adopt a progressively changing alignment to match it with the variations in open pit exit point elevation.

4.4 Water Supply and Distribution

4.4.1 Source of Supply

The water supply source has not yet been identified by AQI. A nominal lump sum cost has been allocated in the estimate to allow for the necessary pumps, piping and associated equipment to deliver water to the plant site.

It is anticipated that the bulk of the plant water demand will be drawn as a recirculating flow from the decant pond in the TSF and from two monitoring ponds that will be built down gradient from the TSF. Water will be supplied to the decant pond from the following sources:

- Water released from the tailings slurry as it settles and consolidates.
- Rain water on the TSF.
- Run-off from rainfall on the undiverted portion of the unlined catchment area surrounding the TSF liner, from rainfall on the exposed liner area, from rainfall on the tailings beaches surrounding the decant pond and rainfall in the monitoring ponds.
- Recirculated drainage water flowing from the tailings underdrainage system laid on top of the HDPE liner.
- Discharge from the package sewage treatment plant.
- Seepage water from the groundwater interceptor drain, beneath the HDPE liner at those times when the quality of this water prevents its being discharged to the receiving waters.

These inputs will be partially offset by evaporation losses from the decant pond surface and from those areas of the surrounding tailings “beaches” which are not entirely

desiccated. The water balance model takes all these contributing factors (apart from occasions of groundwater interceptor drain recirculation or addition of pit dewatering flows) into account through a 30 year series of actual weather conditions incorporating extended wet and dry weather cycles.

4.4.2 Raw Water System

Raw water will be delivered from the water supply source to the open topped raw water tank with a capacity of 800 m^3 located at the plant. High pressure and low pressure pumps each with standby units will supply the raw water needs for the plant. Low pressure (LP) raw water will be filtered through a sand media filtration system and used in the elution circuit and for flocculant and reagent make-up. A bleed stream of the filtered LP raw water will be diverted to the potable water system that is described in Section 4.4.4.

The raw water tank will incorporate a volume dedicated only to the firewater system described in Section 4.11.

4.4.3 Process Water System

Water will be recovered from the tailings pond using a pontoon equipped with duty and standby vertical multistage pumps, and with floating access for power, pipeline and personnel. Normal demand will be met by operation of the duty pump only. The maximum plant process water demand under adverse conditions, e.g. failing to achieve a high tailings thickener underflow pulp density will be met by simultaneous operation of duty and standby pumps. Power will be provided by an overhead line routed from the process plant to the head of the decant pond causeway, and thence by armoured cable laid on the surface. The immediate area will be strongly illuminated and the pontoon will incorporate fencing for security and safety. Pump control will be remote.

Process water will be delivered via an HDPE pipeline, laid on the ground and routed along the decant pond causeway and access road, to the open topped process water tank located at the plant. The tank capacity will be approximately 400 m3. A manual drainage arrangement will allow this pipeline, including the whole process water tank if necessary, to be drained into the decant pond.

Process water will also be used to supply a standpipe for filling water trucks.

4.4.4 Potable Water System

Potable water will be drawn from the filtered raw water supply to the process plant. Ultraviolet sterilisation and chlorination will be used to produce potable quality water. Potable water will be stored at the process plant in a closed tank and pumped, in a system incorporating a central accumulator, to all safety showers and eyewash stations, changing/washing facilities, toilets, kitchens and drinking fountains. The tank capacity will be approximately 60 m3. Potable water piping in the plant area will either be buried below the frost line, routed through heated buildings or heat traced and insulated. Manual drain points will be included to allow emptying of pipelines should weather conditions dictate.

4.5 Effluent

4.5.1 Sources and Disposal

Sewage from the plant site buildings will gravity flow via a pipe network buried below the frost line into the epoxy coated mild steel tank connected with a package activated sludge sewage treatment plant. The outflow will either be pumped to the plant tailings hopper or gravity fed via surface drains to the TSF. Surplus sludge will be periodically transferred to the tailings hopper.

Hydrocarbon waste will arise from equipment maintenance, sumps in the bunded fuel storage area, and operation of the oil/water separator in the vehicle wash down facility. This material will be pump and gravity transferred to a waste oil tank from which it will be pumped to a contractor's bulk tanker for off-site disposal in accordance with the applicable regulations. Chemical waste from the laboratory will be collected and stored for off-site disposal in the same manner.

Office waste and waste from the meals areas will be collected by a cleaning contractor who will dispose of the waste materials off-site in accordance with the applicable regulations.

4.6 Fuel Supply, Storage and Distribution

4.6.1 Usage

During construction, the diesel fuel consumption by all participants is anticipated to be approximately 400 m^3. This is a significant volume and justifies the early scheduling of the installation of the permanent fuel facility.

During operation, the diesel consumption for the mining fleet is anticipated to range between 1000 and 2000 m^3/year, with an average of about 1600 m^3/year. Fuel use for process plant equipment (strip solution heater, carbon regeneration kiln and barring furnace) will be relatively steady at about 260 m^3/year.

The power station will consume approximately 15,000 m^3/year of diesel.

Supplementary uses will be for some light vehicles and plant mobile equipment. These will average about 87 m^3/year for a grand total averaging 16,947 m^3/year or approximately 326 m^3/week.

Petrol use will be minor, as required for a few light vehicles used on the public roads, and will be satisfied by purchase from local retail suppliers.

4.6.2 Fuel Source and Quality

Diesel fuel will be sourced from major suppliers able to provide certification as to its quality to ensure minimisation of environmental impact. It will be delivered in bulk tankers every week.

4.6.3 Storage and Distribution

The facility will comprise two 170 m^3 capacity tanks. A skid mounted fuel supply mechanical/piping package with controls and interlocks will contain tanker unloading pump, fuel transfer pumps, strainers and filters. A light and a heavy vehicle bowser, on opposing sides to separate traffic, will complete the installation. The complete facility will be located within an impermeable bunded area able to contain the full volume of the tanks

with a sump draining to the adjacent vehicle wash down facility hydrocarbon sump. It will be located 25 m clear from the nearest building. Fire suppression sprays will be permanently piped into the firewater distribution system.

Fuel for use in the process plant will be intermittently pumped to a 1000 L day tank within the process plant building, from whence it will gravity feed to all the demand points.

4.6.4 Security

A security fence will prevent access to all but the loading point and bowsers. An electronic card system, preventing fuel withdrawal by unauthorized personnel and recording use against each card, will be installed.

4.7 Vehicle Wash down Facilities

4.7.1 General Description

A vehicle wash down facility will be provided adjacent to the diesel fuel refuelling area. It will comprise a bunded slab sloped to a sump, into which rainwater and fuel spillage from the adjacent fuel storage and distribution area will also drain. A sump pump will transfer dirty water to an oil/water separator. Collected oil will be pumped to a waste oil tank. Separated water and solids will be pumped to the tailings thickener.

High pressure water cleaners will be provided to spray down the vehicles with process water for:

- Regular cleaning.
- Cleaning prior to maintenance.
- Cleaning of wheels and tyres prior to the vehicle or machine leaving the site.

4.8 Power Supply and Reticulation

4.8.1 Power Station

A power station consisting of ten 1.0 MW high speed diesel generators will be installed. No more than eight units will be required to operate at one time. Power will be generated at 11 kV.

The power requirement has been calculated as follows:

- Process plant installed power = 6.7 MW.
- Estimated plant consumed power = 5.8 MW (This is when the plant is operating and is the power used to size the power station).
- Estimated plant average power = 4.5 MW (This is the value used for estimating operating cost).
- Eight 1MW generator sets (operating at a typical 80% load) will produce 8 x 1 x 0.8 = 6.4 MW, sufficient for 5.8 MW of plant power plus an allowance for mine workshop, camp, etc. Ausenco recommends that two stand-by units are provided, giving a total of ten 1 MW generators.

4.8.2 Power Distribution

An aerial distribution arrangement in the plant switchyard, will distribute the 11 kV supply to the following:

- The SAG and ball mill drives.
- An 11/0.4 kV, 2 MVA transformer serving the main plant motor control centre (MCC).
- A 150 m long underground 11 kV line to the crushing plant 11/0.4 kV, 500 kVA transformer.
- A 1.5 km long 11 kV overhead line routed along the TSF access haul road to the 11/0.4 kV, 100 kVA transformers at the TSF embankment and TSF decant pond.

4.9 Buildings

4.9.1 Administration Buildings

These buildings will be brick or block-work construction with on-ground concrete floor slabs finished to suit vinyl flooring and skirted below the frost line. Upper floors will be on similarly finished elevated concrete slabs. They will use local materials and building techniques to the maximum possible extent. The external finish will be painted. Roofing will be timber framing clad with corrugated steel.

Ceilings under roofs will be painted plaster board. Internal partitions will be either timber framed and clad with plaster board or bricks/blocks, all with a painted finish. Skirting and scotia boards will be provided only if the construction technique demands it. Electrical and communication wiring may be surface mounted in conduits inside and out. Internal lights against concrete or block-work will be bulkhead units. Internal doors will be painted hollow core, external doors painted solid core.

Door hardware will be local domestic standard. Floor finishes will be vinyl sheeting with local ceramic tiling to wet areas.

The arrangement of windows in the buildings will allow natural ventilation and be local standard. Heating will be provided by electrical heaters.

Administration Office

This building will house managers, administration and finance personnel and a proportion of the operating and mining staff, and will be approximately 570 m^2 in area.

The building will include 12 private offices, 22 open plan offices, meeting and training rooms, guardhouse/security area, kitchen, toilets, first aid facilities and ambulance parking bay. It will be equipped with furniture, emergency first aid facilities and office and fixed communications equipment.

Plant Offices and Ablution Block

This lean-to structure will be located at one end of the main process plant building.

Approximately 100 m^2 will be dedicated to male and female ablutions, change rooms and lunch room for plant operating and maintenance personnel. The facilities will include showers, basins, toilets, benches and lockers.

Approximately 160 m^2 will be dedicated to offices for the mill and maintenance managers, maintenance foremen, metallurgists, maintenance planners and warehouse personnel. Furnishings will be provided.

4.9.2 Control Rooms

Crusher Control Room

A basic air conditioned and furnished control room will be provided at the primary crusher. It will be approximately 6 m^2 and of a transportable style constructed of insulated sandwich panel or similar.

The crusher operator will use the plant ablution block toilets about 150 m away. Continuous operator presence within the crushing area will not be required for uninterrupted plant feed.

Process Plant Control Room and Titration Area

This room will be located above the gold room within the process plant building overlooking the grinding and regeneration area and close to the access way between the process building to the CIL area. It will be about 20 m^2, have block-work walls and a timber frame and plaster board ceiling. It will be fitted out with appropriate furniture and fixings but have no provisions for heating or cooling.

An area outside the enclosed control room will be fitted with benches, sinks, shelves and equipment and serve as the titration area for process control by operations personnel.

4.9.3 Industrial Buildings

These buildings will have steel frame construction with galvanised cold-rolled sections for purlins and girts, zincalume or colorbond steel cladding and roofing, with concrete and bitumen flooring.

Process Plant Building

The process plant building will be 60 m long (6 x 10 m bays) by 26 m wide and with an eaves height of 13.5 m. One 15 tonne overhead electric travelling crane along its full length will provide complete coverage for operations (ball loading), plant and mine equipment maintenance and warehouse access.

The two 10 m bays at its north east end will house the grinding, desorption, goldroom and regeneration sections of the process plant. A 20 m long by 6 m wide lean-to annex along the north west wall at this point will contain the covered sections of the reagent mixing equipment and be fitted with its own 1.5 tonne monorail crane for reagent addition.

The remaining four 10 m bays of the building will be dedicated to the plant maintenance and light vehicle workshop and warehouse facility. The workshop area will be fitted out with a small lathe and milling machine, benches, tools and welding machines and will include a lay down space accessed by a roller door on each side of the building.

The warehouse section will cover a 30 m by 10 m fenced off floor space along the north west wall with a 20 m by 10 m mezzanine floor at about 3.5 m height. The storage volume below the mezzanine floor will be fitted with a proprietary multi-row, full height shelving system providing the support for the mezzanine floor itself. An elevated walkway at mezzanine floor level will provide access direct from the process plant office area to the

grinding area eliminating unnecessary traffic through the workshop. A fence around the mezzanine floor will provide security to that storage space.

Tools and materials will be dispensed from a bench in an interior wall of the warehouse at floor level. Ground and mezzanine floor areas accessible to the overhead crane hook will be used to store bulky/heavy items. 600 m^2 of fenced outdoor storage will also be provided.

A full length closable ridge vent and peripheral wall louvres on the building will provide summer ventilation and temperature control but allow warmth to be contained in winter. No heating will be provided in the building other than in enclosed office spaces and the control room.

Mine Workshop Building

The mine workshop building will be 40 m long (4 x 10 m bays) by 15 m wide and with an eaves height of 7.5 m.

Two full bays will be dedicated to full size mine equipment stalls and one bay to light equipment and tyre fitting. Each stall will be accessed through a roller door, and 8 m by 9 m concrete slabs in each of the mine equipment stalls will allow jack operation without floor damage. Appropriate fixed benches, tools and lubrication equipment will be provided.

One full bay (180 m^2) will be dedicated to mine personnel offices, lunchroom and ablution facilities.

Reagent Storage

The reagent storage building will be situated adjacent to the process plant building opposite the reagent mixing area. 300 m^2 of floor area will be provided.

Drains and a sump pump will be provided for spill management.

The area and the building will be lit and access will be strictly controlled. The internal areas will be divided to allow separate access to and control of the cyanide storage area in compliance with relevant regulations.

Core Storage and Laboratory

The core storage and laboratory will be situated adjacent to the process plant and will be 750 m2 in area. It will be a bare industrial building shell only, with a section of concrete floor for the sample preparation area and with basic lighting and a power board. The laboratory contractor will provide and install a fully fitted out transportable analytical laboratory building and office space in the allocated area within the building shell. Laboratory staff will use the plant ablution block toilets. The sample preparation facility will be built up from individual items of equipment installed on the concrete slab with dust collection and suppression equipment as required for safe operation.

4.9.4 Motor Control Centre Rooms

Main Plant Motor Control Centre

The maim MCC room will be 11 m by 7 m with 4 m ceiling height and it will be located on the north side of the process building, constructed with block-work walls. Double doors opening to the east will provide access for installation of equipment. The project switchyard will be on the northern end of the room. Additional single doors will provide

personnel and safety ingress/egress. A ventilation fan will provide cooling when necessary.

Crusher Area Motor Control Centre

This will comprise a transportable building pre-fitted with all equipment and provided with doors, lighting and ventilation.

4.10 Man camp

A fully equipped man camp capable of housing up to 200 people will be constructed 1 km from the plant site to the south west of Nelson deposit. The camp will comprise buildings with 4 bedrooms and 2 bathrooms, with each bathroom shared by 2 rooms. Junior personnel will share bedrooms (2 to a room) and senior personnel will not share rooms. The heating in the rooms will be by electrical heaters and there is no provision for airconditioning or fans for cooling.

The camp will have a single lounge area and there will be a single mess supplying all the personnel the same meals using a contract catering service.

4.11 Fire Protection

Fire protection will be a “wet” system. Jockey, duty and diesel-powered standby pumps will pump from a dedicated firewater volume in the raw water tank. The jockey pump will maintain the minimum pressure of 5 bar in the system at all times.

A buried ring main around the plant area will circle the main process plant building and allow “dry” type hydrants suitable for cold weather installation to be located outside the main building, the reagent annexe, the offices, the reagent store, the core shed/lab building, the fuel area and the tankage area. Fire hose reel cabinets, fed from buried ring main branch pipes, will be located within all buildings. Supplementary hand held fire extinguishers, each suitable for its specific area, will be mounted throughout all buildings, with special emphasis on MCCs, control rooms, transformer areas, diesel fired equipment and fuel storage locations. No sprinkler systems will be installed in buildings.

A fire indicator panel will be provided in the main control room, cabled to very early smoke detection alarms in the following areas:

- Process Plant MCC Room.
- Crushing Area MCC.
- Gold Room.
- Administration building and crib room.
- Workshops and stores.
- Power station.

These, and other enclosed areas will be provided with emergency exits and appropriate illuminated exit signs. Other enclosed areas will all be fitted with standard smoke detection alarms and break glass alarm panels will be mounted externally to all buildings adjacent to access ways.

No fire suppression systems will be provided.

4.12 Security

4.12.1 Fencing and Gates

The plant area, incorporating the crushing plant, main process building, leach area, administration building, core shed and laboratory and reagent store will be surrounded by a 2.4 m high chain mesh fence topped with four strands of barbed wire.

The plant access gate with a guard/security house will be on the plant access road immediately adjacent to the security centre in the administration building. The gate will consist of a sliding gate and a boom gate and will be overlooked by a security camera. Vehicle access through the gate will require identification and search/clearance of the vehicle at the security centre immediately prior to entry and exit.

The guard house will be a prefabricated building with separate entrance and exits doors, small washroom with toilet and sink. Exterior and interior finishings will include all plumbing and electrical wiring.

People entering and leaving the plant on foot will report to the security staff and be subject to identification and search as is necessary.

Visitor car parking bays are provided next to the administration building. There is also car parking and truck bays outside the guard house.

Additional fencing within the plant area compound will separate the outdoor storage section of the warehouse, the reagent store building, the switchyard and the transformer compounds. Access to all these areas will be restricted to very few personnel with a single key to each area held by a senior staff member.

Areas external to the process plant compound also will be fenced with 2.4 m high chain mesh topped with four strand barbed wire. These will include the diesel fuel and vehicle wash down facility, the mine maintenance workshop, the TSF seepage pond pump station and the water supply pump station. All will be fitted with single swinging gates with a single key held by a senior staff member.

The magazine area will be fenced with a double security fence with a single sliding gate. A solar powered proximity alarm system with radio link will bring up an alert in the security centre in the event of penetration of the peripheral barrier. The single key to this gate will be held by a senior staff member.

4.12.2 Secure Buildings

Higher security areas within the process plant will comprise the cyanide storage section of the reagent store and the process plant gold room.

The cyanide storage area will be a secure area within the fenced and secured reagent store. It will be separated off by locked door or gate with controlled access.

The goldroom will be reinforced concrete and block work construction with lockable access doors and secure airlock compound for armoured vehicle loading. Access will be by key, fingerprint access and radio contact with the security centre. Continuous surveillance cameras will monitor and record all activity within the gold room.

High security buildings external to the process plant compound will comprise the two modified containers in the magazine area. These intrinsically secure types of construction will be fitted with lockable doors with keys held by individual senior members of staff. It

will require two staff members to access these areas – one holding the key for the security fence gate and one having the key for a magazine container.

4.12.3 Fuel

Fuel bowsers will be locked and all fuel dispensed by a single individual per shift under surveillance by camera and recorder.

4.13 Plant Light Vehicles and Mobile Equipment

The following light vehicles will be supplied:

Table 4.1 Light vehicles to be supplied

Administration	Vehicle Type	No. of vehicles
Resident Manager	4x4 Wagon	1
Administration Manager	4x4 Wagon	1
Security Supervisor	4x4 Wagon	1
Roaming Security	4 Wheel Motorbike	1
Pool Cars	4x4 Dual Cab	3
OHSE Group	4x4 Dual Cab	1
Communications Manager	4 x 4 Dual Cab	1
First Aid (ambulance)	4 x 4 Troop Carrier	1
Transport	Bus	1
Process Plant	**Vehicle Type**	**No. of vehicles**
Processing Manager	4 x 4 Wagon	1
Shift Supervisor	4 x 4 Utility	1
TSF Engineer	4 x 4 Utility	1
Maintenance Manager	4 x 4 Wagon	1
Senior Met/Maintenance Supervisor	4 x 4 Dual Cab	1
Total Administration and Process		***16***

The following mobile equipment will be supplied:

Table 4.2 Light vehicles to be supplied

Administration	Vehicle Type	No. of vehicles
Warehouse	All terrain 3 tonne forklift	1
Process Plant	**Vehicle Type**	**No. of vehicles**
Maintenance	5 t Yard Crane	1
Maintenance	5 t flat bed truck with Hiab	1
Operations	CAT 966G FEL	1
Operations	CAT 980 FEL	1
Operations	Skid steer loader	1
Total Administration and Process		***6***

5 PROCESS PLANT OPERATIONS AND ADMINISTRATION

5.1.1 General

Personnel will be recruited locally to fill the majority of the available positions. Given the requisite skills, or the ability to quickly acquire such skills, Argentinean nationals living in the Río Negro province would be ideally placed to fill the available positions.

Those positions requiring experience in gold plant and related operations will be filled by nationals with the suitable experience, in preference to expatriate labour. Expatriate positions will account for a very small, but essential, proportion of the total personnel employed.

The Calcatreu operations and manning organisation charts are shown in Appendix 5. Labour costs (summary) are provided in Section 6.4.2 for Process Plant and Section 6.5.1 for Administration.

5.2 Administration

The administration development will provide support services to the mining, operations and maintenance departments. Key functions and services will include: human resources management, accounting, payroll, purchasing, warehouse and inventory control, communications support, security environmental support, and accommodation management. Administration staffing has been discussed with AQI and agreed.

The department will be under the supervision of an Administration Manager who will report to the Resident Manager, Reporting to the Administration Manager will be the Information Technology (IT) Manager, Environmental Officer, Human Resources (HR Manager and Finance Manager.

The HR Manager's duties and functions include community relations, security, Occupational Health Safety and Environment (OHS&E), HR (medicals, inductions, training, accommodation) and travel. The HR Manager will have 35 permanent staff to carry out this administration duties shown in Table 5.1.

The Financial Manager's duties and functions include all payroll and accounting functions, capital maintenance, and purchasing and stores inventory. Seven permanent staff will assist to carry out these duties.

All community development and accounting facilities will be based at an office in Jacobacci, with the remaining administration functions being directed from site.

Table 5.1 Administration Employment Numbers Operations Manning

Administration Position	Location	Employees
Administration Manager	Jacobacci	1
Subtotal		***1***
Secretary/Receptionist	Jacobacci	1
Secretary/Receptionist	Site	1
Subtotal		***2***
IT Manager	Both	1
Subtotal		1
Environmental Officer	Site	1
Environmental Technicians	Site	2
Subtotal		***3***
HR Manager	Jacobacci	1
Community Development Officer	Jacobacci	1
OHS&E Manager	Site	1
Drivers	Both	3
Company Nurse	Site	1
Safety and Training Officers	Site	2
Security Supervisor	Site	2
Security Guards	Both	18
Accommodation Supervisor	Site	1
Housekeepers/Cleaners	Site	6
Subtotal		***36***
Finance Manager	Jacobacci	1
Chief Accountant	Jacobacci	1
Cost Accountant	Jacobacci	1
Payroll & Accounts Payable	Jacobacci	2
Supply Manager	Site	1
Purchasing Officer	Site	1
Storeperson	Site	1
Subtotal		***8***
Total		**51**

The process plant shift crew labour of 20 employees, shown in Table 5.2, provides a continuous operational coverage for 24 hours per day, 7 days per week, over 52 weeks per annum. Operations manning has been discussed with AQI and agreed.

The 4-panel roster is structured on a three shift, 24 hours/day basis. A typical shift contains 5 employees. Each shift consists of 1 Shift Supervisor and 4 Process Operators. Each shift covers operations of all continuously operated areas of the process plant (crushing, grinding, leaching and adsorption, and tailings management).

In addition to the operating shifts, 9 operators have been included to handle reagents/water requirements, carry out elution and goldroom duties, provide leave coverage for shift crew, and additional tailings management on a one shift per day basis.

The Assistant Process Manager, Mill Superintendent and Senior Metallurgist will report to the Process Facility Manager who in turn reports to the Resident Manager. The Mill

Superintendent for the Process Plant has 4 Shift Supervisors reporting directly to him/her. The manning numbers for the Process Plant are shown in Table 5.2. Key personnel will be employed sufficiently early in the implementation schedule to achieve an effective involvement in the development of operating and training programmes/procedures. Contract laboratory labour is included in the overall process plant manning levels. The costs for these personnel are included in the monthly assaying services fee.

Table 5.2 Process Plant Employment Numbers

	Process Plant Position	Employees
Supervision	Process Facility Manager	1
	Senior Metallurgist	1
	Assistant Process Manager	1
	Mill Superintendent	1
	Metallurgical Clerk	1
Shift Crew	Shift Supervisors	4
	FEL Operator	4
	Crusher Operator	4
	Mill Operator	4
	CIL/CN Detox Operator	4
Day Crew	Goldroom Operator	2
	TSF Labourer	2
	Reagents/Water Operator	1
	Shift Relief	3
	General Labourer	1
	Metallurgist	1
	Junior Metallurgist	1
	Chemist and Supervisor (Contract)	2
	Sample Preparers (Contract)	6
	Laboratory Technicians (Contract)	4
	Total Process Plant	***48***

5.3 Maintenance

The Maintenance Manager will oversee the plant maintenance activities and will report directly to the Resident Manager. The Maintenance Superintendent reports to the Maintenance Manager and supervises the Maintenance Planner, shift maintenance crew and the Day Foreman. Breakdown maintenance will be performed by the shift crew comprising of one fitter and one electrician. The Day Foreman has a total of 3 Trades people and 2 Trades assistants under his supervision. The maintenance manning numbers are shown in Table 5.3.

Maintenance, during start-up, can be high due to a higher than normal number of equipment failures. Modifications of various kinds (chutes, pipes, instruments, etc) are

required at this time and it is necessary to maintain certain construction crews to assist with commissioning activities.

Table 5.3 Process Plant Maintenance Employment Numbers

	Maintenance Position	Employees
Supervision	Maintenance Manager	1
	Maintenance Superintendent	1
	Maintenance Planner	1
Shift Crew	Electrician	4
	Fitter	4
Day Crew	Day Foreman	1
	Fitter	2
	Instrument Technician	1
	Trades Assistant	2
	Total Maintenance	**17**

6 OPERATING COST ESTIMATE

6.1.1 Summary

The Operating Cost Estimate is presented in United States dollars (USD) and uses prices obtained in 2Q06. All references to dollars or $ are to USD. The overall accuracy of the project initial capital cost estimate is ±25%.

The estimate excludes doré shipping, mining, insurance and refining costs, escalation, accuracy provisions, corporate overhead charges, financing costs, royalties, income taxes or similar imposts as well as expenditures classified as capital, sustaining capital, or rehabilitation and closure costs.

Operating cost estimates were prepared by a number of consultants for the various components of the Project. Major contributions to the estimate were made by:

- AQI participated in advice about and enquiries to local suppliers and advised on labour rates, labour loadings and manning schedules, and security needs.
- Ausenco provided the operating cost components associated with the Process Plant.

Ausenco has combined these separate inputs to create an operating cost estimate for the processing plant and administration combined.

Table 6.1 summarises the average annual costs for processing Calcatreu ore at a rate of 750,000 tonnes per year over the life of the mine.

A detailed operating cost estimate is appended in Appendix 6.

Table 6.1 Operating Cost Summary

Cost Centre	Average Annual Costs	Unit Cost
	USD M	USD/t ore treated
Processing	11.16	14.89
Maintenance	1.30	1.74
General & Administration	2.19	2.92
Total	***14.65***	***19.55***

6.2 General Estimating Parameters

6.2.1 Exchange Rates

The estimate is expressed in USD. Conversions of all quoted foreign currencies have been based upon the foreign exchange rates shown in Table 6.2.

Table 6.2 Currency Exchange Rates

Country	Unit of Currency	Exchange Rate (USD)
USA	Dollar ($)	1.00
Australia	Dollar (Aus$)	0.75
Argentina	Dollar (Arg$)	3.00

6.2.2 Escalation

Operating costs have a base case of 2Q06, with no allowance for escalation.

6.2.3 Accuracy Provision

There is no accuracy provision included in the costs.

6.2.4 Inclusions

The operating cost estimate **includes** the following:

- Rehandle of ore from run of mine stockpile to primary crusher.
- Labour cost calculations include regular bonuses, holiday and sick provisions (by way of additional “standby” staffing), social security contributions, and personal tax offsets.
- Labour costs for supervision, management and reporting of on-site organisational, commercial, technical, environmental, training and occupational health and safety activities.
- Labour costs for operating and maintaining the mobile equipment and light vehicles, process plant and supporting infrastructure as well as for monitoring of the environment.
- All power, fuels, reagents, consumables and maintenance materials utilised in operating the mobile equipment and light vehicles, process plant and supporting infrastructure as well as for monitoring of the environment.
- Operating costs of an on-site contract assay laboratory.
- Operating payments to miscellaneous minor contractors such as waste collection, specialised maintenance groups and similar.
- On-site general and administration (G&A) costs. These are detailed in Section 6.5 below. They include:
  - additional personnel costs such as safety clothing and first aid costs,
  - office costs such as communications, postage, stationery and computer supplies, and
  - operation and maintenance of the administration light vehicles and mobile equipment.

6.2.5 Exclusions

The operating cost estimate **excludes** the following:

- Costs for exploration and assessment of the viability of other potential ore resources.
- Costs for all mining activities.
- In-country or overseas corporate head office, financing, legal, banking, insurance, accounting costs and charges.
- Insurance, shipping costs and refining charges for the doré bars.
- Royalties, Value Added Tax (VAT), income taxes or similar imposts.
- Enterprise fees, licenses, land use, water use or other charges to State or Local Body Authorities other than those specifically provided for.
- Pre-production mining costs, which are included in the capital costs.
- Activities covered by the sustaining capital and closure/rehabilitation provisions.

6.3 Development of Estimated Rates and Costs

6.3.1 Labour

Development of the labour cost estimate is built around the site administration and processing plant manning list as shown in Section 5, excluding the contract laboratory labour.

AQI then provided a base monthly remuneration rate in USD for all local and expatriate personnel on that manning list according to skill level and responsibility. AQI also provided the loadings applicable to those base rates to enable determination of the total amounts directly and indirectly payable for each individual. These included a standard bonus equivalent to an additional base monthly rate. Indirect payments comprised a single loading, understood to cover elements such as social security benefits and payroll taxes.

The spreadsheet showing the development of these costs for the administration and process plant operation and maintenance, with detail of the loadings and their application, is attached as Appendix 6.

6.3.2 Power

Due the location and projected life of the project, power will provided onsite by ten 1.0 MW high speed diesel fired generators.

The cost of power generation has been calculated from a diesel consumption of 0.25 L per kWh generated plus running costs (including operating labour and maintenance costs) of USD 0.13/kWh.

6.3.3 Diesel Fuel

The cost of diesel used in this cost estimate was provided by AQI.

6.3.4 Grinding Media

Costs based on quote received from regional supplier, Molycop Chile.

6.3.5 Reagents

AQI provided pricing information for various reagents including quicklime, cyanide, carbon, caustic soda, flocculant, diatomaceous earth, and zinc dust. International prices were used for all other reagents.

6.3.6 Estimate Basis

Estimated costs have been developed as a matrix of cost type and expenditure area.

Rates and costs were generally determined in the manner described in Section 6.3.1 to Section 6.3.5. The following sources were used to arrive at the operating cost estimate:

- AQI.
- Plant design criteria.
- Vendor data.
- Operating practice, industry standards.
- Engineering Handbooks.

Processing plant operating costs are presented at two levels of detail. Section 6.4 presents a summary of the cost centres, whereas Sections 6.4.2 to 6.4.7 provide detailed cost breakdowns for the individual cost categories.

6.4 Process Plant

6.4.1 Operating Cost Estimate Summary

A summary of the total annual process operating costs, broken down by cost category, is presented as Table 6.3. A split between variable and fixed costs for process plant cost categories is also given in this table.

An alternative summary, with the same costs broken down by process plant area, is shown in Table 6.4.

Table 6.3 Summary of Processing Operating Costs by Cost Type

Cost Centre	Annualised Operating Cost for 0.75 Mt/a				
	Total		Fixed	Variable	
	USD/year	USD/t	USD/year	USD/t	USD/year
Production Labour	919,620	1.23	919,620		
Power	5,151,095	6.87	1,790,273	4.48	3,360,822
Operating Consumables	4,440,510	5.92	438,602	5.34	4,001,908
Laboratory & Assays	653,490	0.87	653,490		
Subtotal - Process	11,164,715	14.89	3,801,985	9.82	7,362,730
Plant Maintenance Labour	440,505	0.59	440,505		
Maintenance Consumables	862,010	1.15	862,010		
Subtotal - Maintenance	1,302,515	1.74	1,302,515		
Total	***12,467,230***	***16.62***	***5,104,500***	***9.82***	***7,362,730***

Note: Operating costs have been calculated on an annualised basis assuming full production. Initial ramp up periods will result in higher unit costs until throughput is stabilised

Table 6.4 Summary of Operating Costs by Plant Area

Cost Centre	Crushing USD/yr	Grinding USD/yr	CIL USD/yr	Elution/ Regeneration/ Goldroom USD/yr	Tailings/CN Destruction USD/yr	Reagents and Plant Services [2] USD/yr	General & Administration USD/yr	Laboratory USD/yr	Annual Total USD/yr
Production Labour	140,400	70,200	35,100	35,100	35,100	94,770	508,950	0	919,620
Plant Maintenance Labour	0	0	0	0	0	212,355	228,150	0	440,505
Power	180,810	3,556,737	578,404	73,950	215,813	499,829	45,552	0	5,151,095
Operating Consumables	204,185	1,838,880	958,041	1,057,131	356,789	0	25,486		4,440,510
Maintenance Consumables	196,820	289,863	75,932	36,777	41,503	221,114	0	0	862,010
Laboratory & Assays	0	0	0	0	0	0	0	653,490	653,490
Total	***722,215***	***5,755,680***	***1,647,477***	***1,202,958***	***649,205***	***1,028,068***	***808,138***	***653,490***	***12,467,230***

Notes:

1. Operating costs have been calculated on an annualised basis assuming full production. Initial ramp up periods will results in higher unit costs until throughput is stabilised.

2. Maintenance labour costs are not distributed by area.

6.4.2 Labour

The labour rates for plant operation and maintenance were developed as described in Section 6.3.1. These were applied to the labour numbers as detailed on the Organisation Chart shown in Appendix 5. A labour cost break-down is provided in Appendix 6.

A summary of labour cost by department is presented in Table 6.5.

Table 6.5 Labour Cost Summary

Labour Distribution	Annualised Cost USD/y
Plant Operation	919,620
Plant Maintenance	440,505
Total	1,360,125

6.4.3 Light Vehicles and Mobile Equipment

A schedule of light vehicles for staff and plant mobile equipment, excluding mining requirements, has been developed for the administration and process plant groups. It is detailed in Section 4 and its substance is repeated in the operating cost tables below.

Whilst mobile equipment and vehicle operating costs are inclusive of fuel and ongoing maintenance, the cost of the mobile equipment operators and maintenance staff has been included under plant labour costs.

The costs shown in these tables are distributed into several summary costing tables.

Light Vehicles

Table 6.6 shows the estimated hourly consumption of diesel and estimated hourly cost for spares and consumables for each vehicle type.

Table 6.7 includes a fleet listing and an annual estimate of running times to allow derivation of annual operating costs for each vehicle and the total fleet.

Table 6.6 Light Vehicle Fuel Use, Spares and Consumable Costs

Light Vehicle Category	Fuel Use (L/h)	Spares & Consumables USD/h
4 x 4 Wagon	7	4
4 x 4 Dual Cab	6	3
4 x 4 Utility	7	4
4 Wheel Motorbike	2	2
4 x 4 Troop Carrier	10	5

Table 6.7 Light Vehicle Running Cost

Light Vehicles	Vehicle Type	No.	Operating Time h/a	Fuel Use L/a	Fuel Cost USD/y	Spares & Cons USD/y
Administration						
Resident Manager	4x4 Wagon	1	500	3,500	1,470	2,000
Admin. Manager	4x4 Wagon	1	368	2,576	1,082	1,472
Security Supervisor	4x4 Wagon	1	1,095	7,665	3,219	4,380
Security	4 W M/Bike	1	2,190	4,380	1,840	4,380
OHS&E Group	4x4 Dual Cab	1	730	4,380	1,840	2,190
IT Manager	4x4 Dual Cab	1	368	2,208	927	1,104
Pool Cars	4x4 Dual Cab	3	730	13,140	5,519	6,570
First Aid (ambulance)	4x4 Troop Carrier	1	365	3,650	1,533	1,825
Admin	Bus	1	1,095	16,425	6,899	10,950
Total Administration		***11***		***57,924***	***24,328***	***34,871***
Process Plant						
Processing Manager	4 x 4 Wagon	1	500	3,500	1,470	2,000
Senior Metallurgist	4 x 4 Dual Cab	0.5	365	2,190	920	1,095
Shift Supervisor	4 x 4 Utility	1	1,300	9,100	3,822	5,200
TSF crew	4 x 4 Utility	1	1,300	9,100	3,822	5,200
Maintenance Manager	4 x 4 Wagon	1	500	3,500	1,470	2,000
Maintenance Superintendent	4 x 4 Dual Cab	0.5	365	2,190	920	1,095
Total Process Plant		***5***		***29,580***	***12,424***	***16,590***
Total		***16***		***87,504***	***36,752***	***51,461***

Mobile Equipment

Table 6.8 shows the hourly consumption of diesel and the estimated hourly spares and consumables costs for each piece of equipment. These have been taken from Australian standard costs, and from mines operating similarly sized equipment. A full breakdown of vehicle allocation and cost build-up is included in Appendix 6.

Table 6.9includes a fleet listing and an annual estimate of running times to allow derivation of annual operating costs for each unit and the total fleet.

Table 6.8 Mobile Equipment Fuel Use, Spares and Consumable Costs

Light Vehicle Category	Fuel Use (L/h)	Spares & Consumables USD/h
All terrain 3 t fork lift	5	10
5 t Yard crane	15	15
5 t flat bed truck with Hiab	15	10
CAT 966G FEL	35	15
CAT 980 FEL	42.5	15
Skid steer loader	20	15

Table 6.9 Mobile Equipment Fuel, Spares and Consumable Costs

Mobile Equipment	Vehicle Type	No.	Op Time h/a	Fuel Usage L/a	Fuel Cost USD/y	Spares & Cons USD/y
Administration						
Stores	3t forklift	1	1,095	5,475	2,300	10,950
Admin Subtotal		***1***		***5,475***	***2,300***	***10,950***
Process Reclaim						
Operations	Cat 966 FEL	1	1,095	38,325	16,097	16,425
Operations	Cat 980 FEL	1	8,200	348,500	146,370	123,000
Reclaim Subtotal		***2***		***386,825***	***162,467***	***139,425***
Process Plant						
Maintenance	5 t Yard crane	1	550	8,250	3,465	8,250
Maintenance	5 t flat bed truck with Hiab	1	550	8,250	3,465	5,500
Operations	Skid steer loader	1	730	14,600	6,132	10,950
Process Subtotal		***3***		***31,100***	***13,062***	***24,700***
Total		***6***		***423,400***	***177,828***	***175,075***

6.4.4 Power

The unit cost for power supply is as shown in Section 6.3.2.

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

factors, depending on drive type and duty. The equipment listing with connected and drawn power demand is attached as Appendix 6.

A summary of the installed power, anticipated average continuous power draw and annual power costs by area has been provided in Table 6.10.

Table 6.10 Power Cost Summary

Power Usage	Average kW	Annual kWh	kWh/t	Annual Cost USD	USD/t
Buildings	**40**	**350,400**		**45,552**	
Subtotal Buildings	***40***	***350,400***	***0.47***	***45,552***	***0.06***
Process Plant					
Crushing	159	1,390,847		180,810	
Grinding	3,123	27,359,518		3,556,737	
Leach and Adsorption	508	4,449,263		578,404	
Desorption and Regeneration	25	222,812		28,966	
Precipitation and Goldroom	40	346,034		44,984	
CN Detox /Tailings	190	1,660,099		215,813	
Reagents	13	113,979		14,817	
Plant Services	426	3,730,857		485,011	
Subtotal Process Plant	***4,483***	***39,273,408***	***52.36***	***5,105,543***	***6.81***
Total	***4,523***	***39,623,808***	***52.83***	***5,151,095***	***6.87***
Fixed Cost	***1,572***	***13,771,329***	***18.4***	***1,790,273***	***2.39***
Variable Cost	***2,951***	***25,852,480***	***34.5***	***3,360,822***	***4.48***

6.4.5 Operating Consumables and Reagents

Unit costs of these items were determined in the manner described in Section 6.3.3, Section 6.3.4, and Section 6.3.5. The consumption rates used in this estimate is a weighted average of the individual ore type consumptions, based on their distribution in mill feed over the life of project.

Unit costs, consumption rates, annual consumptions and annual costs have been developed and are presented in Table 6.11.

Table 6.11 Consumable and Reagent Costs Summary

Area	Price		Consumption Rate		Annual Consumption		USD/y	USD/t
	USD	Unit						
Crushing								
Jaw Liners	6,953	USD/set	6	sets/a	6	sets/a	41,718	0.056
FEL Fuel	420	/kL			386.8	kL	162,467	0.217
Subtotal							***204,185***	***0.272***
Grinding								
SAG Media	955	USD/t	0.66	kg/t	495	t	472,725	0.630
SAG Liners	102,825	USD/set	1	sets/a	1	sets	102,825	0.137
SAG Lifters	176,271	USD/set	1.33	sets/a	1.33	sets	234,969	0.313
SAG Grates	136,610	USD/set	1	sets/a	1	sets	136,610	0.182
Ball Mill Media	800	USD/t	1.33	kg/t	998	t	798,000	1.064
Ball Mill Lifters	28,432	USD/set	1	sets/a	1	sets	28,432	0.038
Ball Mill Liners	65,318	USD/set	1	sets/a	1	sets	65,318	0.087
Subtotal							***1,838,880***	***2.452***
CIL								
Lime	50	/t	1.03	kg/t	775	t	38,726	0.052
Lime Mill Media	800	/t	0.30	kg/t lime	0.35	t	278	0.000
Sodium Cyanide	1,210	/t	0.78	kg/t	586	t	708,887	0.945
Carbon	2,700	/t	0.086	kg/t	65	t	174,150	0.232
Aerator Rental	3,000	/month	12	month/ann			36,000	0.048
Subtotal							***958,041***	***1.277***
Tailings + CN Destruction								
Lime	50	/t	0.51	kg/t	384	t	19,178	0.026
SMBS	442	/t	0.71	kg/t	534	t	235,895	0.315
Copper Sulphate	1,190	/t	0.02	kg/t	15	t	17,707	0.024
Flocculant	3,300	/t	0.03	kg/t	25	t	84,009	0.112
Subtotal							***356,789***	***0.476***
Elution, Precip. + Goldroom								
Hydrochloric Acid	194	USD/t	1240	kg/strip	593	t	115,084	0.153
Caustic Soda	410	USD/t	340	kg/strip	163	t	66,089	0.089
Cyanide	1210	USD/t	340	kg/strip	163	t	196,814	0.262
Diesel-Goldroom/regen	0.420	USD/L	19,496	L/week	1014	kL	425,784	0.568
Diamtomaceous earth	85.7	USD/t	150	kg/cycle	54.6	t	4,679	0.006
Zinc	2360	USD/t	2	g/g Au+Ag	42.2	t	99,622	0.133
Lead Nitrate	600	USD/t	2	g/g Au+Ag	42.2	t	25,328	0.034
Crucibles	3375	USD/ea	12	each	12	each	40,500	0.054
Borax	1380	USD/t	130	kg/smelt	47.3	t	65,302	0.087
Sodium Nitrate	1130	USD/t	22	kg/smelt	8.0	t	9,049	0.012
Soda Ash	350	USD/t	65	kg/smelt	23.7	t	8,281	0.011
Subtotal							***1,057,131***	***1.410***
Other								
Plant LV Fuel	420	/kL			30	kL	12,424	0.017
Plant Mob. Fleet Fuel	420	/kL			31	kL	13,062	0.017
Subtotal							***25,486***	***0.034***
Total							***4,440,510***	***5.92***

6.4.6 Maintenance Consumables

Maintenance consumable costs comprise maintenance materials as well as specialist contract labour costs. Maintenance consumable costs have been estimated as a percentage of the direct installed capital cost (percent factor) on an area by area basis. The magnitude of the factors applied is related to the energy input or severity of duty in each area of the plant and is based on typical industry values. Table 6.12 shows the factors that have been applied to each area.

Table 6.12 Maintenance Consumables Annual Cost Factors

Process Area	% of Installed Capital
Crushing	3.0
Reclaiming has no fixed plant	N/A
Grinding	3.0
CIL	2.0
Regeneration	2.0
Goldroom	3.0
Tailings/Cyanide Destruction	3.0
Reagents	3.0
Utilities and Services	3.0

Application of these factors results in the estimated annual costs summarised in Table 6.13. The 'reclaiming' cost is the cost of running the plant feed front-end loader (FEL) throughout the year. These costs exclude:

- Crusher components, mill liners and lifters, and other components listed in Table 6.11, operating consumable costs.
- Maintenance labour costs.

Details of maintenance costs are included in Appendix 6.

Table 6.13 Maintenance Consumable Costs

Area	Capital Value, USDM	Factored Cost (USD'000/y)	Additional Allowance (USD'000/y)	Total Cost, (USD'000/y)	Cost (USD/t)
Crushing	1.91	57.4	0	57.4	0.077
Reclaim	0.00	0	139.4	139.4	0.186
Grinding	9.66	289.9	0	289.9	0.386
CIL	3.80	75.9	0	75.9	0.101
Regeneration	0.95	18.9	0	18.9	0.025
Goldroom	0.89	17.9	0	17.9	0.024
Tailings/CN Detox	1.38	41.5	0	41.5	0.055
Reagents	1.46	43.9	0	43.9	0.059
Utilities and Services	4.53	135.9	41.3	117.2	0.236
Total		***681.3***	***180.7***	***862.0***	***1.15***

6.4.7 Laboratory and Assaying

The annual laboratory operating cost is estimated at USD 653,490, and is based on a fixed analytical service of USD 40,275 per month and a monthly laboratory equipment supply fee of USD 14,183. The costs include provision of the contract laboratory labour, identified in the process plant manning lists.

6.5 General and Administration

6.5.1 Operating Cost Estimate Summary

A summary of the total annual administration operating costs, broken down by cost category, is presented as Table 6.14. A split between variable and fixed costs for process plant cost categories is also given in this table.

Table 6.14 Summary of Administration Operating Costs by Cost Type

Cost Centre	Annualised Operating Cost				
	Total		Fixed	Variable	
	USD/y	USD/t	USD/y	USD/t	USD/y
G & A Expenses	818,599	1.09	818,599		
Administrative Labour	1,370,655	1.83	1,370,655		
Total	***2,189,254***	***2.92***	***2,189,254***		

6.5.2 General and Administration Expenses

General and administration expense estimates have been based on a combination of experience from similar projects and site specific requirements. The total estimated cost is $0.787 M per year or $1.05 per tonne of ore processed. The estimate provisions are summarised in Table 6.15 followed by a brief discussion of the major items.

Note that no allowance has been made for costs specifically associated with mining.

Table 6.15 General and Administration Costs

Item	Annual Costs, USD
Telecommunications	33,600
Stationery	6,000
Postage, Courier and Light Freight	6,000
Computer Supplies	12,000
First Aid Costs	2,675
Metallurgical Testing	24,000
Consultants and Vendors	48,000
Safety, Clothing	16,050
Rubbish Removal	12,000
Administration Fleet Maintenance Cost	45,821
Administration Fuel Cost	26,628
Accommodation and Messing	585,825
Total	***818,599***

Telecommunications

An allowance of $33,600/year for telephone and facsimile transmissions is included to cover all communication costs. This allows for expected monthly landline and mobile telephone costs plus an allowance for international calls by expatriate workers.

Stationery

An allowance of $6,000/year has been included to cover all stationery requirements for the project.

Postage, Courier and General Freight

An allowance of $6,000/annum, has been included for general postage, courier and light freight not included under operating consumables and maintenance materials costs.

Computer Supplies

An allowance of $12,000/year has been included for computer supplies. This allows for computer servicing and minor software upgrades.

First Aid Costs

An allowance of US$25/person on site, or $2,675/year, has been included for first aid supplies and medical requirements. Note this excludes all mining personnel.

Metallurgical Testing

An allowance of $24,000/year has been provided for off site metallurgical testing in support of operations. This is in additional to the routine test-work undertaken by the on-site service provider.

Consultants and Vendors

An allowance of $48,000/year has been provided for consulting services relating to the operations. This includes an estimate for a safety consultant required to set-up a monitoring program in the first year or more of operation.

Safety Clothing

An allowance of $150/person for safety clothing is used for a total of $16,050. Clothing and personnel protective equipment includes overalls, jacket, hard hat, safety boots, safety glasses, earplugs, dust masks and gloves.

Rubbish Removal

An allowance of $1,000/month has been made for rubbish removal by a local contractor.

Administration Vehicle Fleet Maintenance Cost

An allowance of $45,821 for maintenance costs for the administration light vehicle fleet is included. Refer to Table 6.7 for details of the fleet and cost breakdown.

Administration Vehicle Fleet Fuel Cost

An allowance of $26,628 for maintenance costs for the administration light vehicle fleet is included. Refer to Table 6.7 for details of the fleet and cost breakdown.

Accommodation and Messing Cost

An allowance of $15/person per day, or $585,825/year, has been made for accommodation and messing arrangements for personnel onsite. The figure is based on Ausenco experience.

6.5.3 Administrative Labour

The labour rates for administration were developed as described in Section 6.3.1. These were applied to the labour numbers as detailed on the Organisation Chart shown in Appendix 5. A labour cost break-down is provided in Appendix 6.

Labour costs for environmental monitoring and management are included in the administration labour costs.

7 CAPITAL COST ESTIMATE

7.1.1 Summary

The project initial capital cost estimate is presented in United States dollars (USD) and has a base date of 2Q06. All references to dollars or $ are to US dollars. The overall accuracy of the project initial capital cost estimate is ± 25%.

In broad terms the estimate includes design and construction of the plant access road, power supply, mining infrastructure, process plant, TSF, water supply, on-and off-site infrastructure costs.

The capital cost estimate, as summarised in Table 7.1, is USD 66.2M. The detailed estimate and supporting data is provided in Appendix 7.

The estimate excludes escalation, duties, taxes, mining costs, working capital, Owner's costs, sustaining capital, financing costs, rehabilitation and closure costs and allowance for project growth.

Table 7.1 Capital Cost Summary

Cost Element	Estimated Cost (USD M)
Process Plant	26.6
On Site Infrastructure and Utilities	10.4
Off Site Infrastructure and Utilities	8.1
Mining Infrastructure, Haul Roads and ROM Pad Construction	2.8
Mobile Equipment, First Fill Consumables and Capital Spares	3.9
Indirects Temporary Construction Facilities, EPCM, Start-Up and Commissioning	14.4
Total Capital Cost	***66.2***

Major contributions to the estimate were also made by:

- Vendor quotes.
- Indec S.A. - undertook collection of local contractor rates.
- Vector - provided the tailings management facility design and associated MTO.

Ausenco has combined these separate inputs and applied appropriate accuracy provisions to create an overall capital cost estimate for the project.

7.2 General Estimate Parameters

7.2.1 Exchange Rates

The estimate is expressed in USD. Conversion of all quoted foreign currencies has been based upon the foreign exchange rates shown in Table 7.2.

Table 7.2 Exchange Rates

Country	Unit of Currency	Exchange Rate (USD)
USA	Dollar	1.00
Australia	Dollar	0.75
Argentina	Peso	3.00

7.2.2 Escalation

Capital costs have a base date of 2Q06 with no allowance for escalation.

7.2.3 Working Capital

The estimate does not include an allowance for working capital.

7.2.4 Taxes and Duties

VAT and Import Duties are excluded from the estimate.

7.2.5 Sustaining Capital, Rehabilitation and Closure Costs

Sustaining capital, rehabilitation and closure costs have not been included in the project initial capital cost estimate.

7.2.6 Project Growth Contingency

Contingency is excluded from the estimate. Contingency is normally provided by the Owner to allow for potential costs beyond the Engineer's control or for items that are specifically excluded from estimates, such as changes in scope, escalation of the project, currency variations and inflation; or due to delays caused by weather, demands of contractors, industrial actions, escalation of field construction labour costs above the base line escalation of 2Q06, market conditions, non-availability of ore during commissioning, etc.

Some of these items may be estimated by the Owner and included in the Owner's Costs, but in any event should be included somewhere within the overall project cost estimate.

7.2.7 Accuracy Provisions

The component estimates have been developed at bare cost (excluding accuracy provision). An accuracy provision allowance has then been allocated to each area and element of the direct and indirect costs to reflect the level of definition available in the scope of work.

The purpose of the accuracy provisions is to make allowance for uncertain elements of costs to cover such factors as:

- Restricted information on site conditions, most especially concerning sub-surface conditions and the engineering properties of excavated materials.
- Imprecision of sizing and quantities information arising from lack of detailed engineering.
- Arithmetical and procedural errors in quantity take-offs and estimate assembly and consolidation.

- Accuracy of materials and labour rates.
- Accuracy of productivity expectations.
- Accuracy of equipment budget pricing.
- Lack of direct knowledge of local contractor capabilities.
- Lack of direct knowledge of local permitting methods, procedures and outcomes.

The sum of the estimated bare costs and accuracy provisions is the total estimated cost for the project with an overall accuracy of ± 25%.

It is Ausenco's standard practice to include accuracy provision in all estimates and it does not duplicate level of accuracy.

7.2.8 Assumptions

The following assumptions underlie this estimate:

- The design is based upon available testwork information.
- The design is as detailed in the relevant sections of this report.
- The design will meet local environmental requirements.
- All materials from clearing, grubbing and demolition will be disposed of at the project site.
- Sand, aggregate and construction water is available within a radius of 2km from the plant site.
- Fill materials are generally available from mine waste or from quarries located within the lease boundary of the project site.
- Transportation access is available to port/road systems from point of manufacture to project site including border crossings.
- The assessment and understanding of current local costs obtained by the methodologies described below is correct within the accuracy limits adopted.

7.2.9 Exclusions

The project initial capital cost estimate excludes the following:

- Mining, other than mine workshop building.
- Owner's costs.
- Sunk costs for the completion of this report.
- Ongoing exploration or acquisition costs.
- Ongoing test work.
- Ongoing studies.
- Taxes and duties other than those specifically allowed for.
- Project growth occurrences such as foreign currency fluctuations, escalation and others as described above.

- Sustaining capital.
- Corporate costs.
- Rehabilitation and closure costs.

7.3 Direct Cost Development and Detail

Direct costs include:

- Supply of permanent materials and fixed and mobile equipment.
- Labour to undertake and manage the construction activities. This includes wages and salaries with loadings for site labour, supervision and management, including associated expenses such as accommodation and travel, and home and/or satellite office management expenses.
- Contractors and suppliers mark-up and profit.
- Freight and shipping expenses for permanent and temporary equipment and materials.

7.3.1 Development of Supply Costs, Freight Costs and Construction Rates

Local Supply and Contracting Rates and Capabilities

It is intended to use local capabilities to the maximum extent feasible. This may include:

- Construction equipment hire, including that for earthworks, civil works, cranes, and transportation.
- Labour hire.
- Supply and fabrication of building materials, structural steelwork, mechanical platework, mechanical/piping and electrical components.
- Contracting for earthworks, civil, building, structural steel, tankage, mechanical installation, piping and electrical installation.

Supply of Mechanical Equipment

Estimated costs were derived from two main sources:

- Responses to inquiries for budget prices.
- Ausenco historical costs for the same or similar equipment.

Freight Costs

The freight costs for mechanical equipment have been factored using a percentage based on previous project experience.

Freight for rates items is included in the overall rate as quoted by local contractors.

7.3.2 Direct Cost Estimate

Cost estimates for the disciplines represented in the above components have been developed as follows:

Earthworks

Bulk earthworks cost allowance for the plant site was based on preliminary earthwork drawings in sufficient detail to provide approximate quantities for estimation.

Costs for the ROM pad construction and the haul roads were estimated from the rates provided by the mining contractor and the respective take-offs.

Concrete

Quantity take-offs were made from GA drawings combined with knowledge of quantities on similar Ausenco projects. Local construction rates, covering supply and execution of detailed excavation, blinding concrete, formwork, rebar and embedment, concrete, finishing, stripping, curing and detailed backfill and compaction, from local budget prices as described above, were applied to the estimated quantities.

Structural Steelwork

Quantity take-offs were made from GA drawings combined with knowledge of quantities on similar Ausenco projects. Local supply, transport and erection rates, covering shop detailing, supply, fabrication, surface treatment, transport and erection for steelwork, grating and handrails, from local budget prices as described above, were applied to the estimated quantities.

Mechanical Platework and Site Erected Tankage

Mechanical platework and site erected tankage costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Mechanical Equipment Supply

Equipment lists were developed from the process flowsheets. These provided equipment numbers, type, sizing and power. Prices for equipment were sourced from international suppliers using budget quotations or from Ausenco's data base of recent projects and estimates.

Mechanical Installation

Installation of mechanical equipment and platework is based on Ausenco's historical man hour data from similar gold process plants appropriately adjusted to represent local man hours by applying a productivity factor. The adjusted manhours, together with the labour rate, have been used to obtain total labour costs.

An additional allowance for heavy lift cranes has been included in some instances.

Piping Supply and Installation

Process plant piping costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Pipe runs external to the process plant compound have been quantified from GA drawings and have been estimated separately based on local budget costs as described above.

Electrical and Instrumentation

Electrical and instrumentation costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Buildings

Building areas for the process plant, comprising the crusher control room, main process building, main control room and plant building offices and ablutions, were taken from GA drawings. Construction unit rate estimates were obtained and applied to the measured areas. These unit rates include the building shells, internal fit-out, electrics, lighting, HVAC, furniture and office equipment.

The main process building also contains the plant and warehouse and plant workshop. Supplementary provision is made for warehouse shelving and fixed workshop machines, hand tools and specialized electrical/instrumentation maintenance equipment and lubrication gear.

Freight

Freight allowance to transport all equipment and materials to site has been calculated by percentage based on previous project experience. The majority of fabricated items such as structural steelwork, platework and piping will be sourced in-country, and local freight costs are included in the quoted rates or included in the factored cost as applicable.

First Fill Reagents and Consumables

Costs for supply and delivery of reagents and consumables have been factored based upon Ausenco's experience with similar installations.

Capital Spares

A capital spares provision is included in the estimate and has been factored from previous experience with similar installations

7.3.3 Tailings Storage Facility

Cost estimates for the TSF have been developed by Vector, who provided quantity take-offs for ground preparation, earthworks, HDPE liner, pipe work and access tracks from preliminary drawings of the TSF, and applied local rates to arrive at the cost estimate.

7.3.4 Infrastructure

Plant Site Buildings

These comprise the administration building, core storage and laboratory building.

The administration building area was determined from the GA drawings. A construction unit rate estimate was obtained and applied to this measured area. This unit rate includes the building shell, internal fit-out, electrics, lighting, HVAC, furniture and office equipment.

The estimate for the core storage building was obtained in the same manner as for the main plant building, that is, a package budget price for design, supply and packing ex works for a complete steel framed and clad building, from a specialist supplier. Some additional provisions have been included to allow the installation of a laboratory building and sample preparation area within the shell of the core shed.

Mine Buildings

The mine buildings comprise a workshop for heavy mining equipment maintenance and the explosive magazine.

The building area for the workshop was taken from GA drawings. Construction unit rate estimates were obtained and applied to the measured areas. These unit rates include the building shells, internal fit-out, electrics, lighting, HVAC, furniture and office equipment. Supplementary provision is made for warehouse shelving and fixed workshop machines, hand tools, and mining equipment lubrication gear.

The magazine estimate was developed from first principles. It encompasses the earthworks and high grade security fence for the magazine compound, a small ANFO shed and the magazine proper, comprising two modified shipping containers.

Communications System

This includes supply and installation of a local radio base station and repeater station with associated hand held and vehicle mounted units, a microwave mini-link for voice and data communication, a phone system throughout the plant area and a data wiring system throughout the plant offices. Budget prices for these systems were obtained from previous projects.

Access Roads

The cost of the off-site roads was estimated from rates provided by Micon International Ltd., as requested by AQI, for road construction.

Quantities were obtained from the preliminary design of the road and applied rates to arrive at the estimated cost.

Accommodation Camp

The cost of the accommodation village is based upon a previous design including sleeping quarters, recreation facilities, kitchen, mess, first aid post complete with services and utilities. The accommodation village has been sized to accommodate a labour force of 200 people. The camp will be used during construction and, if required, it will be supplemented with temporary trailer type accommodation.

The cost of constructing the village has been estimated based upon the building costs per square meter provided by local contractors. An allowance has been included for fit-out.

7.4 Indirect Cost Development and Detail

Indirect costs have been factored from the direct costs based upon experience with similar previous projects. The indirect costs include:

- EPCM services, together with supervision and commissioning of the plant.
- Temporary construction facilities and services such as site access and laydown, site offices, ablutions, security and storage buildings, fences, security/cleaning/ maintenance services, water, power and fuel supply, sewage treatment/disposal, communications, first aid and safety equipment, contractors' mobilisation and demobilisation and site establishment costs, survey crew and vendor representative attendance.

- Start-up and commissioning.

7.4.1 Indirect Costs Breakdown

Cost estimates for the indirect components have been developed as follows:

Temporary Construction Facilities and Services

Temporary construction facilities and services have been factored from the direct costs based on previous similar projects. The derived cost includes:

- Offices for EPCM staff, including allowances for fit-out, cleaning, maintenance and consumables.
- Construction personnel ablution building, including allowances for fit-out, cleaning, maintenance and consumables were included.
- Hire and operation of a package temporary sewage treatment plant.
- Security hut, (it was assumed that the permanent fence would provide site security).
- Temporary water supply system comprising pumps, piping, tanks and water treatment plant for site ablutions.
- Temporary power supply system (allows for the hire and operating costs for an on-site generator for site ablutions and EPCM contractor's site office).
- Temporary communications (allows for short term hire of a satellite dish communications link prior to installation of the permanent link and long term provision of radios for construction personnel).
- Fuel storage (allows for the supply of a small storage facility until permanent installation is completed in time for start of mine pre-operations phase).
- Safety barriers, safety signs, safety training materials and first aid supplies.
- Security personnel.
- Site clean-up personnel.
- Contractor mobilisation/demobilisation and site establishment.
- Vendor representation costs for on site construction supervision and pre-commissioning activities where applicable.

EPCM Costs

The EPCM cost has been factored from the direct costs based on Ausenco's experience with similar current projects, which varies between 18% and 26%.

The EPCM cost includes:

- EPCM Labour

  Home office and site based time-based costs for project management, project controls, procurement/contracting, site construction management and secretarial.

- EPCM Expenses

EPCM expenses include costs for consultants, inspection/expediting services, home and site office expenses, airfares, phone, postage, copying, stationary and computer systems.

- Sub Consultants

  Costs for sub consultants are also included under the EPCM costs.

- Commissioning

  This covers the estimated costs of construction contractors providing plant start-up assistance during commissioning and associated miscellaneous materials and equipment.

  The costs for the EPCM Contractor's commissioning group are included.

  Costs associated with vendor commissioning assistance are also included.

- Start-up and Commissioning

  This covers the estimated cost of providing plant start-up experts from the EPCM group based on previous projects. This cost will allow for commissioning manager, a senior mechanical engineer and plant metallurgists to provide round the clock supervision and guidance during the pre-commissioning and commissioning phase.

7.5 Owner's Costs

Owner's costs are excluded from the estimate.

8 ALTERNATIVE PROCESS ROUTES

8.1.1 Summary

In parallel with the IFS based on a CIL/zinc precipitation process, AQI requested Ausenco conduct two concept studies covering two alternative process routes. These alternative process flowsheet options are:

- Gravity concentration followed by flotation.
- Heap leach.

The objective of the option studies was to evaluate the technical and economic viability of the two alternate processes compared to that presented in the IFS. The deliverables for these studies are:

- Conceptual flowsheets.
- Plant capital and operating cost estimates.

The operating and capital cost estimates have an accuracy of ±40%.

Table 8.1 lists the key project indicators applicable to each of the alternative process options.

Table 8.1 Summary of Key Project Indicators

Criteria	Units	Gravity/Flotation	Heap Leach
Mining rate	t/y	750 000	750 000
Cut off grade	Au g/t	1.25	1.25
Average LOM grade, Au	Au g/t	3.86	3.86
Average LOM grade, Ag	Ag g/t	33.20	33.20
Overall metal recovery			
Au	%	78	65
Ag	%	78	65
Product produced		Filter cake	Doré
	t/y	252 000	18.07
Metal produced			
Au	t/y	2.26	1.88
Ag	t/y	19.42	16.19
Operating cost	USD/t	14.89	11.56
	USDM/y	11.32	8.67
Project capital cost (direct & indirect cost)	USDM	60.5	56.1
Deferred capital	USDM	n/a	17.0

Further details on the operating cost estimate for the gravity/flotation option and operating cost estimate for the heap leach option are included in Section 8.2.3 and Section 8.3.3, respectively.

The details on capital cost estimates for the gravity/flotation option and the heap leach option are presented in Sections 8.2.4 and Section 8.3.4, respectively.

The preliminary design criteria, conceptual process flowsheets and preliminary equipment lists for the respective flowsheet options are included in Appendix 8.

8.2 Gravity and Flotation Process Option

8.2.1 Design Basis

Gravity and Flotation

The basis for the gravity/flotation process flowsheet development is the testwork results obtained by Gekko in Australia. Gekko performed a number of scouting tests to determine the suitability of gravity separation followed by flotation for the recovery of gold. The test work was conducted on a composite sample prepared by AMMTEC. The composite sample consisted of similar amounts of oxidised vein (Cal-1) and unoxidised vein (Cal-2) material.

This testwork consisted of gravity concentration of a milled sample by shaking table followed by the upgrading of the table tailings by a batch centrifugal bowl style of gravity concentrator. Subsequently, a flotation test was carried out on the tailings stream from the batch centrifugal bowl concentrator.

The gravity separation testwork demonstrated that a gold concentrate of 44.2 g/t can be produced at a 33.7% mass recovery.

The results obtained from shaking table tests are indicative of the expected recovery of coarse high specific gravity (SG) particles and simulate the performance of an in-line pressure jig (IPJ). The flowsheet for this conceptual study is based on the use of an IPJ followed by the scavenging of the IPJ tail (float) fraction by means of a batch centrifugal gravity bowl unit i.e. Falcon or Knelson concentrator. The IPJ model size is based on treating as much as possible of the equivalent of the new feed within a single unit. For that reason the largest IPJ unit currently available was selected, which accommodates a feed rate of 88 t/h solids, equivalent to 91% of new feed.

The model size of the batch centrifugal gravity bowl is selected to match the feed tonnage of the gravity section.

The results of the flotation tests carried out on the table tailings show a 31.3% mass pull at a concentrate grade of 4.29 g/t Au with a recovery of 44.4%. The Gekko report contains little information on the test conditions under which the flotation test was carried out. For this reason the flotation cell residence time and reagent additions are based on the tests carried out by AMMTEC. However, metal recovery is assumed to be as per the tests results reported by Gekko. This is done because Gekko performed the flotation tests on the tailings of the gravity circuit, whereas AMMTEC performed the flotation tests on the composite sample.

The critical design parameters for the combined gravity and flotation flowsheet are summarised in Table 8.2.

Table 8.2 Critical Design Parameters

	Unit	Value	Mass Pull %	Grade, g Au/t	Recovery, % Au
Gravity concentrate			2.3	44.2	33.7
Flotation concentrate			33.6	7.0	78.0
Flotation Conditions					
Time	min	10			
Reagent addition					
Activator	g/t	100			
Promotor	g/t	100			
Collector	g/t	100			
Frother	g/t	5			

A detailed description of the testwork performed by Gekko is included in Section 2.14.3 and the original report is included in Appendix 2.

Gekko assayed the samples for gold content only. For this conceptual study however, it has been assumed that silver follows the gold in both the gravity and flotation processes.

All further assumptions and input data relevant for the sizing of the flowsheet are included in the design criteria in Appendix 8.

At the Universidad Nacional de San Juan (UNSJ) similar gravity followed by flotation tests were performed. However, high recoveries of gold to the gravity concentrate, much higher than those obtained in the test work carried out by AMMTEC and Gekko, were achieved. It should be noted that the AMMTEC and Gekko work was conducted on portions of the same sample composites, whereas different samples were sent to UNSJ.

Further details on the testwork carried out at UNSJ are included in Section 2.15.

Crushing and Grinding

The design in this alternative flowsheet is identical to the crushing and grinding plant section in the CIL option. The design basis of this circuit is covered Section 3.3.1.

Concentrate Handling

In the absence of applicable testwork data, the sizing of the concentrate thickener and concentrate filter is based on data from the Ausenco in-house database.

A concentrate storage shed has been sized to store six days of concentrate production with an additional 1 day's storage capacity provided in the filter building.

It has been assumed that a low grade gold/silver concentrate can be sold to a third party. However, this assumption requires confirmation prior to further development of this option.

Tailings Handling

The tailings thickener has been sized using a higher settling rate than that determined by Supaflo on the CIL tailings, as the particle size of the tailings from the gravity and flotation circuit is likely to be coarser (less particle degradation due to leaching).

8.2.2 Process Plant Description

The following narrative provides a brief description of the gravity and flotation flowsheet option and should be read in conjunction with the conceptual process flow diagram (1497-F-201) and mechanical equipment list included in Appendix 8.

Ore Receipt and Crushing

Run of mine (ROM) ore is delivered by the mine haul trucks onto a ROM pad. A front-end loader (FEL) is used to feed the ROM ore over a static grizzly located above a ROM bin. The grizzly is used to remove large oversize particle/rocks, which may choke the primary jaw crusher. The jaw crusher is sized to accommodate a rock having a maximum single dimension of 900 mm.

The jaw crusher product gravitates onto a sacrificial conveyor. A sacrificial conveyor is used to minimize damage to the belt in the event that an occasional piece of tramp iron bar is trapped in the crusher and thus causing the belt to rip.

Milling Circuit

Primary grinding is carried out in a single milling line consisting of a SAG mill closed circuit with a trommel screen followed by a ball mill in closed circuit with cyclones.

The SAG mill discharges onto a trommel screen. Coarse oversize, pebbles and worn media, is captured on the screen and is re-circulated by means of a conveyor to the SAG mill feed for another pass through the SAG mill.

The ball mill discharges via another trommel screen. The ball mill trommel screen undersize gravitates to the cyclone feed hopper and is combined with the SAG mill trommel undersize slurry. The ball mill trommel screen oversize product is captured in a bunker. The combined trommel underflow slurry is directed to the cyclones.

The cyclone overflow from the grinding circuit gravitates to the trash screen.

The cyclone underflow stream passes through a distribution box with two outlets, one outlet directs the slurry to the ball mill feed spout and the second outlet directs the slurry stream to the gravity separation circuit. The gravity separation feed stream has throughput capacity for 88 t/h solids.

Gravity Circuit

The gravity circuit feed stream flows through a distribution box onto a scalping screen. The scalping screen oversize fraction (>6 mm) reports to the ball mill feed chute, and the undersize gravitates into an IPJ for coarse gold and silver particles recovery. This concentrate reports to the sink fraction (concentrate) outlet and is pumped to the concentrate thickener.

The float (tailings) fraction of the IPJ is pressurised and is directed into a surge tank which feeds a batch centrifugal bowl concentrator. The batch centrifugal bowl concentrator recovers fine gold and silver that are present in the IPJ tailings stream. Either a Falcon or Knelson concentrator could be used for this duty. The operation of the batch centrifugal bowl concentrator is such that the concentrate is periodically recovered. This is a fully automated process and no manual interaction is required. The bowl concentrate is combined with the IPJ concentrate in a hopper and pumped to the concentrate thickener. The tailings stream of the batch centrifugal bowl concentrator gravitates into the cyclone feed hopper.

As an alternative to feeding the gravity circuit with a portion of the cyclone underflow stream, a provision is made such that a bleed from the cyclone feed can be directed to the gravity circuit. This can be achieved by directing one or more of the outlets of the cyclone distributor to the IPJ. The advantage of having a dedicated port feeding the IPJ is that the pressure in the cyclone feed distributor is maintained. As the result of the consistent pressure in the cyclone distributor, the cyclone cut point and the mill circulating load remain consitent.

Flotation Circuit

The cyclone overflow slurry gravitates onto a trash screen which prevents coarse material from entering the downstream flotation circuit.

The trash screen undersize gravitates through a pipe launder into a conditioner tank to which flotation reagents are added. The conditioner tank overflows to a bank of rougher flotation cells. The tailings stream of the rougher section gravitates to a bank of middlings flotation cells and similarly the tailings of the middlings bank gravitate to a bank of scavenger cells.

The gold-silver concentrates from each of the three flotation banks are combined in a hopper and pumped to the concentrate thickener.

Concentrate Handling

The concentrate from the gravity circuit and the flotation cells is pumped to a concentrate thickener for dewatering. The recovered water from the thickener overflows into the process water tank. The thickened concentrate is pumped by a peristaltic pump into a mechanically agitated concentrate stock tank.

The contents of the concentrate stock tank are pumped to a recessed plate and frame filter press. Only when the filter is ready to receive the slurry, an actuated valve opens and the slurry flow is directed into the filter. Once the filter is full, the slurry is diverted back to the stock tank and the slurry continues to be recycled.

The filter cake is formed in the filter chambers and is air dried using compressed air. After the air blow, the filter press is opened and the cake discharged into a concentrate bunker below. The filtrate gravitates to the process water tank.

A concentrate storage shed is provided to store the filtered concentrate. The storage shed has a capacity of 6 days of production. The filter building is designed such that a further one day's concentrate production may stored in the bunker below the filter press.

Tailings Handling

The scavenger flotation tailings slurry is collected in the flotation tailings hopper and pumped to the tailings thickener for water recovery. The overflow from the thickener gravitates into the process water tank. The thickened tailings are transferred by two-stage pumping to a tailings storage facility.

Reagent Area

Day storage tanks for the various reagents and the mixing facilities for collector and activator are located in the reagent area of the plant.

Reagent dosing pumps are located next to the day tanks and transfer the reagents to the conditioner tank and downstream in the flotation circuit.

A storage shed is provided for reagent storage and protection.

Infrastructure and Services

The infrastructure and services for this flowsheet option is identical to the CIL based flowsheet and is described in detail in Section 4.

Plant Control System

This plant is operated and controlled via a PLC/Scada plant control system. The control system is centralised in a control room.

8.2.3 Operating Cost Estimate

An operating cost estimate has been prepared for the stand alone Calcatreu concentrating plant treating 750,000 t/y ROM ore producing 252,000 t/y of a gold and silver rich concentrate. The costs are expressed in USD and are current during 2Q06. The cost estimate is accurate to ±40%.

The operating cost estimate is based on the process design criteria and the information obtained from the following sources:

- Metallurgical test work.
- Data provided by AQI.
- Quotations from suppliers.
- Ausenco's in-house database.

A summary of the estimated operating costs for the gravity/flotation option is presented in Table 8.3.

Table 8.3 Operating Cost Estimates

Cost Centre	USDM/y	USD/t
Electrical power	4.71	6.28
Labour - General and Administration	1.37	1.83
Labour - Process and Maintenance	1.54	2.05
Consumables	2.43	3.24
Maintenance	0.33	0.44
Sample assays	0.15	0.00
General and Administration	0.79	1.05
Total	***11.32***	***14.89***

Basis of Operating Cost Estimate

The process plant design criteria and conceptual plant mass balance provided the basis for the operating cost estimate.

Unit rates for process consumables have been estimated from vendor information, current projects and in-house data bases.

The unit labour and diesel fuel costs for the gravity/flotation process option are identical to those used in the CIL process option, as supplied by AQI.

The cost of maintenance consumables has been calculated as a percentage of the plant mechanical equipment, piping and electrical equipment cost. The factors are based on industry standard estimating experience with similar metallurgical operations.

A mechanical equipment list has been compiled from which an electrical load list, based on estimated annual operating hours and, service factors, has been developed. This enabled the determination of the annual electrical energy consumption from which the power plant has been sized. An allowance for power demand has been included to cover the site lighting and infrastructure power requirements.

The unit cost of electrical power is US$0.13/kWh.

Qualifications

The operating cost estimate excludes the following costs:

- Mining and ore haulage.

- Marketing, transportation and sales of the gold/silver concentrate.
- Royalties.
- Levies and taxes.
- Bonds.
- Permits.
- Currency fluctuations.
- Insurances other than workers compensation.
- Support services of the head office.
- TSF maintenance.
- Community and heritage related issues.
- Environmental requirements.
- Working capital.
- Contingency allowance.

8.2.4 Capital Cost Estimate

Summary

The project initial capital cost estimate is presented in US dollars, has a base date of the 2Q06, and is to an accuracy of ±40%. All references to dollars or $ are to US dollars.

In broad terms the estimate includes design and construction of the plant access road, power supply, process plant, TSF, water supply, on and off site infrastructure costs.

The estimate excludes escalation, duties, taxes, mining costs, working capital, sustaining capital, financing costs, rehabilitation and closure costs and project contingency. The estimate is summarized in Table 8.4.

Table 8.4 Breakdown of Capital Cost Estimate

Cost Centre	USDM
Direct Cost	
Process Plant	
Crushing	2.1
Grinding	11.2
Gravity Concentration	2.7
Flotation	1.8
Concentrate Handling	0.8
Reagents	0.5
Utilities	4.7
On-Site Infrastructure	10.1
Off-Site Infrastructure	6.8
Mining Miscellaneous	2.8 3.7
Sub-total Direct Cost	***47.2***
Indirect Cost	
Temporary Construction Facilities	2.0
EPCM	10.4
Start up and Commissioning	0.8
Sub-total Indirect Cost	***13.2***
Total Project Cost	***60.4***

Exchange Rates

The estimate is expressed in USD and the exchange rates used are those given in Section 7.2.1.

Escalation

Capital costs have a base date of 2Q06 with no allowance for escalation.

Working Capital

The estimate does not include an allowance for working capital.

Tax and Duties

Value added tax (VAT) and Import Duties are excluded from the estimate.

Sustaining Capital, Rehabilitation and Closure Costs

Sustaining capital, rehabilitation and closure costs are excluded from the estimate.

Project Contingency

Project contingency for changes, which arise from outside or unpredictable circumstances such as exchange rate variations from the estimate basis and escalation of field construction labour costs above the base line estimate, has been excluded.

Accuracy Provisions

The component estimates have been developed at bare cost (excluding accuracy provision). An accuracy provision allowance has then been allocated to each area and element of the direct and indirect costs to reflect the level of definition available in the scope of work.

The purpose of the accuracy provisions is to make allowance for uncertain elements of costs to cover such factors as:

- Restricted information on site conditions, most especially concerning sub-surface conditions and the engineering properties of excavated materials.
- Imprecision of sizing and quantities information arising from lack of detailed engineering.
- Arithmetical and procedural errors in quantity take-offs and estimate assembly and consolidation.
- Accuracy of materials and labour rates.
- Accuracy of productivity expectations.
- Accuracy of equipment budget pricing.
- Lack of direct knowledge of local contractor capabilities.
- Lack of direct knowledge of local permitting methods, procedures and outcomes.

Assumptions

The following assumptions underlie this estimate:

- All costs are based on new equipment.
- The design is based upon available test work information.
- The design is as detailed in the relevant sections of this report.
- The design will meet local environmental requirements.
- All materials from clearing, grubbing and demolition will be disposed of at the Project site.
- Fill materials are generally available from mine waste or from quarries located within the lease boundary of the project site.
- Transportation access is available to port/road systems from point of manufacture to project site including border crossings.
- The assessment and understanding of current local costs obtained by the methodologies described below is correct within the accuracy limits adopted.

Exclusions

The Project Initial Capital Cost Estimate excludes the following:

- Owner's costs.
- Sunk costs to the completion of this report.

- Ongoing exploration or acquisition costs.
- Ongoing test work.
- Ongoing studies.
- Taxes and duties other than those specifically allowed for.
- Project growth occurrences such as foreign currency fluctuations, escalation and others as described above.
- Sustaining capital.
- Corporate costs.
- Rehabilitation and closure costs.

8.2.5 Direct Cost Development

Direct costs include:

- Supply of permanent materials and fixed and mobile equipment.
- Labour to undertake and manage the construction activities. This includes wages and salaries with loadings for site labour, supervision and management including associated expenses such as accommodation and travel, and home and/or satellite office management expenses.
- Contractors and suppliers mark-up and profit.

Development of Supply and Freight Costs

Supply of Mechanical Equipment

Estimated costs were derived from two main sources:

- Responses to inquiries for budget prices.
- Ausenco historical costs for the same or similar equipment.

Freight Costs

The freight costs for mechanical equipment have been factored using a percentage based on previous project experience.

Freight for rates items is included in the overall rate as quoted by local contractors.

Direct Cost Estimate

Cost estimates for the disciplines represented in the above components were developed as follows:

Earthworks

Bulk earthworks cost allowance for the plant site was estimated based on previous similar projects.

Costs for the ROM pad construction and the haul roads were estimated from the rates provided by the mining contractor and the respective take-offs.

Concrete

Concrete costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Structural Steelwork

Structural steel costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Mechanical Platework and Site Erected Tankage

Mechanical platework and site erected tankage costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Mechanical Equipment Supply

Equipment lists were developed from the process flowsheets. These provided equipment numbers, type, sizing and power. Prices for equipment were sourced from international suppliers using budget quotations or from Ausenco data base of recent projects and/or estimates.

Mechanical Installation

Installation of mechanical equipment and platework is based on Ausenco historical man hour data from similar gold process plants appropriately adjusted to represent local man hours by applying a productivity factor. The adjusted manhours, together with the labour rate, have been used to obtain total labour costs.

An additional allowance for heavy lift cranes has been included in some instances.

Piping Supply and Installation

Piping costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Electrical and Instrumentation

Electrical and instrumentation costs have been factored as a percentage of the mechanical equipment drawn on experience from similar plants.

Buildings

Building areas have been estimated. Construction unit rate estimates were obtained and applied to the estimated building areas.

Freight

Freight allowance to transport all equipment and materials to site has been calculated by percentage based on previous project experience.

First Fill and Consumables

Costs for supply and delivery of reagents and consumables have been factored based upon Ausenco experience with similar installations.

Capital Spares

A capital spares provision is included in the estimate and was factored from previous experience with similar installations

Tailings Management Facility

Cost estimates for the TSF have been developed for the gravity/flotation option from estimates provided by Vector Engineering for the CIL process option.

Infrastructure

Communications System

This includes supply and installation of a local radio base station and repeater station with associated hand held and vehicle mounted units, a microwave mini-link for voice and data communication, a phone system throughout the plant area and a data wiring system throughout the plant offices. Budget prices for these systems were obtained from previous projects.

Access Roads

Quantities were derived from the preliminary road design and rates provided by Micon International Pty, as requested by AQI, for access road construction.

Accommodation Camp

The cost of the accommodation village is based upon a previous design. The accommodation village is sized such to accommodate the construction labour force and will also be used by operations upon completion of construction.

The cost of constructing the village has been estimated based upon the building costs per square meter provided by local contractors. An allowance has been included for fit-out.

Indirect Costs Breakdown

Indirect have been factored from the direct costs based upon previous experience with similar past projects. The cost estimates for the indirect components were developed as follows:

Temporary Construction Facilities and Services

Temporary construction facilities and services have been factored from the direct costs based on previous similar projects. The derived cost includes the following:

- Offices for EPCM staff.
- Construction personnel ablution building.
- Temporary sewage system (allows for hire and operation of a package treatment plant).
- Security hut (it was assumed that the permanent fence would provide site security).
- Temporary water supply system.
- Temporary power supply, (allows for the hire and operating costs for an on-site generator for site ablutions and EPCM contractors site office).

- Temporary communications.
- Fuel storage (allows for the supply of a small storage facility unit until permanent installation is completed in time for start of mine pre-operations phase).
- Safety barriers, safety signs, and safety training materials and first aid supplies.
- Security personnel.
- Site clean-up personnel.
- Contractor mobilisation/demobilisation and site establishment.
- Vendor representation costs for on site supervision and pre-commissioning activities where applicable.

EPCM Costs

The EPCM cost has been factored from the direct costs based on Ausenco's experience with previous similar projects. The EPCM cost includes:

- EPCM labour.
- EPCM expenses.
- Sub consultants.

Start-up and Commissioning

This covers the estimated cost of providing plant start-up experts from the EPCM group based on previous projects.

Owner's costs

Owner's costs are excluded form the estimate.

8.2.6 Recommendations

The following recommendations suggested here are applicable to this conceptual gravity/flotation option only:

- Due to the limited gravity and flotation test work carried out for this study it is recommended that a series of tests are conducted to determine the concentrate grades and associated recoveries with varying head grades and ore types.
- Further development of the flotation flowsheet by carrying out re-grinding, reagent and pH screening tests as well as locked cycle flotation tests.
- It should be investigated if the concentrate can be sold economically to a third party.

8.3 Heap Leach Process Option

This section of the report presents the conceptual study findings covering the heap leach process option.

8.3.1 Design Basis

Samples of oxidised mineralised andesite and unoxidised mineralised andesite were leached for five days at coarse crush sizes to assess amenability for treating by heap leach. Full details of the heap leach testwork can be found in Section 2.12. The head grades of the samples were much lower than intended. A trend of increasing gold extraction with finer crush size is evident; although poor leach extraction was found in all tests.

The testwork programme had allowed for a column leach test on each sample at a selected crush size. However, it was decided not to proceed with this work, as the sample grades were below that of a likely feed to a heap leach operation.

The testwork programme had also provided for heap leaching on oxidised and unoxidised vein material, but samples were not available for this work.

For the above reasons the heap leach conceptual study is based on the assumption that the ore is amenable for heap leaching. The following key design parameters were assumed and used:

- Assumed average metal recovery of 65% gold and silver.
- Leach duration 180 days.
- Irrigation rate 0.0015 $m^3/m^2/h$.

8.3.2 Process Description

The following narrative provides a brief description of the main heap leach process option and should be read in conjunction with the conceptual process flow diagram (1497-F-801) and mechanical equipment list included in Appendix 8.

Ore Receipt

Run of mine (ROM) ore is delivered by the mine haul trucks onto a ROM pad. A FEL is used to feed the ROM ore over a static grizzly located above a ROM bin. The grizzly is used to remove large oversize particle/rocks which choke the primary jaw crusher. The jaw crusher is sized to accommodate a rock having a maximum single dimension of 900 mm.

The jaw crusher product gravitates onto a sacrificial conveyor which is used to bare the brunt of tramp and oversize material and thus prolong the life of the downstream equipment.

Crushing and Screening

Further size reduction of the primary crushed ore is achieved by secondary and tertiary crushing in closed circuit with a screen. The closing size of the screen is 12 mm.

A conveyor transfers the primary crushed ore to a triple deck sizing screen. The top deck acts as a relief deck to prevent damage of the screen deck by large particles. The oversize from the top and middle decks combines and is conveyed to a surge feed bin ahead of the secondary crusher. The oversize from the bottom deck is conveyed to a surge feed bin ahead of the tertiary crusher. Metal detectors are installed above the crusher feed conveyors to prevent tramp metal entering the cone crushers.

Both crushers are choke fed to achieve maximum crushing efficiency. The crushed products from the crushers gravitate onto the sizing screen feed conveyor.

The undersize from the bottom deck of the sizing screen gravitates onto a product conveyor which discharges into an agglomerator feed hopper.

A dust collector while a wet dust scrubber captures dust generated at the sizing screen.

Agglomeration and Stacking

The agglomerator is fed from the feed hopper by a variable speed belt feeder. Quicklime for pH control and cement for binding of the fine particles are added onto the ore on the belt feeder. Both the lime and cement silos are located above the agglomerator feed belt.

Water is added to the agglomerator to aid in the agglomeration process. The water addition is ratio controlled with the feed rate such that an agglomerate moisture content of 10% w/w is obtained.

The agglomerator discharges onto a transfer conveyor which in turn, discharges onto the first of a series of stacking conveyors. The stacking conveyors transport the agglomerates to a horizontal bandwagon conveyor and radial stacker conveyor with a “stinger”. The agglomerated ore is then stacked on a heap leach pad, with the spreading action being controlled by the stacker operator.

The stacker and horizontal bandwagon conveyor are retracted daily using a front end loader. Once the stacker and bandwagon are fully retracted the stacking system must be shut down in order to remove a stacking conveyor. As no surge facility exists, between the crushing plant and the heap, crusher maintenance should be undertaken to coincide with the movements of the stacking equipment. Consideration of alternate stacking options (i.e. truck stacking) is recommended if this option is to be progressed.

The agglomerates are stacked in cells, each 60 m wide, 430 m long, and 6 m high, with a capacity of 248,400 tonnes (equivalent to 90 days production). The agglomerates are placed on the heap at the down slope end of the cell and the stacking progresses upslope on the heap, thus allowing irrigation of freshly stacked ore to commence.

Heap Leaching

An initial topographical assessment of the likely plant site reveals that flat open land is scarce. For this reason, the conceptual layout is based on a two level heap leach operation to minimise the real estate requirements.

The leach pad is constructed in stages to avoid excessive rainfall catchment and containment requirements in initial years and also to minimise initial capital expenditure. The stage 1 leach pad is constructed as part of the initial plant construction and is large enough to allow one year’s operation. To allow counter-current leaching and assist in metallurgical balancing, the individual cell has separate, metered solution application systems. Internal solution containment berms direct the leach solution from each cell to discrete off-flow solution collection systems. The collected leach solution from each cell will be directed to either the Pregnant Leach Solution (PLS) or Intermediate Leach Solution (ILS) return pipes, depending on solution tenor.

A two stage counter-current leach cycle will be utilised to reduce PLS flow rates and increase gold solution tenors reporting to clarifying filter. In this arrangement, Barren Leach Solution (BLS) is applied to the oldest, most extensively leached cells (nearly exhausted) to form a low grade ILS. The ILS is then fortified with cyanide and anti¬scalant and applied to fresher cells. The high grade leach solution from the cells under primary leach is directed to the PLS pond while the lower grade leach solution from the cells under secondary leach are recycled to the ILS pond.

Caustic and cyanide solutions will be metered to the BLS pond to maintain a pH of >9 and a free cyanide concentration of 300 - 500 mg/L. The BLS is pumped to the cells under secondary leach. If necessary raw water will be added to maintain the level in the BLS pond and to replenish for the water lost to evaporation and hold-up within the heaps.

Solution draining from the heaps under secondary leach is collected in the high density polyethylene (HDPE) lined ILS pond. The ILS is pumped to the cells under primary and secondary leach. Raw water make-up and caustic, cyanide and anti-scalant metering are identical to the BLS circuit.

Solution draining from the heaps under primary leach is collected in the HDPE lined PLS pond. The PLS is pumped to the clarifying filter feed tank.

Leach solution discharging from the collection system in each individual cell is directed to either the PLS or the ILS open return launder, depending on the solution tenor/leach cycle.

During periods of excessive rainfall, the off-flow solution from the heaps will exceed the capacity of the various ponds. The PLS pond and BLS ponds overflow to the ILS pond, which in turn overflows to the storm water pond. Solution will be reclaimed from the storm water pond as processing capacity becomes available in the period following the rainfall event.

Low annual rainfall, evaporation losses, and water retention in the heap all conspire to create an overall negative water balance. Supplementary water addition is, therefore, required.

Solution form the PLS pond is in batches transferred to a clarifying filter surge tank.

Barren solution from the precipitation filter is returned to the BLS pond.

Precipitation and Goldroom

Prior to the recovery of precious metals, the PL solution is fed through a clarifying filter to ensure that solid particles are removed. The clarified solution is directed to a surge tank.

The stored solution is pumped through a de-aeration vessel ahead of the mixing point where zinc dust, catalyst and precoat are introduced.

Zinc dust is metered into the gold and silver rich solution as it is pumped from the precipitate filter feed tank through a precipitation filter. Gold and silver precipitates are formed on contact with the zinc dust and collected in the precipitation filter. Lead nitrate is added during precipitation to act as a catalyst for the precipitation reaction, while diatomaceous earth is added to the filter feed stream to pre-coat the filter surface prior to each filtration cycle. During filtration, further diatomaceous earth is added to act as a body aid, which maintains filter cake porosity.

Filtrate solution, barren of gold and silver, discharges from the filter into the barren solution tank, where the filtrate is either used for mixing with zinc dust, lead nitrate and diatomaceous earth, or pumped back to the BLS pond.

Two recessed plate filter presses are used for the precipitation filtration duty with one filter operating, whilst the other filter is off line for harvesting and/or in preparation for the next batch. Each filter is sized such that 36 hours of production at design maximum grades can be accommodated.

Filter cake recovered from the precipitate filters is collected in trays. The trays are provided with castors to assist with their withdrawal from beneath the filters and with lifting

points for a fork-lift. The filter cake is dried in a drying/calcining oven before direct smelting with fluxes in a diesel-fired furnace. The smelted precious metals will be poured into a cascade of five 600 oz doré moulds. The drying oven and the furnace are located in the ground floor area of the secure gold room and are designed for a maximum of two smelts per day, seven days per week.

Fume extraction equipment will be provided to remove gases from the drying oven and barring furnace and exhaust to atmosphere.

Doré bars will be cleaned, weighed, stamped, sampled and then stored in a safe while awaiting dispatch.

The gold room has a concrete floor complete with sump trap which overflows to a sump. A sump pump transfers all wash down or spillage to the ILS pond. The heavy metals collected in the sump trap are returned to the drying oven.

Plant Control System

For a plant of this nature a basic hard wired interlock control system is adequate. The plant operation relies on manual stop start of conveyors and pumps. Hard wired interlocks are in place to protect equipment from overloading.

8.3.3 Operating Cost Estimate

An operating cost estimate has been prepared for the stand alone Calcatreu heap leach plant treating 750,000 t/y ROM ore producing 18.07 t/y of gold and silver in doré. The costs are expressed in USD and are current during 2Q06. The cost estimate is accurate to ±40%.

The operating cost estimate is based on the process design criteria and the information obtained from the following sources:

- Metallurgical performance assumption made by Ausenco.
- Data provided by AQI.
- Quotations from suppliers.
- Ausenco's in-house database.

A summary of the estimated operating costs for the heap leach option are presented in Table 8.5.

Table 8.5 Operating Cost Estimates

Cost Centre	USDM/y	USD/t
Electrical power	1.80	2.40
Labour - General and Administration	0.92	1.23
Labour - Process and Maintenance	1.02	1.36
Consumables	3.64	4.85
Maintenance	0.34	0.45
Sample assays	0.32	0.43
General and Administration	0.63	0.84
Total	***8.67***	***11.56***

Basis of Operating Cost Estimate

The process plant design criteria and a conceptual mass balance provide the basis for the operating cost estimate.

Compared with the gravity/flotation process option, the heap leach operation requires less personnel, vehicles, mobile equipment and accommodation.

The unit labour and diesel fuel costs for the heap leach option are identical to those used in the CIL process option, as supplied by AQI.

Unit rates for process consumables are estimated from vendor information, current projects and in-house data bases.

The cost of maintenance consumables is calculated as a percentage of the plant mechanical equipment, piping and electrical equipment cost. The factors are based on industry standard estimating experience with similar metallurgical operations.

A mechanical equipment list has been compiled from which an electrical load list, based on estimated annual operating hours and service factors, has been developed. This enabled the determination of the annual electrical energy consumption from which the power plant has been sized. An allowance for power demand has been included to cover the site lighting and infrastructure power requirements.

The unit cost of electrical power is US$0.13/kWh.

It is assumed that quicklime and cement are available in bulk quantities.

Qualifications

The operating cost estimate excludes the following costs:

- Production loss when the stacking conveyors are relocated.
- Mining and ore haulage.
- Marketing, transportation and sales of the gold/silver concentrate.
- Royalties.
- Levies and taxes.
- Bonds.
- Permits.
- Currency fluctuations.
- Insurances other than workers compensation.
- Support services of the head office.
- TSF maintenance.
- Community and heritage related issues.
- Environmental requirements.
- Working capital.
- Contingency allowance.

8.3.4 Capital Cost Estimate

Summary

The capital cost estimate for the heap leach estimate has been developed in the same manner as the gravity flotation option as described in Section 8.2.4. The capital costs estimate for the heap leach option is summarized in Table 8.3.2.

The heap leach option contains a deferred capital component for the expansion of the leach pad area and associated items. Details of the deferred capital cost also appear in Table 8.3.2.

Table 8.6 Breakdown of Capital Cost Estimates

Cost Centre	USDM
Direct Cost	
Heap Leach Plant	
Crushing	4.7
Agglomeration and Stacking	8.9
Heap Leach Pad and Ponds	8.3
Zinc Precipitation and Goldroom	3.0
Reagents	0.3
Utilities	3.9
On-Site Infrastructure	4.9
Off-Site Infrastructure	4.4
Mining	2.1
Miscellaneous	3.3
Sub-total Direct Cost	**43.8**
Indirect Cost	
Temporary Construction Facilities	1.9
EPCM	9.6
Start up and Commissioning	0.8
Sub-total Indirect Cost	**12.3**
Initial Project Costs	**56.1**
Deferred Capital	17
Total Project Cost	**73.1**

8.3.5 Recommendations

The following recommendations suggested here are applicable to this conceptual heap leach option only:

- A heap leach testwork programme should be developed to obtain engineering design parameters. For that reason it is recommended that the testwork programme covers the following aspects:
  - Testing of all major ore types.
  - Leach response variation versus ore depth and ore location in the deposit.
  - Leach response variation versus feed size distribution.
  - Permeability of the heap.
  - Reagent consumption rates.
  - Agglomeration tests to determine residence time, moisture content and cement addition rate.
  - Leach response versus temperature.

As the entire heap leach operation requires a surface area several orders larger in size than an equivalent CIL based process, an investigation should be conducted to determine a suitable site location.

CALCATREU GOLD PROJECT

Initial Feasibility Study
Volume 5: Financial Analysis

5 April 2007

Prepared by

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Executive Summary

The financial analysis shows that Aquiline Resources Incorporated's (AQI's) Calcatreu Project offers a modest level of profitability using the assumptions and findings of the IFS. The Project delivers free cashflow of USD16.26 M and an NPV of USD6.07 M with an IRR of 14%. All costs and revenues have been modelled and reported in $US unless otherwise stated.

The project as it stands does not have an attractive risk profile. When analysed for the impact of identifiable significant risks, it is apparent that there is only a 37% probability of the project delivering better than a break-even financial result (i.e. zero NPV) and a 24% probability that the project will deliver the IFS NPV or greater.

Analysis using recent 6 month average gold and silver prices produced free cashflow of USD73.54, NPV of USD54.68 with an IRR of 74%.

Risk mitigation strategies should be considered and incorporated into the project before a decision is made to proceed with development. In particular, the identification of additional resources for conversion into mining inventory, through discovery or upgrading the confidence classification of existing resources, provides an opportunity to significantly improve the risk profile. It is understood that Aquiline is pursuing this strategy.

Tables

Figures

Appendices

1 Project Scope

A financial model was prepared to evaluate the viability of AQI's Calcatreu Project assuming the operational and economic assumptions established during the Initial Feasibility Study (IFS). The financial model was used to identify the sensitivity of the project to key input variances and to define the project's risk profile.

2 Project Description

The Calcatreu Project is situated near the southern border of Rio Negro Province, Argentina. The deposit is classified as an epithermal gold and silver vein system which extends over more than 73,000 hectares. Currently 11 outcropping epithermal gold and silver bearing veins have been identified. The two most promising veins are the 49 and Nelson Veins which contain all of the currently drill defined resource ounces at Calcatreu. The other 9 veins remain partly tested and will soon become the focus of the AQI's aggressive project wide exploration program.

The project is planned to mine by open pit methods 3.50 Mt of mineralised material for 395 koz of recovered gold and 2,804 koz of recovered silver over a life of mine (LOM) of 4.67 years (56 months).

3 Base Case

The project financial model assumes the following Base Case parameters:

- mining inventory:
  - 3.50 Mt ore
  - 3.86 g/t Au head grade
  - 33.22 g/t Ag head grade.
- discount rate 8%
- operational costs:
  - mining $41.85 M or $11.95/t ore
  - processing $68.43 M or $19.55/t ore.
- process recovery:
  - gold 90%
  - silver 74%.
- gold price $500/oz
- silver price $8/oz
- royalties 5.5% on NSR
- discount rate 8%
- total capital expenditure $79.15 M.

No provision has been made for exploration expenditure attributable to the operation.

No provision has been made for environmental bonds, rehabilitation costs or salvage revenues.

No provision has been made for project financing arrangements or sales arrangements other than spot price contracts.

Depreciation of capital expenditure has been provided for on a reducing balance over mine life basis.

No provision has been made for inflation of costs with time or taxation of profits.

The Base Case financial model results are summarised in Table 3.1 and are shown in greater detail in Appendix A.

Table 3.1 Base Case financial modelling results

Item	Unit	Value
Net cashflow	$ M	16.26
NPV @ 8% discount	$ M	6.07
IRR	%	14
Payback period	Months	40

The Base Case represents the financial outcome considering the cost, revenue and financial factors derived or assumed by the IFS for the project.

4 Sensitivity and Risk Analyses

Due to the inherent uncertainty in estimating some of the key factors which influence the viability of mining projects of this type, a sensitivity and risk analysis has been undertaken to determine the robustness of the viability of the project and the risk of the project becoming non-viable in the event of foreseeable and reasonably assumed variability in the project assumptions.

The sensitivity analysis identifies the impact on project value for simple variances in the assumed values for the key project factors. An indication of the variance required for each variable to produce a break-even result can be derived.

The risk analysis uses Monte Carlo Simulation to assess the probability of the project achieving a certain outcome, such as a positive NPV, after assigning range limits to key variables. A statistical analysis of the range of likely NPV results provides an insight into the robustness of the viability of the project.

The range of possible variability for each key factor has been estimated on the basis of knowledge and experience with similar projects, consideration of physical attributes of the project and consideration of historical and consensus forecasts for external financial factors. Each factor offers complementary, though not necessarily balanced, risk of downside or upside impacts.

4.1 Sensitivity Analysis

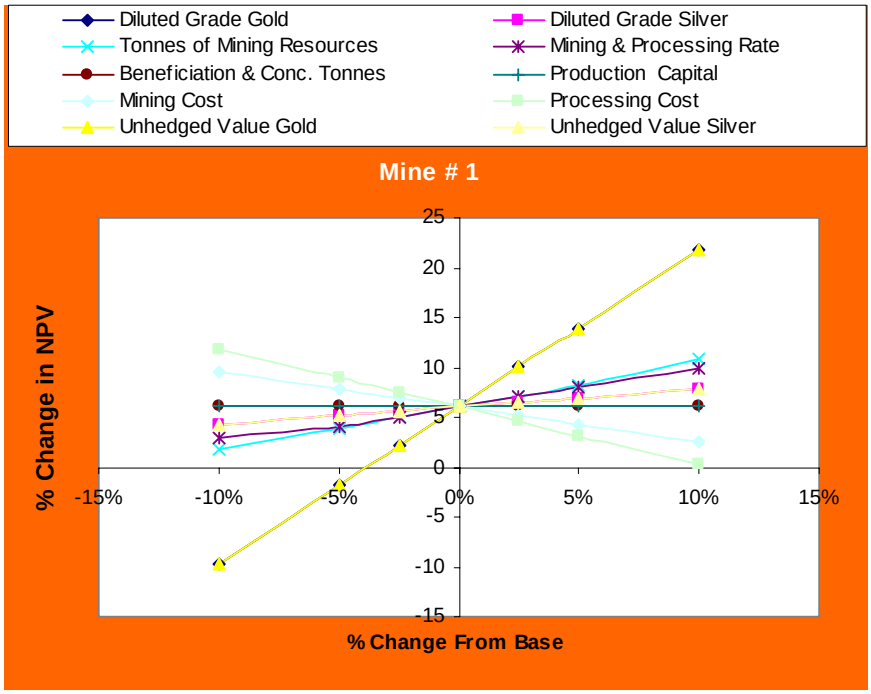
The sensitivity of the project NPV has been established for increments of +/- 2.5%, +/- 5% and +/- 10% of key project variables. The results are shown as a spider diagram in Figure 4.1. Figure 4.1 shows that the project NPV is very sensitive to the key revenue based variables of gold value (price) and gold grade. The project is similarly sensitive to processing recovery of gold though any upside potential is limited by physical constraints to increasing recovery. The project becomes NPV break-even following a 4% negative movement in value of any of these variables. A 4% positive movement sees a 100% increase in project NPV to $12.2 M.

The project is sensitive to variation in the mining inventory. A 5% change in mining inventory sees a 35% or $2.1 M change in NPV. A 12% increase in mining inventory will see a 100% increase in project NPV.

The project is also sensitive to variations in the pre-production capital requirement. An 8% change in pre-production capital will see the project value reduced to break-even, or alternatively, doubled.

An additional sensitivity scenario was run using 6 month (to 31 December 2006) average gold and silver prices of $624 /oz and $12.38 /oz respectively. The projected results of free cashflow of $73.54 M, NPV of $54.68 M and IRR of 74% further demonstrate the sensitivity of the project to commodity prices.

Figure 4.1 Sensitivity results



4.2 Risk Analysis

The key risks associated with the project have been identified and an estimate made of the range of potential variability to consider for each variable.

4.2.1 Resource and reserve risk factors

Narrow veins, generally less than 3 m wide, are complex geological phenomena, which commonly display unpredictable geometry and grade distribution. Variations in structural continuity, dip, strike, width, mineralogy and specific gravity are common. Veins may be composite, with ore-grade mineralisation restricted to specific structural domains. Branches, intersections and braided zones are common features. Potentially high-grade zones are often localized within ore shoots, which are surrounded by barren or low-grade regions. Although the overall deposit grade can be predicted with some confidence, these features make accurate local grade prediction difficult.

The Calcatreu deposit benefits from a low value of the geostatistical parameter “relative nugget ratio”, which distinguishes it from many other narrow vein deposits where this ratio is large and reliable grade estimation is consequently difficult. The gold mineralisation is fine grained and varies in size between 0.2 µm and 6 µm, and this contributes to confidence in the grade estimation.

The mineral resource reported for Calcatreu (Micon International Limited: A Preliminary Assessment and Economic Evaluation for the Calcatreu Gold-Silver Project, October 2004) has been classified as Indicated in accordance with the CIM Standards on Mineral

Resources and Reserves (2000), with no part classified as Measured. This means that the project estimate of resource tonnes and grade is at a medium level of confidence by virtue of the relationship between the sampling density and the geostatistical characteristics of the mineralisation. Accordingly, any assessment of sensitivity or risk should consider commensurate potential variances in the resource.

Narrow vein mining operations can also be subject to high levels of unplanned dilution depending upon geological variability and mining engineering issues, such as blasting impacts and the selectivity of the mining method and equipment. Unplanned high levels of dilution can have negative consequences for the cash flow and NPV of a project.

The IFS assumes average mining dilution from adjacent material of 5% volume to high grade mineralisation and 8% volume to low grade mineralisation. Mining losses of 5% of low grade mineralisation to waste are provided for.

For the risk analysis, a range of variability from the Base Case of - 5% to + 3% in the gold grade has been considered, biased towards the lower side of the grade distribution, which is close to the standard error as calculated for Vein 49. For silver grade a - 5% to + 5% variability has been considered. The mining inventory has a variability of - 4% to + 5% considered. These variabilities reflect the uncertainty in the resource estimate and in the unplanned dilution included in the mining inventory estimate.

4.2.2 Market/price risk factors

The annual revenue generated by the project as reported by the Base Case financial model comprises:

- 85% to 89% revenue from Au
- 11% to 15% revenue from Ag.

The major proportion of the revenue stream is derived from gold sales and so the project value is leveraged more to variability in the gold market price than that of silver. Historically, the market price for gold has fluctuated widely and has been affected by numerous factors, over which the producer has no control, including:

- the demand for gold for industrial uses and jewellery
- actual or expected purchases and sales of gold bullion holdings by central banks or other large gold bullion holders or dealers
- speculative trading activities in gold
- the overall level of forward sales by other gold producers
- the overall level and cost of production by other gold producers
- international or regional political and economic events or trends
- the strength of the U.S. dollar (the currency in which gold prices generally are quoted) and of other currencies
- financial market expectations regarding the rate of inflation
- interest rates.

Changes in the market price for gold will affect the cash flows and profit generated by Calcatreu. A representation of the recent gold price in USD/oz, from 1973 to 2006, is shown in Figure 4.2.

Figure 4.2 Gold price 1973 – 2006 (Source: www.thebulliondesk)

In addition, the current demand for, and supply of gold affects the price of gold, but not necessarily in the same manner as current demand and supply affect the prices of other commodities. Central banks, financial institutions and individuals historically have held large amounts of gold as a store of value and, historically, production in any given year has constituted only a small portion of the total potential supply of gold. Since the potential global supply of gold is largely unrelated to mine production in any given year, normal variations in current production will not necessarily have a significant effect on the supply of gold or the gold price. Historically, gold has tended to retain its value in relative terms against basic goods in times of inflation and monetary crisis.

In 2004, 15 European central banks entered into a gold sales agreement pursuant to which they restrict their annual sales of gold to specified limits. This agreement will be reviewed in 2009. The content and effect of any new agreement on the market in terms of gold sales and prices is unclear but presents a risk which will crystallise during the current scheduled life of the Calcatreu Project.

For the risk analysis, a range of variability in the gold price from the Base Case of - 15% to + 15% has been considered. Due to its small contribution to project value, no variability in the silver price has been provided for.

4.2.3 Production schedule

The analysis indicates that the project is extremely sensitive to variation in gold grade and recovery. As most of the mineralised resource is currently classified as Indicated, the ability to accurately estimate the local distribution of grade in the ore body, and hence short term variability of feed grade to the process plant, is limited. To this extent, the project carries some risk of operating below cash break-even for periods during its life.

An in-depth analysis of this risk would be undertaken during the preparation of a project definitive feasibility study to identify the magnitude of this risk and establish appropriate strategies for its management.

No provision has been made for modelling the risk of variability in the short term mined grades as the resource estimate has not been prepared to the level of confidence required to enable meaningful interpretation of risk.

Recovery of metal is subject to some uncertainty until the process plant has been commissioned and is operating at a steady state performance level. To reflect this uncertainty, variability in recovery from the Base Case has been modelled with a range of - 2% to + 1.5% for gold and - 5% to + 3% for silver. The upside potential for recovery is limited by physical constraints within the selected process path.

4.2.4 Operational cost

The estimated operating costs for the project comprise:

- mining cost $11.95/t ore
- processing cost (includes Administration) $19.55/t ore.

The results of the sensitivity analysis shown in Figure 4.1 indicate that the NPV is relatively insensitive to changes in mining and processing costs. The degree of detail used to estimate these costs in the IFS is such that a relatively low variability can be expected.

For the risk analysis, a range of variability from the Base Case of - 3% to + 5% in the operating costs is considered appropriate.

4.2.5 Capital cost

The total capital cost including costs for the mining equipment, treatment plant and infrastructure is estimated to be $79.15 M. The major part of the capital expenditure ($78.87 M) is scheduled as a pre-production cost. There is a small provision of $0.28 M for road construction during Year 3 of the operation. The largest component of the capital expenditure (84%) is due to processing plant, administration and infrastructure costs.

Recent experience suggests that project capital costs can be volatile and subject to significant short term movement. Capital costs are impacted by global supply and demand forces which have produced significant over-run outcomes in recent times. For the risk analysis, it is assumed that the capital expenditure could vary between -5% and +25% of the expected $79.15 M.

4.2.6 Variable ranges

The Monte Carlo Simulation risk model has been established using the variable ranges shown in Table 4.1 and by running the model on a multivariate basis. These variable ranges are considered to represent a reasonable expectation of the range of variability the project may encounter during operations.

The range of values used for the variables have been modelled as a uniform distribution with a selection bias of 70% (approximately 1 standard deviation) towards the mean of the range for each variable. In this manner, the impact of data outliers is moderated. Ten thousand iterations were run to ensure an adequate distribution of results was obtained to enable meaningful interpretation.

Table 4.1 Variable ranges for risk modelling

Variable	Base Case		Variable ranges	
	Qty	Units	+	-
Diluted Grade				
Gold	3.86	g/t	3.0%	5.0%
Silver	33.22	g/t	5.0%	5.0%
Mining inventory	3.50	Mt	5.0%	4.0%
Recovery				
Gold	90	%	1.5%	2.0%
Silver	74	%	3.0%	5.0%
Metal Price				
Gold	500	$ /oz	15%	15%
Silver	8	$ /oz	0%	0%
Capital				
Pre-Production	78.87	$ M	25%	5%
Production	0.28	$ M	0%	0%
Operating Costs				
Mining	11.95	$ /t	5%	3%
Processing	19.55	$ /t	5%	3%

Further Monte Carlo analyses were undertaken to determine the sensitivity of the simulated NPV outcome to the most influential factors identified by the sensitivity analysis, i.e. the gold price, grade, recovery, mining inventory and capital cost. This analysis was undertaken by changing the range of variability of each of these key variables in isolation to determine the effect on the NPV distribution.

4.2.7 Results

The multi-variate Monte Carlo analysis indicates that, based on the selected variable ranges, the project has a mean (expected) NPV of -$4.59 M, that is, a 50% probability of achieving an NPV of -$4.59 M or lower, and a 50% probability of achieving an NPV of -$4.59 M or higher. The project has a 37% probability of achieving a positive NPV, and a 24% probability of achieving or exceeding the Base Case NPV of $6.07 M. There is a 95% probability of the project delivering an NPV between -$36.51 M and $27.32 M.

The probability distribution of NPV outcomes produced by the Monte Carlo analysis is shown in Figure 4.3.

Figure 4.3 NPV simulation distribution

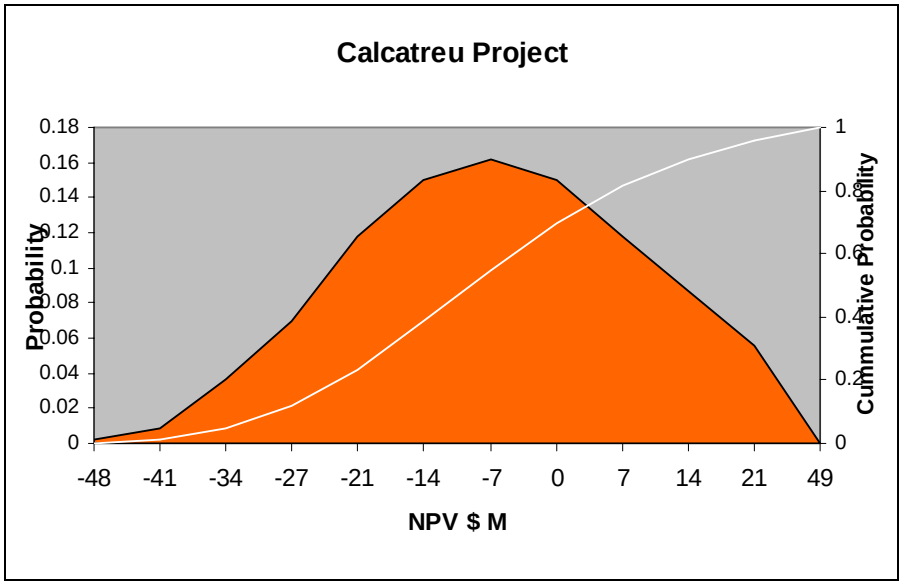


Table 4.2 to Table 4.6 show the results of the additional key variable simulations undertaken to further investigate the sensitivity of the project to the key parameters of:

- gold price
- grade
- recovery
- mining inventory
- capital cost.

The key parameter variance ranges were adjusted to those shown in Table 4.2 to Table 4.6 while the remaining variable ranges were maintained at those shown in Table 4.1

Table 4.2, Table 4.3, and Table 4.4 show little difference in NPV distribution for the key revenue factors (gold price, grade, recovery) apart from a widening of the range of NPVs reported as the range of variability of each factor widens. The mean NPV stays within a narrow range close to the base simulation mean NPV of -$4.59 M.

Table 4.5 shows the effect on NPV distribution from varying the mining inventory, with the upside variance representing the results of potential exploration success. The mean NPV increases by about $3 M for each 5% increase in mining inventory.

Table 4.6 shows that a significant increase in mean NPV may be achieved by limiting the capital expenditure variability over-spend, with a positive mean NPV of $2.69 M reported for a variability range of +/- 5%.

Table 4.2 NPV simulation results with varying gold price range

Gold price range	Mean NPV	95% probability range of NPV
± 5% from base	-$4.33 M	-$22.2 M to $13.5 M
± 10 % from base	-$5.19 M	-$30.0 M to $19.6 M
± 15% from base	-$4.59 M	-$36.5 M to $27.3 M
± 20% from base	-$5.89 M	-$44.3 M to $32.5 M

Table 4.3 NPV simulation results with varying grade range

Grade range	Mean NPV	95% probability range of NPV
-3% to + 1% from base	-$4.52 M	-$35.9 M to $26.9 M
-4% to + 2% from base	-$5.35 M	-$37.0 M to $26.3 M
-5% to + 3% from base	-$4.59 M	-$36.5 M to $27.3 M
-6% to + 3% from base	-$5.88 M	-$37.9 to +$26.1 M

Table 4.4 NPV simulation results with varying recovery range

Varying recovery	Mean NPV	95% Probability Range of NPV
-2% to + 1.5% from base	-$4.59 M	-$36.5 M to $27.3 M
-4% to + 1.5% from base	-$7.20 M	-$38.2 M to $23.7 M
-6% to + 1.5% from base	-$8.35 M	-$40.6 M to $23.9 M

Table 4.5 NPV simulation results with varying mining inventory

Varying mining inventory	Mean NPV	95% Probability Range of NPV
-4% to + 5% from base	-$4.59 M	-$36.5 M to $27.3 M
-5% to + 15% from base	-$1.14 M	-$35.0 M to $32.8 M
-5% to + 25% from base	$1.84 M	-$33.6 M to $37.3 M

Table 4.6 NPV simulation results with varying capital expenditure

Varying capital expenditure	Mean NPV	95% Probability Range of NPV
-5% to + 5% from base	$2.69 M	-$26.8 M to $32.1 M
-5% to + 15% from base	-$0.76 M	-$31.4 M to $29.9 M
-5% to + 25% from base	-$4.59 M	-$36.5 M to $27.3 M

5 Risk Reduction

A number of strategies can be employed to minimise the exposure of the project to the significant risks and uncertainties identified in this financial analysis. These strategies include:

- increasing the mining inventory tonnage and grade. The current mining inventory is operating cash positive though not of a magnitude to produce a very attractive project outcome. Any increase in the mining inventory, by defining additional economic resources to contribute to the mining inventory, will add to the project's viability and reduce the sensitivity of the project NPV to the capital requirements for development.
- reducing the capital cost. This can be achieved by conducting value engineering reviews to identify opportunities to reduce capital cost requirements.
- reducing the project exposure to gold price volatility. This can be achieved by entering into arrangements which secure the price to be received for future mine production. There are a number of methodologies available to achieve this outcome.
- increasing the confidence of the resource estimate. Further resource development work should aim to increase the resource classification from Indicated to Measured. This can be achieved with further drilling and resource estimation and can be aided by the application of conditional simulation techniques to quantify the uncertainty in the resource estimate.
- reducing the risk of grade dilution through mining. This can be achieved by conducting geotechnical investigations which quantify the risk of dilution and provide a basis for establishing the means by which the dilution can be controlled.
- increasing the certainty and currency of capital and operating cost estimates. Cost estimates need to be updated progressively as decision points for project development are approached and firm cost commitments secured to reduce uncertainty. In particular, capital costs are impacted by global supply and demand forces which have produced significantly negative outcomes in recent times.

6 Conclusions

- The financial analysis shows that the Calcatreu Project Base Case delivers $16.26 M in free cashflow with an NPV of $6.07 M and an IRR of 14%. Payback is achieved in 40 months.
- The sensitivity analysis shows the financial outcome is highly sensitivity to the key revenue factors of grade, gold price and recovery. A 4% favourable movement in any of these key factors produces a 100% improvement to the project NPV, to $12.2 M. A 4% negative movement in any of these factors produces a below break-even outcome.
- The analysis shows the project viability to be sensitive to the mining inventory. A 12% increase in mining inventory produces a 100% increase in NPV.
- The analysis shows the project to be quite sensitive to capital costs, with an 8% increase in capital costs resulting in a below-break even outcome.
- Analysis using recent 6 month average gold and silver prices produced free cashflow of $73.54 andNPV of $54.68 with an IRR of 74%.
- The risk analysis shows the profitability of the project is vulnerable within the range of variable movements modelled. The simulation modelling shows the project has only a 37% probability of delivering a break-even result and a 24% probability of achieving the Base Case NPV. The modelled mean project NPV is -$4.59 M. There is a 95% probability of the project delivering an NPV between -$36.51 M and $27.32 M.
- Achieving a reduction in the risk of capital over-expenditure (increasing the certainty of capital cost estimates) will have a significant impact on reducing the risk of the project delivering a negative NPV result.
- Risk reduction strategies should be considered and incorporated into the project before a decision is made to proceed with development.

A Financial model detail

Multi Mine - Advanced Financial Analysis			**21-Feb-07**									
Calcatreu												
Rio Negro Prov. Argentina		**Calcatreu**					**YEARLY DATA**					
Initial Feasibility Study S		**IRR**	**14%**	**Year Ore**				**1.00**	**2.00**	**3.00**	**4.00**	**4.67**
		NPV (Ex Finance)	**$ 6.070**	**L/G Years**				**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
		NPV (All Cashflows)	**$ 6.070**									
		Cash	**$ 16.263**									
Project Type			**0**									
Currency			**USD**									
Pre-Prodn Period		years	0.00				0.00	0.00	0.00	0.00	0.00	0.00
MINEABLE RESOURCES/RESERVES							**TOTALS**					
Grade of Metal	Gold	g/t	3.86									
Grade of Metal	Silver	g/t	33.22									
Tonnes of Mining Resources		million t	3.500									
Strip Ratio		?:1	7.26	7.26			7.26	7.85	9.54	6.86	7.62	2.98
Recovery & Reserves of Ore		%	100.0%	3.500			3.500	0.750	0.750	0.750	0.750	0.500
Grade of Mineralised Waste	Gold	g/t	0.00									
Grade of Mineralised Waste	Silver	g/t	0.00				Remaining Tonnes	3.500	2.750	1.999	1.247	0.494
MINING												
% Production from this Mine				100%			3.500	0.750	0.752	0.751	0.753	0.494
Mining Rate, Remaining Ore & Min Waste		million t/year		6.197			6.197	2.750	1.999	1.247	0.494	-
Tonnes Waste		million t		25.424			25.424	5.889	7.173	5.152	5.738	1.472
Total Tonnes Mined		million t		28.925			28.925	6.640	7.925	5.904	6.491	1.966
Bulk Density Ore		t/m3	2.48					2.48	2.48	2.48	2.48	2.48
BCM Ore		BCM		1.414			1.414	0.303	0.304	0.303	0.304	0.200
Bulk Density Waste		t/m3	2.49					2.49	2.49	2.49	2.49	2.49
BCM Waste - Pre- Strip		BCM		10.193			10.193	2.361	2.876	2.066	2.300	0.590
PROCESSING												
Process Plant Availabilty		%	91%									
Thruput, Mine & Project Life		tpa & yrs	0.750	4.67	4.67	4.67	3.500	0.750	0.752	0.751	0.753	0.494
Wt % Concentrate & Ore Retained	t	%	100.0%		Min. Waste		3.500	0.750	0.752	0.751	0.753	0.494
% Distribution to Conc. & Grade	Gold	%	100.0%	3.86	100.0%	100.0%		100.0%	100.0%	100.0%	100.0%	100.0%
% Distribution to Conc. & Grade	Silver	%	100.0%	33.22	100.0%	100.0%		100.0%	100.0%	100.0%	100.0%	100.0%
% Recovery	Gold						M/W Recov	90.0%	90.0%	90.0%	90.0%	90.0%
Recovered Grade & Commissioning	Gold	%	90.0%	3.47	100.0%	0	100.0%	3.90	3.99	3.81	2.68	2.77
						90.0%		2.922	2.995	2.859	2.018	1.370
% Recovery	Silver							74.0%	74.0%	74.0%	74.0%	74.0%
Recovered Grade & Commissioning	Silver	%	74.0%	24.58	100.0%	0	100.0%	19.18	25.92	33.09	19.48	25.59
						74.0%		14.388	19.479	24.860	14.666	12.654
Recovered	Silver	ozs	2.766					0.463	0.626	0.799	0.472	0.407
SPOT MARKET SALES												
Price & Rate of Change/Year	Gold	USD	$ 500.00	0.00%				$ 500.00	$ 500.00	$ 500.00	$ 500.00	$ 500.00
Payable Concentrate Grade	Gold	g/t						-	-	-	-	-
Value of Recovered Metal / Tonne Conc	Gold	USD/t						$ 62.62	$ 64.06	$ 61.17	$ 43.08	$ 44.53
Value	Gold	million $					$ 195.533	$ 46.979	$ 48.146	$ 45.956	$ 32.437	$ 22.015
Price & Rate of Change/Year	Silver	USD	$ 8.00	0.00%				$ 8.00	$ 8.00	$ 8.00	$ 8.00	$ 8.00
Payable Concentrate Grade	Silver	g/t						-	-	-	-	-
Value of Recovered Metal / Tonne Conc	Silver	USD/t						$ 4.93	$ 6.67	$ 8.51	$ 5.01	$ 6.58
Value	Silver	million $					$ 22.131	$ 3.701	$ 5.010	$ 6.394	$ 3.772	$ 3.255
Total Product Value					**$ 62.18**		**$ 217.665**	**$ 50.680**	**$ 53.156**	**$ 52.350**	**$ 36.210**	**$ 25.270**
Currency/USD			**$ 1.00**					**$ 1.00**	**$ 1.00**	**$ 1.00**	**$ 1.00**	**$ 1.00**
USD Value							**$ 217.665**	**$ 50.680**	**$ 53.156**	**$ 52.350**	**$ 36.210**	**$ 25.270**
PRE-PRODUCTION CAPITAL												
Year	-3		$ -				$ -	$ -	$ -	$ -	$ -	$ -
Year	-2		$ -				$ -	$ -	$ -	$ -	$ -	$ -
Year	-1		$ 78.872				$ 78.872	$ 78.872	$ -	$ -	$ -	$ -
Total Pre-Production Capital			**$ 78.872**		**$ 22.53**		**$ 78.872**	$ 78.872	$ -	$ -	$ -	$ -
PRODUCTION CAPITAL												
Inflation Rate (On Capital Costs)			0.00%					1.0000	1.0000	1.0000	1.0000	1.0000
Year	1		$ -				$ -	$ -	$ -	$ -	$ -	$ -
Year	2		$ 0.279				$ 0.279		$ 0.279	$ -	$ -	$ -
Year	3		$ -				$ -			$ -	$ -	$ -
Year	4		$ -				$ -				$ -	$ -
Year	5		$ -				$ -					$ -
Year	6		$ -	FV	$ 0.279		$ -					
Total Production Capital			**$ 0.279**		**$ 0.08**		**$ 0.279**	$ -	$ 0.279	$ -	$ -	$ -
Capital for Depeciation			$ 79.151		$ 22.61		$ 79.151	$ 78.872	$ 0.279	$ -	$ -	$ -
OPERATING COSTS					Cost / tonne of ore							
Mining Waste		$/t	$ 1.45	33.4%	$ 10.51	$ 10.39		$ 1.32	$ 1.21	$ 1.49	$ 1.40	$ 2.85
							$ 36.367	$ 7.756	$ 8.701	$ 7.693	$ 8.028	$ 4.189
Mining Ore		$/t	$ 1.45	4.6%	$ 1.45	$ 1.57		$ 1.32	$ 1.21	$ 1.49	$ 1.40	$ 2.85
							$ 5.480	$ 0.987	$ 0.911	$ 1.121	$ 1.053	$ 1.408
Total Mining Cost							**$ 41.847**	**$ 8.744**	**$ 9.612**	**$ 8.813**	**$ 9.081**	**$ 5.597**
Mining Cost / tonne							**$ 11.95**	**$ 11.65**	**$ 12.79**	**$ 11.73**	**$ 12.06**	**$ 11.32**
Processing		$/t ore	$ 14.89	47.3%	$ 14.89	$ 14.89	$ 52.121	$ 14.89	$ 14.89	$ 14.89	$ 14.89	$ 14.89
								$ 11.172	$ 11.190	$ 11.187	$ 11.210	$ 7.362
Maintenance		$/t	$ 1.74	5.5%	$ 1.74	$ 1.74	$ 6.091	$ 1.74	$ 1.74	$ 1.74	$ 1.74	$ 1.74
								$ 1.305	$ 1.308	$ 1.307	$ 1.310	$ 0.860
Administration		$/t ore	$ 2.92	9.3%	$ 2.92	$ 2.92	$ 10.221	$ 2.191	$ 2.194	$ 2.194	$ 2.198	$ 1.444
Total Processing Cost							**$ 68.433**	**$ 14.668**	**$ 14.693**	**$ 14.688**	**$ 14.719**	**$ 9.666**
Processing Cost / tonne							**$ 19.55**	**$ 19.55**	**$ 19.55**	**$ 19.55**	**$ 19.55**	**$ 19.55**
Total Operating Cost							**$ 110.280**	**$ 23.411**	**$ 24.304**	**$ 23.501**	**$ 23.800**	**$ 15.263**
Creditors Paid Operating Cost				0.00			$ 110.280	$ 23.411	$ 24.304	$ 23.501	$ 23.800	$ 15.263
Operating Cost / tonne					**$ 31.50**	**$ 31.50**		**$ 31.20**	**$ 32.34**	**$ 31.28**	**$ 31.61**	**$ 30.87**
ROYALTIES												
% Metal Value After NSR			5.50%		$ 3.42		$ 11.972	$ 2.787	$ 2.924	$ 2.879	$ 1.992	$ 1.390
Total Royalties					**$ 3.42**		**$ 11.972**	**$ 2.787**	**$ 2.924**	**$ 2.879**	**$ 1.992**	**$ 1.390**
GOVERNMENT TAXES												
Tax Deductions							$ 122.252	$ 26.199	$ 27.228	$ 26.381	$ 25.791	$ 16.653
Cummulative Tax Deductions				$ -				$ 26.199	$ 27.228	$ 26.381	$ 25.791	$ 16.653
Taxable Gross Income							$ 217.665	$ 50.680	$ 53.156	$ 52.350	$ 36.210	$ 25.270
Taxable Income							$ 95.413	$ 24.481	$ 25.927	$ 25.970	$ 10.419	$ 8.617
Tax Payable in Current Year & Rate		75%	0.0%		$ -		$ -	$ -	$ -	$ -	$ -	$ -
Total Taxes					**$ -**		**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**
FINANCIAL ANALYSIS												
Cash Inflow					$ 62.18		$ 217.665	$ 50.680	$ 53.156	$ 52.350	$ 36.210	$ 25.270
Cash Outflow					$ 57.54		$ 201.402	$ 105.071	$ 27.507	$ 26.381	$ 25.791	$ 16.653
Cash Cost / Unit of Metal	Gold	ozs					$ 280.83	$ -	$ -	$ -	$ -	$ -
Cash Cost / Unit of Metal	Silver	ozs					$ 4.49	$ -	$ -	$ -	$ -	$ -
Total Cost / Unit of Metal	Gold	ozs					$ 462.64	$ -	$ -	$ -	$ -	$ -
Total Cost / Unit of Metal	Silver	ozs					$ 7.40	$ -	$ -	$ -	$ -	$ -
Total Cost/Tonne Mined								$ 140.04	$ 36.60	$ 35.11	$ 34.26	$ 33.68
Net Cashflow					**$ 4.65**		**$ 16.263**	**-$ 54.391**	**$ 25.649**	**$ 25.970**	**$ 10.419**	**$ 8.617**
Cumulative Cashflow								-$ 54.391	-$ 28.742	-$ 2.773	$ 7.646	$ 16.263
Maximum Working Capital							**-$ 54.391**	-$ 54.391	-$ 28.742	-$ 2.773	$ -	$ -
Operating Cashflow					**$ 27.26**		**$ 95.413**	**$ 24.481**	**$ 25.927**	**$ 25.970**	**$ 10.419**	**$ 8.617**
NPV (Excluding Finance)			8.00%				**$ 6.070**	-$ 52.338	$ 22.852	$ 21.424	$ 7.958	$ 6.173
NPV (Including Finance)			**8.00%**				**$ 6.070**	-$ 52.338	$ 22.852	$ 21.424	$ 7.958	$ 6.173
IRR							**14%**				8%	14%