ASX ANNOUNCEMENT

Koka Gold Project – Feasibility Study Full Summary Report

Chalice Gold Mines Limited ABN 47 116 648 956

13 August 2010

Further to Chalice's ASX Release of 13 July 2010 of the results of the Feasibility Study for its Koka Gold Project in Eritrea, East Africa, the Company is pleased to provide the full Project Summary from the Feasibility Study for reference by investors and shareholders.

The full Koka Gold Project Feasibility Study Summary is appended to this release and is also available on the Company's website.

The key financial outcomes of the Feasibility Study, which was undertaken by Lycopodium Minerals Limited ("Lycopodium") with inputs from prominent industry consultants AMC Consultants Pty Ltd ("AMC") and Knight Piésold Pty Ltd ("KP"), are set out in the announcement of 13 July 2010.

The highlights of that announcement were:

- Average life-of-mine total cash operating costs of US$338 per oz of gold
- After-tax $NPV_{5\%}$ of US$196 million, life-of-mine EBITDA of US$589 million and an after-tax IRR of 35% at current gold price of ~US$1,200/oz
- After-tax $NPV_{5\%}$ of US$99 million, life-of-mine EBITDA of US$381 million and an after-tax IRR of 22% at base case gold price of US$900/oz
- Average annual gold production of approximately 104,000oz per year with gold production totalling 731,000oz
- Forecast mine life of seven years at a mill throughput of 600,000 tonnes per annum, rising to 700,000 tonnes per annum from year 5
- Open pit Ore Reserves of 4.63Mt grading 5.1g/t for 760,000oz contained gold with a waste to ore ratio of ~10:1
- Estimated start-up capital cost of US$122M

As advised in the previous announcement, the Company will now proceed to apply to the Eritrean Government for a Mining Licence in respect of the Koka Deposit. In parallel with this application, the Company will assess its various funding options for development of the Koka Project.

The full Feasibility Study was recently presented to the Eritrean Ministry of Energy & Mines as the first step in the Company's application for a Mining Licence for the Koka Gold Deposit.

Lycopodium Minerals Limited, AMC Consultants Pty Ltd and Knight Piésold Pty Ltd have consented to the release of the Summary of the Feasibility Study.

INVESTMENT HIGHLIGHTS

High grade Indicated gold Resource (840,000 oz @ 5.3 g/t)

Feasibility Study completed:

- **Low cash costs of $338/oz**
- **7 year mine life at >100,000 oz average production per year**

Drilling at near mine Konate Prospect in progress

Large unexplored ground position in the Arabian Nubian Shield

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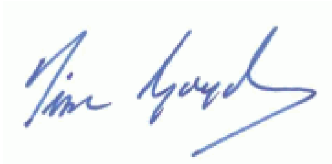

TIM GOYDER
Executive Chairman

Attachment: Koka Gold Project Feasibility Study Summary

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Competent Persons' Statement

The information in this report that relates to Exploration Results is based on information compiled by Dr Doug Jones, a full-time employee and Director of Chalice Gold Mines Limited, who is a Member of the Australasian Institute of Mining and Metallurgy and is a Chartered Professional Geologist. Dr Jones has sufficient experience in the field of activity being reported to qualify as a Competent Person as defined in the 2004 edition of the Australasian Code for Reporting of Exploration Results, Minerals Resources and Ore Reserves, and consents to the release of information in the form and context in which it appears here.

The Mineral Resource estimate was prepared by Mr. John Tyrrell who is a Member of the Australasian Institute of Mining and Metallurgy. Mr. Tyrrell is a full time employee of AMC and has sufficient experience in gold resource estimation to act as Competent Person as defined in the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code)'. Mr. Tyrrell consents to the inclusion of this information in the form and context in which it appears.

The information in this statement of Ore Reserves is based on information compiled by Mr David Lee who is a Member of the Australasian Institute of Mining and Metallurgy and a full time employee of AMC. Mr Lee has sufficient relevant experience to be a Competent Person as defined in the JORC Code. Mr Lee consents to the inclusion of this information in the form and context in which it appears.

ZARA PROJECT
Koka Gold Deposit

DEFINITIVE FEASIBILITY STUDY

PROJECT SUMMARY

July 2010
Lycopodium Minerals Pty Ltd

ZARA PROJECT
Koka Gold Deposit

DEFINITIVE FEASIBILITY STUDY

1609-STY-003

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2.0 PROJECT SUMMARY

2.1 Introduction

The Koka Gold Deposit, part of the Zara Project, is situated in the State of Eritrea. The project is wholly owned by Chalice Gold Mines Limited (Chalice) and its Australian incorporated subsidiaries Sub-Sahara Resources (Eritrea) Pty Ltd and Chalice Gold Mines (Eritrea) Ltd (formerly Dragon Mining (Eritrea) Ltd). Chalice is listed on the Australian Securities Exchange.

Chalice engaged Lycopodium Minerals Pty Ltd (Lycopodium) to coordinate the production of a Feasibility Study to assess the viability of the project.

Contributors to the Study have been AMC Consultants (AMC), Global Resources Development and Management Consultants (GREDMCO) of Asmara, Knight Piésold, and Orway Mineral Consultants (OMC). The contributions are as summarised in Table 2.1.1 as follows:

Table 2.1.1 Study Contributions

Ownership and Leases	Chalice
Geology	Chalice / AMC
Resource model	AMC
Mine design, scheduling and mining cost estimates	AMC
Preliminary metallurgical testwork	Chalice
Metallurgical interpretation	Lycopodium
Mining geotechnical assessment	AMC
Tails dam design	Knight Piésold
Hydrological assessment	Knight Piésold
Environmental assessment	Global Resources Development and Management Consultants (GREDMCO) / Knight Piésold
Process plant design	Lycopodium / Orway
Infrastructure design	Lycopodium / Knight Piésold
Access road	Knight Piésold
Project implementation	Lycopodium
Capital cost estimation	Lycopodium
Operating cost estimation	Lycopodium
Risk assessment	Lycopodium / Chalice
Financial analysis	Lycopodium / Chalice

2.2 Project Background

2.2.1 Geography and Climate

The project is located in the northern region of Eritrea, in north east Africa, as shown in Figure 2.2.1.

Figure 2.2.1 Project Location

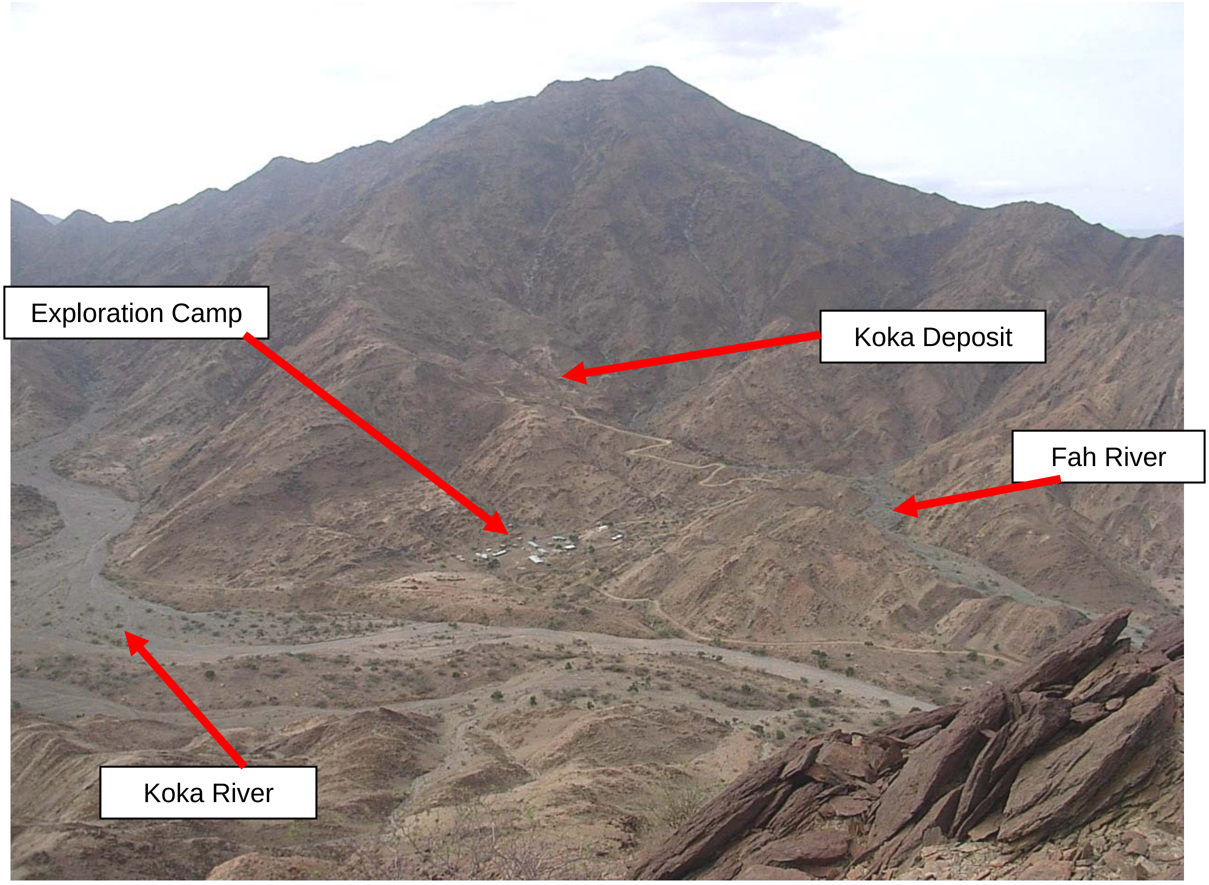
The project site is located about 165 km north of the capital city Asmara in sub-Zoba Sela, Zoba Anseba. The area is situated in the arid lowland zone with a subtropical climate with distinct dry and rainy seasons. The rainy season in the project area is very short duration and erratic and drought is common. Based on site data, the total rainfall in 2008 was 117.4 mm and temperatures at site in 2008 ranged from a low of 16.75ºC in March and a high of 44.75ºC in May.

There are no perennial rivers or streams within the project area; however major seasonal rivers like the Zara, Fah and Koka are dry for more than 9 months and flow of water occurs for a very short period of time during the rainy season.

2.2.2 Site Topography

The topography of the project area is dominated by steep slope mountains, ridges and valleys, with the elevation ranging between 500 m above sea level (asl) along the river bed and 2000 m asl at the mountain crests. The Koka deposit is located on the north-west slopes of Debre Konate, one of the larger mountain peaks in the area (Figure 2.2.2).

Figure 2.2.2 Koka Deposit



2.2.3 Tenure

The current Exploration Lease covers 147 square kilometres.

Chalice was granted a 12 months extension expiring on 25 May 2011 subject to a feasibility study and a socio-economic impact assessment (SEIA) being completed by the 25 May 2011.

In August 2009, an additional 468 square kilometres of tenure was granted covering the northern and southern extensions of the Koka trend.

On application for a Mining Licence, the Eritrean Government is entitled to a 10% free carried interest and, in addition, the Government has the right, by agreement, to purchase a further 20% equity participation interest, at market value, in any mining project.

2.2.4 Fiscal Regime and Taxation

Taxable income from mining operations is subject to income tax at the rate of 38% of taxable income. All capital expenditure and pre-production costs may be depreciated. Any financial loss resulting from mining operations of a licensee in an accounting year may be carried forward and deducted from gross income in the next ten accounting years.

Service tax of 10% is payable on contracted services and import duties of 0.5% are payable on all imports of equipment and materials (except sedan cars) for mining operations.

2.3 Geology

2.3.1 Introduction

The Zara Project is situated within rocks assigned to the Nakfa and Adobha Abiy terrains and to a flexure in the Elababu Shear Zone, which separates the Adobha Abiy and Nakfa terrains, and where there is an abrupt change in azimuth from northeast to north-northeast.

The eastern and central parts of the Zara exploration licences are underlain by meta-volcanic and meta-sedimentary rocks metamorphosed to greenschist facies, together with post-tectonic granitoids, assigned to the Nakfa terrain. The western part is underlain by predominantly siliciclastic rocks, together with minor meta-chemical sedimentary rocks, basalt and syn-tectonic granitoids assigned to the Adobha Abiy terrain.

The Koka mineralised zone has a total strike length of more than 650 m and lies adjacent to the sheared and altered contact between a sequence of meta-sedimentary and meta-basaltic rocks in the west (footwall) and a meta-volcanic and meta-volcaniclastic sequence, intruded by granitoid bodies, to the east (hangingwall) within the Nakfa terrain. The meta-sedimentary rocks comprise tuffaceous greywackes, siltstones, and shales with minor mafic intrusive rocks. This sequence is isoclinally folded. The meta-volcanic and meta-volcaniclastic sequence comprises more massive, principally intermediate and acidic, pyroclastic rocks and intrusions of microgranite and micrographic microgranite together with minor rhyolite and dacite.

The Koka deposit is located on the Zara Project which covers the boundary between the Neoproterozoic Adobaha Abiy and Nakfa terrains. A prominent north-trending ridge of basalt outcrops at the boundary between the western sequence of siliciclastic and chemical sediments and the volcanic-sedimentary pile in the east. Texturally the rock varies from fine-grained, locally vesicular, aphanitic basalt to very coarse plagioclase-phyric to trachytic basalt. The Zara area is underlain by approximately 65% volcano-sedimentary strata and 35% syn-tectonic to post-tectonic granite and gabbro intrusions.

The rocks at Koka have been affected by at least two stages of deformation:

- Compression and folding with weak to no structural foliation development.
- Penetrative deformation related to at least one episode of east-over-west thrust faulting. Foliation fabrics dip at high angles towards the east.

The deformation-related structures are well developed in the metasediment as well as in the volcanic rocks.

There are five principal alteration types associated with gold mineralisation at Koka. The main hydrothermal alteration in the area can be recognised as one or multiple phases of carbonate, sericite, silica, chlorite or pyrite alteration. Generally the alteration haloes are gradational, vary according to proximity to the mineralisation, and are dependant on parent rock type.

The gold mineralisation at Koka is developed principally within an elongated lensoid body of microgranite intruded along the western margin of the meta-volcanic and meta-volcaniclastic succession. This unit has been strongly silicified and brecciated and is cut by a stockwork of quartz veins. There is a considerable competence contrast between this unit and the meta-sedimentary and meta-basaltic sequence immediately to the west. This competence contrast is believed to be significant in locating both deformation and mineralisation. The meta-sedimentary rocks behaved competently, whereas the meta-volcanic and meta-volcaniclastic sequence behaved incompetently resulting in brecciation and multiple phases of quartz veining.

Drilling and Sampling

All drilling at the Koka Gold Project has been diamond drilling. 134 drillholes totalling 20,446 m were used for the 2010 Mineral Resource estimate, which includes drilling up to the end of February 2010. Eight drillholes for 1,078 m completed for metallurgical testwork have no associated assays. Four drillholes designed by AMC have been drilled for geotechnical purposes only and have not been assayed.

Drilling campaigns are summarized in Table 2.3.1.

Table 2.3.1 Number of Drillholes and Year Drilled

Drillhole Number	Year Drilled
ZARD001 to ZARD004	2005
ZARD005 to ZARD026	2006
ZARD027 to ZARD111	2007
ZARD112 to ZARD127	2008
ZARD128 to ZARD157	2009 - 2010

Drilling has been completed to an average depth of about 165 m below surface and drillholes close out the mineralisation at depth on most sections. Drilling completed between October 2009 and March 2010 was aimed at intermediate sections between the previous 40 m spaced sections resulting in a drillhole pattern of 20 m x 20 m at surface. The drilling was aimed at understanding the short range variability of the gold grade and a better understanding of geological controls.

The local grid has been surveyed by qualified surveyors and substantial survey data has been collected to ensure the local grid is robust and consistent relative to the UTM.

Downhole surveys were completed using a Reflex EZ-Shot® tool. All diamond drillholes were surveyed at intervals of 30 m downhole.

Diamond core recovery ranged between 88.5% and 99.7%, with the majority of drilling having a core recovery of 95%.

Each drillhole has been logged by qualified geologists using a standardised logging format and geological codes and core has been photographed to provide a permanent record.

Diamond drillcore was sampled on 1m intervals over intersections of the microgranite that hosts gold mineralisation.

Core sample were prepared in Asmara by Eritrean company Africa Horn, a joint venture with Genalysis Laboratory Services Pty Ltd (Genalysis) of Perth, Australia. QA assessment of the assay results suggest that the laboratory has performed very well from the macro scale (sample preparation of received field samples) to the analytical scale (both pulp preparation handling and aliquot preparation/analysis).

Modelling

Chalice provided AMC with data for drillhole collars, downhole surveys, gold assays and geological logging in comma separated format. The data review found extensive use of descriptive geology codes that were not specified on the Chalice Gold Geology Code list, but concluded it is unlikely that inconsistent use of geology codes would have an impact on the resource estimate. Minor depth record discrepancies were discovered in the data, but AMC found no errors in regard to samples reporting gold grades higher than the detection limit.

The interpretation used to develop domains for resource estimation was based on gold grades and the geological relationship of gold grade to sulphide minerals associated with quartz veining.

A probability-based method was used to assist in interpreting wireframe boundaries to three domain-types hosted within the microgranite. The probability model was created by assigning an indicator to sample intervals where gold grade was above 0.3 g/t Au and total sulphide content exceeded 1%. The indicator values were estimated into a model within the microgranite envelope. The gold and sulphide domains were combined into one mineralisation domain for grade estimation.Gold composite statistics were analysed to determine if any domain had unusually high grade outliers that would need topcutting. A topcut of 200 g/t Au was applied to the composites within mineralisation domains. The final choice of parent cell size for volume modelling was based upon the drillhole spacing, the descriptive statistics and variography. A block size of 10 m x 10 m in easting and northing was applied, supported by quantitative kriging neighbourhood analysis. The 5 m parent cell height reflected possible mining bench height.

For tonnage calculations, the dry bulk density of 2.74 t/m^3 was assigned to all parent cells in the block model, based on averaging of Koka bulk density determinations from drillcore.

AMC ran kriging efficiency tests for the gold domain only, with the results of the testing showing that generally, gold would be well estimated using the parameters finally selected. Gold grades were estimated into the volume model using ordinary kriging. Grades were estimated into parent cells, with all subcells receiving the same grade as its parent. The maximum number of composites allowed for each estimate was 30, with estimation of most cells within the mineralisation domain completed with 30 composites.

Resource classification was set in two passes using some of the estimation parameters for the first pass, then rationalised section by section. The final classification resulted in all of the mineralisation domain being classified as Indicated Resource, due to the proximity of drilling, the number of samples used in the estimate and the confidence in the interpretation. None of the estimate was classified as Measured Resource, as the estimate is very sensitive to changes in the estimation parameters.

The microgranite outside the mineralisation domain is classified as Inferred Resource although none of this estimate reports above a 1.2 g/t Au cut-off. There has been no significant mining at Koka although there has been some artisanal mining and surface disturbance due to drilling. Depleted volumes are not known but in AMC's opinion, the volumes are unlikely to be material to the Mineral Resource estimate.

The Mineral Resource estimate for the Koka gold deposit, classified and reported in accordance with the JORC Code[1] is listed in Table 2.3.2. Mineral Resources are reported inclusive of Ore Reserves.

Table 2.3.2 Koka Gold Deposit Mineral Resource Estimate as at 1 June 2010 Reported at 1.2 g/t Au Cut-Off

Category	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (oz)
Indicated Resource	5.0	5.3	840,000

The resource classification is based upon a 1.2 g/t Au cut-off which may not be applicable at higher cut-off grades.

2.4 Mining

2.4.1 Geotechnical

The assessment of the slope parameters was based on information available from dedicated geotechnical drillholes, core logging and laboratory tests.

The pit was divided into two sectors considering the geology and the pit wall dimensions. Based on a preliminary assessment of the probable shape of the pit, the base of the pit was located at 740 mRL.

Detailed design parameters are provided in Table 2.4.1.

1 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, The JORC Code 2004 Edition, Effective December 2004, Prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

Table 2.4.1 Design Parameters for Detailed Design

Sector	Depth (mbs)	Batter Angle (BFA) (º)	Batter Height (H) (m)	Berm (B) (m)	IRA (º)	IRA Height (m)	Geotech Berm Width (Rw) (m)	OSA (º)
E -	0 - 25	60	10	5	42.9	60	15	45
Hangingwall	>25	75	10	5	54.8	80	15	
W -	0 -25	60	10	5	42.9	60	15	48
Footwall	>25	75	10	4.5	56.5	80	15	

2.4.2 Pit Optimisation

Pit optimisation was performed using Whittle Four-X software. The pit optimisation excluded Inferred material.

Ore Parameters

The following ore related parameters were used in the optimisation:

- Process and administration cost of $33.49/t processed as supplied by Lycopodium assuming a 0.5 Mtpa processing rate (based on the Scoping Study).
- No off-site costs were included.
- A metallurgical recovery of 96.2% (at a head grade of 5 g/t) as indicated by Lycopodium and based on metallurgical testwork conducted by AMMTEC.
- A gold price of US$900/oz.
- Foreign exchange rate of US$0.85 to A$1.
- A government royalty of 5% of revenue as per the Eritrean Mining Regulations, Clause 35.

For the purpose of creating discounted cash flow scenarios in Whittle, a discount factor of 10% was applied and a maximum processing rate of 0.5 Mtpa was applied.

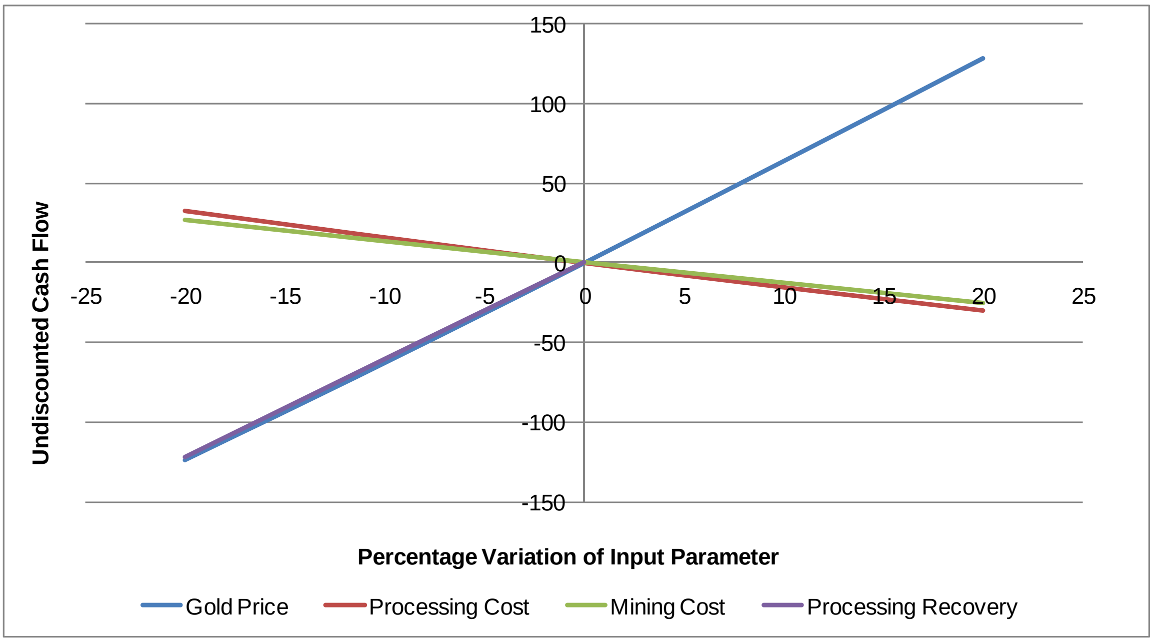
The treatment plant breakeven COG was estimated to be 1.27 g/t.

Costs

Optimisation Run No. 11 was used as the basis for the FS design and included all parameters required for the FS and used the reference gold price of US$900/oz. From review of the optimisation results, pit Shell 6 (starter pit) and 24 (final) were selected as the basis for the pit designs.

Sensitivity of the project outcomes to changes in the main project parameters was tested and the results of the sensitivity analysis are shown by changes in the undiscounted cash flow, Figure 2.4.1.

Figure 2.4.1 Sensitivity of Undiscounted Cash Flow



Pit Design Staging

A smaller starter pit (Stage 1) was developed to provide ore to the plant in the first year of operation. The starter pit contains:

- 2.4 Mt of ore at a grade of 6.1 g/t Au.
- Total tonnage of 20 Mt.

A final optimisation run was performed on the mining model with the results showing an increasing number of ore tonnes and decreasing Au grade contained within pit shells of increasing size. Evaluation of the results shows that:

- The selection of Shell 24 is valid due to immaterial difference when compared to shell 18.
- Updates to the mining cost, processing cost and slope angles do not result in a material difference in overall discounted cash flow.

Sections through the starter and final pits are shown in Figure 2.4.2 and Figure 2.4.3.

Figure 2.4.2 Cross Section Through Starter and Final Pit

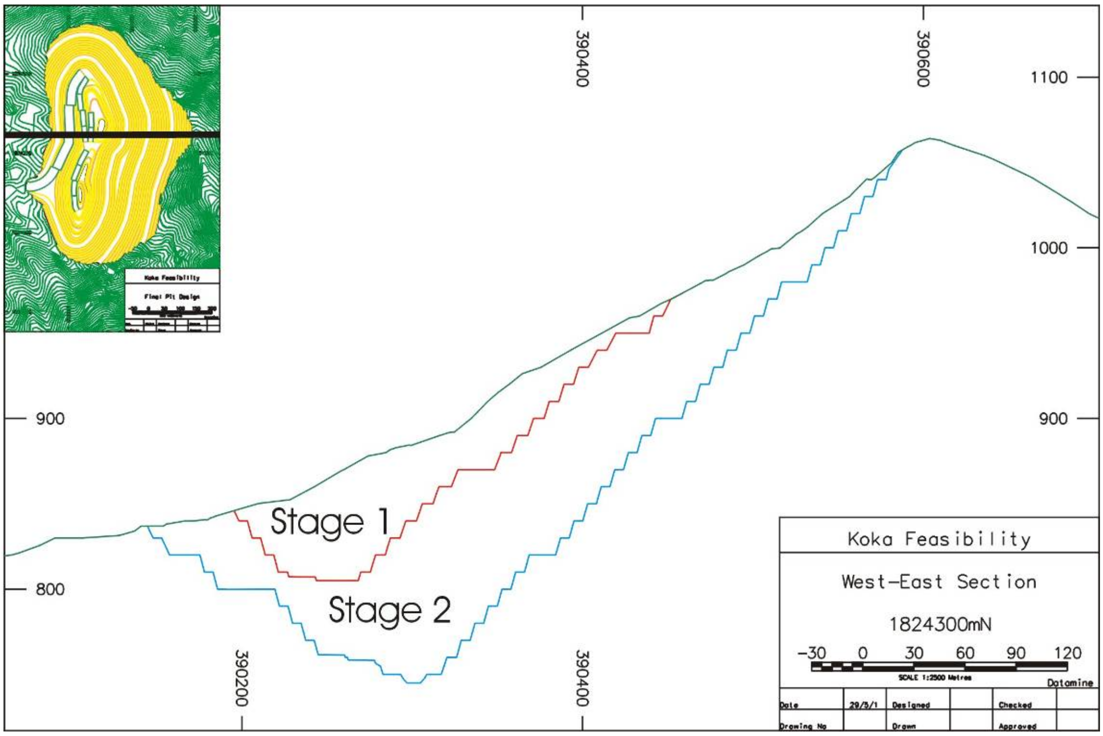
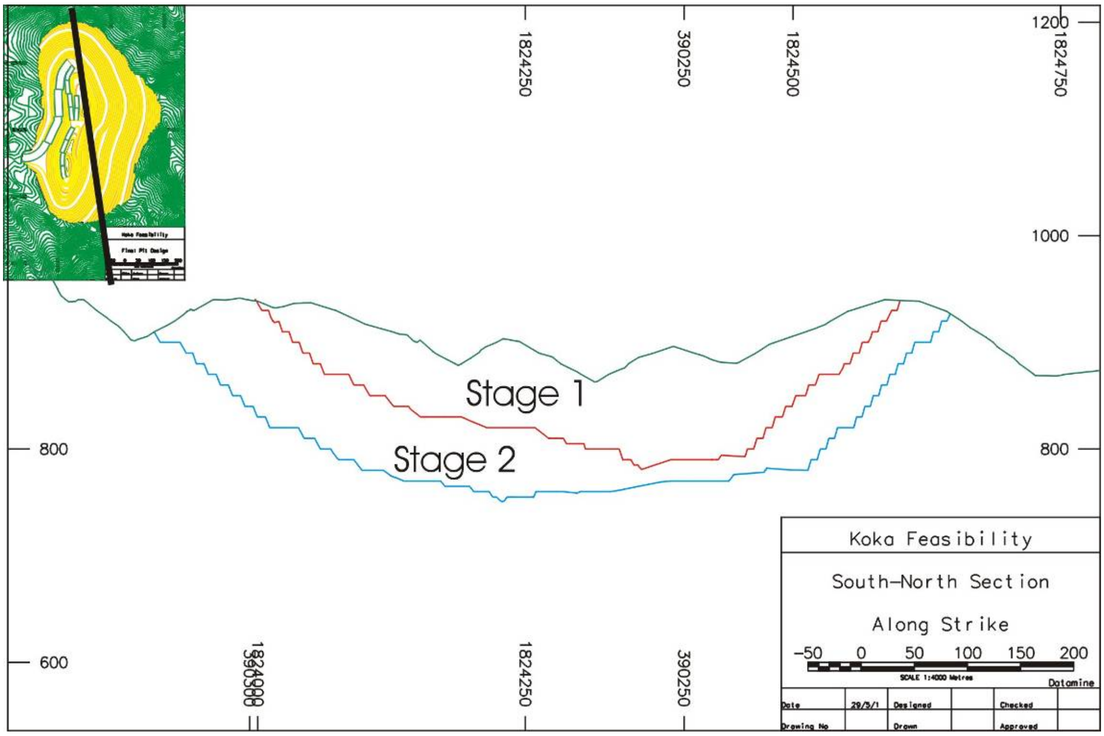


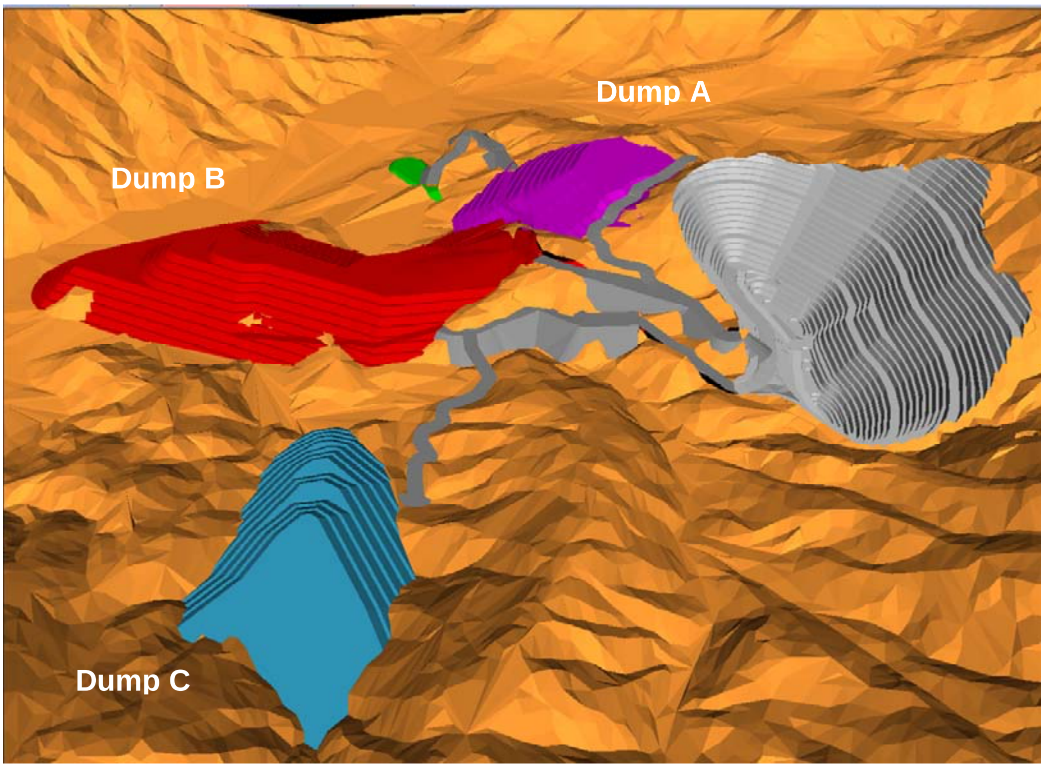
Figure 2.4.3 Long Section Through Starter and Final Pit



Waste Dump Design

Due to the steep topography there are limited locations available to locate waste dumps. Mine waste will be used to develop haul roads, construct the tailings dam, other miscellaneous earthworks and then three main dumps will be developed as shown in Figure 2.4.4.

Figure 2.4.4 Waste Dump Locations (Looking North)



Mine Schedule

The production schedule is summarised in Table Figure 2.4.5, based on:

- Staged pit designs.
- Maximum mining rate of 9 Mtpa.
- Processing rate of 0.6 Mtpa rising to 0.7 Mtpa as the head grade decreases.
- Minimising the ramp up time from commencement of mining to achieving full mill production rates.
- In Year 2 of the schedule Stage 1 mining is slowed to a rate to match mill ore requirements and production in the Final pit is commenced. The total mining rate in Year 2 drops to 8 Mtpa as mining is constrained by the vertical rate of advance in the Final pit cutback.

- In Years 3 and 4 mining rates are maintained at 9 Mtpa with Stage 1 supplying ore to the mill whilst the Final pit undergoes waste removal. The Stage 1 pit is completed mid Year 5.

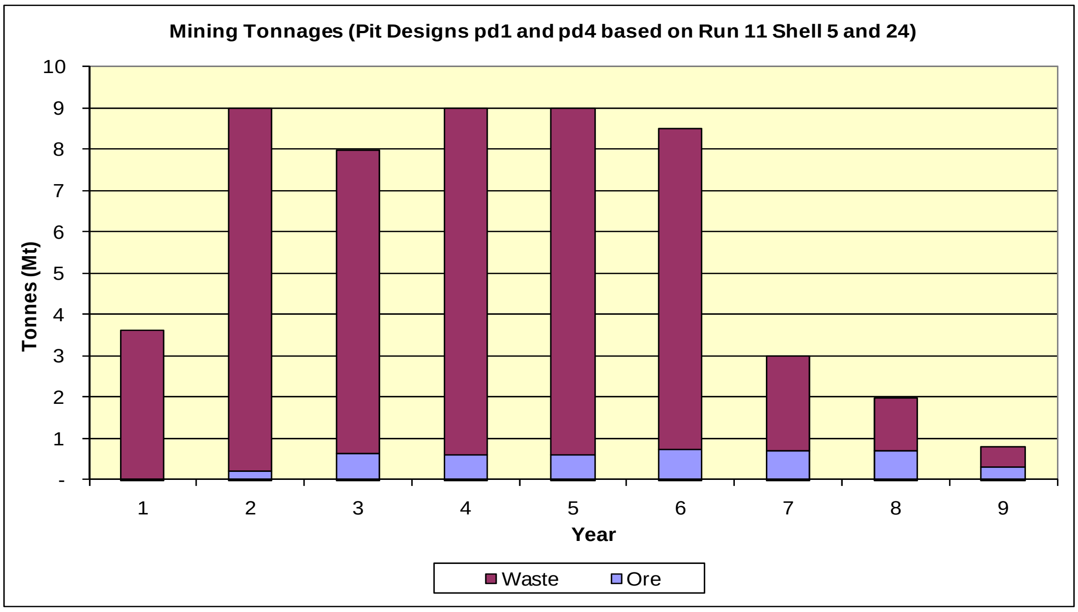
- From Year 5 onwards the strip ratio is low and the mining rate declines rapidly through Years 6, 7 and 8, however a significant ore stockpile is built.

Table 2.4.2 Mine Production Schedule

Annual	Units	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mining											
Ore Tonnes	t	0	224,129	637,436	627,436	619,391	757,539	723,977	720,248	320,606	4,630,761
Gold Grade	g/t	0.00	5.42	7.29	6.39	5.52	4.18	3.54	3.87	5.61	5.10
Contained Gold	oz	0	39,053	149,391	128,977	109,864	101,873	82,403	89,504	57,828	758,891
Ore Volume	bcm	0	81,799	232,641	228,991	229,667	331,087	278,667	228,104	79,103	1,690,059
Waste Tonnes	t	3,625,000	8,775,871	7,362,564	8,372,564	8,380,609	7,742,461	2,276,023	1,279,752	497,582	48,312,428
Waste Volume	bcm	1,335,109	3,203,608	2,886,027	3,079,372	3,182,925	2,683,712	734,702	395,874	110,446	17,611,775
Strip Ratio	t:t	0.00	39.16	11.55	13.34	13.53	10.22	3.14	1.78	1.55	10.43
Total Tonnes	t	3,625,000	9,000,000	8,000,000	9,000,000	9,000,000	8,500,000	3,000,000	2,000,000	818,189	52,943,189
Total Volume	bcm	1,335,109	3,285,407	3,118,668	3,308,363	3,412,592	3,014,799	1,013,369	623,979	189,549	19,301,834
Milled											
Tonnes	t	-	224,129	600,000	600,000	600,000	700,000	700,000	700,000	506,632	4,630,761
Grade	g/t	-	5.42	7.32	6.45	5.64	4.35	3.66	3.70	5.04	5.10
Ounces	oz	-	39,053	141,120	124,338	108,802	97,802	82,401	83,246	82,130	758,891
Recovered oz*	oz	-	37,699	136,544	120,199	105,067	93,810	78,871	79,691	78,901	730,780

** Note: the Recovered oz shown in the mining tables in the appendices were calculated at constant recovery; the recovery varies with grade as detailed in Sect 6.11. Actual recovered oz are as shown in Table 2.4.2.*

Figure 2.4.5 Annual Mining Tonnages



(Note: Year number designations are mining years, not processing years as in other figures)

Mining Operations

The operation will be 'owner operated', with the exception of some specialist activities including the supply of explosives and tyre maintenance services. The mining operations will be conducted on a 24 hours per day, 7 days per week, year round basis.

The conventional open pit truck and excavator mining method has been adopted for the Zara project, with the loading fleet has been sized to match the schedule requirements:

- Caterpillar 777, 90 t class haul trucks.
- Bucyrus RH 40-E, 110 t class, diesel hydraulic excavators with 6.7 m^3 buckets.

Mine Operating Cost

AMC estimates an average mine operating cost of $1.99 per tonne mined for the pd1c.dm pit design, as shown in Table 2.4.3.

Table 2.4.3 Summary of Estimated Mine Operating Costs

Item	Units	Total
Total Material		
Waste	kt	48,539
Ore	kt	4,631
Total Movement	**kt**	**53,169**
Capital Expense Material Movement	**kt**	**3,662**
Operational Expense Material Movement	**kt**	**49,507**
Mining Activity		
L&H (incl. aux., fixed & o'head costs)	US$ '000	76,820
D&B	US$ '000	26,812
Grade Control	US$ '000	454
Crusher Feed	US$ '000	1,981
Subtotal	**US$ '000**	**106,067**
Total Capital Expense	**US$ '000**	**11,321**
Total Operational Expense	**US$ '000**	**94,747**
Mining Activity		
L&H (incl. aux., fixed & o'head costs)	$/t mined	1.44
D&B	$/t mined	0.50
Grade Control	$/t mined	0.01
Subtotal	**$/t mined**	**1.99**
Total Capital Expense	**$/t mined**	**3.09**
Total Operational Expense	**$/t mined**	**1.91**
Mining Activity		
Production Drill	US$ '000	12,359
Production Blast	US$ '000	14,452
Load	US$ '000	11,368
Haul	US$ '000	26,394
Ancillary	US$ '000	18,340
Overhead Salaries and Wages	US$ '000	18,347
Miscellaneous Operational Overheads	US$ '000	2,371
Grade Control	US$ '000	454
Crusher Feed	US$ '000	1,981
Subtotal	**US$ '000**	**106,067**
Total Capital Expense	**US$ '000**	**11,321**
Total Operational Expense	**US$ '000**	**94,747**
Mining Activity		
Production Drill	$/t mined	0.23
Production Blast	$/t mined	0.27
Load	$/t mined	0.21
Haul	$/t mined	0.50
Ancillary	$/t mined	0.34
Overhead Salaries and Wages	$/t mined	0.35
Miscellaneous Operational Overheads	$/t mined	0.04
Grade Control	$/t mined	0.01
Crusher Feed	$/t mined	0.04
Subtotal	**$/t mined**	**1.99**
Total Capital Expense	**$/t mined**	**3.09**
Total Operational Expense	**$/t mined**	**1.91**

Mine Capital Cost

AMC estimates a life of mine (LOM) capital cost of US$22.2M as shown in Table 2.4.4 to an accuracy of ±15% as at Quarter 1 of 2010 exclusive of GST and undiscounted.

Table 2.4.4 Summary of Estimated Mine Capital Costs

Capital Element	LOM Total (US$'000)	Establishment Cost (US$'000)	Production Cost (US$'000)	Sustaining Capital (US$'000)
Mobile Equipment	20,488	12,268	4,699	3,521
Maintenance Equipment	1,471	1,471	-	-
Technical Equipment	256	256	-	-
Total	**22,214**	**13,994**	**4,699**	**3,521**

Mining Risk

AMC conducted a risk assessment including both threats and opportunities in the assessment. The assessment identified only one risk with a residual ranking of Extreme: a lack of an available workforce during the mine life, driven by market forces. The control strategy recommended is to adopt appropriate marketing and retention strategies.

2.4.3 Reserve Statement

The Koka Gold Deposit Ore Reserve estimate, classified and reported in accordance with the JORC Code, is listed in Table 2.4.5. This is the first Ore Reserve estimate reported for the Koka Gold Deposit.

Table 2.4.5 Koka Gold Deposit Ore Reserve Estimate as at 1 June 2010

Category	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (oz)
Probable Reserve	4.6	5.1	760,000

2.5 Metallurgy

A comprehensive metallurgical testwork program was conducted on primary ore samples from the Koka Prospect at the laboratory of Australian Metallurgical and Mineral Testing Consultants (AMMTEC) in Perth, Western Australia.

The detailed testwork programme was undertaken for the Definitive Feasibility Study with the following objectives:

- Select the most suitable processing route.
- Determine the optimum plant operating parameters for the ore being processed.
- Evaluate the variability in metallurgical performance for the primary ore source.
- Obtain data required for plant design.

A total of seven diamond drillhole cores were transported from site to AMMTEC for comminution and metallurgical sample preparation. The drill core consisted of whole HQ core intercepts that had been cut into 200 mm sticks and individually labelled and bagged.

Samples for metallurgical testing were selected after crushing and assaying every second 200 mm core sample interval. A master composite sample was formed to allow development of an optimised treatment route. The master composite was formed from portions of eight variability composites selected to confirm metallurgical response throughout the orebody and with respect to gold head grade.

Comminution samples were selected from the uncrushed core intervals remaining from the seven diamond drillhole cores. A total of ten composites were formed for comminution testwork.

Salient outcomes of the metallurgical and comminution testwork programmes conducted on Koka Prospect primary ore were:

- The ore is moderately competent and abrasive with above average comminution energy requirements.
- The ore is ‘free-milling’ with a high gravity-recoverable free gold component and high gold extraction from the gravity tails by cyanidation leach with low reagent consumption.
- Anticipated lime and cyanide consumption are low and are typical of operations conducted with good quality water treating clean ‘free milling’ (non refractory) primary ores.

2.5.1 Head Grade Analysis

Detailed head grade analysis of the master and variability composites indicated the following:

- Total sulphur ranged from 0.69% to 2.84%.
- Silver grades are variable (0.3 to 32 g Ag/t) and will need to be considered in design of the CIL and elution circuits.
- Mercury (<0.10 ppm) and arsenic levels (<88 ppm) are low and should not present an environmental or occupational health risk in the elution or electrowinning circuits.
- Concentrations of some base metals such as copper (109 to 387 ppm), lead (25 to 3,163 ppm) and zinc (44 to 3,481 ppm) are slightly elevated, and whilst these may contribute in a small way to increased cyanide consumption, the levels are unlikely to adversely affect carbon performance.
- Organic carbon levels (<0.03 ppm) are low and preg-robbing should not be a problem.

2.5.2 Comminution Testwork

The comminution testwork program was conducted under the supervision of Orway Mineral Consultants Pty Ltd (OMC). Outcomes from the comminution testwork programme were as follows:

- Unconfined compressive strength (UCS) ranged from 30 MPa to 135 MPa, with an average UCS of 74 MPa. The UCS values indicate the ore does not have high competency and can be primary crushed using a jaw crusher.
- Crushing work indices (CWI) ranged from 6.6 to 20.5 kWh/t, with an average CWI of 9.8 kWh/t. Most samples tested showed low to average resistance to impact breakage.
- Bond rod mill work indices (RWI) ranged from 14.9 kWh/t to 21.9 kWh/t, with an average RWI of 17.5 kWh/t.
- Bond ball mill work indices (BWI) ranged from 14.5 kWh/t to 18.0 kWh/t, with an average BWI of 16.8 kWh/t. The Bond rod and ball mill indices are relatively high indicating a high grinding energy requirement.
- The abrasion indices measured range from 0.156 to 0.407, with an average abrasion index of 0.305. The abrasion indices are relatively high indicating moderate to high liner and media wear rates in crushing and grinding circuits.
- The A x b values ranged from 32.3 to 80.5.

In summary Koka prospect primary ore is an abrasive, moderately competent ore with above average comminution energy requirements.

On the basis of the comminution testwork programme and modelling of several comminution circuit configurations OMC recommended a single stage jaw crushing circuit followed by single stage semi-autogenous (SAG) grinding mill in closed circuit with hydro cyclones to produce ground product at the selected grind P_{80} size of 106 microns. The selected circuit offers the lowest capital and operating costs of the circuits considered.

2.5.3 Gravity / Cyanidation Testwork

A detailed testwork program was conducted on the master composite to determine the optimum treatment route and conditions for treatment of Koka Prospect primary ore. The testwork program investigated parameters such as the propensity for preg robbing, gravity / leach versus direct leach, optimum grind P_{80} size, residence time, cyanide optimisation, air versus oxygen addition, leach density and the effect of site bore water.

The optimum conditions determined for conducting the leach extraction and reagent consumption tests on the master and variability composites based on the preliminary testwork outlined above were as follows:

- P_{80} grind of 106 microns.
- Gravity concentration by centrifugal concentrator.
- Leach pulp density of 50% solids.
- pH 10.0 – 10.5 adjusted with commercial lime (60% available CaO).
- Initial cyanide dosage of 0.5 kg NaCN/t with residual cyanide levels maintained at or above 200 ppm.
- Dissolved oxygen levels of 20 ppm or greater.
- 36 hour leach duration.

The master composite sample and eight variability composite samples were treated using the optimised treatment scheme. Salient outcomes of testwork were as follows:

- Gravity gold recoveries were moderate to high and ranged from 45 to 72%.
- Overall gold extractions were excellent and ranged from 95.3 to 99.2% for gold head grades ranging from 2.33 to 14.51 g Au/t.
- Lime consumption was low at 0.33 kg/t (60% available CaO basis).
- Cyanide consumption was low at 0.29 kg NaCN/t.

2.5.4 Ancillary Testwork

A suite of ancillary testwork was completed to assist with process plant design. Ancillary testwork completed included the following:

- Oxygen uptake testing – the ore has a low oxygen demand. Oxygen addition to the leach circuit has been included in design to improve leach kinetics and overall gold extraction and to help minimise cyanide consumption.

- Viscosity tests – the primary ore has low viscosity. A pre-leach thickener ahead of the CIL circuit has been incorporated to improve mixing in the CIL tanks and reduce overall reagent consumption. The pre-leach thickener also allows classification efficiency in the grinding circuit to be maximised by operating at optimal cyclone feed densities.

- Carbon loading kinetics – carbon loading kinetics are good and high carbon loadings are achievable.

- Cyanide destruction – cyanide destruction testwork using the air / SO_2 process was conducted on leach slurries and weak acid dissociable (WAD) cyanide levels were readily reduced to below 50 ppm – the recommended discharge level in the International Cyanide Management Code (ICMC).

  Assessment of an air / SO_2 cyanide destruction circuit versus a three stage counter current decantation (CCD) wash circuit for reduction of WAD cyanide in the CIL discharge stream indicated that the CCD wash circuit offers benefits such as reduced operating costs, recovery of a portion of the cyanide in the tailings stream for re-use in the process plant as well as recovery of a portion of any gold in solution.

  A CCD wash circuit that reduces WAD cyanide to less then the 50 ppm required by the ICMC was adopted for plant design.

- Thickening testwork – the primary ore has excellent settling characteristics (2.0 $t/m^2.h$) due to the clean nature of the primary ore. Thickening to the pre-leach feed density of 50% w/w solids and underflow densities of 60% w/w solids in the CCD circuit will present no difficulties.

2.5.5 Plant Recovery

The testwork results suggest that primary ore calculated gold head grade and gold residue grades are mildly correlated and that a linear model to predict tailings grade based on head grade is appropriate to estimate gold recoveries. Allowing for large scale plant inefficiencies and solution losses, the anticipated recovery from a head grade of 6.0 g Au/t is 96.6% at the nominal design throughput of 600,000 tpa. At the higher throughput of 700,000 tpa expected in Years 5 to 7, the reduced residence time in the CIL circuit and possible coarsening of the grind will lower the anticipated recovery by 0.4%.

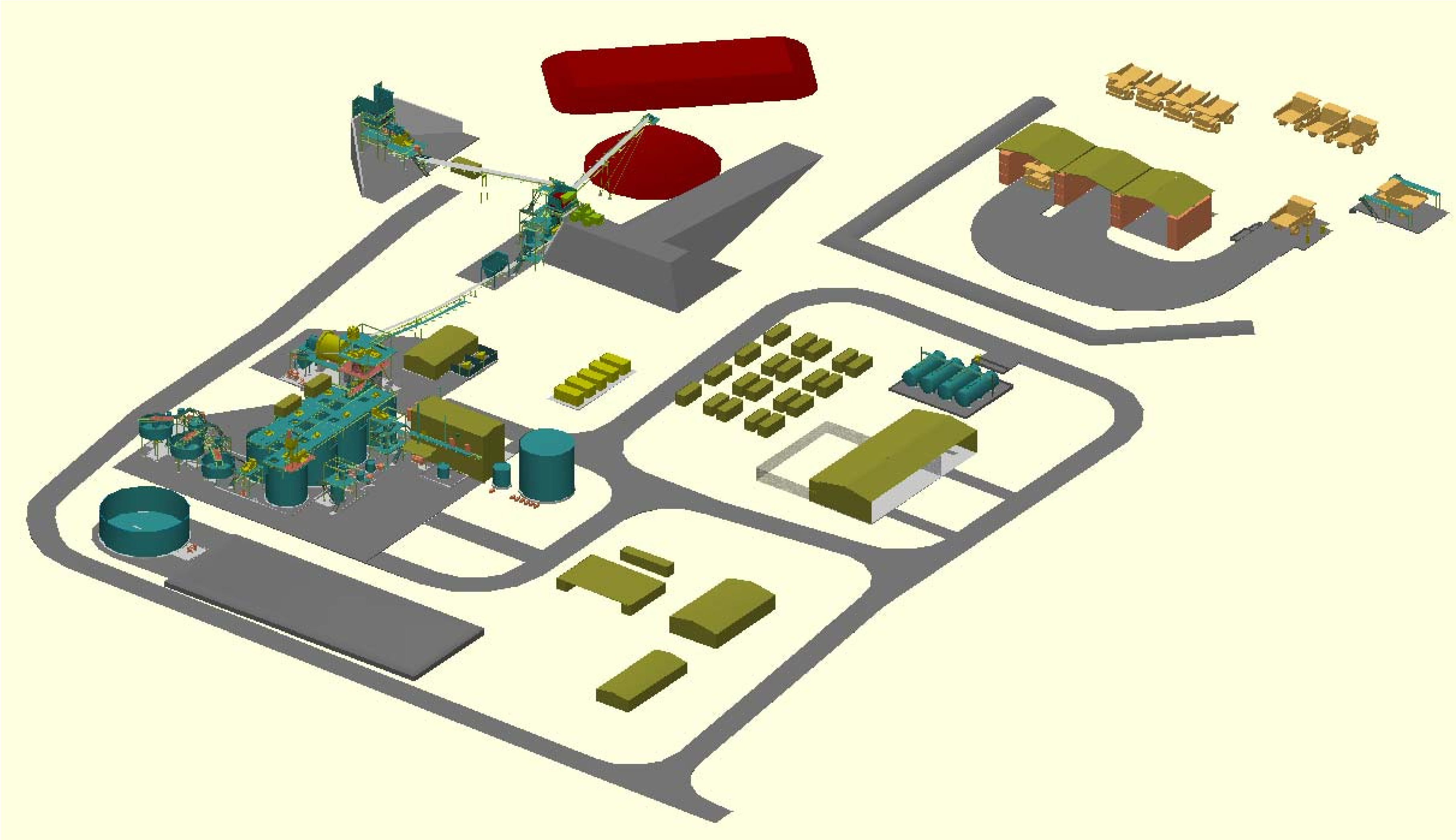
2.5.6 Process Flowsheet

The overall process flowsheet adopted for the Zara Project has been based on the outcomes of the detailed metallurgical testwork program. The flowsheet selected is based on industrially proven unit processes and presents low technical risk. The major unit processes incorporated in the flowsheet are as follows:

- Single stage jaw crushing circuit.
- Single stage SAG mill in closed circuit with hydro cyclones.
- Gravity concentration circuit treating a portion of cyclone feed. The gravity circuit will comprise a centrifugal gravity concentrator and an intensive cyanidation and electrowinning module for recovery of gold from the gravity concentrate.
- Pre-leach thickening.
- 7 stage carbon-in-leach circuit.
- Zadra elution circuit for recovery of gold on carbon.
- Three stage counter current decantation circuit to treat CIL discharge slurry and lower WAD cyanide in the tailings stream to below 50 ppm.

A view of the process plant is given in Figure 2.5.1.

Figure 2.5.1 Process Plant



2.6 Support Facilities and Services

2.6.1 Access

The preferred site access by road from Asmara is via Keren and Akurdet, a distance of 440 km. The existing Government roads are in a good condition to Kerkebet Bridge with the final 100 km of road to the project site needing to be upgraded. A primary upgrade has been effected to support exploration activities, but a second stage of upgrade is needed for construction and operations traffic.

Plant site access from Rikeb and the accommodation village will also be upgraded, but will still pass through the Koka River bed, which may cut access for short periods during flow events.

A gravel airstrip will be constructed at Rikeb for personnel transfers to and from Asmara. A helipad will also be built near the plant site for secure air transport.

2.6.2 Plant Site

The plant site will be located on the site of the existing exploration camp and will be constrained by the mine waste dumps and the Koka River. The site will be terraced, using mine waste where necessary, to take advantage of the topography. Safety berms and catch drains will isolate the plant site from the waste dump and run-off will be diverted around the plant. Within the plant area, drainage will be collected and recovered to the process.

2.6.3 Accommodation Village

A 250 person permanent village will be established to house 84 senior staff in single rooms and a further 168 junior staff and shift workers in six-person dormitories, with the necessary kitchen, mess, recreation and laundry facilities. The buildings will be predominantly flat-pack construction, but some containerised units will be provided (e.g. freezer units).

The village will be supplemented by a tent camp constructed very early in the project to house the construction workers building the village and mining staff assembling the fleet. Once the permanent village is complete, the mining staff and senior construction staff will use the facility with additional construction crews being housed in the tents.

2.6.4 Tailings Facility

The proposed tailings storage facility (TSF) site is approximately 3 km west northwest of the Koka deposit in a small valley tributary on the north side of the Koka River. The facility will be constructed in stages to store a total of 4,600,000 t, using suitable mine waste being placed as part of the mining operations. The TSF basin is located in a stream-valley that is bounded by ridges on all sides.

The coarse nature of the tailings indicates a relatively high permeability and the geochemical assessment indicates a moderately low potential to form acid but sufficient to lower the pH within the facility if full oxidation of the tailings is permitted. This may lead to dissolution of the contained heavy metals. The design and operating procedures for the facility will minimise the potential for acid generation.

Bores will be established to monitor seepage and the wall will be surveyed regularly. The facility will be fenced and capped at the end of its life to minimise water ingress.

2.6.5 Water Supply

With no perennial rivers or permanent surface water features in the region, the Project water demand must be satisfied by groundwater.

A hydrogeological assessment identified that the most promising aquifer targets lie adjacent the main river channel of the Zara River, north and south of the Rikeb settlement. The Zara river has a significant catchment area (977 km^2) and groundwater can therefore be developed from surface water leakage, groundwater storage and groundwater throughflow.

In years one to three of plant operation the average water demand for the project is estimated to be 31 L/s with a plant throughput of 600,000 tonnes per annum (tpa). In years four to seven the average water demand is estimated to be 34 L/s with a plant throughput of 700,000 tpa.

Five test production bores have been installed into the Zara valley alluvium, with airlift yields ranging from 7 to over 20 L/s. Pumping tests have been conducted and base don the results and the preliminary groundwater balance, a bore field extending along a three kilometre stretch of the Zara valley is recommended.

Raw water will be pumped about 8.5 km up the Koka River valley to the process plant with a supply taken to the village located about 5 km along this route.

Potable water will be obtained from the raw water supply pumped from the Zara River. The preferred treatment system based on the water quality from the production bores will be simple filtration and disinfection.

Sewage from the village will be treated in a package treatment plant and treated effluent disposed of in evaporation ponds. Wastewater from small ablution facilities in the plant will be treated by local septic tanks and leach drain systems. At the main crib area a small aerobic treatment unit will be installed to treat wastewater. The treated water will be disposed of by ground soakage.

Thickener overflows will be used as process water in the plant. Make-up to the process water system will come from the raw water system. Any decant water collected from the tailings will be returned to the process water system.

2.6.6 Surface Water

Stormwater runoff from undisturbed areas of the plant site will be allowed to run into the Koka River, through diversion channels and sediment control structures if necessary. Rainfall onto bunded areas will be recovered by the sump pumps and used in the process. Rainfall onto other plant areas will flow into a secure event pond and returned to the process, as it may be contaminated with chemicals or ore spillage.

2.6.7 Power

Power will be generated for the operations by diesel gensets at 11 kV to meet the expected maximum demand of 5.1 MW. The mill motor will operate at this voltage and power will be transmitted to the village at this voltage. The borefield will have a separate generator and a stand-by generator will be located at the village.

2.6.8 Site Buildings

Wherever practical, flat-pack type buildings will be used, imported in containers and assembled on site. This construction method will be used for the laboratory, administration and training facility.

Larger facilities such as the workshop and warehouse will be made up of stacked shipping containers, with roofing over the enclosed spaces and concrete floors as required. The containers will be used as stores and offices.

Specialised buildings such as the plant control rooms and titration laboratory will be prefabricated in shipping containers and brought to site fully assembled.

MCC rooms will likewise be prefabricated and pre-wired inside shipping containers, with the wiring tested in the factory before despatch, to minimise site work.

2.6.9 Mobile Equipment

In addition to the mining fleet, eight 4WD vehicles will be provided for use by the treatment plant and administration department as well as two 20-seater buses to transfer staff around the site, to the airport and any workers recruited from Rikeb. An ambulance will be provided.

Two 5 t trucks with hoists will be provided to carry light freight to and from Asmara.

Other equipment to be provided to the operations include two mobile cranes, portable stand-by generators and lighting towers, diesel welders and HDPE pipe fusion welder, an Integrated tool carrier / forklift for reagent bag handling and a small FEL / Bobcat for plant clean-up.

2.6.10 Fuel Storage

Four 110 kL diesel storage tanks will provide approximately two weeks supply for the mine and power station.

The facility will be bunded with unloading pumps, light vehicle and heavy vehicle rapid fill bowsers. Mining trucks will refuel at the plant site, but other mining equipment and remote generators will be refuelled by a service truck. Transfer pumps will be used to supply the plant diesel genset day tanks.

During construction, the existing 20 kL tank will be supplemented by two additional 20 kL storage bladders, until such time as the permanent fuel storage tanks can be constructed.

2.6.11 Communications

The existing satellite service for voice and data will be used until a new expanded system can be installed for use during construction and later for operations.

A two-way radio system will be installed at the site, covering the mine operations and extending as far as the borefield and the airport. A solar powered transmitter tower will be installed on a convenient hill and the system will provide additional security for the area. Vehicles travelling between Asmara and the site will be equipped with satellite telephones for safety reasons.

The village will have a satellite television service distributed by cable to the single rooms and a communal TV viewing area. Senior rooms will have telephones and internet connections.

2.6.12 Security

The whole Koka Valley has been cleared of artisanal mine workers and the remaining nomadic herders will be relocated to an adjacent valley and provided with an alternate water source. A security check point will be established across a narrow part of the valley, to the west of the accommodation village to secure the mining lease and the authorities will secure the surrounding valley from this point.

Within the plant, additional security will be provided around the gold room.

2.6.13 Rehabilitation and Closure

At the end of the mine life, major items of process equipment and mining plant will be sold. Structures will be dismantled and the construction materials sold for use in other projects. The accommodation units will be offered to the Government for use in the region. Waste dumps will be battered during operation to conform to typical slopes in the region and the tailings dam will be capped.

Seepage bores will be monitored for several years after closure to ensure that no drainage problems arise.

The project will construct an airstrip close to the site, for bullion and personnel transfer to and from Asmara and also for emergency evacuations of medical cases. Chalice are entering into negotiations with airline operators in the region to provide the necessary service.

2.7 Environmental

A comprehensive Terms of Reference in accordance with government guidelines was prepared and submitted with the Scoping Study completed last year. Following minor amendments from the Ministry these accepted Terms of Reference have been the basis upon which a comprehensive Draft Social and Environmental Impact Assessment (SEIA) and Draft Social and Environmental

Management Plan (SEMP) have been developed as stand alone documents in support of this feasibility study and mining license application.

Compilation of environmental baseline data for the Zara Project commenced in 2007 and has continued through 2010 and will be on-going throughout the life of the project. Due to the remote location no regional data or historical records on environmental or meteorological conditions are available for comparisons.

The scope of the SEIA covers and addresses the following key components:

Potential Development Areas (PDAs) are defined as those areas directly affected by Project activities. Environmental baseline studies are an integral part of this as they provide the data in order to assess impacts and this data is generally focused on two spatial study areas as follows:

1. Local study area (LSA) – detailed study areas centring around PDAs and linear corridors; or potentially affected communities

2. Regional study area (RSA) –areas used to assess indirect impacts resulting from Project activities and road access

Baseline studies were completed to:

- define pre-development social and environmental conditions
- identify potential social and environmental issues and sensitivities
- provide information to input into Project design and decision making
- surveys of local communities on socio-economic issues, observations of wildlife, etc.

Socio-economic components of the study addressed:

- Population demographics
- Infrastructure
- Land use
- Employment and training
- Community health
- Social conditions
- Economy
- Governance
- Archaeology

Ecosystem components of the project area addressed:

- Air Quality
- Noise
- Water availability and quality
- Soils
- Vegetation
- Wildlife, including mammals, birds and reptiles
- Livestock

Outcomes from these investigations have concluded that there are overwhelming benefits to the project proceeding as the regional community will benefit with the creation of employment, education and training opportunities and the proposed community development programmes will assist the local residents with much needed infrastructure (sanitation and water quality) and resources. Financial benefits to the Eritrean government are also significant through the Governments interest in the project, royalties and taxes paid by Chalice and its contractors.

Furthermore, investigations have highlighted no material risks that cannot be appropriately managed with the assistance of Government and the project can comply with all Eritrean and Australian environmental guidelines.

Approval and acceptance of the SEIA and SEMP are anticipated following the normal community and government consultation protocols and Chalice will continue to address any outstanding issues raised in the course of seeking grant of the mining license.

2.8 Project Implementation

The EPCM approach is recommended for development of the Project, with horizontal packaging. A small Owner’s team will supervise the EPCM Engineer, and under this arrangement the Owner will pay for all direct costs of plant, equipment, materials, supply, fabrication and installation as approved by the Engineer. The Engineer will not derive any profit from this direct expenditure.

With limited resources available in Eritrea, the preliminary implementation plan is based on the use of an expatriate trade workforce to carry out key activities.

The most likely source of construction materials is summarised in Table 2.8.1, while nearly all construction equipment will need to be imported:

Table 2.8.1 Construction Material Sources

Description	Origin
Concrete materials	Cement: Eritrea or KSA Aggregate, sand: Local sources
Steelwork / Platework / Tankage	South Africa, Kenya or Asia
Piping and Valves	South Africa, Australia or Europe
Mechanical and Electrical equipment and Materials	Various international suppliers
Building Supplies	South Africa, Kenya, Turkey, Middle East or KSA

The project schedule is governed by the 12 months of mine pre-stripping activities required to develop the mine and a further 6 months of mine production ramp-up before sufficient ore can be produced for the process plant to be commissioned. Prior to this, the mining fleet must be procured, shipped to site, assembled and commissioned. In order to mobilise the mining fleet to site, the access road must be upgraded. Other early activities necessary will be procurement and construction of the accommodation village and the erection of temporary mining services facilities to support the pre-stripping activities. The process plant and remaining infrastructure will be constructed during the mine pre-stripping operations.

The overall project is estimated to take 24 months to commissioning from approval of finance, which is expected six months after submission of the final Feasibility Study.

2.9 Operations

A core group of expatriate experts will be recruited for the initial training and management of the operation. A small number of key Eritrean personnel will receive approximately three months pre-operations training followed by at least two months experience on the plant during testing and commissioning.

An expatriate mining team has been allowed for start-up and establishment of procedures for up to two years, after which it is anticipated that Eritrean nationals will be able to take over many of the functions. Local staff will be recruited with the help of local village administrators.

Expatriate staff as well as senior national staff will be domiciled in Asmara and will be flown to site to work a weekly roster. Expatriates will be provided with housing rental assistance and receive two flights per year back to their place of hiring. Day shift staff will reside in the towns and villages along the road between Asmara and site and will be bussed to and from site to work a nine days on five days off shift roster. Continuous shift personnel in both the mine and process plant will have a two weeks on / one week off roster and similar bussing arrangement. Allowance has been made in estimating staff numbers for annual and sick leave needs.

The entire operations workforce will be under the control of a general manager who will be supported by five main departments each with a manager heading the department:

- Mining
- Processing
- Community Relations
- Administration
- Security.

A heavy emphasis will be placed on training to increase the skill levels of national staff, not just in mining and process operations, but also in maintenance and support trades, such as diesel fitting and general trades work. Safe work practices will receive priority in training. Community relations will be of vital importance to the operation and a separate department will manage all aspects of the interactions with the local and wider community.

At least two months stocks of consumables and spares will be held on site, because of the extended logistics chain, with all items having to be imported.

2.10 Operating Cost Estimate

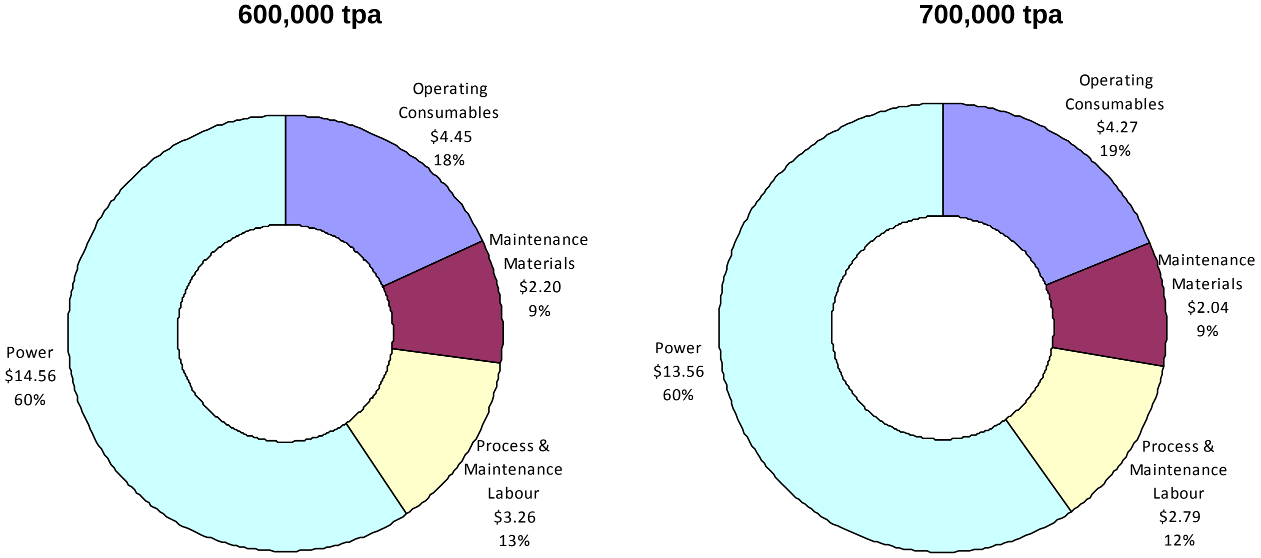
Mining, processing and administration costs as at 2Q10 are summarised in Table 2.10.1 for feed rates of 600,000 and 700,000 tpa and represented in Figure 2.10.1 and are considered to have an accuracy of ±15%.

Table 2.10.1 Operating Cost Estimate Summary (US$, 2Q 2010, ±15%)

Cost Centre	600,000 tpa		700,000 tpa	
	US$/year	US$/t Ore	US$/year	US$/t Ore
Mining Activity Cost [1]	12,632,886	21.05	12,632,886	21.05
Total Mining Operating Cost	**12,632,886**	**21.05**	**12,632,886**	**21.05**
Operating Consumables	2,669,888	4.45	2,987,033	4.27
Maintenance Materials	1,319,650	2.20	1,424,881	2.04
Labour	1,955,235	3.26	1,955,235	2.79
Power	8,735,225	14.56	9,494,480	13.56
Total Processing Cost	**14,679,380**	**24.47**	**15,861,628**	**22.66**
General and Administration Costs	3,496,189	5.83	3,496,189	4.99
Administration Labour	792,192	1.32	792,192	1.13
Total Operating Cost	**31,601,265**	**52.67**	**32,782,895**	**46.83**

[1] Mining activity costs vary by year, as per the mining cost schedule, the figure quoted is the average annual cost over the life of the mine.

Figure 2.10.1 Processing Cost by Cost Centre – (US$/t ore, 2Q 2010, ±15%)



The processing costs exclude:

- head office costs
- escalation and currency fluctuations
- contingency allowance
- land compensation and closure costs
- royalties, licence and Government fees.

2.11 Capital Cost Estimate

The project capital cost estimate is summarised in Table 2.11.1.

The following items are specifically excluded from the capital cost estimate:

- Sunk costs, including pre-feasibility and feasibility costs.
- Exchange rate variations.
- Licence costs and fees
- Site security.

Table 2.11.1 Capital Cost Estimate Summary (US$, 2Q10, ±15%)

Main Area CCC Code	Plant Area CCC Code	Subtotal Cost including Duties / Taxes US$	Project contingency US$	Escalation Cost US$	Project Total US$
000 Construction Indirects	001 Construction Indirects - Contractors P & Gs	5,867,808	704,105	103,985	6,675,898
	010 Site Construction Indirects General	663,462	122,240	57,836	843,537
	020 Site Construction Facilities	98,080	24,430	-	122,510
	030 Site Construction Facilities Other	429,956	51,228	28,781	509,964
	040 Construction Operations	353,578	88,078	-	441,655
	050 Construction Accommodation	3,090,317	430,954	109,501	3,630,771
000 Construction Indirects Total		**10,503,200**	**1,421,034**	**300,102**	**12,224,336**
100 Treatment Plant	101 Treatment Plant - General	929,288	232,322	92,929	1,254,539
	120 Feed Preparation	2,230,284	258,542	198,555	2,687,380
	130 Milling	7,390,839	782,304	651,702	8,824,846
	140 Screening / Tailings	2,540,251	284,953	225,419	3,050,623
	160 Leaching	2,349,021	253,199	207,655	2,809,875
	170 Desorption	1,212,154	151,548	108,769	1,472,471
	180 Refining	1,678,470	181,898	148,379	2,008,748
100 Treatment Plant Total		**18,330,308**	**2,144,766**	**1,633,408**	**22,108,481**
200 Reagents and Plant Services	201 Reagents and Plant Services - General	4,320	1,080	432	5,832
	210 Reagents	873,081	103,612	77,929	1,054,622
	230 Water Services	1,755,736	218,163	157,571	2,131,470
	250 Air Services	766,368	87,657	68,094	922,119
	260 Fuels	588,828	87,011	43,684	719,523
	270 Electrical Services	925,834	101,480	81,922	1,109,237
200 Reagents and Plant Services Total		**4,914,168**	**599,003**	**429,633**	**5,942,804**
300 Infrastructure	301 Infrastructure - General	3,057,497	545,069	177,537	3,780,103
	310 Environmental	99,575	14,292	8,475	122,343
	320 Utilities and Services	1,472,901	165,715	130,754	1,769,371
	330 Power Supply	6,011,569	658,744	531,693	7,202,005
	340 Tailings Dam	3,265,327	700,304	313,298	4,278,929
	350 Buildings - Admin & Security	1,896,040	195,626	166,765	2,258,431
	360 Buildings - Plant	154,644	15,659	13,586	183,889
	370 Buildings - Services	750,478	170,145	68,314	988,937
	380 Village	6,188,350	762,758	553,858	7,504,966
300 Infrastructure Total		**22,896,380**	**3,228,313**	**1,964,280**	**28,088,973**
400 Mining	401 Mining-Equipment	14,063,970	699,700	-	14,763,670
	420 Mine Establishment	11,321,000	-	452,840	11,773,840
	430 Mining Pre-Production	4,699,000	234,950	197,358	5,131,308
	450 Mining Facilities	16,973	1,697	1,494	20,163
400 Mining Total		**30,100,943**	**936,347**	**651,692**	**31,688,981**
500 Management Costs	501 EPCM - Home Office	4,762,235	476,223	419,077	5,657,535
	520 EPCM - Site	2,379,010	237,901	209,353	2,826,264
500 Management Costs Total		**7,141,244**	**714,124**	**628,430**	**8,483,798**
600 Owners Project Costs	601 Owners Costs - General	3,479,000	-	276,800	3,755,800
	620 Plant and Admin Pre-Production	4,033,097	310,439	346,281	4,689,818
	630 Admin Pre-Production Other	319,000	-	25,520	344,520
	640 Spare Parts	742,177	73,876	65,011	881,065
	650 Fees / Taxes / Duties	-	-	-	-
	690 Community Development	175,000	-	7,000	182,000
600 Owners Project Costs Total		**8,748,274**	**384,316**	**720,612**	**9,853,202**
700 Owners Operations Costs	710 Working Capital	3,330,065	-	266,405	3,596,470
700 Owners Operations Costs Total		**3,330,065**	**-**	**266,405**	**3,596,470**
Grand Total		**105,964,582**	**9,427,903**	**6,594,561**	**121,987,046**

2.12 Financial Analysis

A simple cash flow model has been used. The results are summarised in Table 2.12.1.

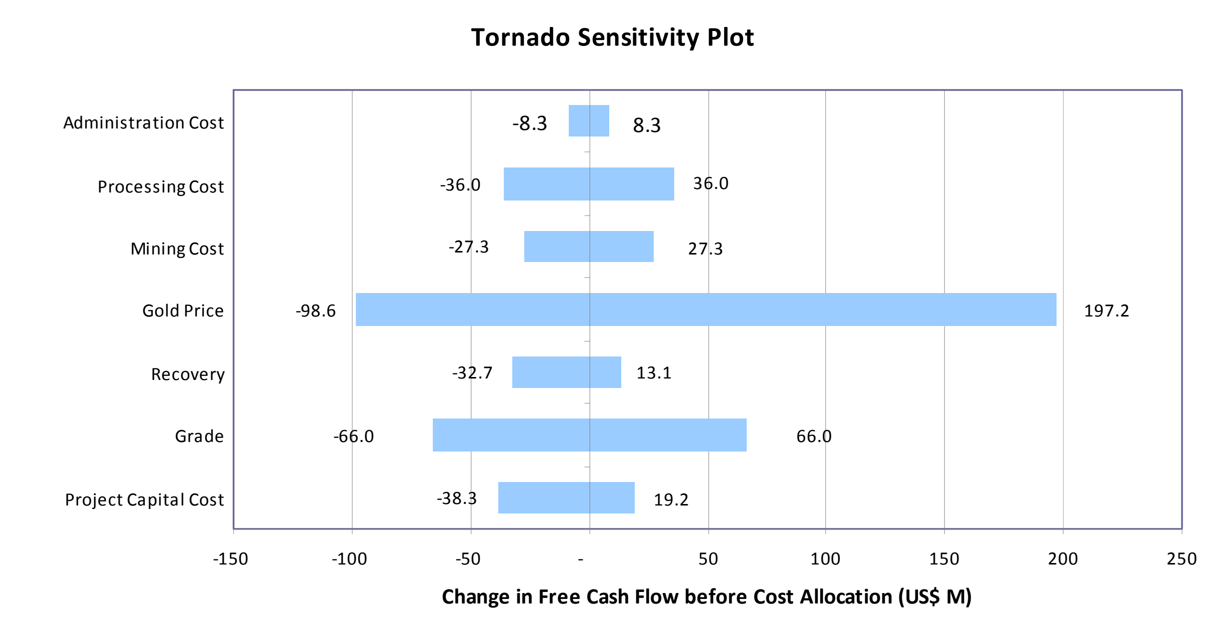
Table 2.12.1 Base Case Financial Analysis

Total Mined	tonnes/million	52.94			
Ore Milled	tonnes/million	4.63			
Strip Ratio		10.4			
Grade Gold g/t	g/tonne	5.10			
Contained Gold oz	oz	758,900			
Recovery Gold%	%	96.31			
Recovered Gold oz	oz	730,780			
Gold Price $/oz	$/oz	900			
Revenue from Gold Sales	US$ million	**657**			

		US$ Million	$/oz sold	$/t Processed	$/t Mined
Mining Cost		94.7	129.8	20.5	1.92
Process Cost		114.8	157.2	24.8	2.33
Smelting and Refining Charges		2.88	4.00	0.62	0.06
G&A Cost		34.09	46.69	7.36	0.69
Cash Operating Costs		**246.5**	**337.7**	**53.23**	**5.00**
Other (Including Royalties)		29.7	40.7	6.41	0.60
Total Cash Costs		**276.2**	**378.4**	**59.64**	**5.60**
Depreciation and Amortisation		131.3			
Total Production Costs		**407.5**			
Earning Before Interest Taxes **(EBIT)**	US$ million	249.5			
Income Tax Expense	US$ million	94.8			
Net Profit After Tax	**US$ million**	**154.7**			
NPV (5.0%)	US$ million	**99**			
IRR	%	**22**			

The results are highly sensitive to the gold price, as shown in Figure 2.12.2. The analysis is based on a gold price significantly below the current spot price, providing significant upside to the project. If deemed appropriate, mechanisms could be implements to secure a stable gold price.

Figure 2.12.2 Project Sensitivity



2.13 Risk Analysis

A risk assessment workshop was held addressing project risks identified during earlier studies as well as risks associated with the operations which became apparent with more detailed design. Risks were assessed from the points of view of health and safety, environment and financial risks. The review endeavoured to identify major risks that could place the project in jeopardy or cause significant personal injury or environmental damage, to allow these to be addressed early in the design The review procedure categorised risks according to the severity of the consequences of an event, the likelihood of it occurring and the control measures in place.

Only one risk was identified as Extreme: a road traffic accident occurring outside the controlled project area. This risk is outside the control of Chalice; however, Chalice is upgrading significant portions of the road and is seeking the Eritrean Government's cooperation to assist in improving the standard of roads which will be used by the project.

A number of other risks were identified and rated as 'High', despite the current control measures in place. Some of the risks are beyond the control of the project, but others can be mitigated by on-going emphasis on safety awareness and training. These aspects will be emphasised in the training programs to be conducted on site.