ASX ANNOUNCEMENT
National Instrument 43-101 Technical Report on the Koka Gold Deposit

Chalice Gold Mines Limited ABN 47 116 648 956 — 8 October 2010

Chalice Gold Mines Limited (ASX: **CHN**) would like to advise that in support of the Company's application to list on the Toronto Stock Exchange it has completed a National Instrument 43-101 Technical Report on the Koka Gold Deposit. This report, prepared by AMC Consultants Pty Ltd, is appended to this announcement.

TIM GOYDER
Executive Chairman

Attachment: 43-101 Technical Report on the Koka Gold Deposit

About the Zara Gold Project

The Zara Project comprises four Exploration Licenses and two Prospecting Licenses covering an area of 615km^2 situated in northern Eritrea, approximately 160km northwest of Asmara city. Chalice holds a 100% interest in the project subject to Eritrean government participation rights.

The Koka Gold Deposit within the project contains a Probable Reserve of 4.6 million tonnes of ore grading 5.1 grams of gold per tonne and containing 760,000 ozs of gold. This is contained within an Indicated Resource of 5.0 million tonnes grading 5.3 grams of gold per tonne containing 840,000 ozs of gold.

INVESTMENT HIGHLIGHTS

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

Feasibility Study completed and permitting commenced:

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

Drilling at near mine Konate Prospect ongoing

Large unexplored ground position in the Arabian Nubian Shield

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Competent Persons' Statement

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.

The Canadian National Instrument 43-101 report appended to the ASX release was compiled by Mr Dean Carville who is a Member of the Australasian Institute of Mining and Metallurgy. Mr. Carville is a full time employee of AMC and has sufficient experience 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 Carville also meets the requirements of a Qualified Person as defined in NI 43-101, and is independent of Chalice. Mr. Carville consents to the inclusion of this report in the form and context in which it appears.

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TECHNICAL REPORT ON THE KOKA GOLD PROJECT

ERITREA

CHALICE GOLD MINES LIMITED

AMC PROJECT 210018

27 July 2010

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QUALITY CONTROL

The signing of this statement confirms this report has been prepared and checked in accordance with the AMC Peer Review Process. AMC's Peer Review Policy can be viewed at www.amcconsultants.com.au.

Project Manager

Dean Carville	[signature]	27 July 2010
	Signed	Date

Peer Reviewer

Pat Stephenson	[signature]	27 July 2010
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IMPORTANT INFORMATION ABOUT THIS REPORT

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1 SUMMARY

The Koka gold deposit forms part of the Zara Project in the State of Eritrea, East Africa. The project is controlled by Chalice Gold Mines Limited ("Chalice"), a company listed on the Australian Securities Exchange ("ASX"). Chalice is preparing to list on the Toronto Stock Exchange and has requested that AMC Consultants Pty Ltd ("AMC") prepares a technical report (Technical Report) that is compliant with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101").

The Technical Report has been prepared by Dean Carville and David Lee of AMC, and David Gordon of Lycopodium Minerals Pty Ltd ("Lycopodium") all of whom meet the requirements of a Qualified Person as defined in NI 43-101, and are independent of Chalice. Dean Carville and David Lee have made recent visits to the Koka site.

The Koka gold deposit contains an Indicated Mineral Resource as listed in Table 1.1. The Indicated Resource category referred to in the JORC Code[1] is directly comparable to the Indicated Resource category defined in the CIM Definition Standards[2] and referred to in NI 43-101. NI 43-101 allows the use of Mineral Resource categories of the JORC Code in a Technical Report if reconciliation with CIM Definition Standards is disclosed.

Table 1.1 Koka Gold Deposit Mineral Resource 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 Mineral Resource estimate was completed by AMC in June 2010. A scoping study was completed by Lycopodium in October 2009 based on a May 2009 Mineral Resource estimate. A feasibility study was completed by Lycopodium in July 2010 with the Mineral Resource and Ore Reserve estimates and geotechnical and mining sections of the study completed by AMC. The scoping study concluded that open pit operation was financially more attractive and a lower risk option than underground mining and only open pit mining was investigated for the feasibility study. The feasibility study was based on the June 2010 Indicated Resource and Ore Reserves[3] that were estimated and reported to the ASX on 4 June 2010. The Ore Reserve estimate at 1 June 2010 is listed in Table 1.2.

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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).

2 CIM Definition Standards on Mineral Resources and Mineral Reserves Prepared by the CIM Standing Committee on Reserve Definitions. 2005.

3 The term Ore Reserves as defined in the JORC Code is equivalent to the term Mineral Reserves as applied in the CIM Standards. NI 43-101 allows the use of JORC Code terms if reconciliation with the CIM Definition Standards is disclosed.

Table 1.2 Koka Gold Deposit Ore Reserve

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

The Zara Project is situated approximately 165 km northwest of Asmara, the capital of the State of Eritrea in northeast Africa. Exploration rights are held under four exploration licenses and two prospecting licenses covering a total of 606 km^2. The licences are currently in good standing and are subject to annual renewal.

On application for a mining license, the Eritrean government is entitled to a 10% free carried interest in the project and, in addition, the government has the right to purchase a further 20% equity participation interest. The pricing of the additional equity participation is by agreement.

There had been no previous exploration tenure over the Koka gold deposit nor any recorded exploration activity by a previous owner or operator. There are no historic Mineral Resource estimates for the Koka gold deposit.

The Zara Project is situated within rocks of the Nakfa and Adobha Abiy neo-Proterozoic metamorphic terrains and to a flexure in the Elababu Shear Zone. The eastern and central parts of the Zara exploration licenses are underlain by meta-volcanic and meta-sedimentary rocks metamorphosed to greenschist facies, together with post-tectonic granitoids. The western part is underlain by predominantly siliciclastic rocks, together with minor meta-chemical sedimentary rocks, basalt and syn-tectonic granitoids.

The Koka mineralised zone has a total strike length of more than 650m 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). The rocks have been subject to lower greenschist facies metamorphism and at least two stages of deformation.

The main zone of mineralisation is hosted within competent microgranite on the western margin of the Koka shear zone. This unit has been strongly silicified and brecciated and is cut by a stockwork of quartz veins. There are five alteration types associated with gold mineralisation. The main hydrothermal alteration can be recognised as one or multiple phases of carbonate, sericite, silica, chlorite or pyrite alteration.

The Koka gold deposit represents a greenschist facies, lode gold deposit in which most of the gold is hosted by quartz-sulphide veins in altered microgranite, with minor gold associated with the altered wall rocks. Gold occurs as free gold and in pyrite in quartz veins.

The first drilling at the Koka discovery commenced in August 2005 and resulted in the confirmation of gold mineralisation beneath the artisanal workings. The drilling consisted of four diamond drillholes about 150m apart, returning significant gold mineralisation in all four holes. A systematic diamond drilling programme covering the known strike of artisanal workings commenced in January 2006 and has been completed in a number of campaigns as listed in Table 1.3.

Table 1.3 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

All drilling has been diamond drilling. A total of 127 drillholes were completed by 2008 and 137 drillholes totalling 20,839m were used for the 2010 Mineral Resource estimate. Drillhole spacing over the strike length of the deposit is mainly 40m x 20m with the central core of the deposit drilled at about 20m x 20m spacing. Eight drillholes for 1,078m completed for metallurgical testwork have no associated assays. Drilling has been completed over a strike length of about 610m and to an average depth of about 165m below surface.

Drillhole collars were surveyed using differential global positioning system and downhole surveys completed about every 30m. Core recovery ranges from 88.5% to 99.7%, averaging 95%. Core recovery is poorer from moderate to highly weathered units and highly fractured and brecciated zones. There is no apparent correlation between gold grade and reduced core recovery.

Each drillhole was logged using standardised logging format and geological codes and core was geotechnically logged. Drill core was sampled over the full length of intersections of the altered microgranite that hosts the gold mineralisation. Competent core was cut with an electric diamond core saw with a sample length mainly of 1m.

The drillholes are distributed over the length and depth extent of the mineralised zone and drilling has been executed following acceptable industry practice supported by assay quality control protocols. The drillhole spacing and sampling provides good representation of the mineralised zone over its strike and depth extent.

Sample preparation of the core samples was conducted in Asmara by Eritrean company Africa Horn Laboratory. The sample preparation laboratory is a joint venture with Genalysis Laboratory Services Pty Ltd ("Genalysis") which is a commercial mineral industry laboratory accredited with the National Association of Testing Laboratories, Australia.

Most gold assaying has been completed using a lead collection 50g fire assay method with an atomic absorption spectroscopy finish. A blank sample is introduced every 20 to 25 routine samples. Four certified reference materials ("CRM") are submitted with all sample batches with two standards submitted at the start and end of each batch. The CRMs are disguised from the laboratory. The CRMs contain low, medium and high gold grades to reflect the grade distribution of the deposit. Five percent of the returned coarse reject samples are routinely submitted to the umpire laboratory to test the analytical precision of the principal laboratory. Five percent of all returned pulps are submitted to the principal laboratory (2.5%) and umpire laboratory (2.5%) to monitor the precision of the principal laboratory.

For drilling prior to 2009, the CRMs show good replication of the certified value. Splits of sample pulps submitted to an umpire laboratory returned results with very good correlation to the original assay. Laboratory pulp duplicates displayed marginally acceptable precision.

Assay quality control data were reviewed by an independent consultant that concluded that CRMs performed very well and sample preparation and laboratory pulp check assays had an overall negative bias with repeats tending to be lower than their original assays. The bias is weighted by the very high grade range and results for these repeats in general correlate very well. The laboratory repeats correlate very well, with no appreciable bias observed.

AMC has reviewed the evaluation of assay quality control data and has concluded that the procedures follow common industry practice and support the data to be used for Mineral Resource estimation.

There are no known significant mineral occurrences in the area surrounding the Zara exploration licenses and Zara North and Zara South prospecting licenses.

A metallurgical testwork programme initiated and supervised by Lycopodium was conducted on primary mineralised samples at the laboratory of Australian Metallurgical and Mineral Testing Consultants ("AMMTEC") in Perth, Western Australia in 2009. The samples came from intercepts in seven diamond drill holes completed for this purpose. An earlier preliminary testwork programme was completed on about 30 kg of drillcore sample in 2007.

The detailed testwork programme was undertaken for the feasibility study with the following objectives:

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

Outcomes of the metallurgical and comminution testwork are:

- The mineralised rock is moderately competent and abrasive with above average comminution energy requirements.
- The mineralised rock is 'free-milling' (non-refractory) 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 primary ores.
- 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 g/t Au to 14.51 g/t Au.
- Lime consumption was low at 0.33 kg/t (60% available CaO basis).
- Cyanide consumption was low at 0.29 kg of sodium cyanide per tonne.

The overall process flowsheet has been based on the results 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 semi-autogenous grind 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.
- Seven stage carbon in leach ("CIL") circuit.
- Zadra elution circuit for recovery of gold on carbon.
- Three-stage counter current decantation circuit to treat CIL discharge slurry.

The Mineral Resource estimate was developed by AMC based on interpretation of the host microgranite and within that interpretation of overlapping gold and sulphide-bearing domains reflecting the association of gold with sulphide mineralisation. A probability model was used to assist with the interpretation of mineralisation continuity. 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.

Assays within the domains were selected and composited to 2m. A topcut of 200 g/t Au was applied to the composites within mineralisation domains.

Gold grade was estimated using ordinary kriging with parameters based on a study of variography. The block model parent cell dimensions were 10m in easting and northing directions and 5m in RL. 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.

A dry bulk density of 2.74 t/m^3 was applied to the model based on averaging of bulk density measurements from drillcore.

The Mineral Resource estimate has been classified as Indicated Resource in accordance with the JORC Code.

AMC considers that the Mineral Resource estimate has been prepared using common industry practices and has been appropriately classified as Indicated Resource in compliance with the JORC Code. AMC is satisfied that the estimate is of a suitable standard for reporting under NI 43-101.

The Mineral Resource model was used as the basis for the Ore Reserve estimation. Only Indicated Resources have been used for the ore reserve estimate. The impact of mining on the anticipated ore tonnage and grade was analysed by considering the impact of reblocking the resource model at the selective mining unit size suitable for the equipment being considered. A minimum block size of 5m in easting and northing directions and 2.5m in RL was used for a 120t type excavator. The result of the dilution and ore loss process was to add 15% dilution material and 5% ore loss.

The slope angles used in the Ore Reserve estimation were based on work by AMC that used core logging information collected from exploration and geotechnical diamond drillholes as well as material property data collected from laboratory tests. The overall slope angles vary between 45º and 48º.

The pit limits for the open pit were selected through analysis using the Whittle Four-X implementation of the Lerchs Grossman algorithm.

The following ore related parameters were used in the optimisation:

- Process and administration cost of US$33.49/t processed assuming a 0.5 Mtpa processing rate.
- A metallurgical recovery of 96.2%.
- A gold price of US$900/oz.
- A government royalty of 5% of revenue.
- The treatment plant breakeven cut-off grade was estimated as 1.26 g/t Au.

The processing and administration cost was developed by Lycopodium as part of the scoping study completed in 2009.

The Ore Reserve is the contents of a pit design, above a cut-off grade of 1.26 g/t Au that was developed based on the optimisation results.

All the Indicated Mineral Resources intersected by the open pit mine design were classed as Probable Ore Reserves after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Koka project. Ore Reserves are included in the Mineral Resource estimate.

Lycopodium completed a feasibility study for Chalice in July 2010 using the June 2010 Ore Reserve estimate. The project schedule is governed by the twelve months of mine pre-stripping activities required to develop the mine and a further six months of mine production ramp-up before sufficient ore can be produced for the process plant to be commissioned. The process plant and remaining infrastructure will be constructed during the mine pre-stripping operations.

The project capital cost has been estimated for the feasibility study at an accuracy of ±15% in first half 2010 costs. Mining, processing and administration costs estimated using costs are considered to have an accuracy of ±15%.

The overall economic viability of the Koka Gold Project has been evaluated using simple cash flow techniques. Results of the financial analysis for the feasibility study base case are presented in Table 1.4. The base case is evaluated at a gold price of US$900 per ounce.

Table 1.4 Base Case Financial Analysis

Item	Unit	Value
Total Mined	Mt	52.94
Ore Milled	Mt	4.63
Strip Ratio	-	10.4
Gold Grade	g/t Au	5.10
Contained Gold	ounces	758,900
Recovery Gold	%	96.3
Recovered Gold	ounces	730,780
Gold Price	US$ per ounce	900
Revenue from Gold Sales	**US$M**	**657**

Item	US$M	US$/oz Sold	US$/t Processed	US$/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)	249.5	-	-	-
Income Tax Expense	94.8			
Net Profit After Tax	**154.7**			
NPV (5.0%)	**99.0**	**-**	**-**	**-**
IRR	**22%**	**-**	**-**	**-**

2 INTRODUCTION

The Koka gold deposit forms part of the Zara Project in the State of Eritrea, East Africa. The project is controlled by Chalice Gold Mines Limited ("Chalice"), a company listed on the Australian Securities Exchange ("ASX"). Chalice is preparing to list on the Toronto Stock Exchange and has requested that AMC Consultants Pty Ltd ("AMC") prepares a technical report (Technical Report) that is compliant with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101").

The Koka exploration licenses are held and operated by Chalice Gold Mines (Eritrea) Limited formerly known as Dragon Mining (Eritrea) Ltd.

Chalice acquired its interest in the Zara Project by way of:

- a merger in August 2009 with ASX-listed Sub-Sahara Resources NL ("Sub-Sahara") which held a 69% interest
- the purchase of all the shares in the 11% interest held by the unlisted company Africa Wide Resources Ltd ("AWR")
- the acquisition in June 2010 of all the shares of Dragon Mining (Eritrea) Ltd, a wholly-owned subsidiary of Dragon Mining Limited ("Dragon") which held a 20% interest.

The Mineral Resource estimate for the Koka gold deposit at 1 June 2010 is listed in Table 2.1.

Table 2.1 Koka Gold Deposit Mineral Resource 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 Mineral Resource estimate is reported and classified as Indicated Resource under the JORC Code[4] including a Competent Person's compliance statement. The Indicated Resource category referred to in the JORC Code is directly comparable to the Indicated Resource category defined in the CIM Definition Standards[5] and referred to in NI 43-101. NI 43-101 allows the use of Mineral Resource categories of the JORC Code in a Technical Report if reconciliation with CIM Definition Standards is disclosed.

A previous Mineral Resource estimate (Coffey, 2009) was reported by Sub-Sahara in May 2009 (Sub-Sahara, 2009). The revised estimate has been completed following further drilling.

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4 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).

5 CIM Definition Standards on Mineral Resources and Mineral Reserves Prepared by the CIM Standing Committee on Reserve Definitions. 2005.

A scoping study was completed by Lycopodium Minerals Pty Ltd ("Lycopodium") in October 2009 based on the May 2009 Mineral Resource estimate. A feasibility study (Lycopodium, 2010) was completed by Lycopodium in July 2010 with the Mineral Resource and Ore Reserve estimates and geotechnical and mining sections of the study completed by AMC. The scoping study concluded that open pit operation was financially more attractive and a lower risk option than underground mining and only open pit mining was investigated for the feasibility study. The feasibility study was based on the June 2010 Indicated Resource and Ore Reserves[6] that were estimated and reported to the ASX on 4 June 2010. The Ore Reserve estimate at 1 June 2010 is listed in Table 2.2.

Table 2.2 Koka Gold Deposit Ore Reserve

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

This Technical Report incorporates the results of the Mineral Resource and Ore Reserve estimates and the feasibility study.

AMC is a firm of independent geological, mining geotechnical, mine engineering and mine management consultants offering expertise and professional advice to the exploration, mining and mining finance industries from our offices in Melbourne, Perth, Brisbane, Adelaide, the United Kingdom and Vancouver, Canada.

AMC's principal capabilities address two core elements - evaluation and operations - and within these, AMC provides consulting services in the following:

- Geology
- Mining
- Geomechanics
- Feasibility studies
- Technical Specialist reports
- Mine optimisation
- Valuations.

Lycopodium is a firm of independent consulting engineers and metallurgists offering expertise in mineral and metal processing, metallurgical testwork, feasibility study management and plant design and construction. The feasibility study was managed from Lycopodium's principal office in Perth, Western Australia.

This Technical Report has been prepared in compliance with NI 43-101, Form 43-101F1 and Companion Policy 43-101CP. AMC contributed to a scoping study on the Koka gold deposit in October 2009. As part of that study, in September 2009 AMC Principal Geologist, Dean Carville and Principal Mining Engineer, David Lee, visited the Koka site,

[6] The term Ore Reserves as defined in the JORC Code is equivalent to the term Mineral Reserves as applied in the CIM Standards. NI 43-101 allows the use of JORC Code terms if reconciliation with the CIM Definition Standards is disclosed.

the offices of Sub-Sahara in Asmara and the sample preparation facility in Asmara. AMC considers that the site inspection satisfies the requirement of current personal inspection required by NI 43-101 and that there has been no material change to the scientific and technical information about the property since the personal inspection.

The site inspection involved a review of diamond drill core, geological logging, surface exposure, and the sample preparation procedures. Neither AMC nor the authors have performed any independent exploration work, drilled any holes or carried out any sampling and assaying.

Dean Carville, David Lee and David Gordon have been involved in the preparation of the Technical Report but neither they, AMC nor Lycopodium have any shareholdings or other interest in Chalice or in any of the assets reviewed and AMC and Lycopodium have no pecuniary interest, association or employment relationship with Chalice other than the payment of fees according to their normal per diem rates and out of pocket expenses for consulting services including preparation of this report. Those fees are not contingent on the outcome of any transaction subject to this report.

Chalice has been provided with drafts of the Technical Report to enable correction of any factual errors and notation of any material omissions.

3 RELIANCE ON OTHER EXPERTS

The description of the status of the tenements in Section 4 of the Technical Report is based on the due diligence report of Fessahaie Habte (2009). Mr Habte is an eminent Asmara-based attorney and legal counsel who previously sat on the High Court of Ethiopia and the Supreme Court and High Court of Eritrea. AMC has seen the document from Mr Habte and relied on his opinion on the status of the tenements.

Information on taxation required for the feasibility study was provided to Lycopodium by Chalice. The Qualified Persons have not verified the taxation information.

The environmental considerations summarised in Section 19.1 are based on information provided by consultant Knight Piésold which was involved in base line environmental studies and preparation of the draft Social and Environmental Impact Assessment and draft Social and Environmental Management Plan.

4 PROPERTY DESCRIPTION AND LOCATION

The Koka gold deposit forms part of the Zara Project in the State of Eritrea, East Africa.

The Zara Project is situated approximately 165 kilometres northwest of Asmara, the capital of the State of Eritrea in northeast Africa (Figure 4.1). The project is located in a range of mountains running parallel to the Zara River which drains north into Sudan.

Figure 4.1 Zara Project Location

The original prospecting license was issued by the Department of Mines of the Eritrean Ministry of Energy and Mines on 2 October 1998 and converted to four exploration licenses (the Zara Main exploration licenses) for a period of five years on 20 October 2000 and reduced by 240 km^2 to a total of 196 km^2. In November 2008 the joint venture reduced its holding in the Zara Main exploration licenses to 147 km^2 (Table 4.1 and Figure 4.2).

Table 4.1 Tenement Details

Tenement Name	Tenement Type	Area (km^2)	Tenement Holder
Zara 1	Exploration License	21	Chalice Gold Mines (Eritrea) Limited
Zara 2	Exploration License	28	Chalice Gold Mines (Eritrea) Limited
Zara 3	Exploration License	49	Chalice Gold Mines (Eritrea) Limited
Zara 4	Exploration License	49	Chalice Gold Mines (Eritrea) Limited
Zara North	Prospecting License	113	Sub-Sahara Resources (Eritrea) Limited
Zara South	Prospecting License	346	Sub-Sahara Resources (Eritrea) Limited

The Mineral Resource is located on the Zara 3 exploration license as shown in Figure 4.2. The property boundaries are specified in the title documents in Universal Trans Mercator ("UTM") coordinates.

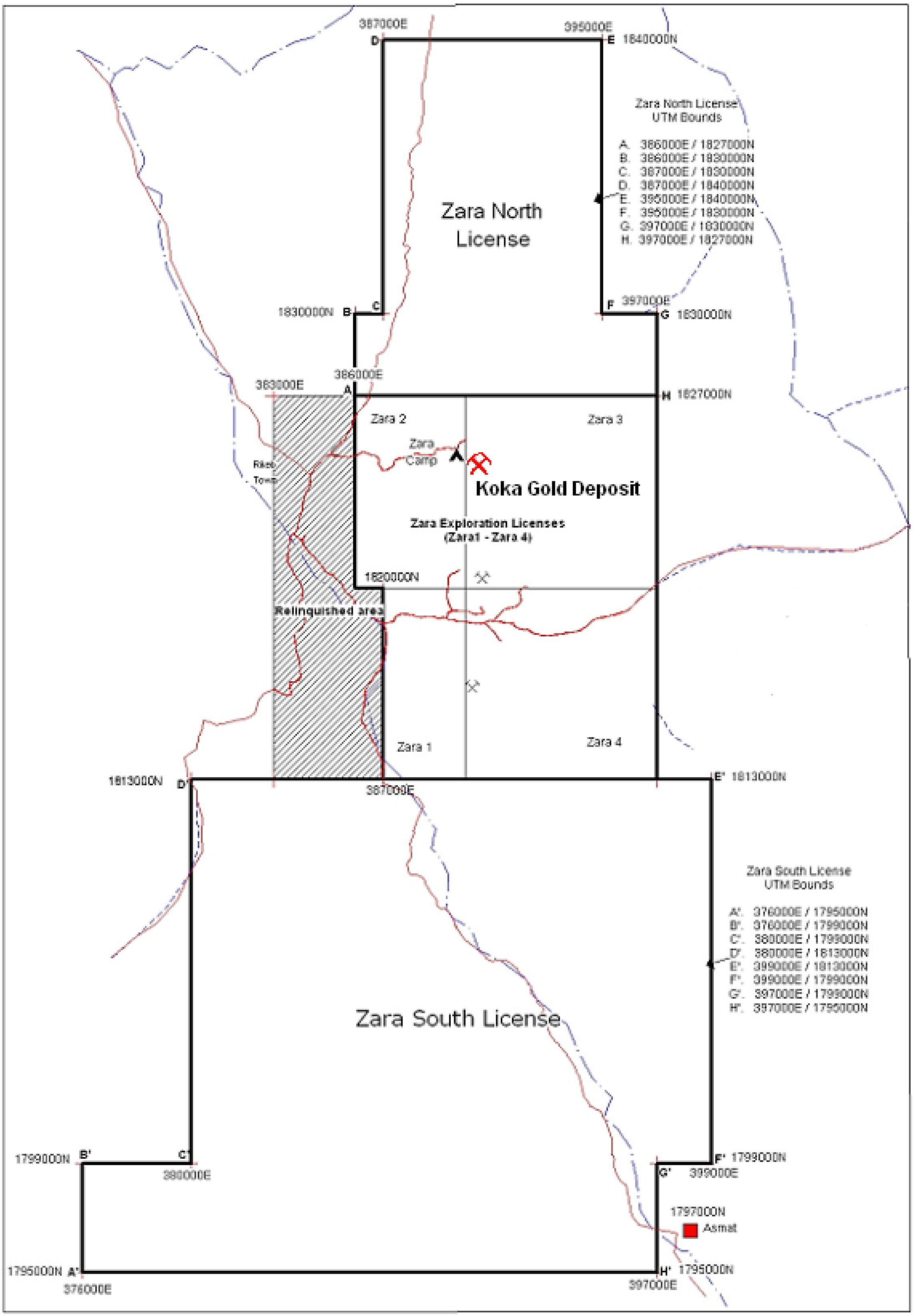
A 12 month extension to the exploration licenses was granted expiring on 25 May 2011 subject to completion of a feasibility study, Social, Environmental Impact Assessment (“SEIA”) and a Social, Environment Management Plan (“SEMP”) for the Koka gold deposit.

Two additional prospecting licenses have been granted over the Zara North (113 km^2) and Zara South (346 km^2) areas (Figure 4.2). The prospecting licenses were granted for a period of one year from approval of work programme with the documents dated 19 August 2009.

Paragraph 35 of Eritrean Mining Proclamation No 68/1995 states that a 5% royalty is payable on revenue from production of precious metals although a lesser rate of royalty may be agreed in order to encourage mining investments in areas given development priority.

Survey data at the Koka gold deposit is recorded in both UTM and local grid coordinates. The coordinate system is UTM North Zone 37 and is based on the WGS 84 ellipsoid. The local grid is rotated 12.5º east from the UTM grid.

Figure 4.2 Exploration and Prospecting License Boundaries



On application for a mining license, the Eritrean government is entitled to a 10% free carried interest in the project and, in addition, the government has the right to purchase a further 20% equity participation interest. The pricing of the additional equity participation is by agreement which has to specify the timing, financing, resulting rights and obligations and other details of the participation.

Chalice has obtained independent legal advice in relation to a number of issues regarding the status of the tenements (Fessahaie Habte, 2009). The letter concludes, among other things, that:

- the Minister of Energy and Mines is authorised to renew an exploration license twice for additional terms of one year
- under certain circumstances the Licensing Authority may allow an extension of renewal periods
- the Licensing Authority may further allow an extension of renewal periods where the licensee documents the necessity for additional advanced exploration activity, or provides information on other circumstances which justify an extension of the duration of the license
- the licensee shall have the right to be granted a mining license in the event that he determines a mineral deposit within the license area which may be mined on an economically viable basis provided that he has:
    - fulfilled all obligations under the exploration license
    - meets all requirements in the application for such a mining license
    - is not in breach of any provisions of the Mining Proclamation, Regulation or directives issued under the Proclamation which would constitute grounds for suspension or revocation of the exploration license
- a Licensee has an exclusive right to explore for all minerals within the area specified in the license apart from construction material, mineral water and geothermal deposits
- the Mining Proclamation enables the Government to acquire without cost to it a participation interest of up to 10% of any mining investment
- additional equity participation of up to a total of 30% including the 10% cost free participation interest, may also be provided to the Government
- the additional equity participation may be provided to the Government by agreement, which has to specify the percentage, timing, financing, resulting rights and obligations and other details of such participation
- the price of acquiring up to 30% of equity participation is determined by agreement.

Subsequent to Fessahaie Habte (2009), the exploration licences were renewed for a further year to 25 May 2011.

The current land tenure is in the form of exploration and prospecting licenses. The licenses are granted with the following environmental obligations:

- During its prospecting activities the Licensee shall comply with all environmental laws and regulations in force in Eritrea.

- The Licensee shall not unduly disturb or interfere with the living conditions of the indigenous population lawfully settled within the License Area and surroundings and shall respect their customs.
- The Licensee shall, upon surrender of its Prospecting License or termination of this Agreement, take commensurate measures to safeguard any pits and such other works so that the health life and property of persons is not endangered.

Mining tenements granted subsequent to the exploration and prospecting licenses may carry different environmental conditions.

The exploration tenements covering the Koka gold deposit and adjacent area cover the proposed open pit, waste rock dumps, plant site, tailings storage facility, and camp area. AMC considers that a mining tenement granted subject to the exploration tenements would cover at least a similar area and would be adequate for these purposes. A proposed bore field for water supply would lie outside the current exploration tenements and would be subject to further licences.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

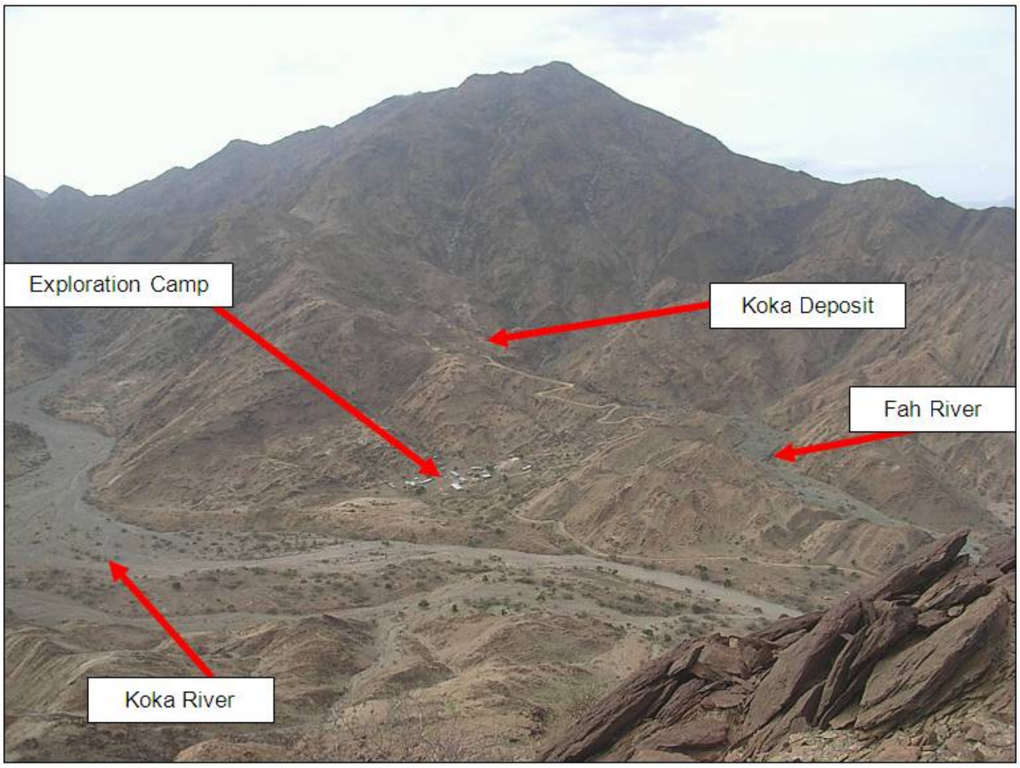
The project site is located about 165 km north of the capital city Asmara. Eritrea is divided into regions (Zobas) and districts (sub-Zobas) and the project is located in sub-Zoba Sela within Zoba Anseba. According to the agro-ecological zoning of Eritrea the area is situated in the arid lowland zone.

The main access to the study area is through the asphalt road from Asmara to Keren and then through the road through Hamelmalo, Melebso and Asmat to Rikeb. There is no public transport system to the area except for one private bus that makes the trip once a week between Keren and Rikeb.

The climate is subtropical with distinct dry and rainy seasons with the wet season spanning May through to September (dominantly July to August). In general the rainy season in the project area is of very short duration and erratic and drought is a common occurrence. The mean annual precipitation in the arid lowland zone ranges from 200 mm to 500 mm, the mean annual temperature ranges from 21ºC to 29ºC and the potential evapo-transpiration ranges from 1,800 mm to 2,000 mm. Based on site data, the total rainfall in 2008 was 117.4 mm and temperatures at site in 2008 ranged from a low of 17ºC in March to a high of 45ºC in May.

The topography of the project area is dominated by steep-sloped mountains, ridges and valleys, with an elevation ranging between 500m above sea level ("ASL") along river beds and 2000m ASL at the mountain crests. Most of the flat areas are located along the river banks or along the river beds (Figure 5.1).

Figure 5.1 Photograph Looking East Across the Koka River Valley to the Koka Gold Deposit



Due to the steep slopes, very shallow soil and low rainfall, the project area is not suitable for farming for most field crops. The overall vegetation cover of the study area is very poor, the area being almost bare due to prolonged drought. The mountains and ridges in the area are covered with seasonal grass and very few scattered shrubs. Shallow Leptosols, Lepto-rock association and bare rock dominate the steep slopes, mountains and hills of the project area. These groups of soils are not suitable for farming and can only be used for grazing and browsing. Fluvisols, situated along the rivers and streams, are relatively fertile and suitable for farming, although areas cultivated with irrigation have limited production. Land use mapping has established that shrub land (47%), grassland (31%) and bare land (12%) are the predominant land cover types.

There are no perennial rivers or streams, however major seasonal rivers like Zara, Fah and Koka are found within the project area. These rivers and streams are dry for more than nine months of the year and flow of water occurs for a very short period of time during the rainy season. The rivers are the main water source for domestic consumption, animal drinking and irrigation purposes. There are no natural lakes or man made dams within the area.

According to the local administration reports, sub-Zoba Sela is one of the most sparsely populated zones of the country. The total population of the sub-Zoba is estimated at 11,598 with a very low population density of 1.2 persons per square kilometre. The total number of households in the sub-Zoba is 2,698 translating to an estimated average household size of 4.2, which is lower than the national average of 4.8. About 506 households live in the socio-economic area studied for the feasibility study and the population of the closest village of Rikeb is 210 households. The main ethnic groups in the project area are Tigre and Hidareb.

Public services and infrastructure in the project area are limited. An elementary school and adult literacy programme operate in Rikeb but enrolments are relatively low. A health centre in Rikeb is staffed by one public health practitioner and two health assistants, and a larger number of trained community health workers and traditional birth attendants. More complicated cases are referred to the regional healthcare facilities in Asmat or Keren.

Until recently the main source of drinking water for Rikeb town was the Zara River, but recently piped water has been introduced in the town. However, only a few of the residents have access to piped water, the rest still depend on the public fountain, public well or Zara River as a source of water. There is no sewage or sanitation system in Rikeb or other villages.

Modern communication facilities such as telephone and internet are not available in the project area. The main source of energy for the community is firewood. Electricity has not yet been introduced to this area, although several local businesses use generators to provide light to operate after sundown.

The livelihood of the people in the project area mainly depends on livestock production (pastoralism). Farming is very limited and covers 0.4% of the project area. With the recent allocation of land for farming, some of the villages are cultivating pearl millet and sorghum. Unlike the central highlands of Eritrea, there is little integration between farming and livestock production. The livelihood of the local people is supplemented by artisan mining activities with some people selling gold or labour. Currently this illegal gold extraction is carried out at Debri Tsa’eda and other scattered localities. In addition trading and small businesses are also sources of income in areas such as Rikeb village.

The Eritrean Land Proclamation (Proclamation No. 58/1994) declared that all land is the property of the state. According to the law any right over land is given by the state. This indicates that the village community has no collective ownership over land, but the government allows villages to have continuous use and control over farmlands, rangeland and water resources. The law ensures that:

- every Eritrean citizen is entitled to land usufruct with regard to agriculture and residential land (Tiesa) regardless of origin, sex and religion
- land allotted according to the proclamation shall be registered and granted in the name of the recipient usufructuary; the usufructuary shall use the land for his/her lifetime
- a usufructuary may, in exchange for a fixed quantity of agricultural products, grant the right to use part or all of his/her land to any person who would contribute labour or oxen, or both, or other farming implements
- a usufructuary may lease his/her right over land in whole or in part and duration of contracts shall be determined by an agreement to be made between the parties
- the government or appropriate government body shall have the right and power to expropriate land that people have been settling on or land that has been used by others, for various development and capital investment aimed at national construction. A government body that expropriates land in accordance with this provision shall pay compensation
- the inhabitants of the study area have equal access to rangelands and arable land. Currently some of the inhabitants in the village own plots of farm land, with estimated sizes of 0.5 ha to 1 ha. Land distribution is carried out by the local administration. Grazing and woodland are communal property and are ruled by customary law.

6 HISTORY

Exploration and artisanal mining activity in the Zara Project area indicates that it is possible that gold has been worked in the area since the Middle Ages or even as far back as the Pharaonic Period.

From 1996 artisanal miners were active on various prospects within the Zara Project, with most activity centred on the Koka and Konate prospects. Estimates from the Ministry of Energy and Mines in Asmara suggest that between 950 ounces and 1,300 ounces of gold were extracted per month at the height of artisanal mining activity. However, since the arrival of Sub-Sahara in 2003, gold recovered by artisanal miners from Koka and Konate has been estimated to be no more than 1 ounce to 5 ounces per day.

The original prospecting license covering the Koka gold deposit was issued to the original joint venture partners by the Department of Mines of the Eritrean Ministry of Energy and Mines on 2 October 1998 and converted to four exploration licenses on 20 October 2000. The exploration licenses are still current although the area covered by the tenements has been reduced to 147 km^2.

Chalice acquired its interest in the Zara Project by way of:

- a merger in August 2009 with ASX-listed Sub-Sahara Resources NL ("Sub-Sahara") which held a 69% interest
- the purchase of all the shares in the 11% interest held by the unlisted Australian company Africa Wide Resources Ltd ("AWR")
- the acquisition in June 2010 of all the shares of Dragon Mining (Eritrea) Ltd, a wholly-owned subsidiary of Dragon Mining Limited ("Dragon") which held a 20% interest.

There had been no previous exploration tenure over the Koka gold deposit nor any recorded exploration activity by a previous owner or operator. There are no historic Mineral Resource estimates for the Koka gold deposit.

7 GEOLOGICAL SETTING

Neoproterozoic metamorphic rocks outcrop over much of northern and central Eritrea. These rocks were accreted onto the Arabian-Nubian Shield and then deformed between 900 Ma and 550 Ma. They can be subdivided into four terrains, which from east to west are referred to as the Nakfa, Adobha Abiy, Hagar and Barka terrains. These terrains are separated by major, crust-breaking, east-northeast trending deformation zones.

The Zara Project is situated within rocks assigned to the Nakfa and Adobha Abiy terrains (Figure 7.1) 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 licenses 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 650m 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 (Figure 7.2). 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.

Figure 7.1 Regional Geology

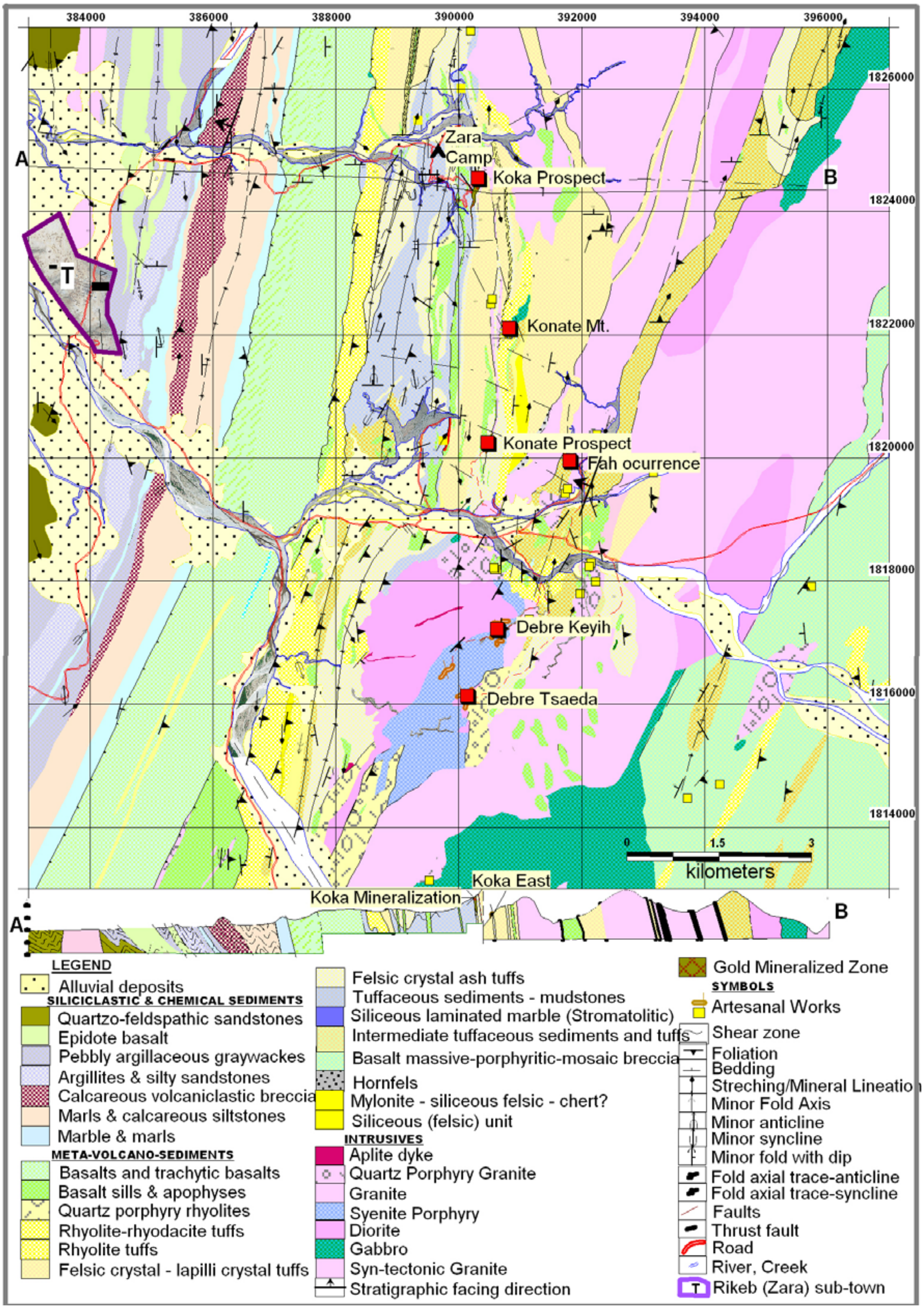
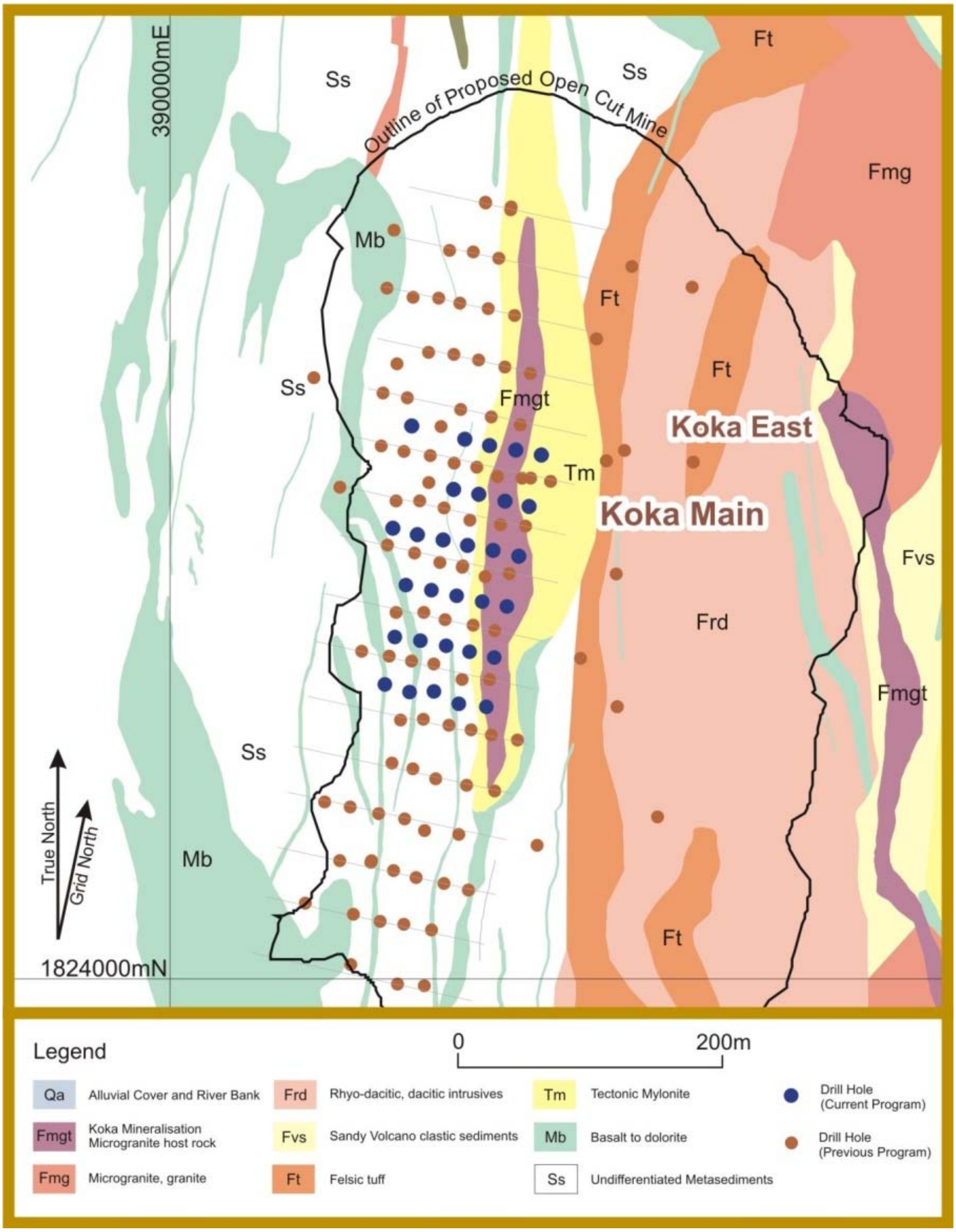
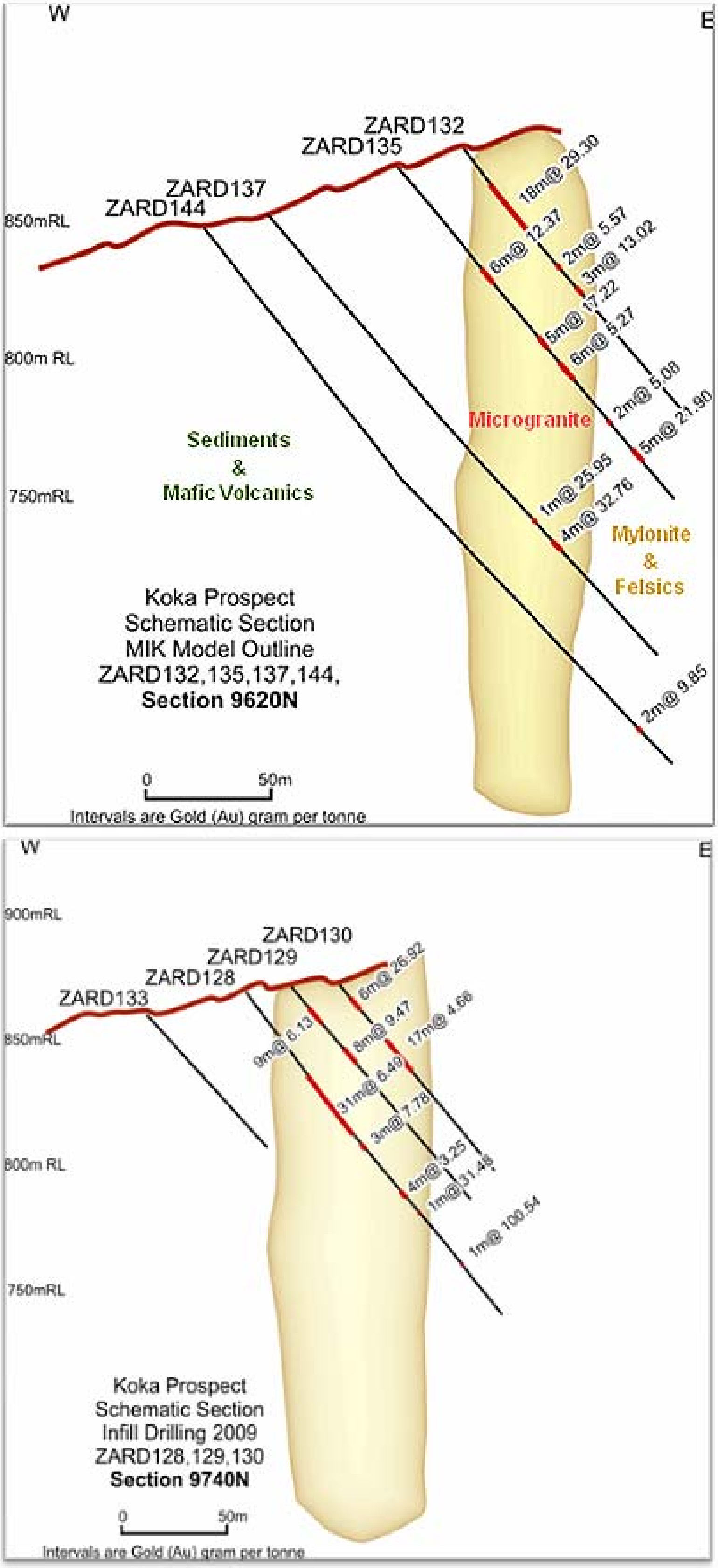


Figure 7.2 Koka Prospect Geology



The contact between these two major sequences is linear, sharp and sub-vertical to steep easterly dipping and orientated approximately north-south (Figure 7.3). It is strongly sheared and mylonitised. The contact is sub-parallel to a pervasive regional fabric, developed particularly in the finer grained rocks, that also dips steeply to the east. A shallow, southerly plunging, regional lineation is also evident.

Figure 7.3 Schematic Geological Sections Looking North



Low-grade regional metamorphism in the lower greenschist facies is considered to have affected the volcanogenic rocks, producing foliated metamorphic assemblages dominated by mats of sericite with micro-granular albite and quartz.

The rocks in the Koka area 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 sedimentary rocks and volcanic rocks. Primary bedding and lamination structures are well preserved in the sedimentary rocks. Systematic repetition of units occurs at approximately 500m intervals.

The fold system is interpreted as a double plunging asymmetrical fold with a long limb to the east and a short limb to the west.

Intense reverse faulting along high angle planes parallel to foliation and sub-reverse faults with east block upwards displacements have been observed. Strong faulting and displacement in the footwall sediments represents the plane along which basalt and microgranite have been emplaced and subsequent hydrothermal activity occurred.

The main zone of mineralisation is hosted within competent microgranite on the western margin of the Koka shear zone.

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 main alteration assemblage associated with mineralisation is silica+pyrite+sericite. Well defined zones of silicification are coincident with Koka's main zone of mineralisation, with a north-south striking silica-pyrite alteration corridor in the tuffs.
- Envelopes of pyrite alteration (with more than 2% pyrite content) are coincident with, and extend beyond, the interpreted zone of mineralisation.
- Sericite alteration envelopes the silica+pyrite alteration haloes and could be coeval with the gold mineralisation.
- Carbonate alteration occurs as pervasive wall rock alteration and veins, and may or may not occur with the other main alteration types.
- Weak chlorite alteration occurs over the known Koka mineralised area.

8 DEPOSIT TYPES

Koka is considered to represent a greenschist facies, orogenic gold deposit.

This style of deposit is present in metamorphic terranes of various ages, and displays variable degrees of deformation. The host geological environments include volcano-plutonic and clastic sedimentary terranes, both of which comprise the sequence of host rocks encompassing the microgranite, which in turn hosts the auriferous quartz veins at the Koka gold deposit.

The host rocks to orogenic gold deposits have characteristically been metamorphosed to greenschist facies conditions and the deposits can be hosted by any rock type. Host rocks to the Koka gold deposit have been metamorphosed to lower greenschist grade, with marginally higher metamorphic grade (mid- to upper-greenschist) evident at the Konate prospect several kilometres along strike to the south of Koka.

Orogenic gold deposits typically occur within, or in the vicinity of, regional, crustal-scale deformation zones with a brittle to ductile style of deformation. The overall geological architecture of northern Eritrea is dominated by a north-northeast to north striking shear zone that separates predominantly gneiss and intrusion-dominated lithologies in the east from dominantly sedimentary and ultramafic (ophiolitic) lithologies in the west. The shear diverges at the southern end and comprises an eastern shear, the Elababu Shear Zone, and a western shear, the Baden Shear Zone. The Koka gold deposit is located in the Nakfa Domain and lies immediately to the east of a major flexure in the Elababu Shear Zone.

Typically, there is a strong structural control to mineralisation in orogenic gold deposits. Depending on physical and chemical conditions, the characteristics of the host and feeder structures can be highly variable. Host structures include brittle faults, ductile shear zones, extensional fractures, stockworks, breccias, and fold hinges. Given that the structural controls to orogenic gold deposits operate at all scales, it is common for more than one structural site to host mineralisation.

At the deposit-scale the competent Koka microgranite has failed in a brittle fashion during contractional deformation, allowing ingress of silica-carbonate bearing fluids and deposition of the veins that host gold. The quartz-carbonate veins comprise a stockwork that exhibits two principal orientations. Quartz veins were deposited progressively, coeval with ongoing deformation, resulting in a second-order stage of fracturing and brittle-ductile shearing that has overprinted the veins. Gold-sulphide mineralisation was deposited during the later stages of this progressive deformation event. At the scale of the veins the gold is hosted by marginal shears on the boundaries of the quartz veins. As with many orogenic gold deposits, the Koka gold deposit exhibits the same style of veining from surface to the vertical extent of current drilling.

9 MINERALISATION

The gold mineralisation at the Koka gold deposit 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.

The western contact of the microgranite is the conduit for a zone of intense carbonation and sericitisation up to 20m wide. Multi-element geochemical patterns indicate that this zone is enriched in Ca, Mg, K and Fe. Within 50m to 80m of this major contact, the multi-element geochemical patterns outline a second zone of intense alteration only 10m wide, also enriched in Ca, Mg, K and Fe, within the microgranite. To the east of this second zone of intense alteration, the microgranite is pinkish in hue due to the appearance of potassic feldspar and shows less evidence of alteration. It is cut by later basaltic intrusives.

The footwall contact of the microgranite hosting gold mineralisation is the contact between the meta-sedimentary rock and the microgranite. No anomalous gold mineralisation has been intersected in any of the footwall rocks. The hangingwall contact of mineralised microgranite is taken as the first appearance of unaltered, pinkish, potassic feldspar-bearing microgranite. The mineralisation lies within this 50m to 80m wide zone. It is preferentially located closer to the footwall contact and is intimately associated with a stockwork of quartz veins. In some of the wider gold intersections the higher grades and more continuous mineralisation are found closer to the sharp footwall contact, whereas the hangingwall contact of the mineralisation is more diffuse (refer to Figure 7.3).

Fracturing, veining and mineralisation particularly affected the microgranite, possibly because it was a structurally competent unit which fractured in response to deformation. Thin brittle fractures and open fractures encouraged invasion of the rock body by abundant mineralising hydrothermal fluid composed mainly of H_2O and CO_2, with minor other dissolved components including S, Zn, Pb, Cu, Au and possible Sb.

Thin fracture networks were sealed by fine-grained foliated sericite ± carbonate (dolomite), and are now observed as thin pale yellowish green fracture fillings in white host rock which suffered selective pervasive replacement by the alteration assemblage albite + minor sericite + carbonate (dolomite) + opaques (pyrite ± inclusions of sphalerite, galena) + trace leucoxene. Primary quartz and zircon are preserved, and primary potassic feldspar has survived in pale pinkish cream rocks that have suffered lower intensity of alteration. Rare small grains of native gold occur adjacent to veins.

Limited post-vein deformation has generated shadowy strain extinction in vein quartz and thin micro-cracks in vein pyrite. It has also caused local remobilisation of galena, sphalerite and trace native gold for short distances along some micro-cracks.

The main mineralisation zone has been divided into three sub-zones:

1. The Northern Zone occurs in the northernmost 50m of the main zone and comprises a flat vein system. Sericite+carbonate+pyrite alteration is common.
2. The Middle Zone comprises a flat vein system and quartz vein stockwork. Stockwork veining is restricted to several meters on the eastern margin of the zone. Strong sericite+carbonate±pyrite alteration is characteristic.
3. The Southern Zone comprises a stockwork system approximately 20m to 35m wide. A silicified and brecciated quartz porphyry dyke hosts a stockwork of sulphide bearing auriferous quartz veins.

The Koka mineralisation is interpreted to represent a progressive brittle-ductile deformation system, as outlined below:

- Basalt injection along structurally permeable zones (lithological contacts, major thrust faults and bedding).
- Tectonic dextral shear stress created en-echelon openings along permeable zones within the more brittle lithologies.
- Silica and carbonate rich, low temperature hydrothermal activity accompanied intrusion of microgranite along tensional openings. The formation of quartz veining and stockwork, strong silicic alteration and selective sericite-carbonate alteration is related to the intrusion of the microgranite.
- Reactivation of the hydrothermal system resulted in brecciation of the microgranite. Fractures developed due to fluid overpressuring, veins formed in dilational zones, and minor gold was precipitated.
- Gold and other metals were deposited in openings within and around existing quartz veins and wider fractures throughout the stockwork and mosaic breccia.

10 EXPLORATION

In 1997, Dragon, AWR and Genesis Resources NL formed a joint venture operated by Dragon. The Zara Project area was applied for and granted in October 1998 as prospecting licenses covering 400 km^2. Only minimal work was completed during this period. Dragon increased its interest when Genesis withdrew leaving two partners: Dragon (66.66%) and AWR (33.33%). All licenses were suspended for two years as a result of a border dispute with Ethiopia and all prospecting licenses were converted to exploration licenses in 2000.

In 2003, Sub-Sahara signed a joint venture with Dragon to earn a 70% interest in Dragon's 66.66% and during August 2005, the first drilling by Sub-Sahara at the Koka discovery commenced and resulted in the confirmation of gold mineralisation beneath the artisanal workings. The drilling consisted of four diamond drillholes about 150m apart, returning significant gold mineralisation in all four holes. A systematic diamond drilling programme covering the known strike of artisanal workings commenced in January 2006 (Table 10.1).

Table 10.1 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

In September 2006, Sub-Sahara agreed to acquire a further 22.35% interest in the project from AWR leaving AWR with a free-carried 11.12% interest. In August 2007, Sub-Sahara advised Dragon that it had earned its 69% in Dragon's interest. At this time the equity stood as follows: Sub-Sahara 69%, Dragon 20% and AWR 11%.

In July 2009, the AWR agreed to sell its interest to Chalice and in August 2009, Sub-Sahara merged with Chalice giving Chalice an 80% stake in the Zara Project. In June 2010, Chalice acquired Dragon's interest resulting in 100% ownership.

Drilling to 2008 was used to estimate the May 2009 Mineral Resource. Further drilling on an infill pattern has been carried out between October 2009 and March 2010 and all available data were used to estimate the June 2010 Mineral Resource.

Licenses are currently renewed on an annual basis requiring compulsory relinquishment. The license was recently renewed for an additional 12 months to May 2011.

At the nearby Konate prospect five diamond drillholes were drilled to test surface geochemical anomalies with intersections of 5 g/t Au over 1m and 2.1 g/t Au over 2.1m. Drillhole intersections did not readily support the intensity of mineralisation apparent at surface.

Exploration has continued to be carried out on the broader area of the Zara exploration licenses and Zara North and Zara South prospecting licenses. Satellite imagery covering

Proterozoic sedimentary and volcanic rock types was interpreted and spectral signatures of known base metal and gold deposits was used to identify similar signatures. In the Zara project area, more than 60 spectral anomalies suggesting alteration or iron enrichment were identified for further work.

A detailed structural geological interpretation of Landsat and Quickbird imagery was compiled in tandem with the spectral interpretation. Country-scale structural geological relations were interpreted and the following geological data sets were created:

- Intrusions including definition of contact metamorphic aureoles and dyke swarms. Several intrusive suites were identified based on geometry, compositional zoning and relationship to countryrock fabrics.
- Major faults, which were divided into populations of different structural ages based on cross-cutting relationships and fieldwork.
- Trends of lithology.
- Strike trend lines of several generations of widely distributed cleavage.
- Strike trend lines for a pervasively developed 'shear fabric' that is continuous across all of the Landsat images.

The faults and shear fabric trends were further interrogated to resolve the movement sense of all structures thus allowing a compilation of fault populations attributed with structural age and kinematics. Based on the structural geological interpretation all of the features known to be conducive to the structural localisation of orogenic gold in other major goldfields were identified. Zones containing a coalescence of favourable features were identified and integrated with spectral targets in order to highlight and prioritise areas of potential for high exploration success.

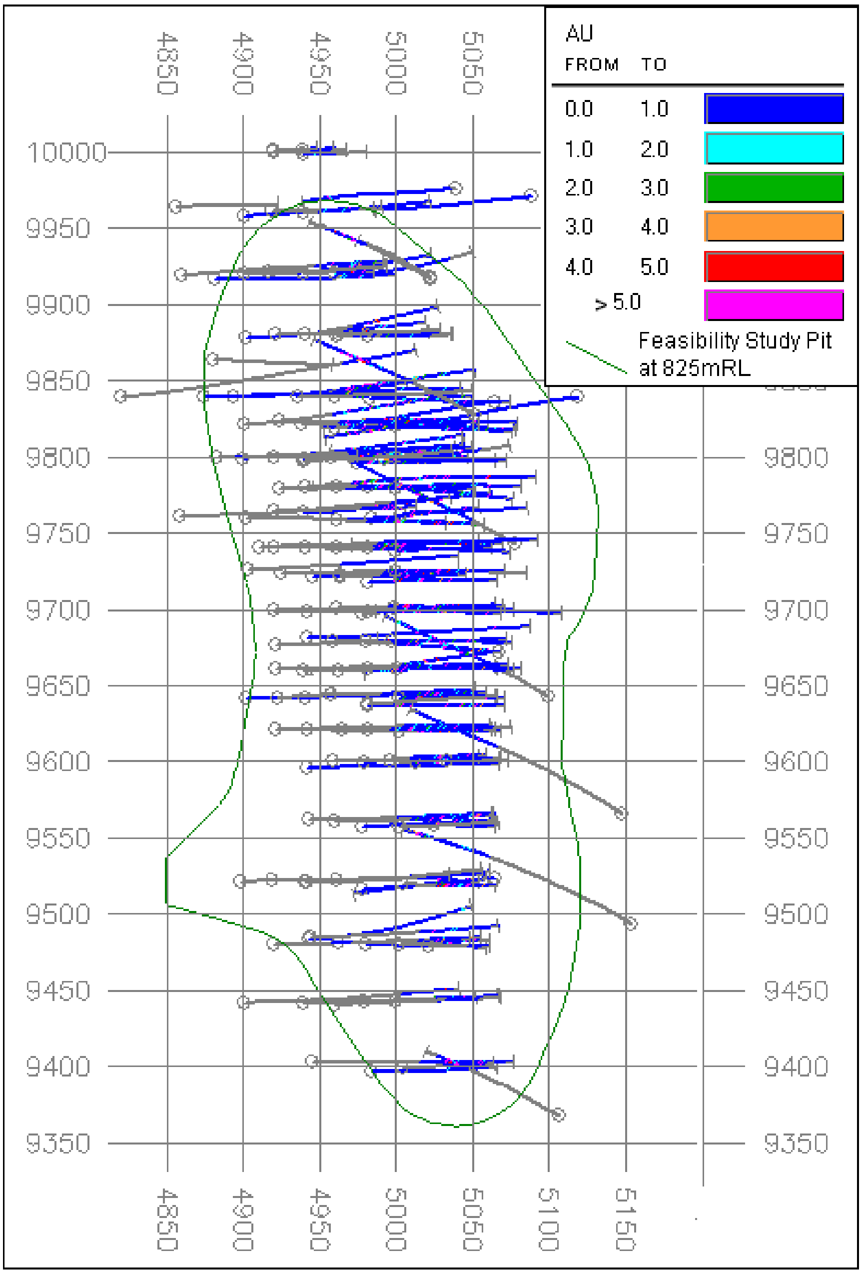
An iron-rich spectral anomaly was identified over a significant gossan in the Zara South prospecting license associated with altered pyritic rhyolite and pyritic chert. The gossan, which indicates a potential massive sulphide deposit, appears to lie within the northern continuation of volcanic stratigraphy that hosts the Bisha volcanic-hosted massive sulphide deposit about 100 km to the south.

Exploration plans to test the spectral anomalies include ground geophysics, geological mapping and drainage geochemical sampling.

11 DRILLING

All drilling at the Koka Gold Project has been diamond drilling. A total of 137 drillholes totalling 20,839m were used for the 2010 Mineral Resource estimate. Eight drillholes for 1,078m completed for metallurgical testwork have no associated assays. Figure 11.1 shows the location of all drillholes. Drilling has been completed over a strike length of about 610m and to an average depth of about 165m below surface. The current drilling has not yet closed out mineralisation at depth and there is potential to extend the interpreted mineralisation.

Figure 11.1 Traces of Diamond Drillholes



The orientation of most drillholes (west to east) intersects the steeply dipping, north-oriented mineralised zone. Mapping of vein orientations showed a number of vein orientations. A dominant orientation averages a strike 151 with a dip of 61º to the west but two significant clusters strike 037 and dip 52º to the east and strike 286 and dip 36º to the north. The dominant drillhole direction is not conducive to intersecting the latter two vein directions at a high angle. Drillholes oriented to intersect 037 and 286 striking veins would not be suitable to intersect 151 striking veins. The steeply west-dipping topography of the Koka site also precludes drilling of many west-dipping drillholes.

Drilling 2005 to 2008

Drilling was carried out by General Exploration Drilling under contract with supervision provided by Sub-Sahara geologists. Two drill rig types were used: GED37 man-portable rig and Longyear 38 drill rigs drilling HQ (63.5 mm) diameter core.

The drilling up to the completion of the Mineral Resource estimate was designed to test the mineralised zone over about 610m along strike by about 320m across strike at about 40m x 20m spacing. Most drillholes have been drilled in a west to east direction (at an azimuth of approximately 100 UTM) with some early holes drilled in an east to west direction (at an azimuth of approximately 280 UTM). Seven holes oblique to the grid were drilled to collect structural information.

Survey data for the Koka area were collected using a Promark3 differential global positioning system ("DGPS"). Survey data are available in both UTM and local grid coordinates. The coordinate system is UTM North Zone 37 and is based on the WGS 84 ellipsoid.

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

Drill sites require regulatory approval before a hole is drilled and a geologist must be present to supervise the drilling whenever a drill rig is operating.

The proposed site was pegged by a qualified surveyor where possible or by the geologist. Drillhole collars of completed holes have been surveyed by DGPS by a qualified surveyor except for a small number of holes where the drill pads were destroyed in preparation of other drill pads before survey. The extreme topographic nature of the Koka site means that not all drillhole collars can be preserved. In these cases the original setup DGPS survey location has been used.

Downhole surveys were completed using a Reflex EZ-Shot® tool. All diamond drillholes were surveyed at intervals of 30m downhole. The downhole survey measurements have been converted from magnetic to UTM North Zone 37 values using an azimuth conversion factor of -5º and also to the local grid which is rotated from UTM by 12.5º to the east.

Diamond core was orientated using a combination of the spear technique and an Ezimark orientation device. All orientation spear marks were scrutinised for reliability before acceptance and the orientation marks were correlated between drill runs. Good correlation between reliable orientation marks was indicated by drawing a solid line along the core. Dashed orientation lines along the core represent less reliable correlation between the orientation marks. Drilling induced or handling fractures were marked on either side of the break.

Diamond core recovery was monitored and ranges between 88.5% and 99.7%. The majority of drilling has a core recovery of 95%. Core from the moderate to highly weathered units and highly fractured and brecciated zones are typically problematic and lower recoveries are recorded. There is no apparent correlation between gold grade and reduced core recovery.

Core was recovered into galvanised core trays marked for the hole number, interval, and hole start. Core runs were marked with plastic markers. Core was photographed to provide a permanent record. Dry and wet core photographs were taken. Logging, sampling and core storage was completed under cover at the Koka camp site.

Each drillhole was logged by qualified geologists using a standardised logging format and geological codes. Lithology, colour, weathering, schistosity, grain size, texture, mineralisation, alteration, structure and veining were observed and recorded. Core structure orientations were routinely recorded to assist in determining the controls on mineralisation and to provide additional geotechnical information.

Data collected from detailed geotechnical logging of orientated diamond core includes:

- discontinuity types and orientation
- rock fabric defects/descriptions
- discontinuity surface roughness and infill
- rock quality designation
- compressive strength
- structures
- core orientation.

Logging is directly entered into an MS Excel template.

Drilling October 2009 to March 2010

Drilling completed from October 2009 to March 2010 was aimed at intermediate sections between the previous 40m spaced sections resulting in a drillhole pattern of 20m x 20m at surface. The drilling was aimed at understanding the short range variability of the gold grade and a better understanding of geological controls.

All drilling was HQ diameter (63.5 mm) diamond drilling. Data collection practices including collar survey, downhole survey, drill core management sampling and assaying followed the same protocols as had been established previously.

12 SAMPLING METHOD AND APPROACH

All drilling at the Koka gold deposit was diamond drilling. Samples were taken over the full length of intersections of the altered microgranite that hosts the gold mineralisation. Less continuous sampling has been completed in footwall metasediments where quartz veins are apparent.

Competent core was cut with an electric diamond core saw with a sample length mainly of 1m. Very soft core was dry cut or cleaved. The orientation line was used as a guide for cutting. The half core with the orientation line was returned to the core tray and the remaining half core was bagged for analysis.

Bulk density determinations were carried out on every metre of core within expected mineralisation and every 10m within waste zones. Determination was on site using the water immersion method, prior to submission for assaying. Two reference standards have been used to calibrate the determinations. The bulk density of the standard material was measured at the start and end of each batch for calibration purposes. Bulk density data are recorded in the drillhole database.

A total of 2,310 bulk density determinations have been collected and range from 1.37 t/m^3 to 4.7 t/m^3, with an average of 2.74 t/m^3. The bulk density was confirmed by 12 samples that were submitted to a commercial laboratory for determination. Those samples returned an average bulk density of 2.75 t/m^3.

Diamond core recovery ranges between 88.5% and 99.7% with an average of 95%. There is no apparent correlation between gold grade and reduced core recovery.

The drillholes are distributed over the length and depth extent of the mineralised zone and drilling has been executed following acceptable industry practice supported by assay quality control protocols. The drillhole spacing and sampling provides good representation of the mineralised zone over its strike and depth extent. Figures 12.1 and 12.2 show log probability and frequency histogram plots for all drillholes within the microgranite available for the Mineral Resource estimate.

Figure 12.1 Log Probability Plot of Gold Assays within Microgranite

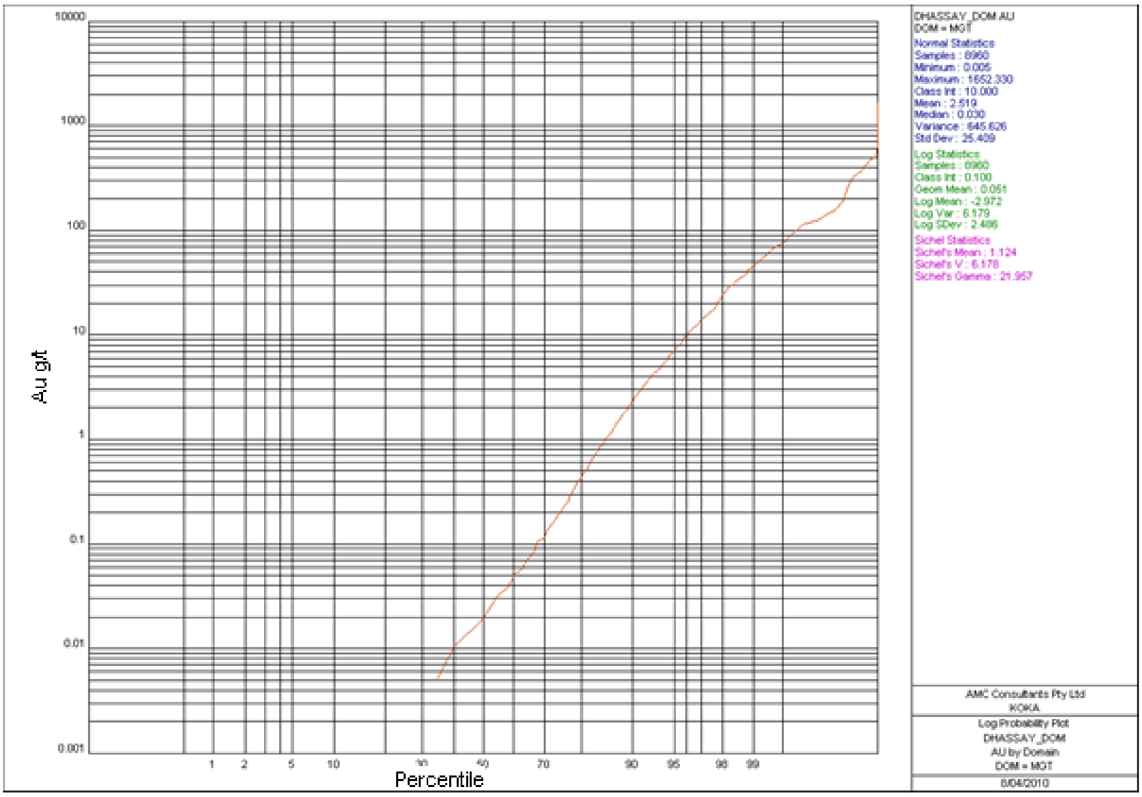
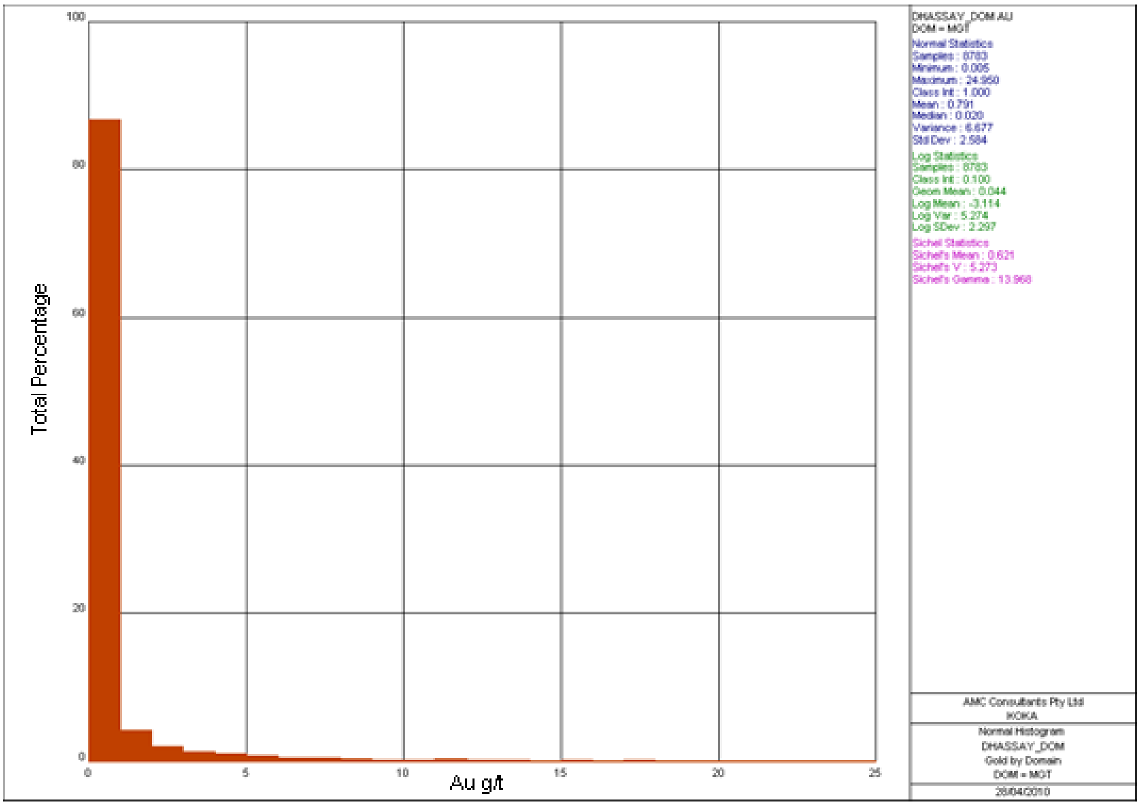


Figure 12.2 Histogram of Gold Assays within Microgranite



13 SAMPLE PREPARATION, ANALYSES AND SECURITY

Sample preparation of the core samples was conducted in Asmara by Eritrean company Africa Horn. The sample preparation laboratory is a joint venture with Genalysis Laboratory Services Pty Ltd ("Genalysis") which is a commercial mineral industry laboratory accredited with the National Association of Testing Laboratories, Australia. The following procedure was followed:

- Approximately 3 kg of sample was dried and jaw crushed to -3 mm.
- The sample was split using a Jones-type riffle splitter to collect a 1 kg to 2 kg sub-sample for pulverisation.
- The sub-sample was pulverised to a nominal 95% passing -75 micron using an LM2 pulveriser.
- The pulp was further split into two 100g to 150g sub-samples; a primary pulp sample for analysis and a duplicate pulp sample as a reference.
- The sample pulps were transported by air to the principal laboratory, Genalysis, in Perth, Western Australia. Ultra Trace Pty Ltd (Perth, Western Australia) was the umpire laboratory for the first 127 drillholes. ALS Laboratory Group (Perth, Western Australia) was the umpire laboratory for subsequent drilling.
- The majority of gold assaying has been completed using a lead collection 50g fire assay method with an atomic absorption spectroscopy ("AAS") finish. Some assaying has been completed using a screen fire assay and Leachwell methods.

A blank sample was introduced every 20 to 25 routine samples as part of the normal sample submission process. Blanks were also submitted after samples with expected high grades to test contamination during sample preparation at the laboratory. Blank samples were disguised to appear as normal samples.

Four certified reference materials ("CRM") supplied by Australian supplier Geostats Pty Ltd in Australia were submitted with all sample batches with two standards submitted at the start and end of each batch. The CRMs were disguised from the laboratory. The CRMs contain low, medium and high gold grades to reflect the grade distribution of the Koka gold deposit.

On site sampling of drillcore was undertaken under the supervision of Chalice geologists. Sample preparation and analyses were undertaken by independent laboratories and was not conducted by an employee, officer, director or associate of Chalice.

14 DATA VERIFICATION

CRMs, blank samples, duplicates and repeats have been submitted to the routine and umpire laboratory as part of the assay quality control procedures.

Five percent of the returned coarse reject samples are routinely submitted to the umpire laboratory to test the analytical precision of the principal laboratory. Standard samples are included at a frequency of 1 every 20 routine samples and there is at least one standard per submission.

Five percent of all returned pulps are submitted to the principal laboratory (2.5%) and umpire laboratory (2.5%) to monitor the precision of the principal laboratory. Standard samples are included at a frequency of 1 every 20 routine samples and there is at least one standard per submission.

Wet sieving analysis was conducted on 5% of all pulps to test the consistency of the pulverisation process at the principal laboratory. Ninety percent of each pulp should pass an appropriate mesh size. Wet sieving analysis was carried out by the umpire laboratory.

Drilling 2005 to 2008

Coffey (2009) evaluated assay quality control data in preparation of the May 2009 Mineral Resource estimate.

Coffey (2009) evaluated the results of 489 assays of 21 CRMs and one blank sample. Results generally showed a small positive bias averaging 2.1% and ranging from -1.19% to +6.14% (Table 14.1). The positive bias is more apparent for lower grade CRMs with most within the expected accuracy tolerance levels. Overall, the CRMs show good replication of the certified value.

Table 14.1 Certified Reference Materials and Blanks

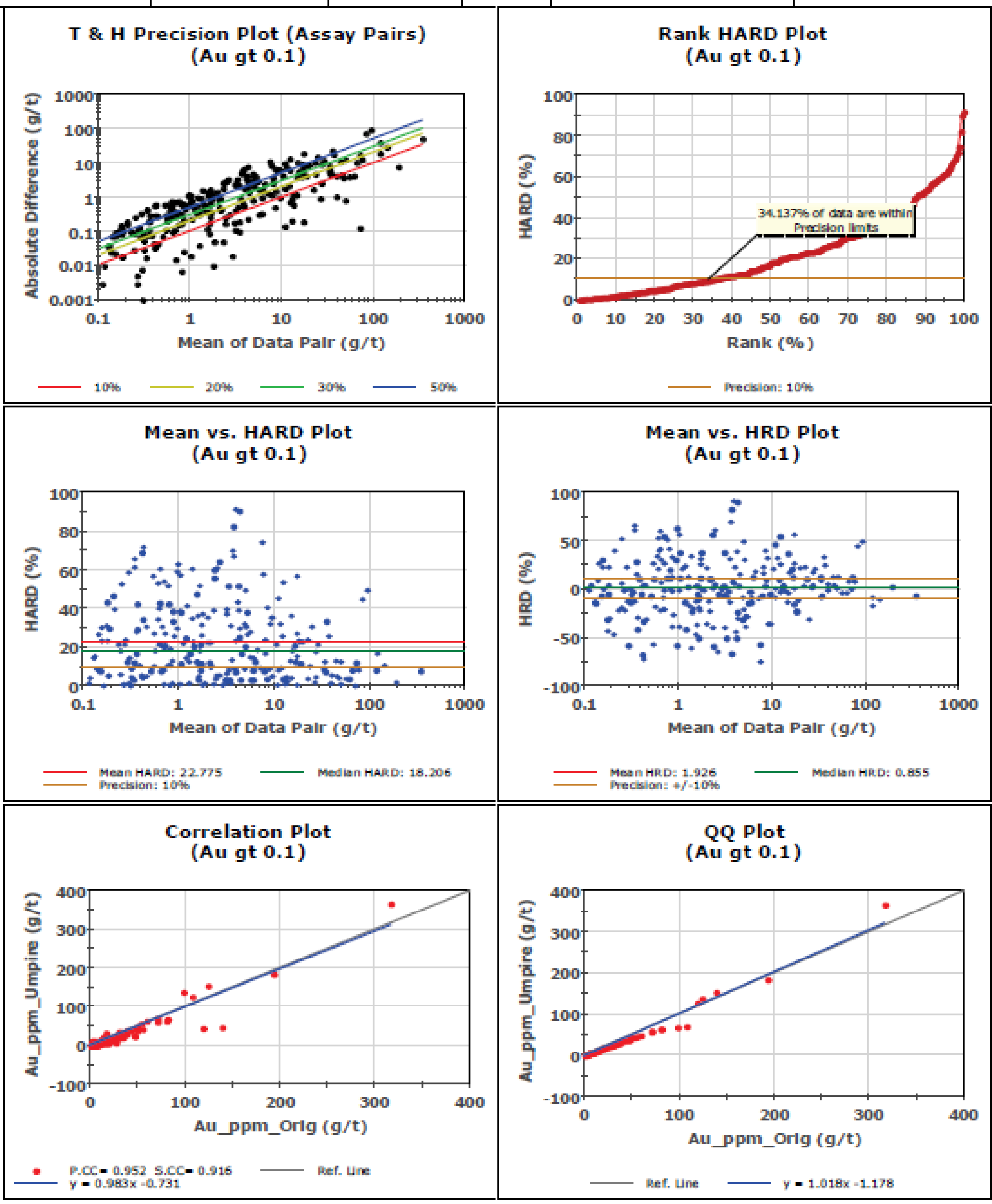
Standard Name	Expected Value (EV)	± 10% of Expected Value (g/t)	No. of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within ±10 of EV (%)	% RSD (from EV) (%)	% Bias (from EV) (%)
Blanks									
Control Blank	0.005	0.004 to 0.005	122	0.005	0.03	0.005	98.36	43.75	4.92
Certified Reference Materials									
OXD27	0.416	0.374 to 0.458	4	0.41	0.42	0.415	100.00	-0.24	1.21
OXE20	0.548	0.493 to 0.603	3	0.52	0.60	0.553	100.00	0.97	6.14
OXE42	0.610	0.549 to 0.671	51	0.58	0.68	0.624	98.40	2.22	2.96
OXE56	0.611	0.550 to 0.672	22	0.61	0.66	0.626	100.00	2.52	2.29
OXE60	1.025	0.922 to 1.127	22	1.01	1.07	1.037	100.00	1.15	1.75
OXH37	1.300	1.170 to 1.430	16	1.26	1.38	1.303	100.00	0.19	2.36
OXH52	1.291	1.162 to 1.420	40	1.20	1.38	1.314	100.00	1.74	2.98
OXJ36	2.398	2.158 to 2.638	3	2.38	2.45	2.416	100.00	0.64	1.19
OXL34	5.758	5.182 to 6.334	25	5.52	6.06	5.828	100.00	1.22	1.88
OXL51	5.850	5.265 to 6.435	38	5.58	6.02	5.813	100.00	-0.64	1.72
OXN49	7.635	6.872 to 8.399	16	7.35	7.78	7.590	100.00	-0.59	1.68
OXP39	14.890	13.401 to 16.379	25	14.68	15.45	14.965	100.00	0.51	1.28
OXP50	14.890	13.401 to 16.379	52	14.58	15.63	15.150	100.00	1.75	1.50
SJ10	2.643	2.379 to 2.907	15	2.57	2.74	2.649	100.00	0.21	1.77
SJ22	2.604	2.344 to 2.864	6	2.51	2.72	2.602	100.00	-0.09	2.36
SJ32	2.645	2.380 to 2.909	60	2.57	2.80	2.676	100.00	1.18	2.01
SK11	4.823	4.341 to 5.305	3	4.82	4.98	4.890	100.00	1.39	1.37
SN16	8.367	7.530 to 9.204	15	8.19	8.65	8.457	100.00	1.08	1.65
SN26	8.543	7.689 to 9.397	71	8.19	8.94	8.586	100.00	0.50	
OXM16	15.150	-	1	-	-	16.050	-	-	-
OXg22	1.035	-	1	-	-	1.110	-	-	-

A total of 249 assay pairs of assays greater than 0.1 g/t Au were available for splits of sample pulps submitted to an umpire laboratory (Ultra Trace). Coffey (2009) concluded that a high degree of correlation (r=0.95 is apparent. Little relative bias is present although more significant differences occur at higher grades. Coffey (2009) concluded that the umpire assays support a finding that the data are accurate. Figure 14.1 shows the summary of umpire assaying.

A total of 147 pairs of laboratory pulp duplicates returned values greater than 1 g/t Au. The data show a relative precision of 18.4 (the mean of the half average relative difference; HARD) with 42.2% of the data within a ±10% range. A target of 90% ± relative precision is considered a good result for pulp duplicates (Figure 14.2). Coffey (2009) recommended a review of sample preparation and evaluation of coarse rejects.

Figure 14.1 Umpire Assaying Summary Statistical Charts

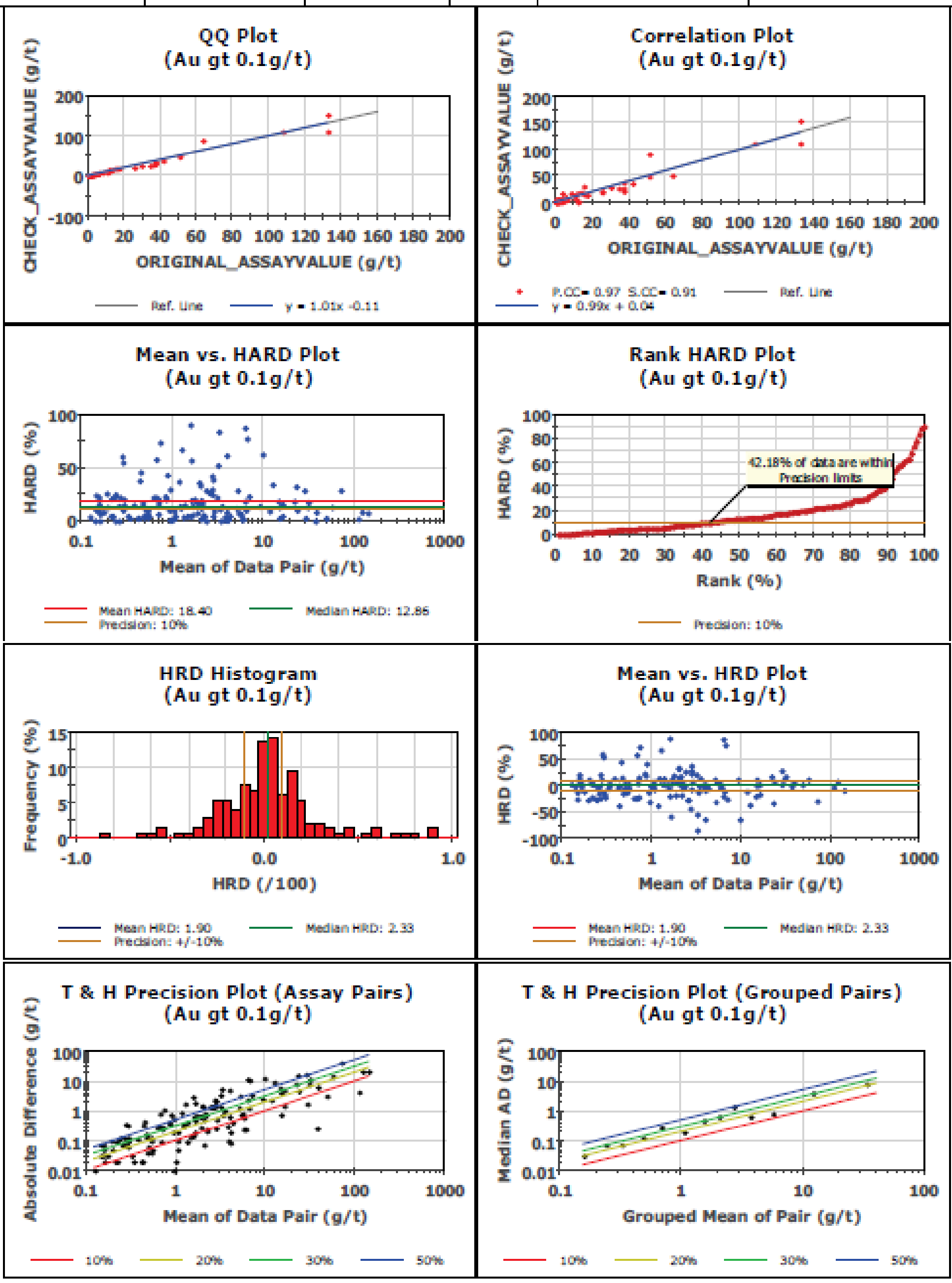
	Original Assay	Umpire Assay	Units	Parameter	Result
No. Pairs:	249	249		Pearson CC:	0.952
Minimum:	0.100	0.104	g/t	Spearman CC:	0.916
Maximum:	316.760	368.000	g/t	Mean HARD:	22.775
Mean:	12.552	11.605	g/t	Median HARD:	18.206
Median	2.030	2.180	g/t		
Std. Deviation:	30.939	31.953	g/t	Mean HRD:	1.926
Coefficient of Variation:	2.465	2.753		Median HRD	0.855



T & H: Thompson and Howarth plot
HARD: half average relative difference
HRD: half relative difference
QQ: quantile quantile plot

Figure 14.2 Laboratory Repeat Assaying Summary Statistical Charts

	Original Assay	Umpire Assay	Units	Parameter	Result
No. Pairs:	147	147		Pearson CC:	0.97
Minimum:	0.11	0.11	g/t	Spearman CC:	0.91
Maximum:	132.58	153.07	g/t	Mean HARD:	18.40
Mean:	7.85	7.83	g/t	Median HARD:	12.86
Median	1.53	1.40	g/t		
Std. Deviation:	20.00	20.55	g/t	Mean HRD:	1.90
Coefficient of Variation:	2.55	2.62		Median HRD	2.33



T & H: Thompson and Howarth plot
HARD: half average relative difference
HRD: half relative difference
QQ: quantile quantile plot

Overall, Coffey (2009) concluded that the quality control protocols are considered to represent good industry practice. The assay data are considered to be accurate although they display marginally acceptable precision. Additional assaying of coarse reject material was recommended.

Drilling October 2005 to March 2010

Assay quality control for drilling between October 2005 and March 2010 was reviewed by Maxwell Geoservices, an independent database and sampling consultant (Maxwell, 2010). Conclusions from the Maxwell (2010) review are:

- Certified Reference Materials:
    - The majority of both the company-inserted and laboratory CRMs have performed very well, and no glaring issues were observed.
    - There was one possibly mislabelled CRM.
    - There are some time trends observed for individual CRMs, but there is certainly no global trend apparent over the whole data set analysed.
    - Blank samples show some outliers above the expected two standard deviation limit, but the assays are below 0.1 g/t Au and are therefore insignificant.
- Repeats:
    - Both the sample preparation and laboratory pulp checks exhibited an overall negative bias, with repeats tending to be lower than their original assays. However this bias is weighted by the very high grade range and results for these repeats actually correlate, in general, very well.
    - The laboratory repeats correlate very well, with no appreciable bias observed.
    - The repeat results suggest that the Genalysis 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).

AMC Comment

Assay quality control data have been evaluated separately for drillhole data available for the May 2008 Mineral Resource estimate and the total data available for the June 2010 Mineral Resource estimate. AMC has reviewed the evaluation of assay quality control data and is satisfied that the procedures follow common industry practice and support the data to be used for Mineral Resource estimation.

15 ADJACENT PROPERTIES

There are no known significant gold occurrences in the Koka area although the tenements are being actively explored. Exploration has continued to be carried out on the broader area of the Zara exploration licenses and Zara North and Zara South prospecting licenses including interpretation of satellite imagery, ground geophysics, geological mapping and drainage geochemical sampling covering Proterozoic sedimentary and volcanic rock types.

There are no known significant mineral occurrences in the area surrounding the Zara exploration licenses and Zara North and Zara South prospecting licenses.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

A comprehensive metallurgical testwork programme initiated and supervised by Lycopodium was conducted on primary mineralised samples from the Koka gold deposit at the laboratory of Australian Metallurgical and Mineral Testing Consultants ("AMMTEC") in Perth, Western Australia in 2009. An earlier preliminary testwork programme was completed on about 30 kg of drillcore sample in 2007.

The detailed testwork programme was undertaken for the feasibility study with the following objectives:

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

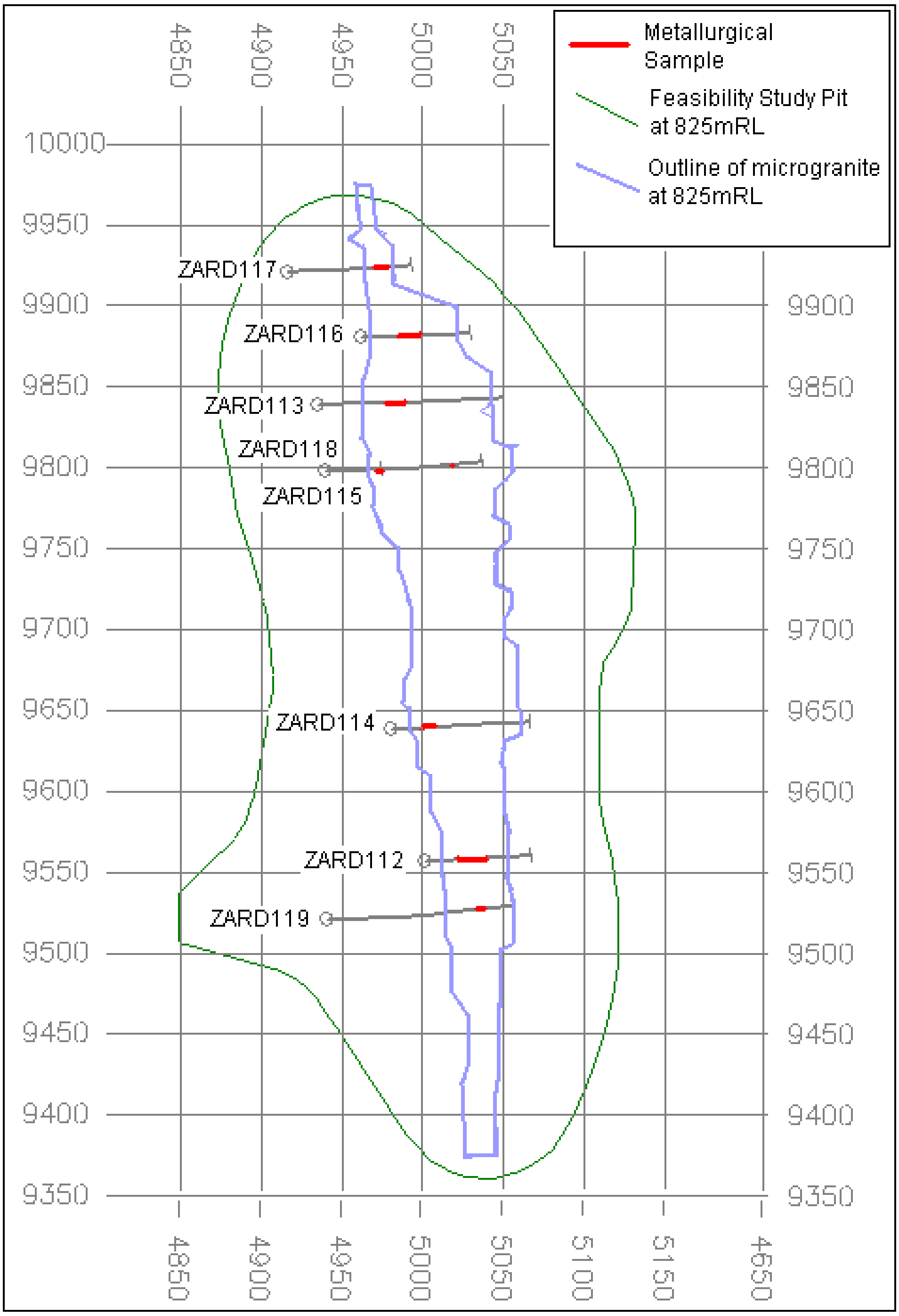
Samples from seven diamond drill core intercepts were transported from the Koka site to AMMTEC for comminution and metallurgical sample preparation. The drill core consisted of whole HQ diameter core intercepts that had been cut from alternate 0.20m intervals over the sample downhole length (Table 16.1) and individually labelled and bagged. The locations of the drillholes complete for metallurgical sampling are shown in Figure 16.1. AMC considers that these samples are representative of the mineralisation.

Table 16.1 Metallurgical Sample Intervals*

Extraction Variability Sample	Drillhole Number	From (m)	To (m)	Mass (kg)
EV1	ZARD112	31.22	60.80	82.1
EV2	ZARD113	75.00	96.75	23.4
EV3	ZARD114	30.20	45.80	42.9
EV4	ZARD116	31.22	50.80	53.2
EV5	ZARD117	88.20	100.80	33.6
EV6	ZARD118	54.20	60.80	20.1
EV7	ZARD119	129.20	131.76	8.1
EV8	ZARD119	148.20	154.80	18.8

*Samples are composited from alternate 0.20m intervals over the selected length of the drillhole

Figure 16.1 Location of Metallurgical Drillholes



Samples for metallurgical testing were selected after crushing and assaying every second sample. 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 mineralisation and with respect to gold head grade.

Ten comminution testwork samples were selected from the uncrushed core remaining from the seven diamond drill core intercepts.

Salient outcomes of the metallurgical and comminution testwork programmes conducted on Koka deposit primary ore are:

- The mineralised rock is moderately competent and abrasive with above average comminution energy requirements.
- The mineralised rock is 'free-milling' (non-refractory) 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 primary ores.

Head Grade Analysis

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

- total sulphur ranged from 0.69% to 2.84% S
- silver grades are variable (0.3 g/t to 32 g/t Ag) although higher grades are of limited occurrence and may need to be considered in the design of the carbon in leach (“CIL”) and elution circuits
- mercury (less than 0.10 ppm Hg) and arsenic levels (less than 88 ppm As) 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 ppm to 387 ppm Cu), lead (25 ppm to 3,163 ppm Pb) and zinc (44 ppm to 3,481 ppm Zn) 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 (less 0.03 ppm C) are low and preg-robbing should not be a problem.

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:

- unconfined compressive strength ("UCS") ranged from 30 MPa to 135 MPa with an average UCS of 74 MPa. The UCS values indicate the mineralised rock does not have high competency and can be primary-crushed using a jaw crusher
- crushing work indices ("CWI") ranged from 6.6 kWh/t 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 (a measure of resistance to breakage with a higher value indicating soft material) ranged from 32.3 to 80.5.

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.

Gravity and Cyanidation Testwork

A detailed testwork programme was conducted on the master composite to determine the optimum treatment route and conditions for treatment of primary mineralisation. The testwork programme investigated parameters such as the propensity for preg-robbing, gravity and leach versus direct leach treatment, optimum P_{80} grind 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 were as follows:

- P_{80} grind of 106 microns
- gravity concentration by centrifugal concentrator
- leach pulp density of 50% solids
- pH 10.0 to 10.5 adjusted with commercial lime
- initial cyanide dosage of 0.5 kg of sodium cyanide per tonne with residual cyanide levels maintained at or above 200 ppm NaCN
- dissolved oxygen levels of 20 ppm O 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:

- 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 g/t Au to 14.51 g/t Au
- lime consumption was low at 0.33 kg/t (60% available CaO basis)
- cyanide consumption was low at 0.29 kg of sodium cyanide per tonne.

Ancillary Testwork

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

- Oxygen uptake testing indicates that the mineralisation 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 show that the mineralisation 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 are good and high carbon loadings are achievable.
- 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 NaCN which is 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 reuse 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 than the 50 ppm NaCN required by the ICMC was adopted for plant design.
- Thickening testwork indicates excellent settling characteristics (2.0 $t/m^2.h$) due to the clean nature of the mineralisation. 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.

Plant Recovery

The testwork results suggest that calculated gold head grade of primary ore 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 g/t Au 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%.

Process Flow Sheet

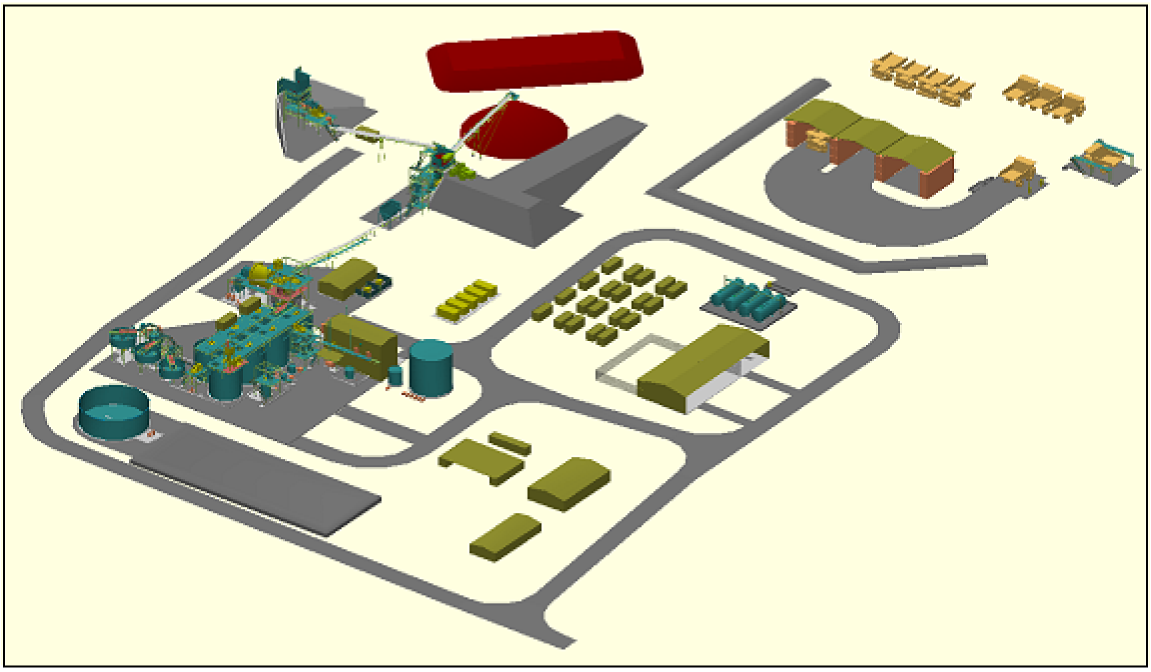
The overall process flowsheet adopted for the Koka gold project has been based on the outcomes of the detailed metallurgical testwork programme. The flowsheet selected is based on industrially proven unit processes and presents low technical risk. The major unit processes incorporated in the flowsheet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

- Single stage jaw crushing circuit.

- Single stage semi-autogenous grind 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 CIL 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 proposed process plant is given in Figure 16.2.

Figure 16.2 Proposed Process Plant Layout



17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Mineral Resource Estimation

The Mineral Resource estimate was completed by AMC using data available at 7 April 2010. The estimate is based on interpretation of overlapping gold and sulphide domains reflecting the association of gold with sulphide mineralisation. The domains were interpreted within a broader domain indicating the limits of the microgranite that hosts the gold mineralisation. Gold grade was estimated using ordinary kriging.

The resource estimate was carried out using Datamine mining industry software.

17.1.1 Data Available

Chalice provided AMC with data for drillhole collars, downhole surveys, gold assays and geological logging in comma separated format. Twenty three drillholes in the database were abandoned prior to reaching the mineralisation and remain unassayed. Chalice also provided a three dimensional wireframe of the interpreted microgranite in dxf format. Original laboratory assay reports and corresponding sample number lists for six drillholes were provided to AMC for validation of part of the assay data. Routine validation checks run on the desurveyed drillhole file and identified that some intervals outside the microgranite and significant mineralisation were unsampled.

17.1.2 Geological Modelling

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 three domains comprised:

- veining (based on geological logging)
- sulphides (based on geological logging)
- gold domain (based on assays).

Using the drillhole data, the probability of a model cell containing veining, sulphides or gold mineralisation was estimated using a preliminary indicator model with estimation parameters based on initial variography (Section 17.1.7.1).

To assist in the indicator estimation a preliminary statistical review was carried out on:

- all the drillhole data
- drillhole data confined with the microgranite wireframe.

17.1.2.1 Veining

Log probability plots and histograms were generated for veining in all the drillholes (Figures 17.1 and 17.2) and veining only within the microgranite (Figures 17.3 and 17.4). The log probability plots show some instability between 0.8% and 3.0% veining and also a rapid drop in observed vein content below 1.0%. The drop below 1% veining is probably attributable to the inability to accurately visually estimate the amount of veining below 1%.

Table 17.1 details the percentage of logged intervals identified to contain veining. The majority of logged intervals within the microgranite contain more than 1.0% total veining.

Table 17.1 Percentage of Veining in Logged Intervals

Domain	Veining (%)	Logged Intervals with Veining (%)
All Drilling	0	41
	0-0.99	8
	≥1.0	51
Within Microgranite	0	26
	0-0.99	5
	≥1.0	69

To estimate the vein indicator, an indicator of 0 was assigned for all logged intervals containing less than 1% veining and an indicator of 1 for all logged intervals containing more than 1% veining.

Figure 17.1 Log Probability Plot of Veining for all Drilling

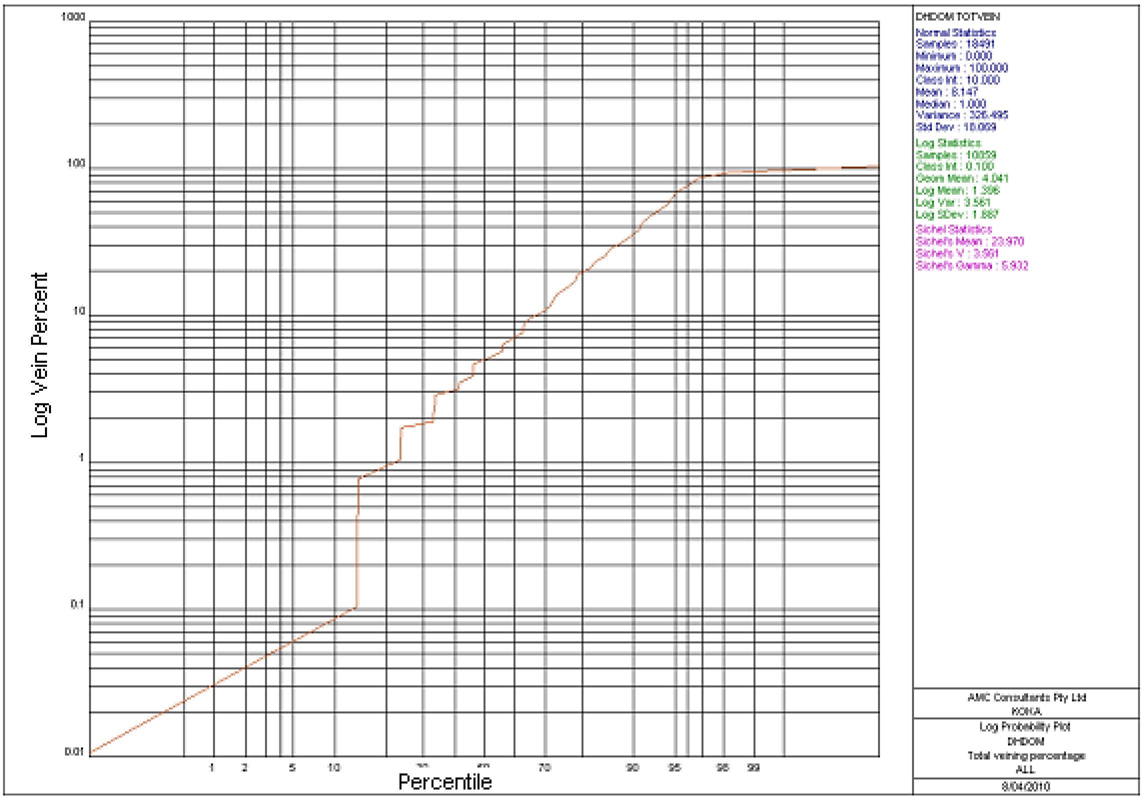


Figure 17.2 Histogram of Veining for all Drilling

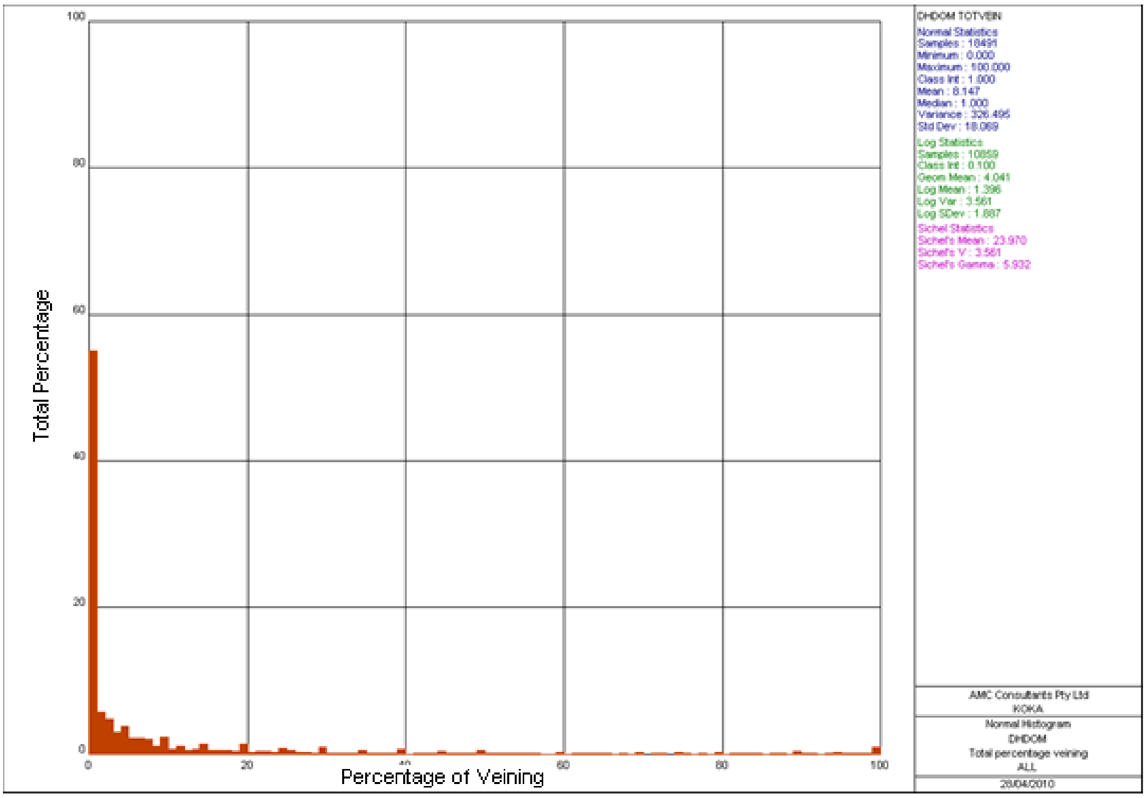


Figure 17.3 Log Probability Plot of Veining within Microgranite

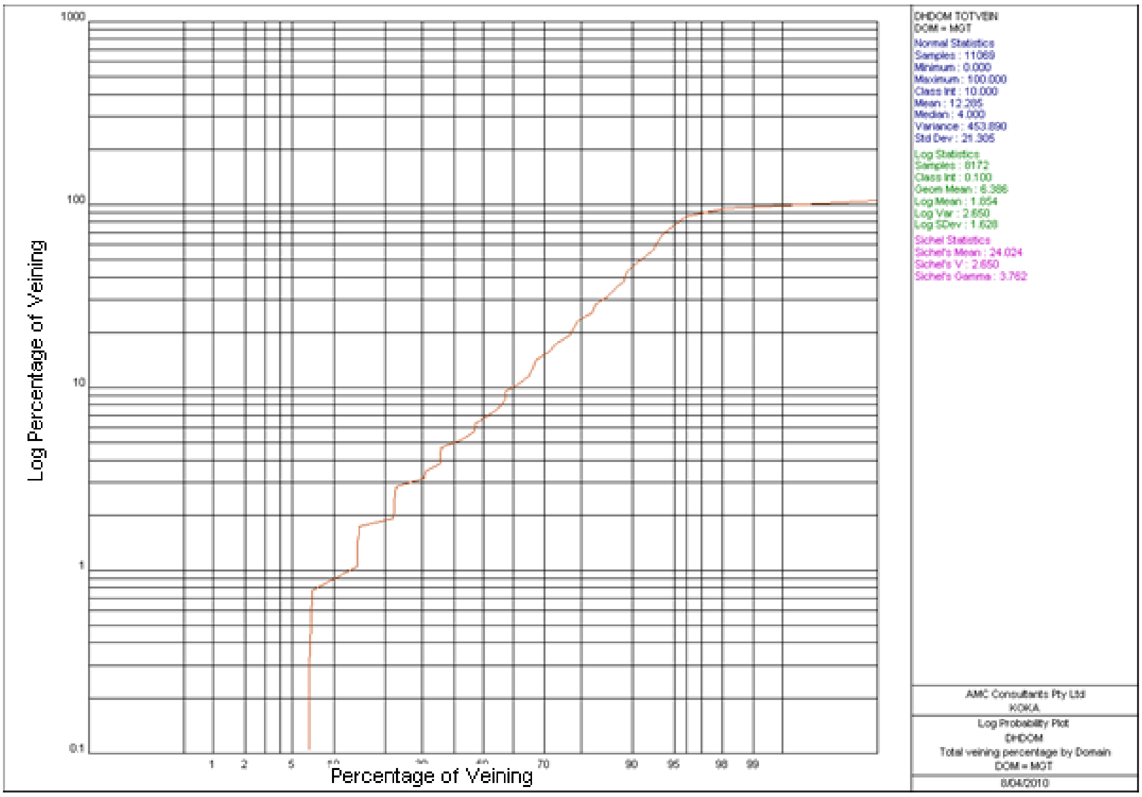
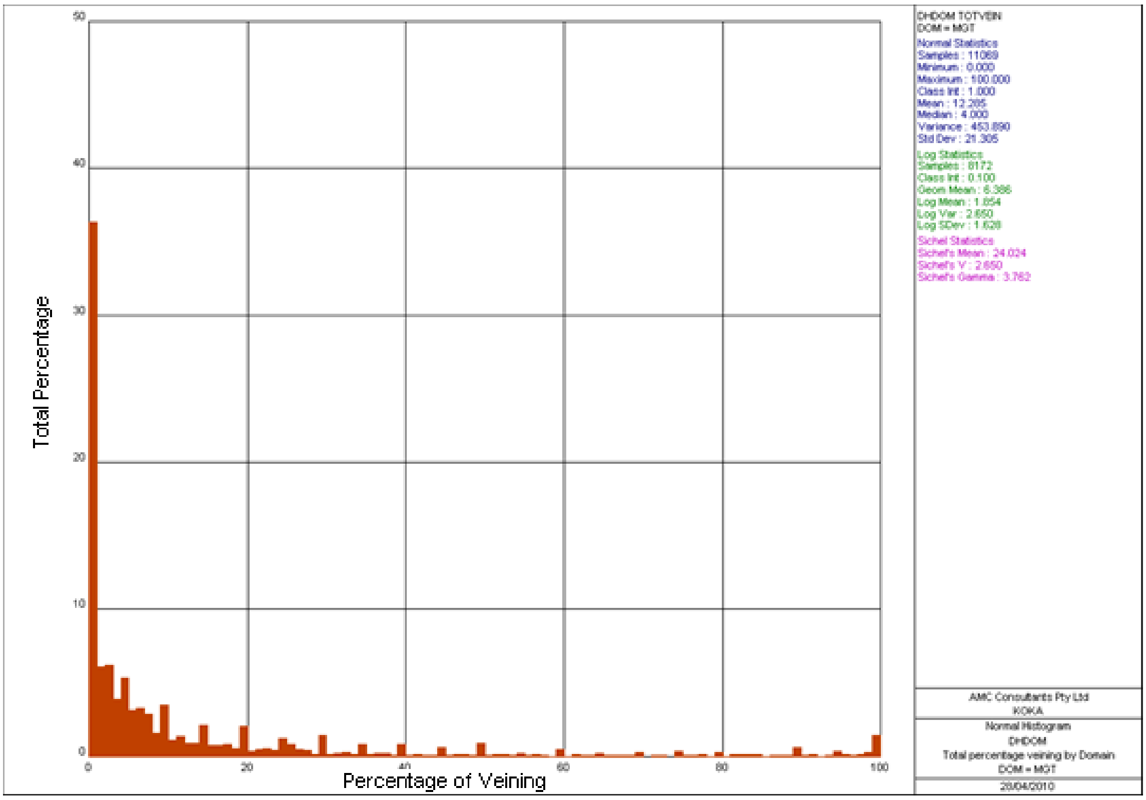


Figure 17.4 Histogram of Veining within Microgranite



17.1.2.2 Sulphides

Log probability plots and histograms were generated for total sulphides in all the drillholes (Figures 17.5 and 17.6) and total sulphides only within the microgranite (Figures 17.7 and 17.8).

Figure 17.5 Log Probability Plot of Sulphides for all Drilling

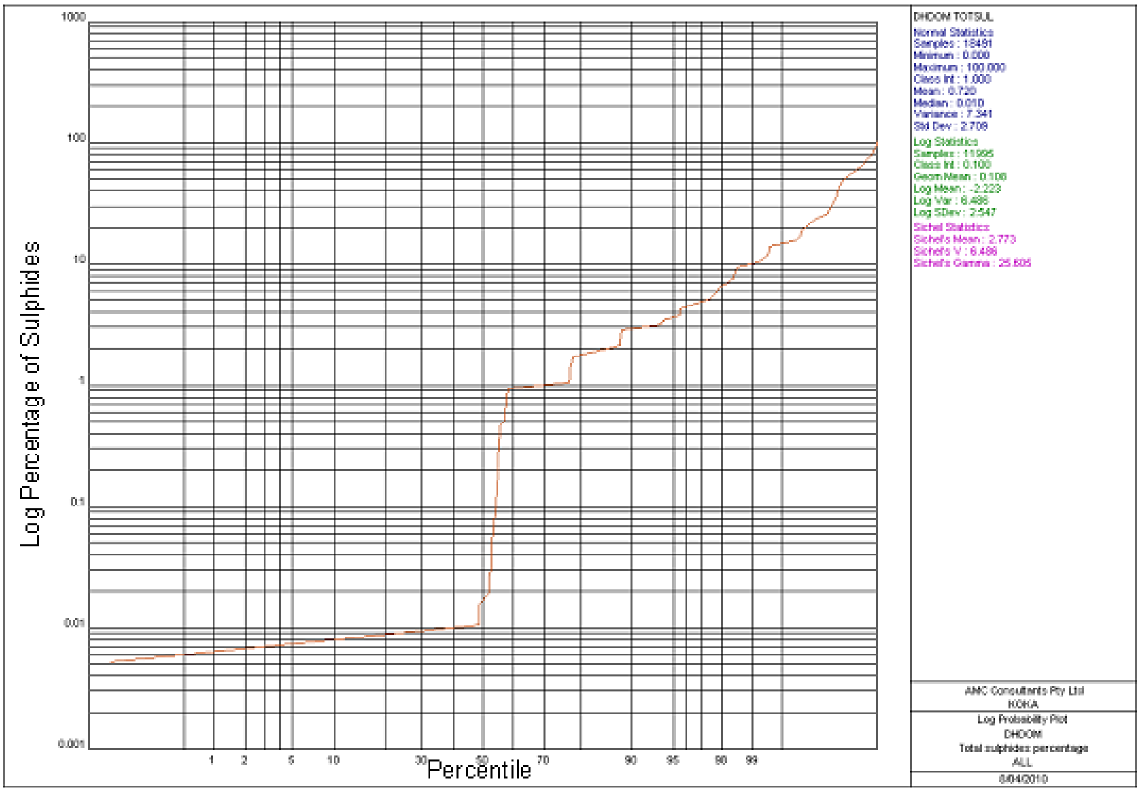


Figure 17.6 Histogram of Sulphides for all Drilling

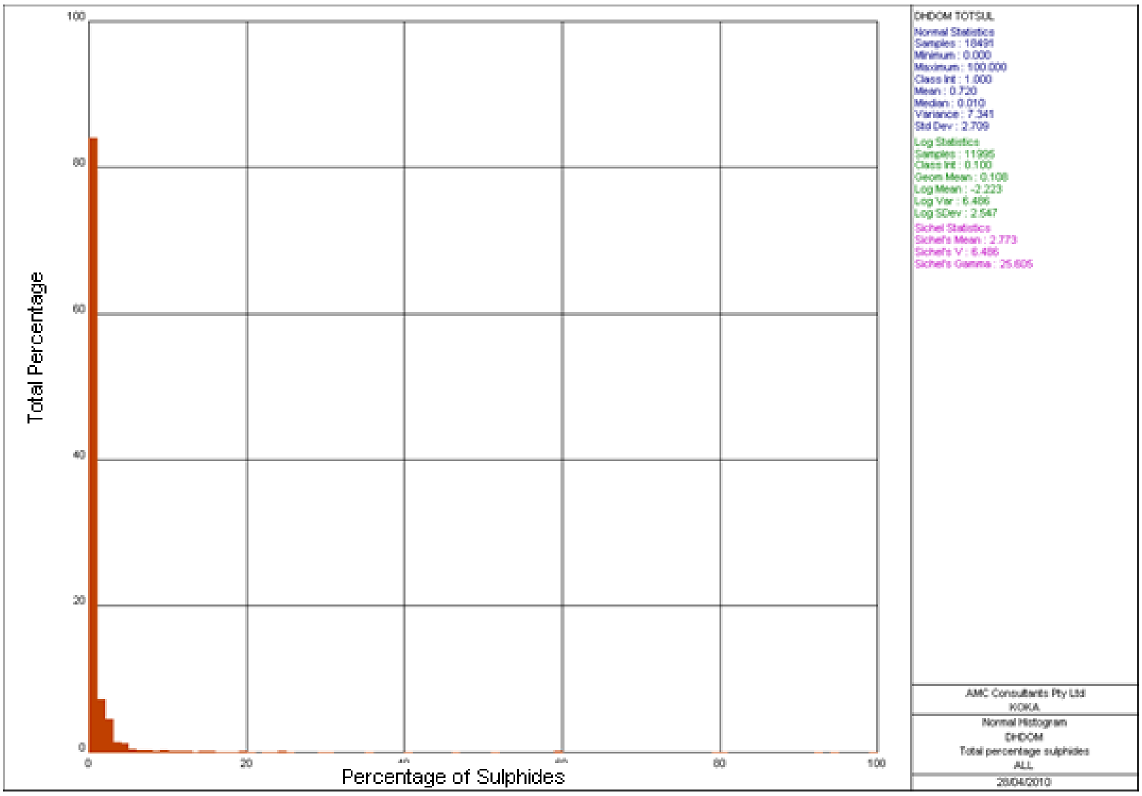


Figure 17.7 Probability Plot of Sulphides within Microgranite

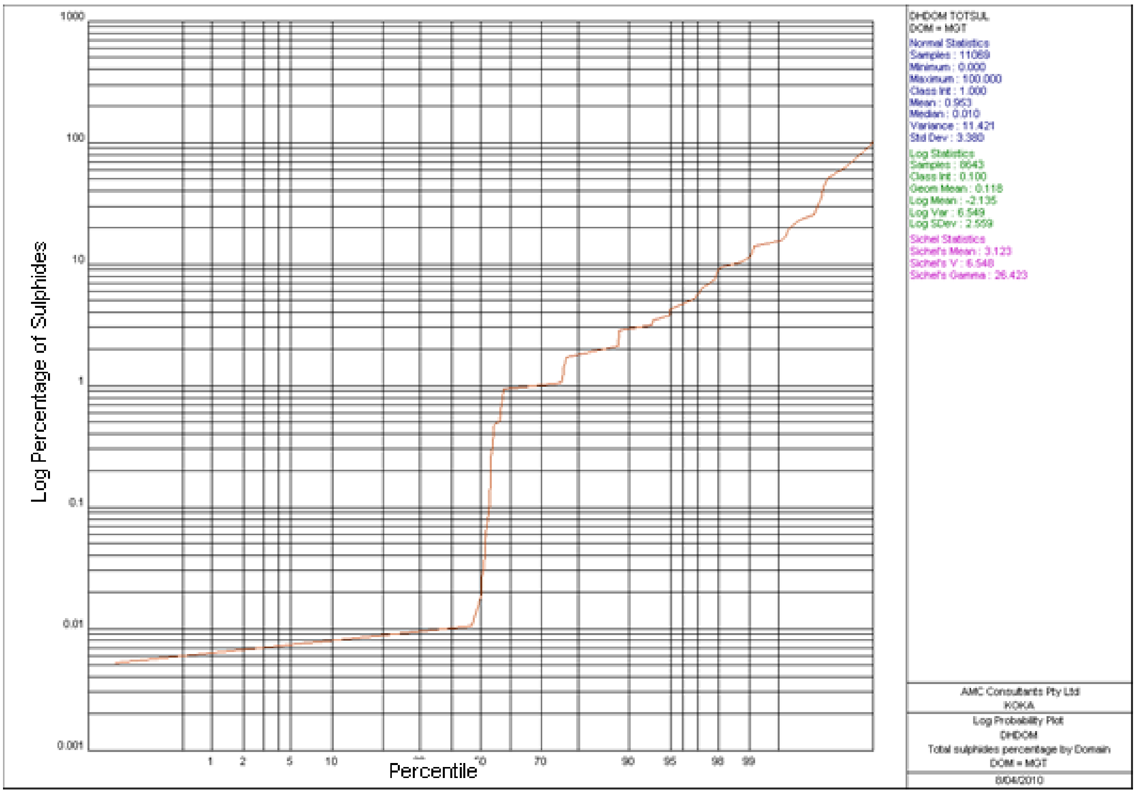
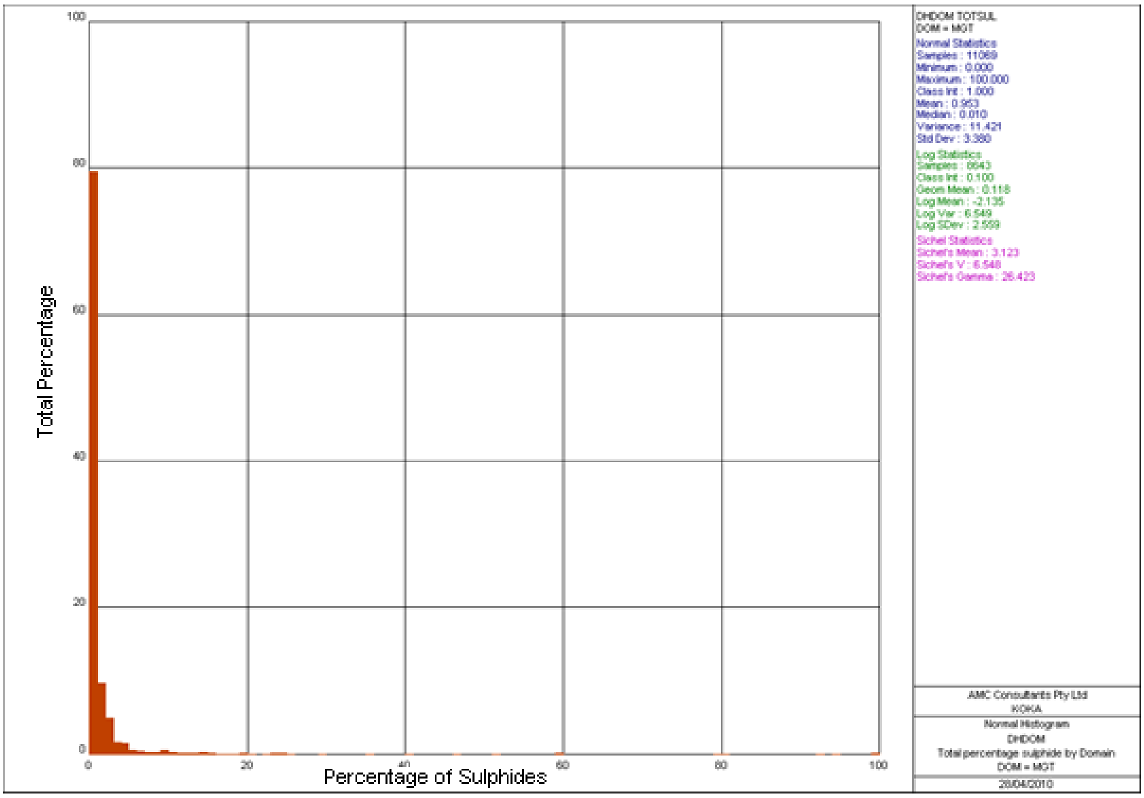


Figure 17.8 Histogram of Total Sulphides within Microgranite



The log probability plots shows some instability between 1% and 3% and also a rapid drop in observed total sulphide content below 1%.

The drop below 1% total sulphide content is probably attributable to the inability to accurately visually estimate the amount of sulphide mineralisation below 1%.

Table 17.2 details the percentage of logged intervals identified to contain sulphides. The percentage of logged intervals within the microgranite that contain 1% or more of total sulphides (33%) is significantly lower than the 69% of intervals that contain 1% or more veining.

Table 17.2 Percentage Of Total Sulphides in Logged Intervals

Domain	Veining (%)	Logged Intervals with Sulphides (%)
All Drilling	0	35
	0-0.99	38
	≥1.0	27
Within Microgranite	0	22
	0-0.99	45
	≥1.0	33

To estimate the sulphide indicator, an indicator of 0 was assigned for all logged intervals containing less than 1% total sulphides and an indicator of 1 for all logged intervals containing more than 1% sulphides.

17.1.2.3 Gold Assays

13,709 gold assay data were available for preliminary statistical review. Figures 17.9 to 17.12 show log probability and histogram plots for all the assays and also for the assays only within the microgranite with the histograms constrained to an upper value of 25 g/t Au. 183 (1.3%) gold assays are higher than 25 g/t Au. 177 of these are hosted within the microgranite. 84% of all gold grades are less than 0.3%.

The log probability plots for gold show a very slight inflection between 0.2 g/t Au and 0.3 g/t Au. A lower cut-off of 0.3 g/t Au was used for the indicator model. An indicator of 0 was assigned to assays less than 0.3 g/t Au and an indicator of 1 was assigned to assays greater than 0.3 g/t Au.

Figure 17.9 Log Probability Plot of Gold Assays for all Drilling

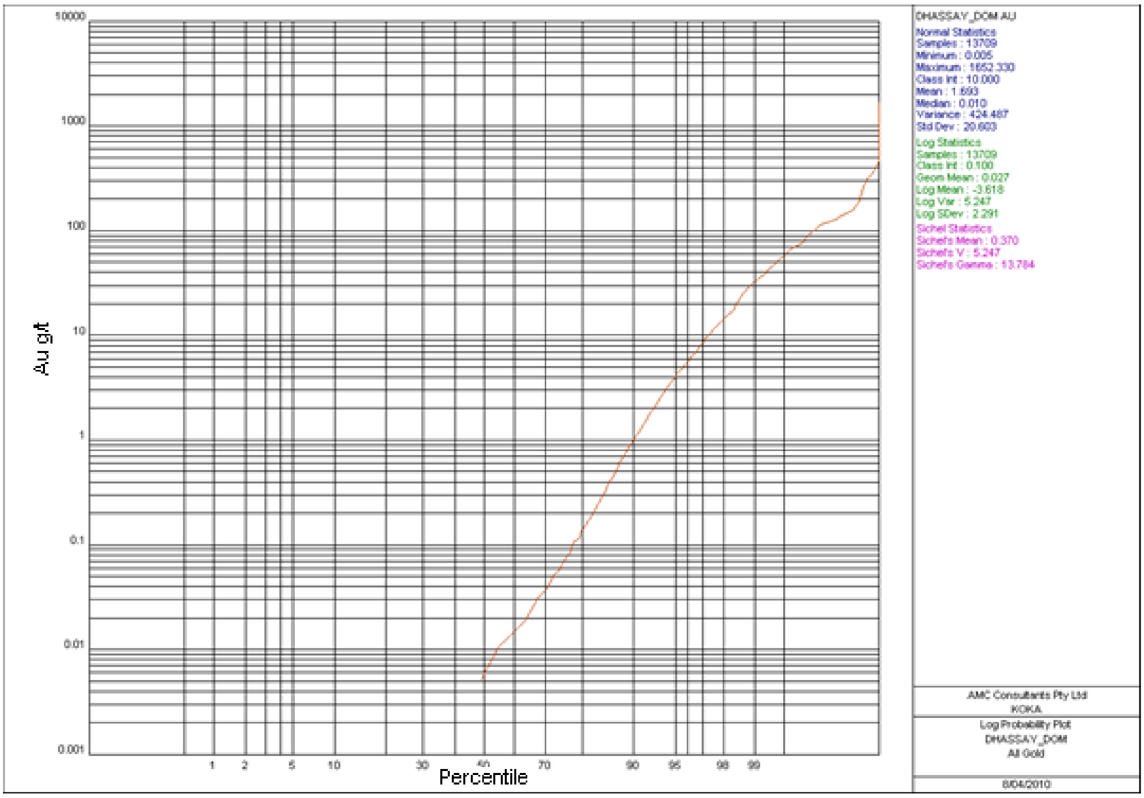


Figure 17.10 Histogram of Gold Assays for all Drilling

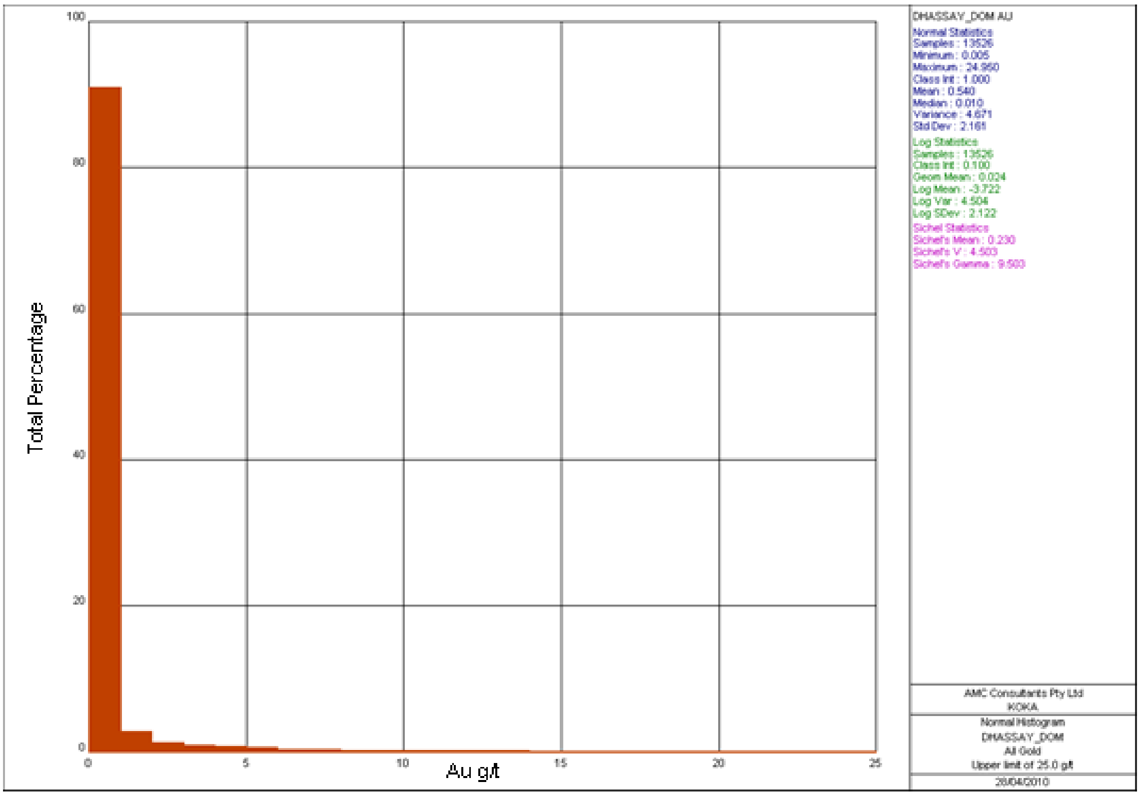


Figure 17.11 Log Probability Plot of Gold Assays within Microgranite

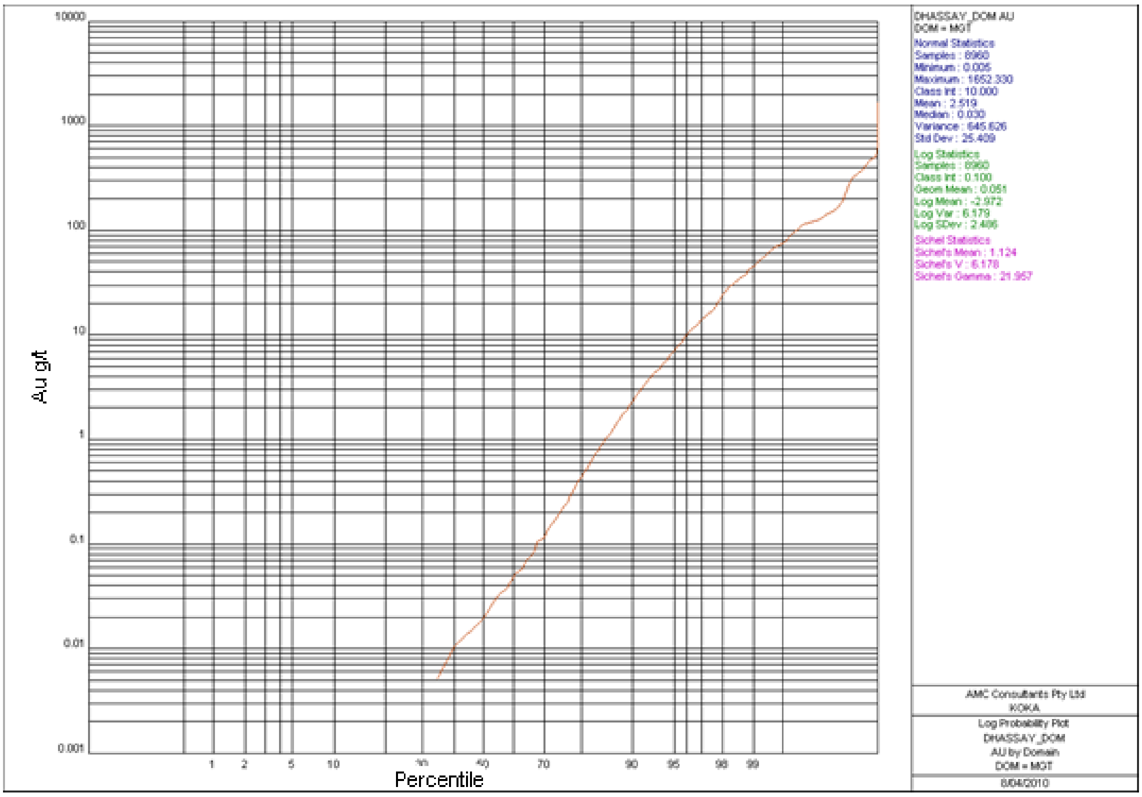
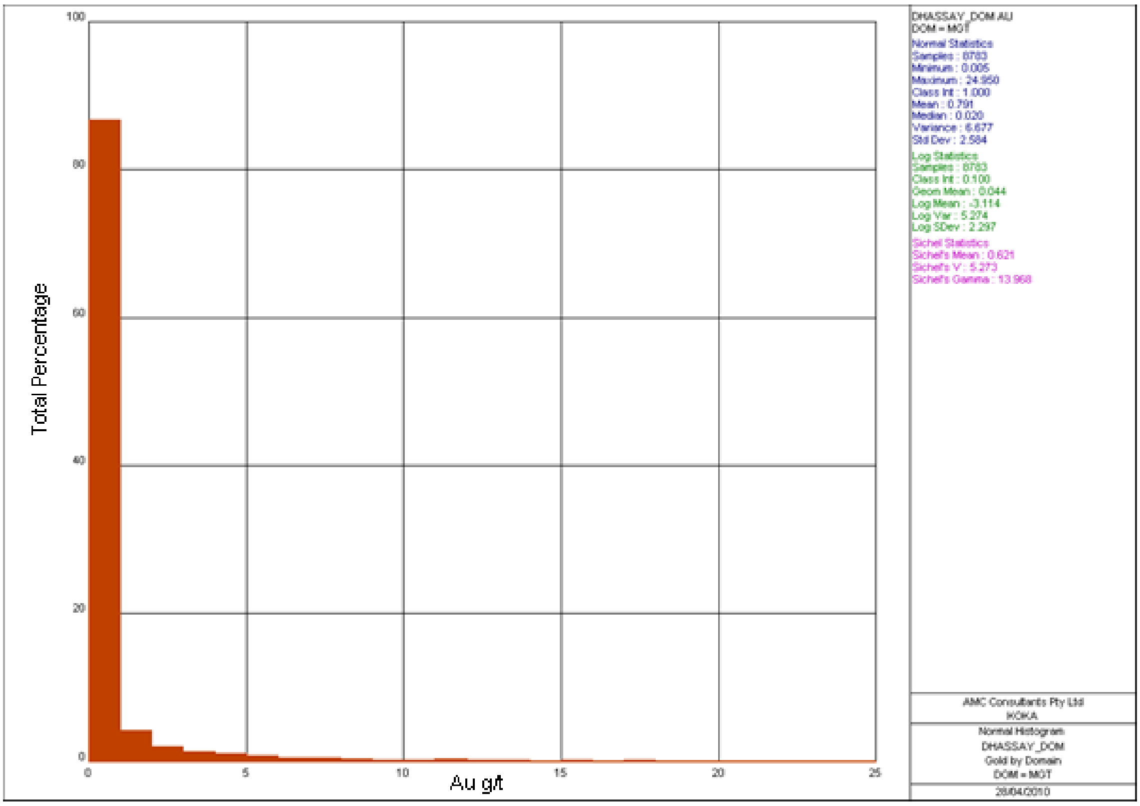


Figure 17.12 Histogram of Gold Assays within Microgranite



17.1.2.4 Interpretation

Indicator values between 0 and 1 for veining, total sulphides and gold were estimated into the block model using inverse distance to the power of two. A series of domain boundaries were interpreted constrained by the microgranite wireframe based on the indicator models, drillhole logging and assay data. The domain boundaries overlap.

Subsequently, only the gold (Figure 17.13) and sulphide (Figure 17.14) domains were used for the resource estimate as the vein domain dominated the microgranite and it was apparent that there was extensive veining in microgranite that contained waste gold grade. The gold and sulphide domains were combined for grade estimation.

An oxidation profile was also interpreted, however the depth of weathering is shallow, consistent with the bulk density data, and the oxidised/fresh rock boundary was not distinguished in grade estimation.

Figure 17.13 Section 9820N Showing Interpreted Gold Domain Wireframe Slice Looking North

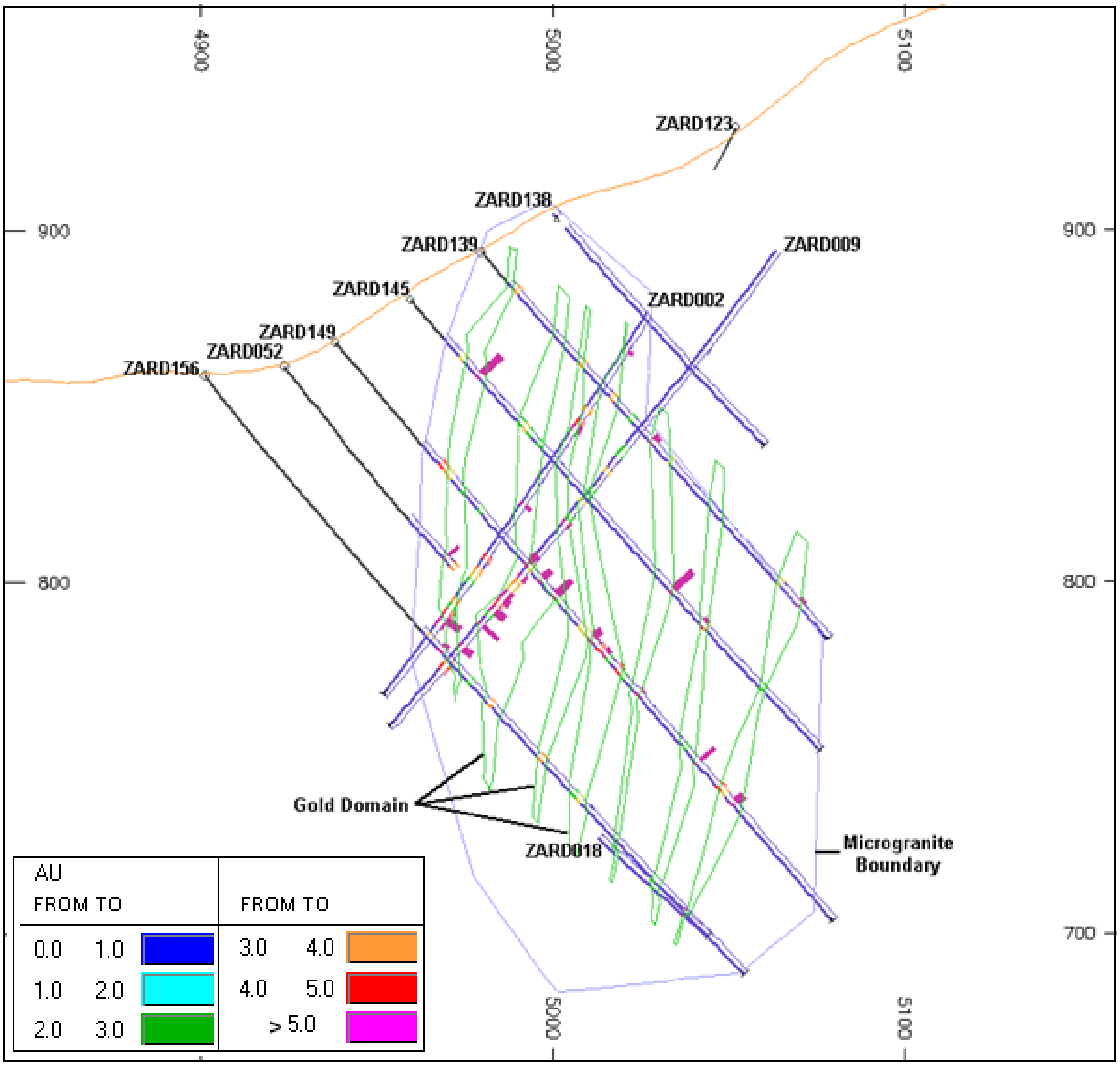
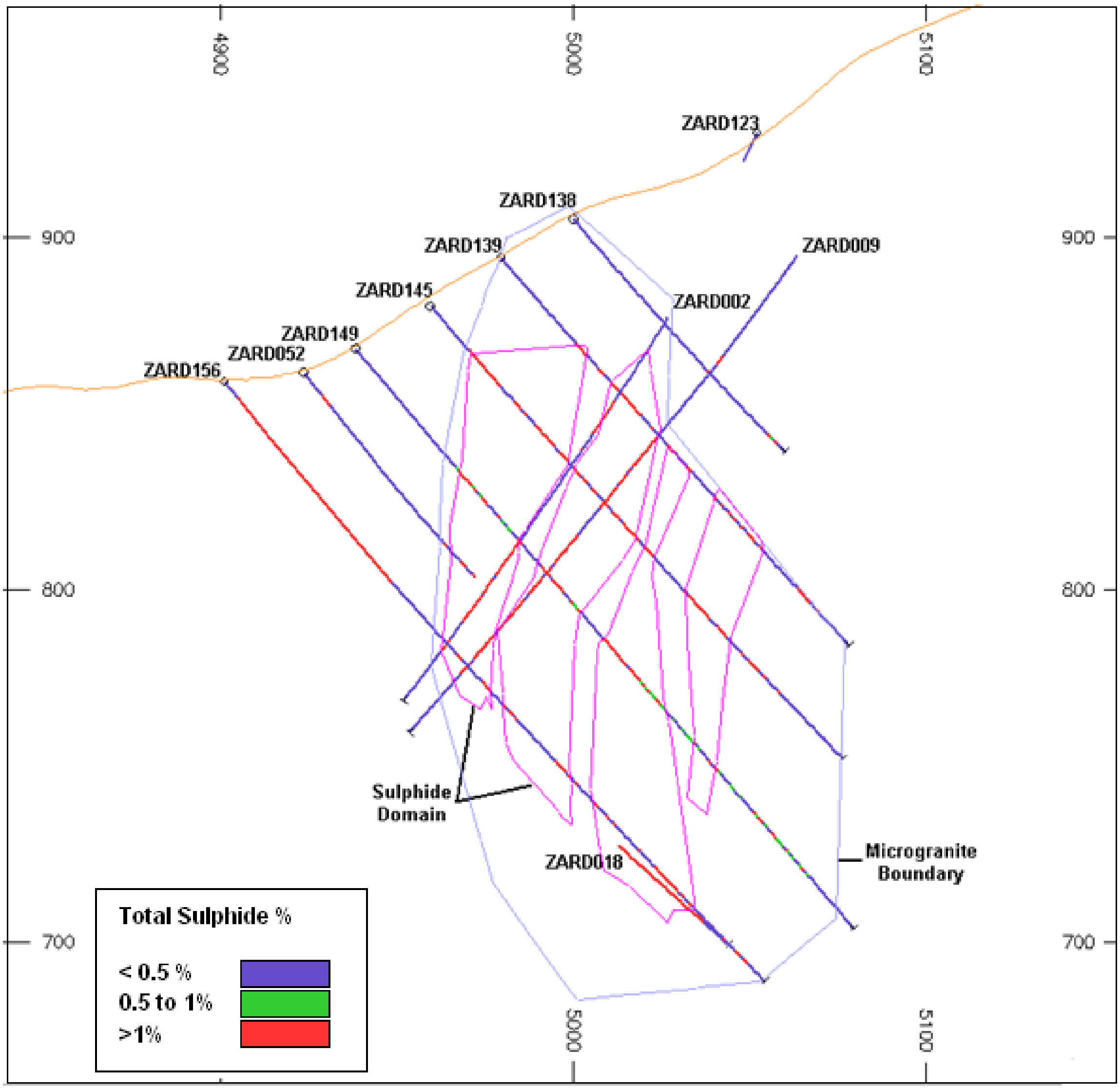


Figure 17.14 Section 9820N Showing Interpreted Sulphide Domain Wireframe Slice, Looking North



17.1.3 Descriptive Statistics

The interpreted gold and sulphide wireframes were used to flag the drillhole data with a numeric domain code to allow statistical evaluation and grade estimation. Where gold and sulphide wireframes overlapped, the domain code of the gold domain was retained.

Table 17.3 shows the summary statistics for gold and sample length for the domained sample data. The coefficient of variance (CV) for gold is very high in all domains, which is not uncommon for a stockwork gold deposit.

Table 17.3 Summary Descriptive Statistics - Domained Samples

Domained Samples									
Domain	Variable	Samples	Minimum	Maximum	Mean	Median	Std Dev	Variance	CV
Gold	Au	2818	0.005	1652.3	8.04	0.75	44.9	2015.9	5.59
	Length	2818	0.6	4.1	1.01	1	0.1	0.0	0.12
Sulphide	Au	2860	0.005	71.3	0.25	0.02	2.0	4.1	8.25
	Length	2860	0.55	3.0	1.00	1	0.1	0.0	0.05
Background	Au	4455	0.005	11.0	0.07	0.005	0.4	0.1	5.27
	Length	4457	0.4	5.0	1.01	1	0.1	0.0	0.14

The drillhole data were composited to a regular sample length to provide equal sample support for estimation. Sample lengths of the raw data vary from 0.4m to 5m, but are most commonly 1m. Of the 2,818 samples in the gold domain, only 35 (1.2%) are less than or greater than 1m in length. Compositing to 1m will have no effect on reducing the CV. A composite sample length of 2m was selected to allow for a reduction in the CV and retain sufficient data for a robust grade estimate.

Approximately 20% of drillhole intercepts for the gold domain are narrower than two metres and a composite length greater than two metres would result in more than 30% of composite lengths being less than the nominated length. By using 2m composites, only 16% of composite lengths were less than the 2m. Most of these samples are at the downhole edge of each intercept and are generally masked in the estimation process by full length composites. The compositing process was checked by calculating the total gold metal in the samples prior to and after compositing. There was no net loss or gain of gold metal by compositing.

Table 17.4 lists summary statistics for domained assays and composites.

Table 17.4 Summary Descriptive Statistics for Domained Assays and Composites

Domained Samples									
Domain	Variable	Samples	Minimum	Maximum	Mean	Median	Std Dev	Variance	CV
Gold	Au	2818	0.005	1652.3	8.04	0.75	44.9	2015.9	5.59
	Length	2818	0.6	4.1	1.01	1	0.1	0.0	0.12
Sulphide	Au	2860	0.005	71.3	0.25	0.02	2.0	4.1	8.25
	Length	2860	0.55	3.0	1.00	1	0.1	0.0	0.05
Background	Au	4455	0.005	11.0	0.07	0.005	0.4	0.1	5.27
	Length	4457	0.4	5.0	1.01	1	0.1	0.0	0.14

2m Composites									
Domain	Variable	Samples	Minimum	Maximum	Mean	Median	Std Dev	Variance	CV
Gold	Au	1544	0.005	851.5	7.81	1.54	31.4	984.8	4.02
	Length	1544	0.2	2.0	1.84	2	0.4	0.1	0.20
Sulphide	Au	1537	0.005	36.2	0.25	0.033	1.6	2.5	6.44
	Length	1537	0.05	2.0	1.86	2	0.3	0.1	0.19
Background	Au	2376	0.003	5.5	0.08	0.013	0.3	0.1	3.88
	Length	2376	0.1	2.0	1.90	2	0.3	0.1	0.16

Figures 17.15 to 17.17 show gold grade statistics plots for composites in the gold domain and Figure 17.18 shows the log histogram for composite length. The 2m composites have a near log normal distribution with a few high-grade outliers. There does not appear to be multiple grade populations and there is no high-grade population that can be domained. This indicates the need for topcutting for the gold domain.

Figure 17.15 Histogram of Gold Grades, 2m Composites, Gold Domain

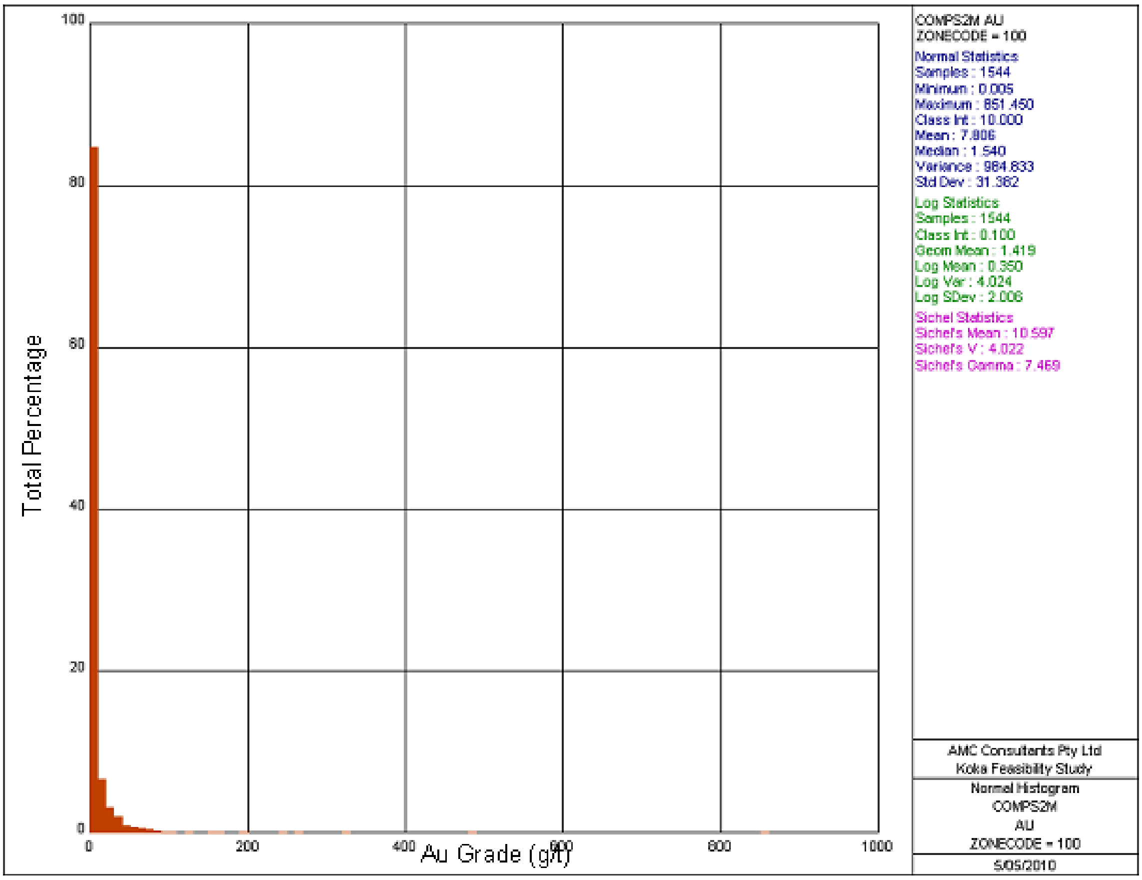


Figure 17.16 Log Histogram of Gold Grades, 2m Composites, Gold Domain

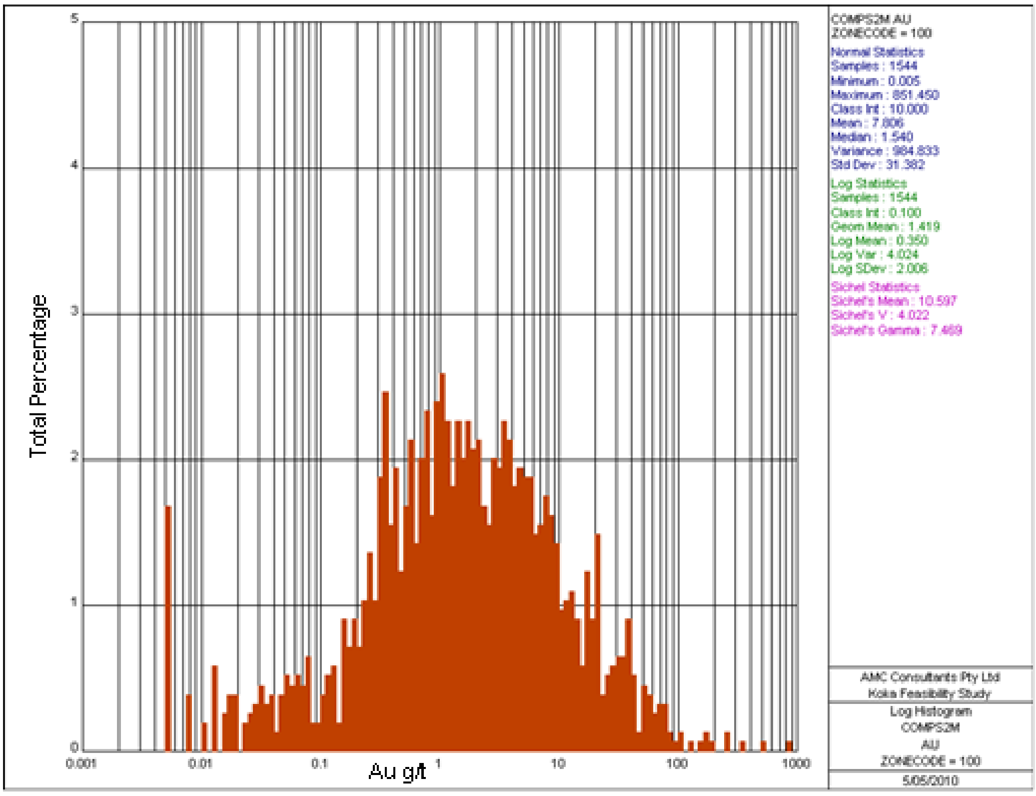


Figure 17.17 Log Probability Plot of Gold Grades, 2m Composites, Gold Domain

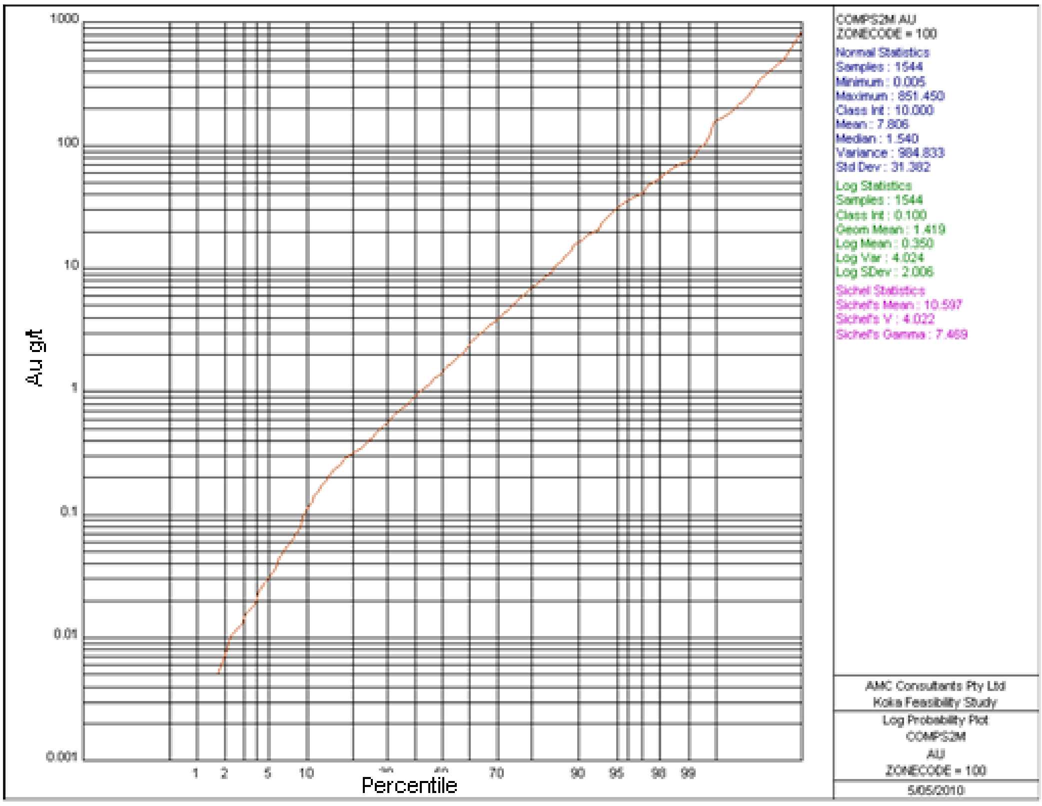
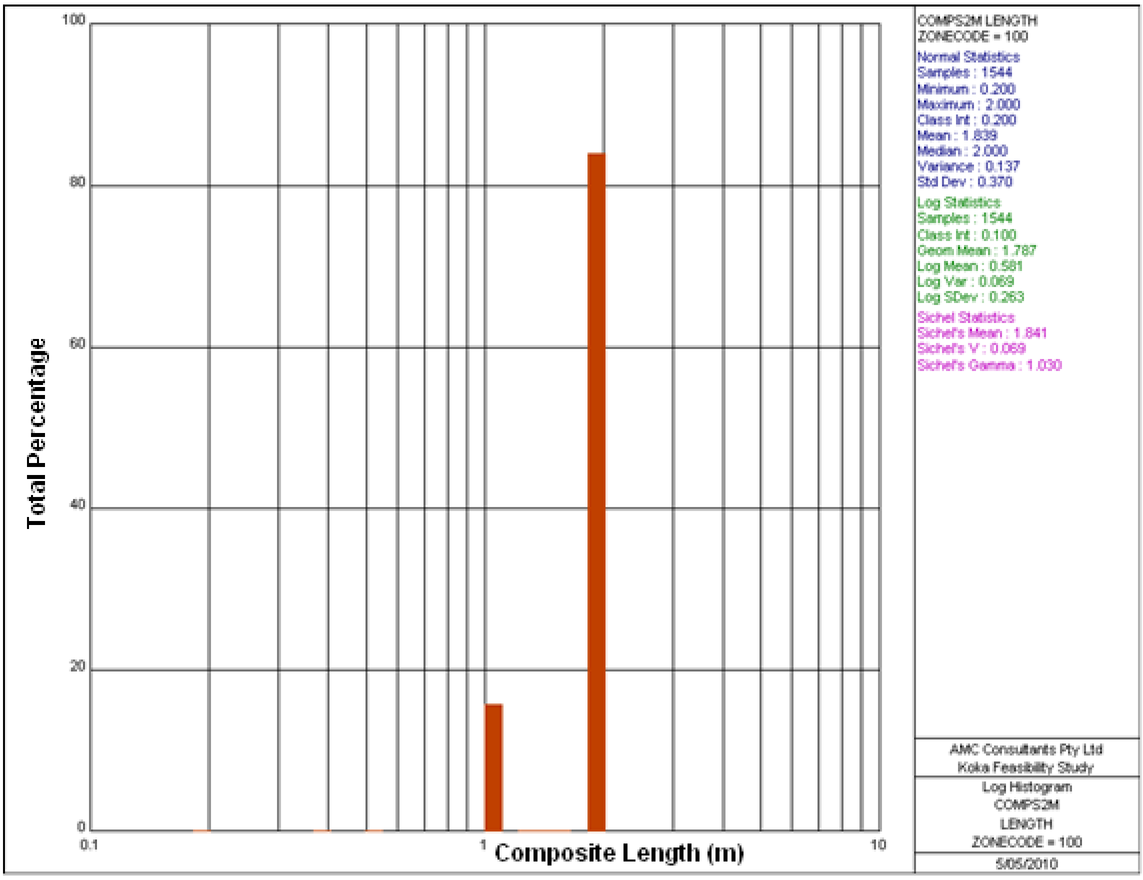


Figure 17.18 Log Histogram of Composite Length, 2m Composites, Gold Domain



17.1.4 Topcuts

Gold composite statistics were analysed to determine if any domain had high-grade outliers that would need to be cut. Topcuts are applied to outliers to avoid undue influence during grade estimation. Values above the topcut value are reset to that value. Topcuts were chosen where the population distribution changed from the primary to the outlier population. These values were initially chosen as points on the log probability plots where the slope of the curve changed dramatically. Figures 17.16 and 17.17 in Section 17.1.3 show a break in the data population at a value of approximately 150 g/t Au to 200 g/t Au.

Possible topcuts were checked against histogram plots, the loss in metal after topcutting and the proportion of the composites affected by the topcut. The composites above the topcut were viewed on screen with the wireframes to check for clustering to ensure that the composites were outliers. Adjustments were made to the topcut values as required. The final gold topcut of 200 g/t Au for the gold domain, represents the 99.7th percentile of the data and resulted in a drop of approximately 10% in gold metal content. Most of this metal is contained in the top two outlier composites of 489.39 g/t Au and 851.45 g/t Au.

A topcut was only applied to the gold domain. Table 17.5 shows summary statistics for the topcut composite gold grades.

Table 17.5 Summary Statistics for Gold for Topcut Composites - Gold Domain

Topcut 2m Composites									
Domain	Variable	Samples	Minimum	Maximum	Mean	Median	Std Dev	Variance	CV
Gold	Au	1544	0.005	200.0	7.04	1.54	18.6	345.9	2.64

The results show a considerable drop in the CV, which is important for the estimation process. Considering that the raw data for this domain had a CV of 5.6, this value shows that after compositing and topcutting, that data are suitable for use in an ordinary kriged estimate.

Figures 17.19 to 17.21 show statistics plots for the topcut 2m composites for the gold domain.

Figure 17.19 Histogram of Gold Grades, 2m Composites with 200 g/t Au Topcut, Gold Domain

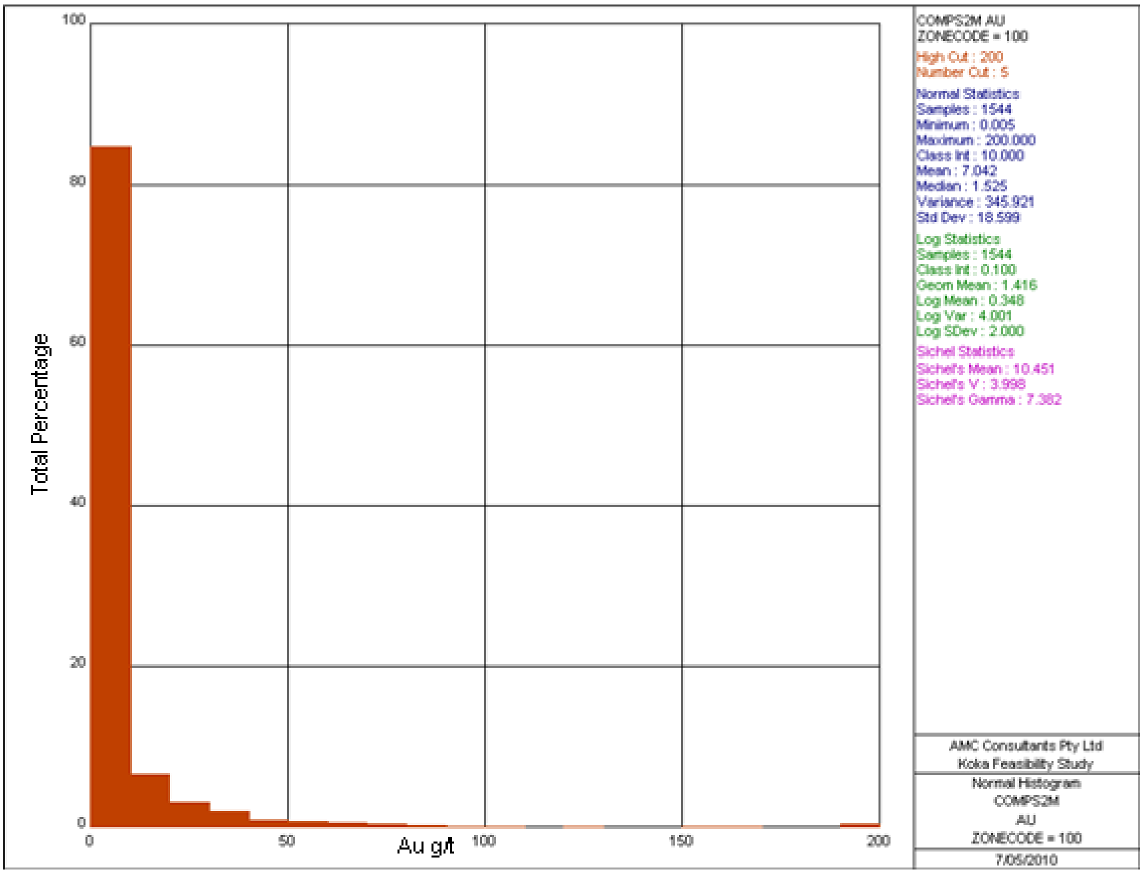


Figure 17.20 Log Histogram of Gold Grades, 2m Composites with 200 g/t Au Topcut, Gold Domain

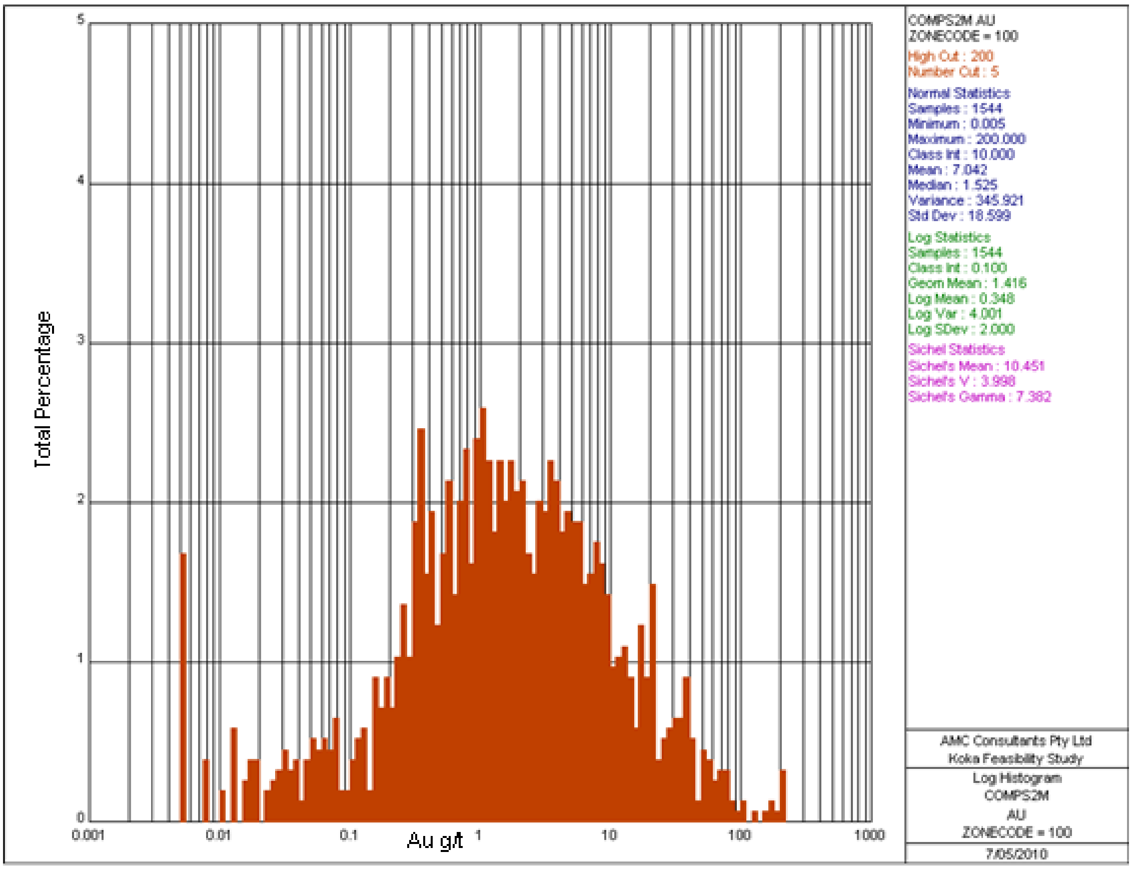
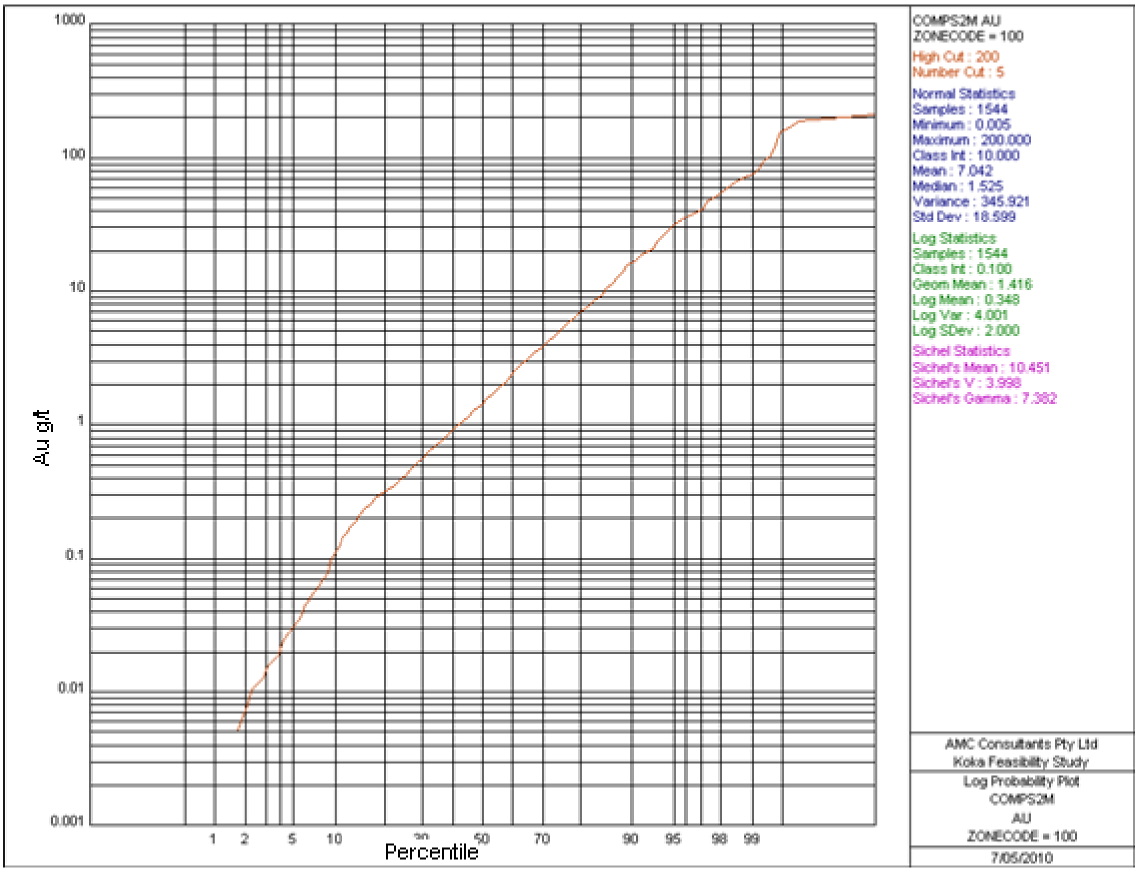


Figure 17.21 Log Probability Plot of Gold Grades, 2m Composites with 200 g/t Au Topcut, Gold Domain



17.1.5 Volume Modelling

A block model (volume model) was created using the wireframes that were used for sample domaining. Each domain wireframe was filled with cells as defined by the model prototype. The parent cells were allowed to split into sub-cells to honour the shape and volume of the domain wireframes. The amount of splitting was optimised to maintain the correct interpreted wireframe volume, without producing excessive sub-cells in the volume model.

The final choice of parent cell size was based upon the drillhole spacing, the descriptive statistics and variography (the measures of local and global variance). The central to northern area of the deposit is the widest part of the mineralisation and has the most closely-spaced drilling. The area from approximately 9600 mN to 9850 mN is drilled at 20m x 20m spacing in easting and northing. The remainder of the deposit, to the north and south along strike of this central area, is drilled at 20m x 40m spacing in easting and northing.

As nearly half the deposit, at the widest area of interpreted mineralisation, is drilled at the 20m x 20m spacing, a block size of 10m x 10m in easting and northing (half the drillhole spacing) is considered to be appropriate. This decision was supported by quantitative kriging neighbourhood analysis (described in Section 17.1.8). The 5m parent cell height reflected possible mining bench height.

Table 17.6 lists the final volume model prototype parameters.

Table 17.6 Volume Model Prototype

Axis	Origin (Local Grid Coordinates)	Parent Cell Dimensions (m)	Number of Parent Cells	Smallest Sub-Cell Size (m)
X	4810 mE	10	42	0.333
Y	9365 mN	10	68	0.333
Z	620 mRL	5	75	0.001

The empty block model was flagged for the domain to match the composites.

The volume model was visually validated by checking the position and the shape of the parent and sub-cells against the wireframes. Each domain in the volume model was also quantitatively validated by comparing the model domain volume to the corresponding wireframe volume.

17.1.6 Density

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 the mean of bulk density determinations from drillcore.

17.1.7 Variography

17.1.7.1 Initial Variography

Initial variography was performed on all data within the microgranite wireframe. This was done to understand general trends in the mineralisation and to aid in the geological interpretation. AMC performed directional variography using median indicator variography and produced variogram fans for the horizontal and vertical planes of the deposit.

The variography showed a direction of greatest continuity with a strike of approximately 350 and a dip of approximately 75º to 80º to the west (Figures 17.22 and 17.23).

Figure 17.22 Horizontal Variogram Fan, Raw Data Within Microgranite, 0.05 g/t Au Indicator

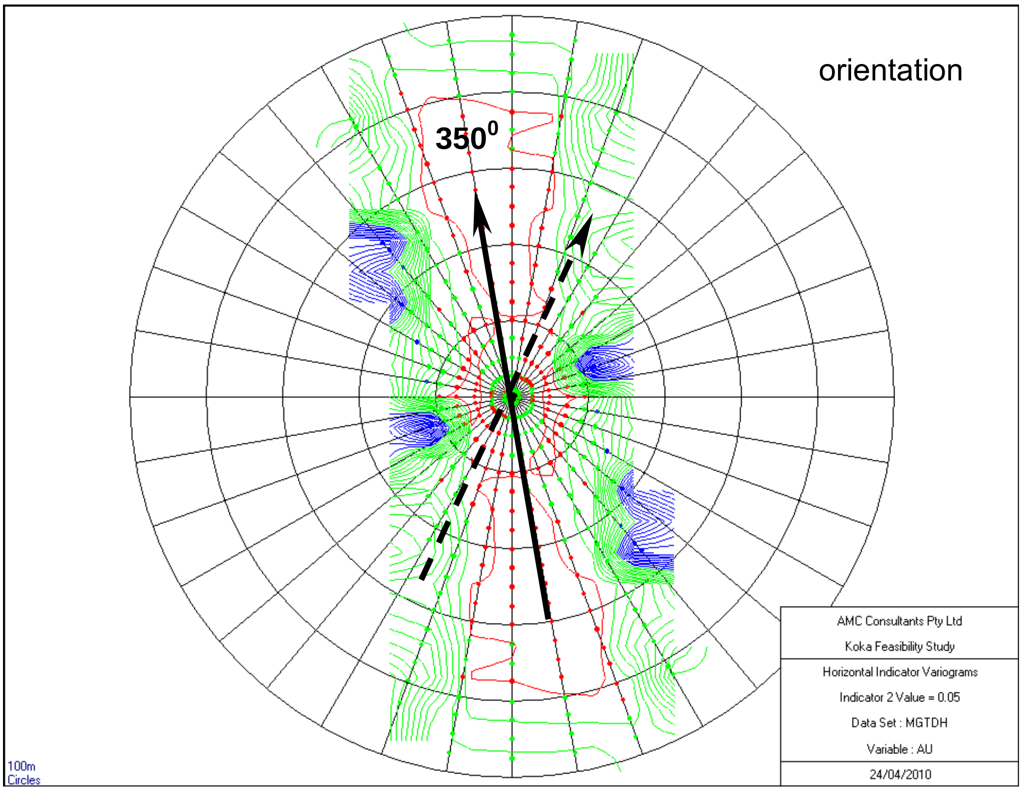
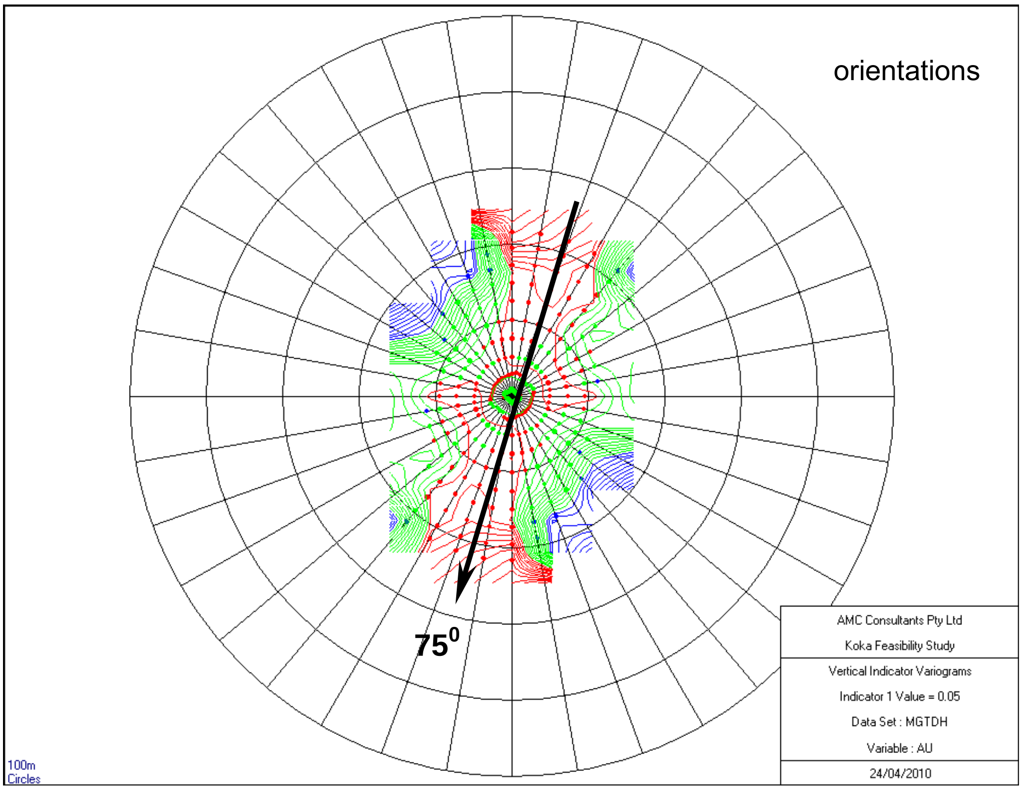


Figure 17.23 Vertical Variogram Fan, Raw Data Within Microgranite, 0.05 g/t Au Indicator

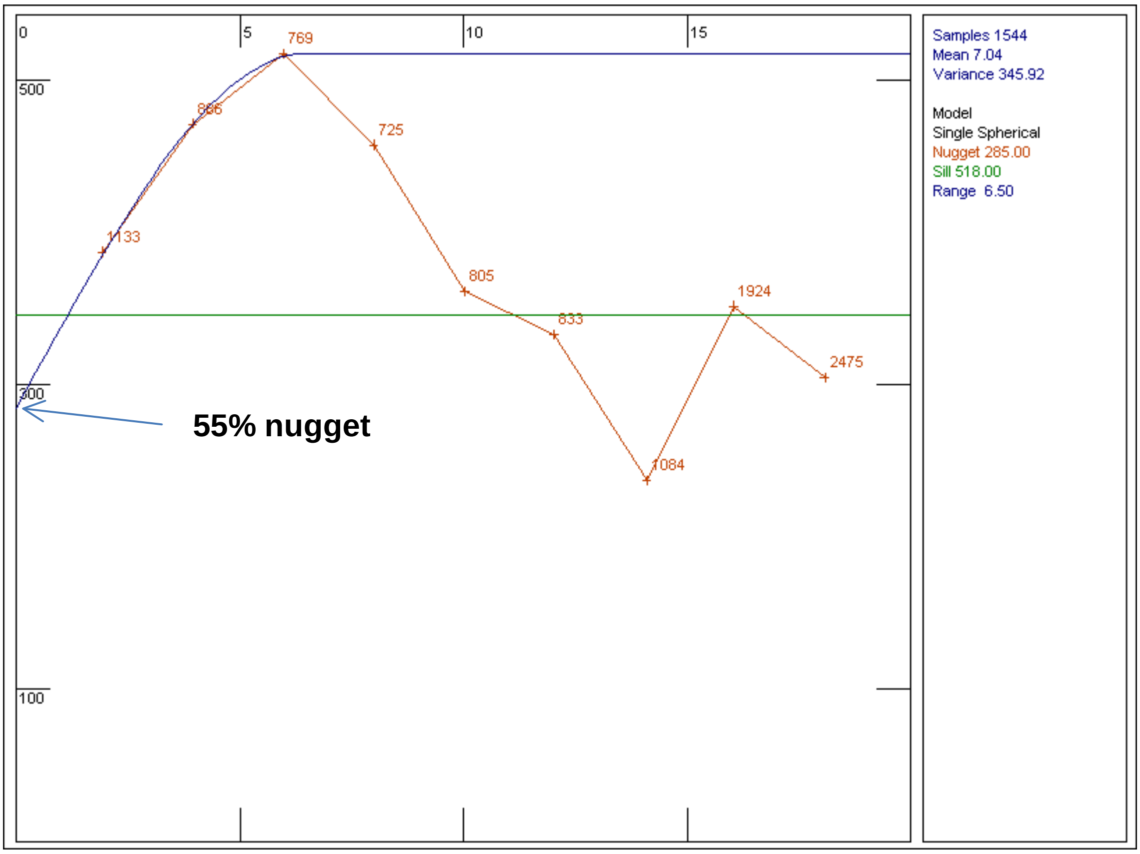


17.1.7.2 Domainal Variography

Experimental variography was performed on the composite gold values to determine the best estimation strategy. Variograms were only produced for the gold domain as the gold and sulphide domains were to use the same variography during grade estimation and the microgranite was essentially waste grade.

Down-hole variography was undertaken to estimate the nugget. Figure 17.24 shows the down-hole variogram produced using a 2m lag distance. The nugget value for the gold domain is indicated as 55%. This is a high nugget and suggests that much closer spaced sampling will be required to reliably define the mineralisation boundaries prior to mark out for mining.

Figure 17.24 Downhole Variogram, 2m Cut Composites, Gold Domain



Directional variography was performed using the nugget derived from the down-hole direction. Generally the variograms were of poor quality. Horizontal and vertical variogram fans were produced to determine the plane of greatest gold grade continuity using a lag distance of 30m. A plane striking towards 175 and dipping 85º to the west was chosen as the major axis direction and a vertical variogram fan in this plane was used to determine the semi-major axis direction. The major and semi-major axis variograms were modelled from the experimental variograms (Figures 17.25 and 17.26). The minor direction variogram is perpendicular to this plane (Figure 17.27).

Figure 17.25 Directional Variogram, Major Axis, 2m Cut Composites, Gold Domain

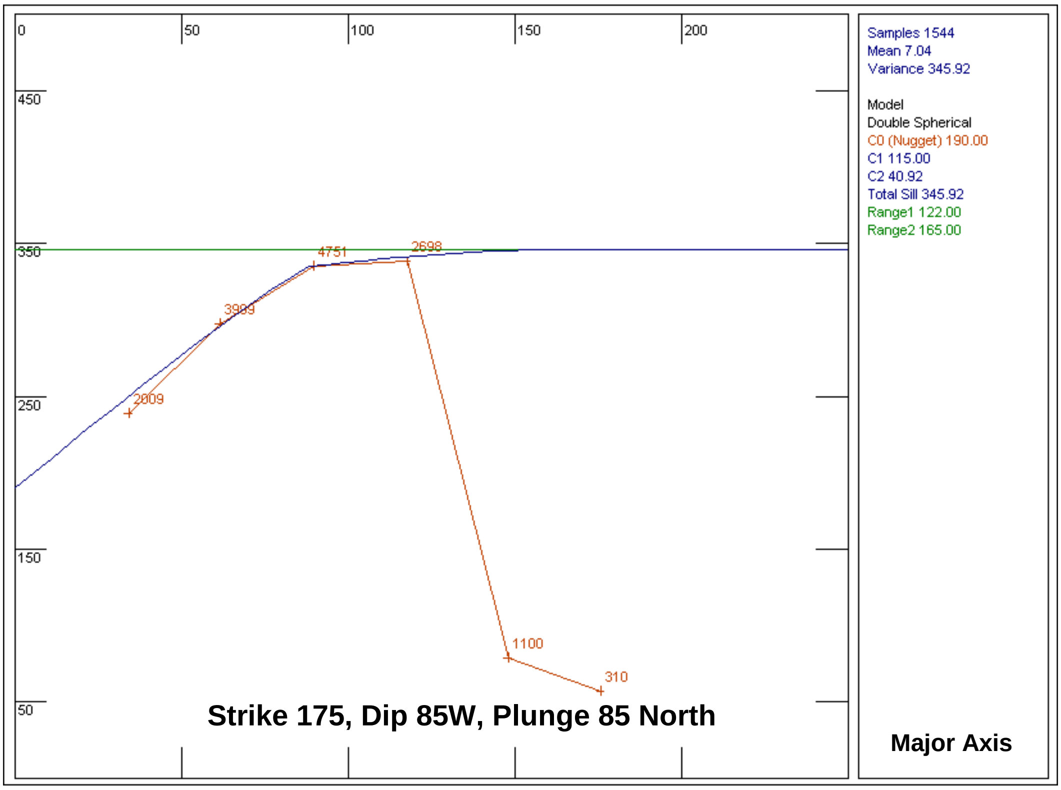


Figure 17.26 Directional Variogram, Semi-major Axis, 2m Cut Composites, Gold Domain

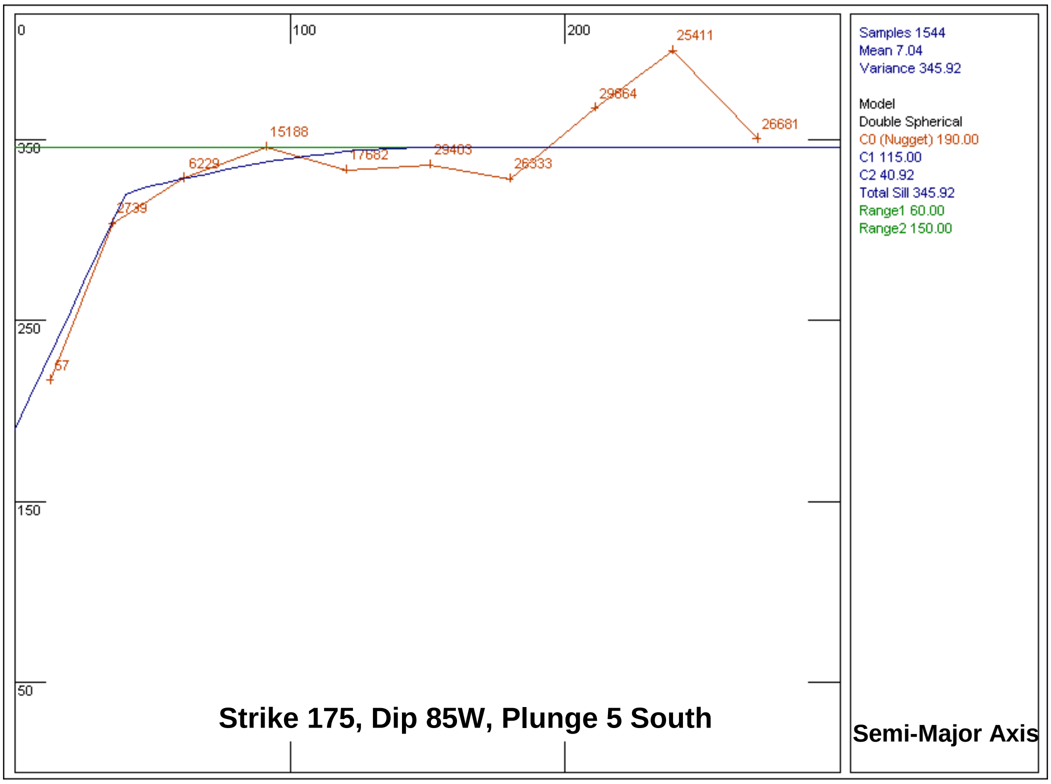
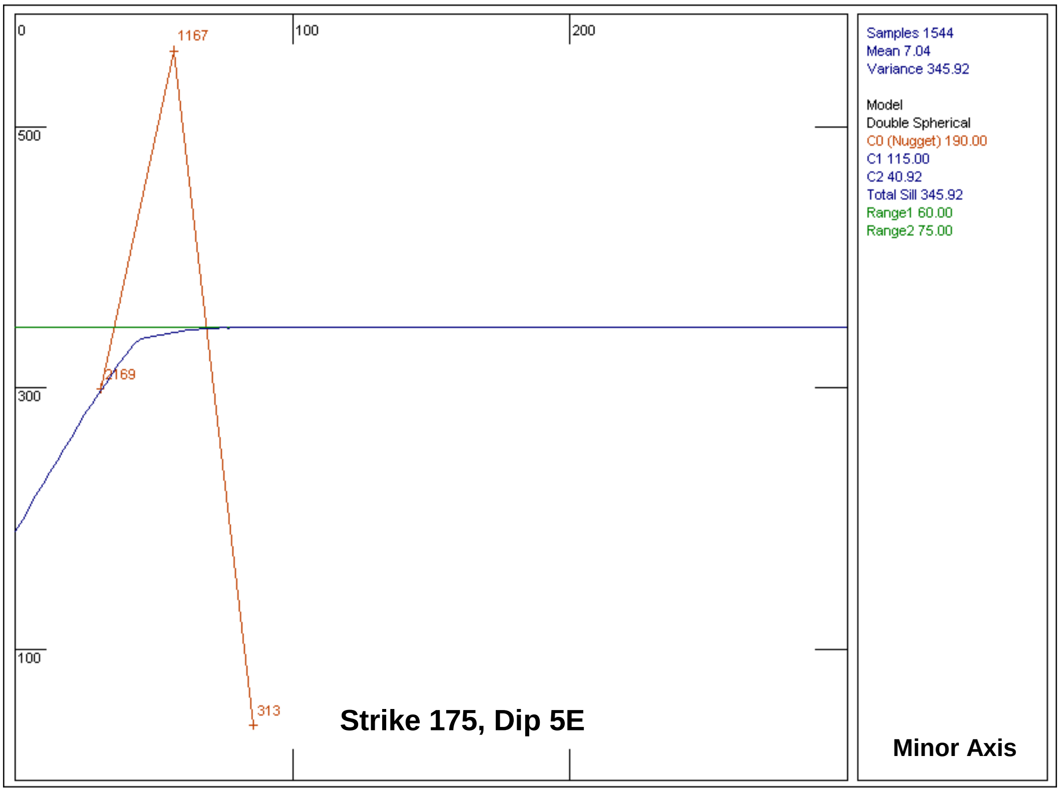


Figure 17.27 Directional Variogram, Minor Axis, 2m Cut Composites, Gold Domain



The final modelled variogram parameters are listed in Table 17.7.

Table 17.7 Directional Variogram Parameters Gold Domain

Axis	Nugget (C0) (%)	C1 (%)	C2 (%)	A1 (m)	A2 (m)	Dip (°)	Dip Azimuth (°)
Major	55	33	12	122	165	83	310
Semi-Major				60	150	5	175
Minor				60	75	-5	265

17.1.8 Quantitative Kriging Neighbourhood Analysis ("QKNA")

AMC used a procedure to test the validity of the kriging parameters used in the estimation of grade. The procedure relies on the variography. Kriging efficiency and slope of regression values are calculated for combinations of parent-cell sizes, search distances, number of drillhole samples used in the search, discretisation points used and any other relevant grade estimation parameters.

Kriging efficiency (KE %) is determined by the use of the following equation:

$$KE\ \% = ((TBV-KV+(2*LAGRANGE))/TBV)*100$$

Where:

TBV =	'true' block variance and is in turn defined as the local variance (of a domain) minus the statistical 'F value'(the average value of the variograms in a block). The F value is output by Datamine during the simple kriging estimation (IMETHOD 101 in the estimation parameter file)
KV =	estimation variance for variable per block (from ordinary kriging).
LAGRANGE =	Lagrange multiplier, a constant, output by Datamine during the kriging runs (IMETHOD 102 in the estimation parameter file).

Generally, kriging efficiency values of 95% to 100%, or slightly higher, are the target range of values for a well estimated model, although this varies with the quality of the data, the position of the estimated block with respect to the data and the quality of the variography. The slope of regression (PSL_A) has been calculated by using the equation:

PSL_A = 1 - LAGRANGE/(TBV-KV+(2*LAGRANGE))

Slope of regression values of 0.95 to 1.0 are in the target range for a well-estimated model. Theoretically a slope of regression of value 1.0 corresponds to an estimate where the estimated grades match the 'true block grades' exactly. As with kriging efficiency, the slope calculation is reliant on factors such as the input data and variography quality and the estimated block position.

AMC ran kriging efficiency tests for the gold domain only. The qualitative kriging analysis process is iterative, with many runs being undertaken and compared, before deciding on a final set of estimation parameters. The procedure inherently tends towards smooth estimates and this is also taken into account when deciding on the final parameters. When choosing final parameters, emphasis is not placed on particular values, but where the rate of change stabilises.

The results of the testing showed that generally, gold would be well estimated using the parameters finally selected.

17.1.9 Grade Estimation

Gold grades were estimated into the volume model using ordinary kriging. Grades were also estimated using inverse distance squared as a check estimate. Only the kriged grades are reported in the Mineral Resource statement.

All sub-cells within a parent cell were assigned the same grades. Parameter files were created for search, variogram and estimation parameters and these were read into a Datamine macro that allowed multiple estimation runs to test the parameters and the final output model.

Search ellipse dimensions were initially selected based on the variogram ranges and aligned according to the modelled variogram axis directions. The initial search ellipse was tested using quantitative kriging neighbourhood analysis and model validation runs to select the final search ellipse dimensions. The search ellipse is described in Table 17.8.

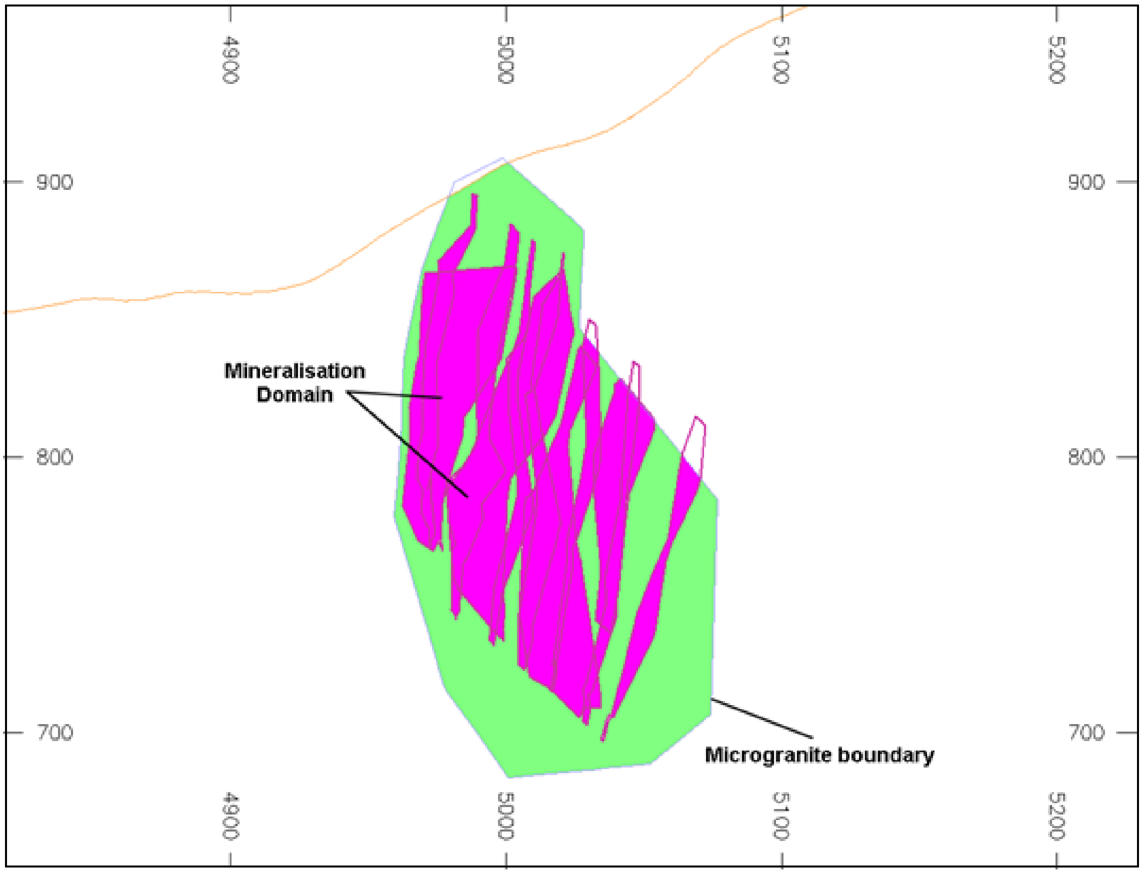
Table 17.8 Estimation Search Ellipse Parameters

Parameter	Value
Distance 1	100
Distance 2	91
Distance 3	45
Angle 1	175
Angle 2	5
Angle 3	-86
Axis 1	3
Axis 2	1
Axis 3	2

During the estimation process, the gold and sulphide domains were combined by re-coding the sulphide domain composites and sulphide domain blocks in the volume model with the same domain value as the gold domain. This ensured that there was one mineralisation domain, surrounded by a waste microgranite domain.

Figure 17.28 shows a cross section with the original gold and sulphide domain wireframe outlines and the combined mineralisation domain outline used in grade estimation.

Figure 17.28 Section 9820N Showing Combined Mineralisation Domain for Grade Estimation



Grade was also estimated into the microgranite domain to allow for the calculation of dilution grade in the subsequent pit optimisation processes. The gold domain variogram was used for the microgranite domain, scaled to the local gold variance.

The following general estimation parameters were used:

- A multiple pass approach was used, with up to one expansion of the initial search ellipse allowed (for a total of two passes).
- The second pass was a two and a half times expansion of the first pass.
- A maximum of 30 composites was used for the grade estimation in all passes.
- A minimum of 10 composites was used in the first pass and 5 composites in the second pass.
- Gold grade was estimated into parent-cell volumes only, with any sub-cells receiving the grade of its parent.
- Kriging weights were allowed to be negative.
- Cell discretisation was set to 5 x 5 x 3 in X, Y and Z respectively.
- A maximum of five composites were allowed from each drillhole for each estimated grade.

The number of samples chosen to be used in each estimate and the number of discretisation points in each dimension were tested using QKNA. During the initial parameter-testing stages, the local estimated grades in the model proved to be quite sensitive to changes in the key parameters.

In the final grade estimate, all parent cells were estimated in the first pass and over 99% of parent cells were estimated using 30 samples.

17.1.10 Depletion

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.

Drilling has not been carried out in near-surface disturbed areas and AMC did not extend the mineralisation domain to surface on every section. There may be some potential for near surface gold mineralisation that is not apparent in the Mineral Resource estimate that might be identified in grade control.

17.1.11 Resource Classification

The Mineral Resource estimate was classified according to the guidelines of the JORC Code based on a combination of inputs. The drillhole spacing, the confidence in the geological interpretation, the number of the estimation pass, the number of samples used in the estimate and the local continuity apparent from the geology, the interpretation and the variography were considered in the resource classification. Table 17.9 lists the criteria considered in the classification of the Mineral Resource.

Table 17.9 Mineral Resource Classification Criteria

Item	Discussion
Drilling Techniques	Diamond drilling - industry standard approach.
Logging	Geological logging is completed using standard nomenclature and is of acceptable quality.
Drill Sample Recovery	Acceptable recoveries determined for the majority of the drilling.
Sub-sampling Techniques and Sample Preparation	Industry standard for diamond drilling.
Quality of Assay Data	Quality control data available and has been assessed. Most quality control data is acceptable although the precision of some pulp duplicates is marginally acceptable.
Verification of sampling and Assaying	Umpire assaying has been completed which indicates robust analytical data.
Location of sampling Points	All drillhole collars and have been surveyed and most drillholes have been downhole surveyed.
Data Density and Distribution	The data spacing is considered to be appropriate for resource evaluation.
Database Integrity	No material errors identified.
Geological Interpretation	The Mineral Resource estimate is based on interpretation of overlapping grade and sulphide domains recognizing the relationship between gold mineralisation and sulphide apparent in core logging. The combined mineralisation domains are constrained within the interpretation of microgranite that hosts all significant gold mineralisation.
Estimation and Modelling Techniques	The gold grade estimation used ordinary kriging on 2m composites with a 200 g/t Au topcut. A comparative estimate was developed using inverse distance squared.
Cutoff Grades	The Mineral Resource estimate has been reported at a cutoff of 1.2 g/t Au.
Mining Factors or Assumptions	The model has been developed for evaluation by open pit mining but no modifying factors have been applied in the Mineral Resource estimate.
Tonnage Factors	An average bulk density of 2.74 t/m^3 has been assigned based on 2,311 determinations. The bulk density is considered to be well established and reasonable for both this style of mineralisation and associated rock type.

Strings were generated on section guided by the estimation inputs and linked to form wireframes that were used to select parts of the model. A numeric field was set in the grade model to indicate classification (Table 17.10).

Table 17.10 Resource Classification Codes

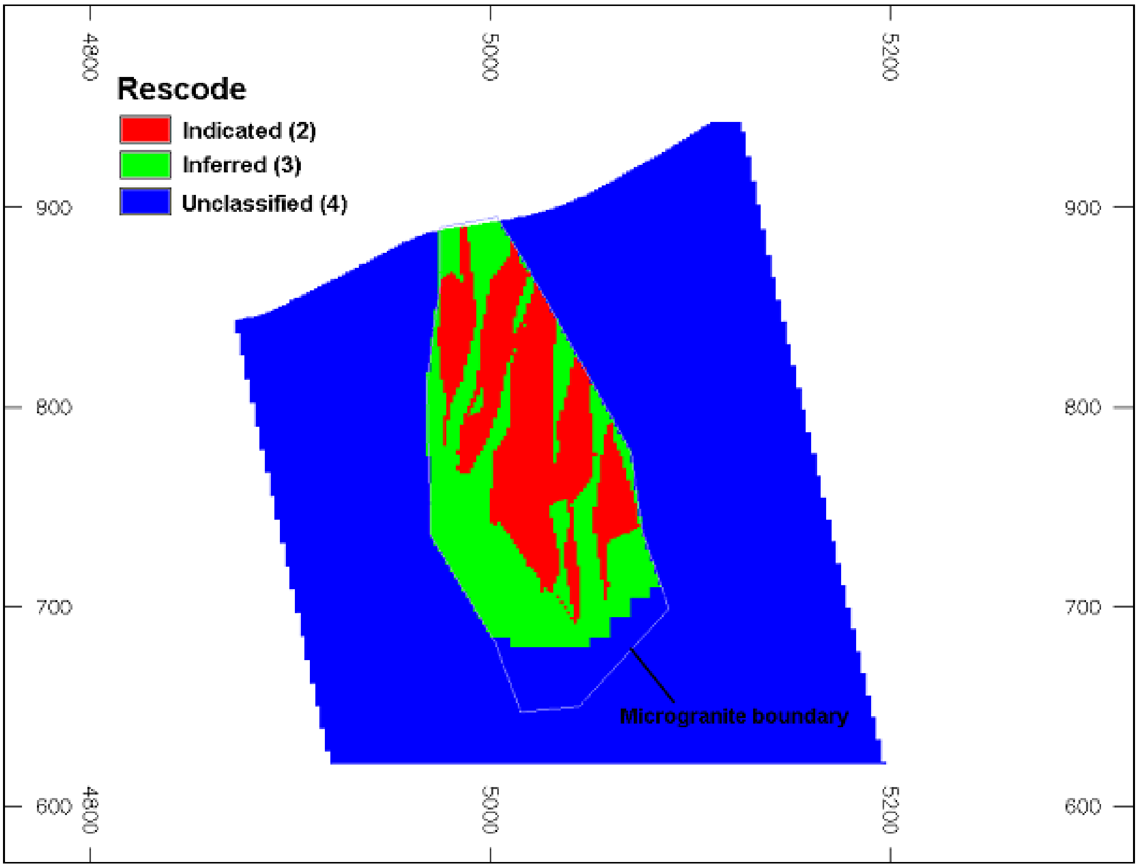
Classification	Rescode	Domain
Indicated	2	All of mineralised domain
Inferred	3	All of the microgranite domain, apart from the extremities
Unclassified	4	All of the background domain and a portion of the microgranite domain

None of that part of the model classified as Inferred Resource has a higher gold grade than 1.2 g/t Au and so no Inferred Resource is included in the Mineral Resource statement.

Due to the stockwork nature of the deposit, the variability of the interpreted lodes within the mineralisation domain, the sensitivity of the grade estimate to changes in the input parameters and the very high nugget value, AMC has not classified any of the Koka Mineral Resource estimate as Measured Resource.

Figure 17.29 shows a typical cross section of the block model coloured by Mineral Resource classification.

Figure 17.29 Cross-Section 9780 mN Showing Resource Classification

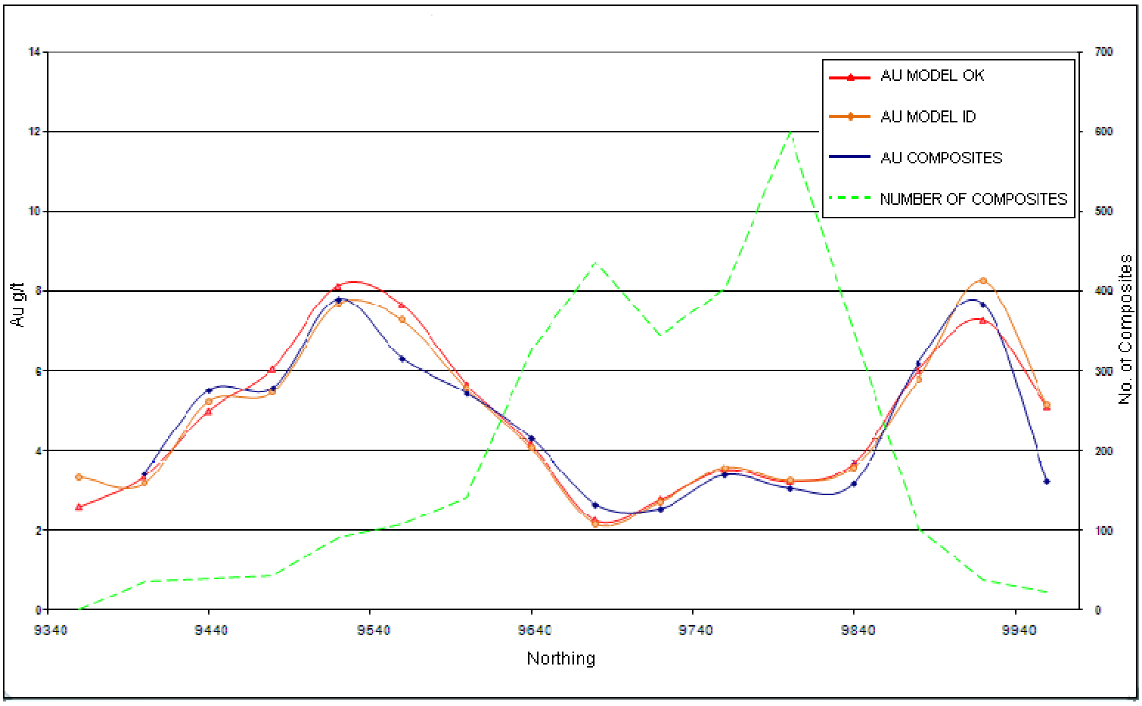


17.1.12 Block Model Validation

The final model was validated visually and statistically. The model was compared with drillholes and wireframes on sections to check for errors.

Plots were produced (Figure 17.30) comparing the estimated model grades with the composite grades in a series of slices through the model and data. The profile plots of 40m slices in northing confirm the general trends in the data and model.

Figure 17.30 Gold Grade Profile by 40m Northing Slice

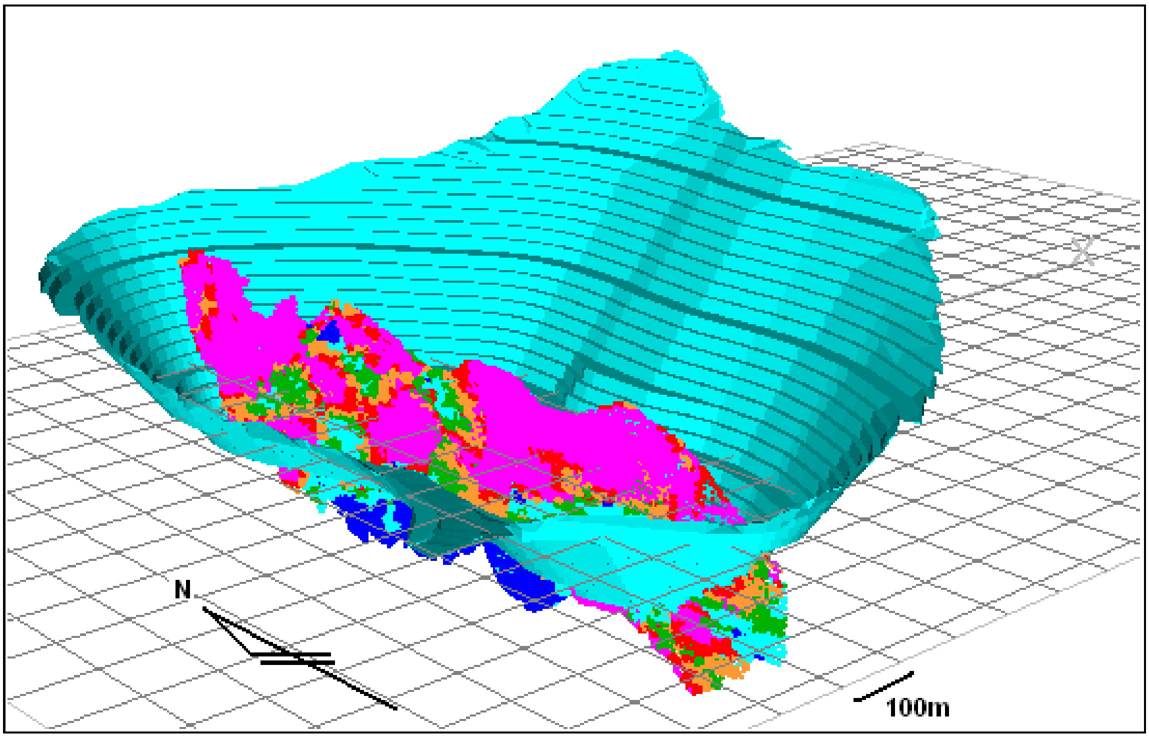


The profile plots show a smoother trend for the model (AU MODEL OK) compared to the composites (AU COMPOSITES), with the model grade profile mostly contained within the peaks of the composite grade profile. The model grade for inverse distance estimate (AU MODEL ID) closely matches the ordinary kriged grade. There are no unusual trends or areas of material concern.

There is a slight divergence of the model grade and composite grade at approximately 9540 mN. This is most likely due to the small number of local composites and greater influence from composites to the north.

Figure 17.31 shows the Mineral Resource model coloured on gold grade within the feasibility study pit.

Figure 17.31 Oblique View of the Resource Model Coloured on Gold Grade within the Feasibility Study Pit



17.1.13 Mineral Resource Statement

The 2010 Mineral Resource estimate for the Koka gold deposit reported at a 1.2 g/t Au cut-off and considering the rounding guidelines in the JORC Code is presented as Table 17.11.

Table 17.11 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.

AMC Comment

The June 2010 Mineral Resource estimate has been prepared using common industry practices and has been appropriately classified as Indicated Resource in compliance with the JORC Code. AMC is satisfied that the estimate is of a suitable standard for reporting under NI 43-101.

To the best of AMC's knowledge, the Mineral Resource estimate is not likely to be materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other non geological issues. The Mineral Resource estimate is not materially affected by the low level of artisanal mining or by metallurgical or infrastructure factors.

17.2 Mineral Reserve Estimation

AMC developed an estimate of Ore Reserves[7] using the 1 June 2010 Mineral Resource estimate assuming open pit mining of the Koka gold deposit.

17.2.1 Geotechnical

The Koka mineralised zone has a total strike length of more than 650m 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 assessment of the slope parameters for open pit optimisation was based on information available from geological drill holes, dedicated geotechnical drillholes, core logging, and laboratory tests.

A geotechnical database comprising 49 drillholes was selected for assessment of the geological and geotechnical domains. These included the drillholes AMC001, AMC002, AMC003, and AMC004 which were designed to target the rock mass located in the high wall east of the mineralisation contact. Geotechnical data associated with infill drillholes ZARD128 to ZARD157 were not available to be included in the assessment.

Three domains were identified:

- Ore Domain ("OD"): is comprised of strongly silicified brecciated microgranites and is located between the Eastern Contact Domain and Western Contact Domain. Typical rock mass weighting (“RMR^{89}”)values range from 70 to 80 with ground conditions described as 'good' to 'excellent'. A prominent mylonitised shear lies in the western margin of the OD. The shear zone appears competent and rehealed, with common rock quality designation (“RQD”) values described as 'good' to 'excellent' with typical RMR^{89} values in the range of 60 to 70. The mylonitised shear zone is estimated to represent about 5% to 10% of the drill core evaluated.
- Eastern Contact Domain ("ECD") is located east of the OD and is comprised of meta-volcanic and meta-volcaniclastic sequence including tuff, intrusions of altered and unaltered microgranite and minor rhyolite and dacite (Hamer, 2007). Typical RMR^{89} values range from 60 to 70, with ground conditions described as 'fair' to 'excellent'.

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[7] The term Ore Reserves as defined in the JORC Code is equivalent to the term Mineral Reserves as applied in the CIM Standards. NI 43-101 allows the use of JORC Code terms if reconciliation with the CIM Definition Standards is disclosed.

- Western Contact Domain ("WCD") is located west of the OD and is comprised of meta-sedimentary rocks, including tuffaceous greywackes-siltstones-shales (Hamer, 2007). Typical RMR[89] values range from 60 to 70 with ground conditions described as 'good' to 'excellent'.

Several shears were identified in the resource drill holes, including the two 'contact' shears bounding the mineralised microgranite. In general the zones of poorer ground associated with such shears tend to be of limited width and as such are not expected to present any special problems for mine development, provided they are avoided as far as possible.

The depth of weathering has been assessed through the comparison of geological logs and density measurements. While the geological logging has identified weathered material between 12m to 61m with an average of 38m, this logging possibly includes alteration and country rock containing areas of oxidised sulphides. The corresponding density measurements suggest minimal weathering with the overall average density of the top 30m equal to the average density of the whole deposit. Drillholes AMC001 to AMC004 indicated the weathering profile in the east wall is less than 30m. The stability analysis was carried out assuming 25m of weathered material.

The pit was divided into two sectors considering the geology and the pit wall dimensions. For each sector a two dimensional model was built and analysed using Phase2 geotechnical software. Overall slope angles were estimated as ranging from 45º to 48º. Detailed pit design parameters are provided in Table 17.12.

The factor of safety ("FOS") used in the analysis was FOS = 1.35 for overall slope angle and FOS =1.25 for inter-ramp slope angle. The design parameters were calculated assuming dry walls and a water table located close to the bottom of the pit.

Table 17.12 Design Parameters for Detailed Design

Sector	Depth (mbs)	Batter Angle (º)	Batter Height (m)	Berm (m)	Inter Ramp Angle (º)	Inter Ramp Height (m)	Geotech Berm Width (m)	Overall Slope Angle (º)
E – Hanging Wall	0 - 25	60	10	5	42.9	60	15	45
	>25	75	10	5	54.8	80	15	
W - Footwall	0 -25	60	10	5	42.9	60	15	48
	>25	75	10	4.5	56.5	80	15	

17.2.2 Mining

As part of a scoping study completed in 2009, AMC undertook open pit and underground evaluations and concluded that open pit mining was more suitable for the Koka gold deposit due to:

- the nature of mineralisation and the Mineral Resource model
- higher discounted operating cash flow.

The feasibility open pit study was based on:

- the 1 June 2010 Mineral Resource model developed by AMC
- the optimisation input parameters supplied by Chalice, Lycopodium and AMC
- geotechnical slope angles developed by AMC.

17.2.2.1 Model Preparation

The Mineral Resource model was converted to a mineable model. The impact of mining on the anticipated ore tonnage and grade was analysed by considering the impact of reblocking the resource model at the selective mining unit ("SMU") size suitable for the equipment being considered for mining. A minimum block size of 5m in both easting and northing directions and 2.5m in RL was used for a 120t type excavator. The result of the dilution and ore loss process was to add 15% dilution and 5% ore loss.

17.2.2.2 Pit Optimisation

The pit limits for the open pit were selected through analysis using the Whittle Four-X implementation of the Lerchs Grossman algorithm. The pit optimisation considered Indicated Mineral Resources only.

The following ore related parameters were used in the optimisation:

- Process and administration cost of US$33.49/t processed assuming a 0.5 Mtpa processing rate.
- Metallurgical recovery of 96.2%.
- Gold price of US$900 per ounce.
- Government royalty of 5% of revenue.
- Treatment plant breakeven cut-off grade estimated as 1.27 g/t Au.

The processing and administration cost was developed by Lycopodium as part of the 2009 scoping study.

Metallurgical testwork was conducted by Australian Metallurgical and Mineral Testing Consultants in Perth, Western Australia in 2009 under supervision of Lycopodium. The 96.2% recovery factor is based on a head grade of 5 g/t Au.

The complete optimisation results for Run No. 11, using the reference gold price of US$900 per ounce are shown in Table 17.13 and Figure 17.32. The results show a flat undiscounted cash flow over a wide range of pit sizes.

Table 17.13 Dilution/Ore Loss from Reblocking 2.5m x 5m x 5m

Global Model

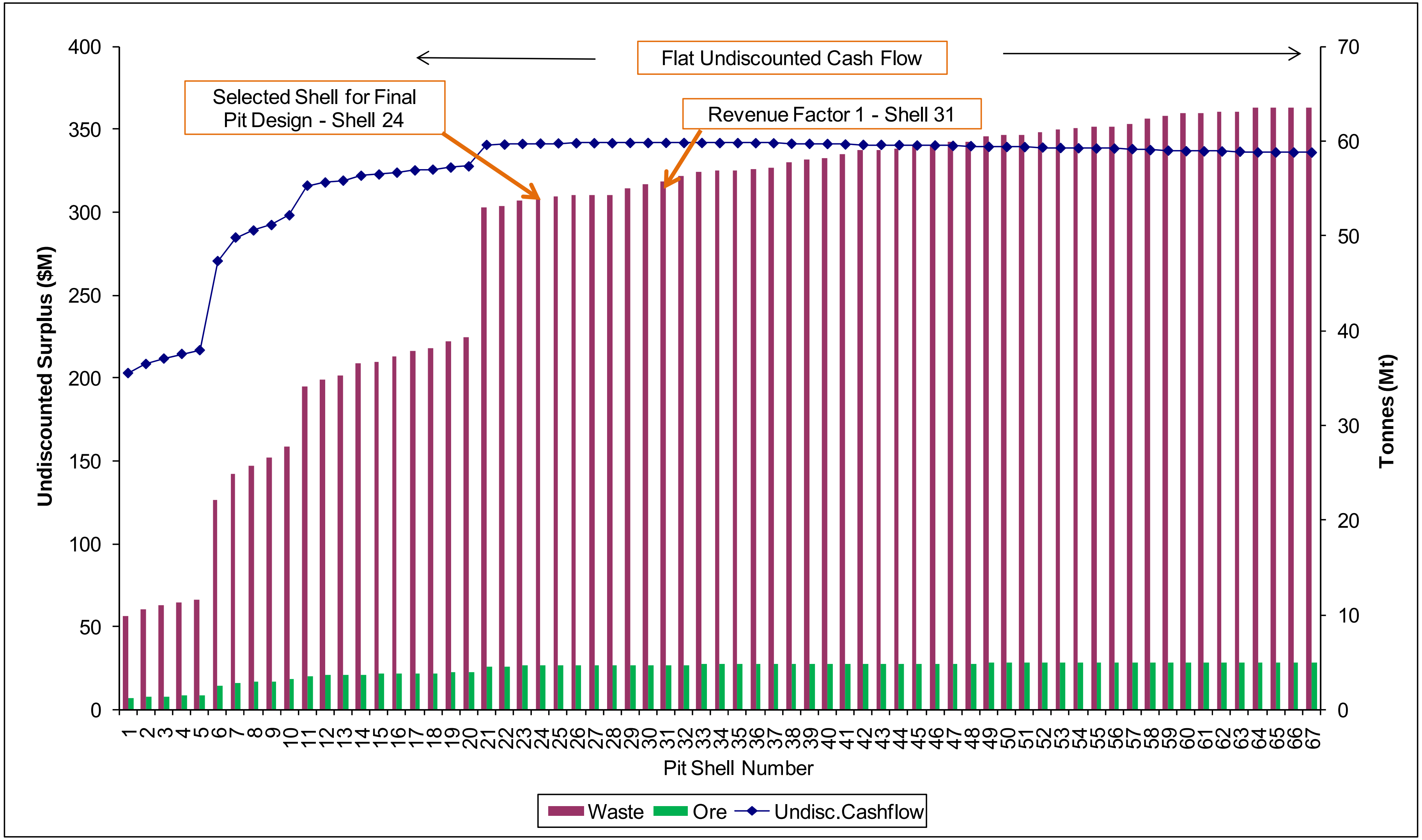
Case	Resource	Ore	Grade	Difference		Metal Difference
			Au	Ore	Au	Au
		(t)	(%)	(%)	(%)	(%)
2.5 X 5 X 5 m cells	Undiluted Resource	4,875,131	5.37	100	100	100
	Ore Loss	337,497	2.52	7	47	3
	Dilution	743,011	0.09	15	2	-
	Diluted Resource	5,280,645	4.81	108	90	97

Koka Dilution and Ore Loss Report
Truncated Model by Pit Shell (ptec31)

Truncated Model by Pit Shell (ptec31) `

Case	Resource	Ore	Grade	Difference		Metal Difference
			Au	Ore	Au	Au
		(t)	(%)	(%)	(%)	(%)
2.5 X 5 X 5 m cells	Undiluted Resource	3,824,070	5.66	100	100	100
	Ore Loss	208,311	2.68	5	47	3
	Dilution	569,278	0.10	15	2	-
	Diluted Resource	4,185,037	5.05	109	89	98

Figure 17.32 Pit Shell Size



From review of the optimisation results, pit shell 6 (starter pit) and 24 (final) were selected as the basis for the pit designs. The pit shells were selected after consideration of:

- pit optimisation results
- smoothing the life of mine stripping ratio
- support early ore production
- minimum mining widths between a stage pit and final pit.

17.2.2.3 Pit Design

Based on results of the open pit optimisation process, AMC designed an open pit that:

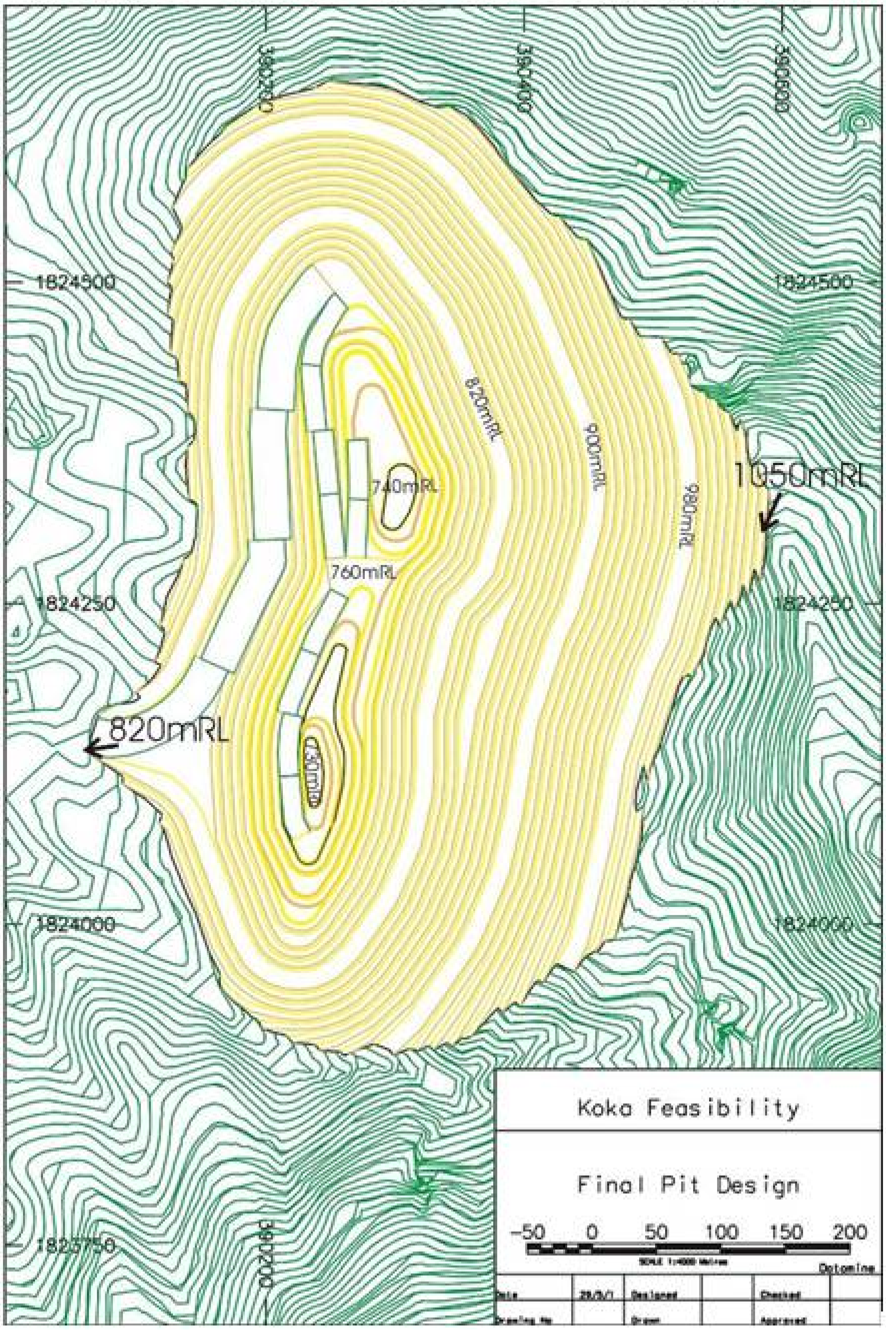
- contains a Probable Ore Reserve of 4.6 Mt grading 5.1 g/t Au and containing 760,000 ounces of gold at a 1.26 g/t Au cut-off grade
- contains 48 Mt of waste at a strip ratio of 10.4 (waste tonnes):1 (ore tonnes).

The mine design is shown in Figure 17.33.

A Stage 1 pit was also designed containing:

- 2.6 Mt of ore grading 6.1 g/t Au and containing 480,000 ounces of gold at a 1.26 g/t Au cut-off grade
- 22.1 Mt of waste at a strip ratio of 8.6 (waste tonnes):1 (ore tonnes).

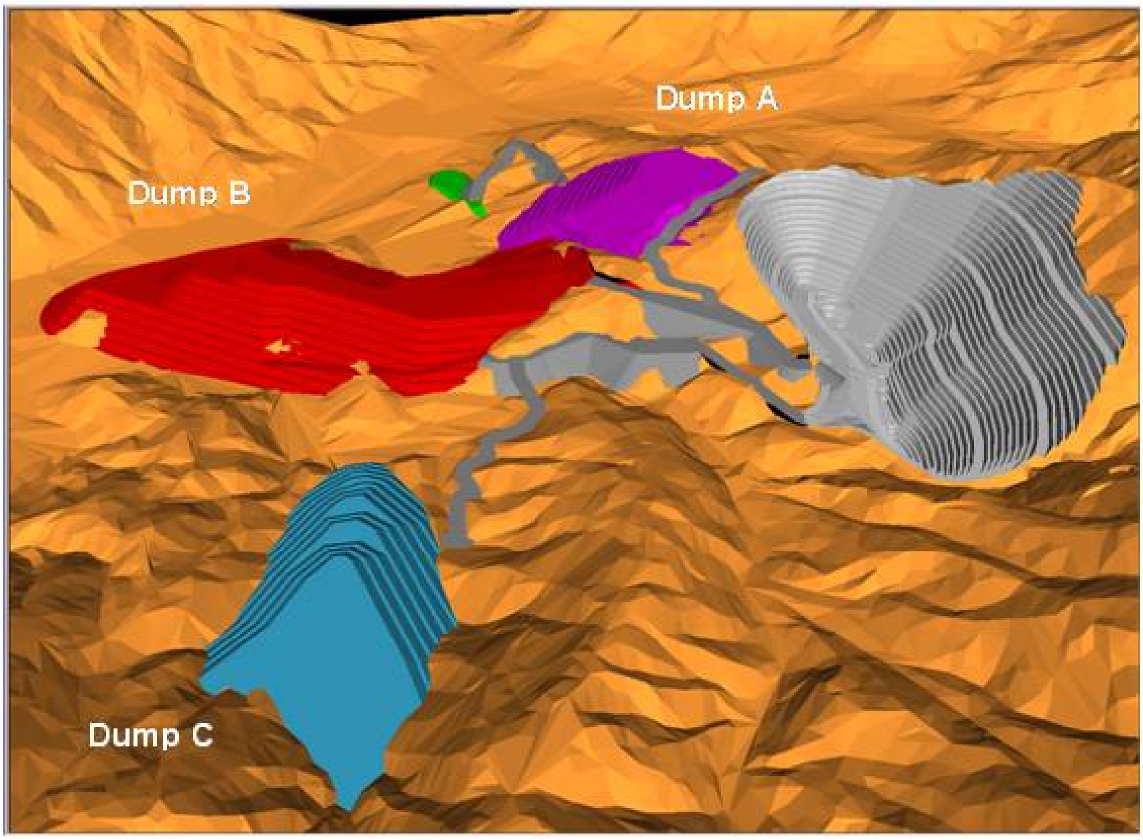
Figure 17.33 Open Pit Mine Design



17.2.2.4 Waste Dumps

Due to the steep topography there are limited locations available to locate waste dumps. Three dumps are required which are shown in Figure 17.34.

Figure 17.34 Waste Dump Locations



Dump A, with a volume of 3.96 million loose cubic metres ("Mlcm"), will be located immediately northwest of the pit in a valley adjacent to the Run of Mine ("ROM") pad.

The main waste dump (Dump B) will be located immediately west of the pit and adjacent to the process plant. The dump will be bounded by the plant site to the north, river channel to the south and small hill and junction of two rivers to the west. The dump will be 140m high and contain a volume of 10.1 Mlcm, suitable for 7.7 Mt (80%) of the pit waste.

A tertiary waste dump (Dump C) will be located further to the south of the main dump site across the river channel. This dump site will be capable of containing the remaining waste requirements of the mine.

Waste dumps have been designed in 10m lifts with a 37° batter angle and a 5m berm between lifts, except for Dump C which has 10m berms. Final batter angles were left at 37° to maximise waste dump capacity in the confined areas available

17.2.2.5 Schedule

The mining and processing schedule is summarised in Table 17.14 and presented graphically in Figure 17.35. The schedule was based on:

- Stage 1 pit.
- Final pit.

- Maximum mining rate of 9 Mtpa.
- Processing rate of 0.6 Mtpa rising to 0.7 Mtpa as the head grade decreases.

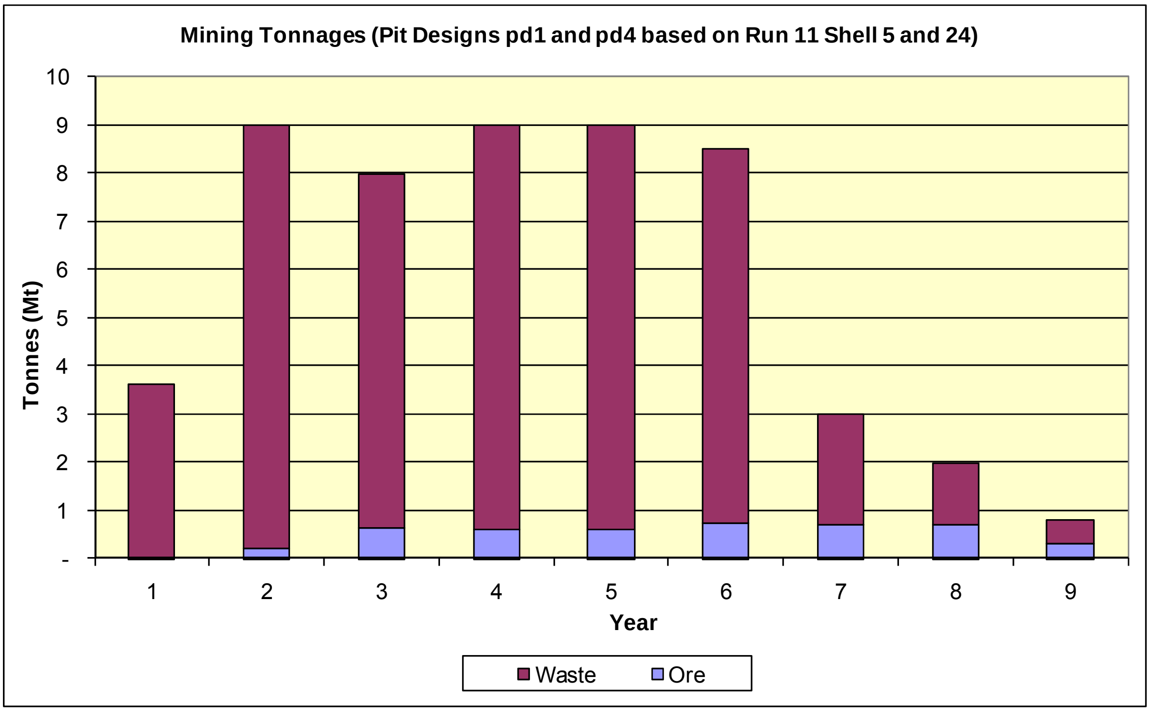
Due to the steep topography on the eastern wall, the open pit has high stripping ratios resulting in low ore production in the early years followed by high production in the last three years. An aggressive mining rate with 130m of vertical advance in the Stage 1 pit over the first two years (Years -1 and 1) is required to provide sufficient ore tonnage on a consistent basis to the process plant by the start of Year 2 of the schedule.

At this point Stage 1 mining is slowed to a rate to match mill ore requirements, and production in the Final pit is commenced. In Years 3 and 4 mining rates are maintained at 9 Mtpa with Stage 1 supplying ore to the mill while 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 excess ore tonnage is produced. The mining rate then declines rapidly through Years 6, 7 and 8 although a significant ore stockpile is built.

Table 17.14 Mining and Processing Schedule

Annual	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mining											
Ore Tonnes	t	-	224,129	637,436	627,436	619,391	757,539	723,977	720,248	320,606	**4,630,761**
Gold Grade	g/t	-	5.42	7.29	6.39	5.52	4.18	3.54	3.87	5.61	**5.10**
Contained Gold	oz	-	39,053	149,391	128,977	109,864	101,873	82,403	89,504	57,828	**758,891**
Ore Volume	bcm	-	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 Ounces	oz	-	37,699	136,544	120,199	105,067	93,810	78,871	79,691	78,901	**730,780**

Figure 17.35 Annual Mining Tonnages



17.2.2.6 Mining Fleet

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 mining method adopted was the conventional open pit truck and excavator system. The loading fleet was selected based on a match with the schedule requirements, consideration of the existing presence of CAT and Bucyrus equipment in Eritrea and remove the reliance on a single excavator. The major mining equipment selected consists of:

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

Ancillary equipment includes:

- 2 x Track dozer Cat D9
- 1 x Wheel dozer Cat 834
- 1 x Wheel loader Cat 966
- 1 x Grader Cat 16
- 1 x Water truck Cat 773
- 1 x Rock Breaker Cat 336
- 1 x Service truck

17.2.2.7 Costs

All currency is shown in 2010 United States dollars (US$) unless otherwise stated, and is undiscounted. The costs were estimated as at Quarter 1 of 2010. AMC estimates the accuracy of this cost estimate to be ±15%.

Operating Cost

AMC prepared an owner mining-based operating cost estimate from first principles using AMC's OPMincost system. The open pit activities and costs assessed within OPMincost include:

- drill, blast, load, haul and dump of ore to the ROM/primary crusher and waste to the waste dump
- auxiliary operations such as haul road construction and maintenance and bench, dump and drainage maintenance
- pit dewatering
- mining department management and supervision up to and including the open pit manager. Technical services personnel such as mining engineers, geologists and surveyors are also included
- operating and maintenance manpower requirements
- consumables including fuel, parts, explosives, etc
- training of personnel.

AMC estimates an average mine operating cost of US$1.99 per tonne mined (including crushing at US$0.04/t mined) for the Final pit design, as shown in Table 17.15, based on a total life of mine (“LOM”) estimated mine operating cost of US$106M.

Table 17.15 Summary of 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**

Capital Cost

AMC estimates a life of mine capital cost of US$22.2M as shown in the breakdown of capital expenditure in Table 17.16. The capital costs include:

- mining fleet
- maintenance equipment
- technical equipment.

Table 17.16 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**

17.2.3 Ore Reserve Statement

The Ore Reserve estimate for the Koka gold deposit as at 1 June 2010 based on the July 2010 feasibility study is listed in Table 17.17.

Table 17.17 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

18 OTHER RELEVANT DATA AND INFORMATION

Lycopodium completed a feasibility study for Chalice in July 2010 for submission to the Eritrean government as part of Chalice's license conditions (Lycopodium, 2010). AMC completed the Mineral Resource and Ore Reserve estimation and geotechnical and mining components of the study. The Ore Reserve estimate and proposed approach to mining are described in Section 17.2 of the Technical Report. Metallurgical testwork and the proposed process flowsheet are discussed in Section 16 of the Technical Report. This section of the Technical Report covers issues relevant to the feasibility study and project proposal that are not covered in previous sections or Section 19. Preliminary designs used as the basis for cost estimates for the access road upgrade, tailings storage facility, airstrip and bore field were developed by geotechnical consultants Knight Piésold of Perth, Western Australia following appropriate site investigations.

18.1 Support Facilities and Services

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 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.

The proposed 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 plant.

A 250 person permanent village will be established supplemented by a tent camp to house construction workers.

18.2 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 4,600,000t, 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.

18.3 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 to 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 1 to 3 of plant operation the average water demand for the project is estimated to be 31 L/s with a plant throughput of 600,000 tpa. In Years 4 to 7 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 River valley alluvium, with airlift yields ranging from 7 L/s to over 20 L/s. Pumping tests have been conducted and based on the results and the preliminary groundwater balance, a bore field extending along a 3 km stretch of the Zara River 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.

18.4 Surface Water

Stormwater run-off 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 plant. Rainfall onto other plant areas will flow into an event pond and returned to the process plant, as it may be contaminated with chemicals or ore spillage.

18.5 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 bore field will have a separate generator and a standby generator will be located at the village.

18.6 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.

18.7 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.

19 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

Lycopodium completed a feasibility study for Chalice in July 2010. The Ore Reserve estimate and proposed approach to mining that applied to the feasibility study are described in Section 17.2 of the technical report. Metallurgical testwork and the proposed process flowsheet are discussed in Section 16 of the technical report. Other issues relevant to the feasibility study and project proposal are covered in Section 18.

19.1 Environmental Considerations

Comprehensive terms of reference for environmental review were prepared in accordance with government guidelines with the 2009 scoping study. Following minor amendments from the government, the accepted terms of reference have been the basis for a comprehensive draft Social and Environmental Impact Assessment ("SEIA") and draft Social and Environmental Management Plan ("SEMP") in support of the feasibility study and mining license application.

Compilation of environmental baseline data for the Koka gold project commenced in 2007 and continued through 2010 and will be ongoing 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 comparison.

The scope of the SEIA addresses the key components outlined below.

Potential Development Areas ("PDAs") are defined as those areas directly affected by project activities. Environmental baseline studies are an integral part of the environmental assessment as they provide the data to assess impacts. The data are generally focused on two spatial study areas:

- Local study area ("LSA") - detailed study areas centring around PDAs and linear corridors; or potentially affected communities.
- Regional study area ("RSA") - areas used to assess indirect impacts resulting from project activities and road access.

Baseline studies were completed to:

- define predevelopment social and environmental conditions
- identify potential social and environmental issues and sensitivities
- provide information to input into project design and decision making
- survey 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 government's 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 is 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.

19.2 Taxes

Paragraph 35 of Eritrean Mining Proclamation No 68/1995 states that a 5% royalty is payable on revenue from the production of precious metals although a lesser rate of royalty may be agreed in order to encourage mining investments in areas given development priority. The royalty was taken into account in pit optimisations for the feasibility study.

Revenue from mining operations is subject to income tax at the rate of 38% of taxable income. All capital expenditure and preproduction 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.

Withholding tax of 10% is payable on services contracted to non-Eritrean residents and import duties of 0.5% are payable on all imports of equipment and materials (except sedan cars) for mining operations.

19.3 Project Implementation

The Engineering, Procurement, Construction Management ("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 EPCM engineer. The EPCM 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 project schedule is governed by the twelve months of mine pre-stripping activities required to develop the mine and a further six 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 twenty four months to commissioning from approval of finance, which is expected six months after submission of the final feasibility study.

19.4 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. Day shift staff will reside in the towns and villages between Asmara and the site and will be bussed to and from site to work on 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.

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, in mining and process operations and maintenance and support trades, such as diesel fitting and general trades work.

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

19.5 Operating Cost Estimate

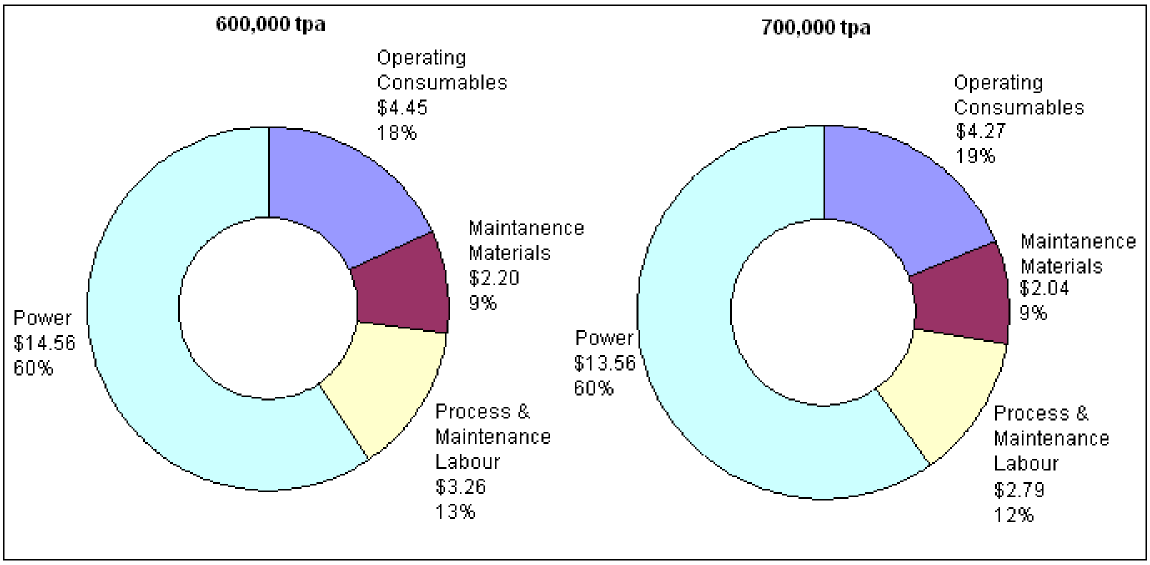
Mining, processing and administration costs as at June 2010 are summarised in Table 19.1 and represented in Figure 19.1 for feed rates of 600,000 and 700,000 tpa and are considered to have an accuracy of ±15%.

Table 19.1 Operating Cost Estimate Summary (US$ - June 2010, ±15%)

Cost Centre	600,000 tpa		700,000 tpa	
	US$/year	US$/t Ore	US$/year	US$/t Ore
Total Mining Operating Cost[1]	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 19.1 Processing Cost by Cost Centre (US$ June 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.

19.6 Capital Cost Estimate

The project capital cost estimate is summarised in Table 19.2.

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

- Sunk costs, including prefeasibility and feasibility costs.
- Exchange rate variations.
- Licence costs and fees.
- Site security.
- Operation and maintenance of the permanent rooms in the village once occupied by operations staff.

Table 19.2 Capital Cost Estimate Summary (US$ June 2010, ±15%)

Area Code	Facility	US$
000	Construction Indirects	12,224,336
100	Treatment Plant	22,108,481
200	Reagents and Plant Services	5,942,804
300	Infrastructure	28,088,973
400	Mining	31,688,981
500	Management Costs	8,483,798
600	Owners Project Costs	9,853,202
700	Owners Operations Costs	3,596,470
Total Project Cost		**121,987,046**

19.7 Financial Analysis

A simple cash flow model has been used to evaluate the project. The results, which are summarised in Table 19.3, are based on the cash flow model outlined in Table 19.4.

Table 19.3 Base Case Financial Analysis

Item	Unit	Value
Total Mined	Mt	52.94
Ore Milled	Mt	4.63
Strip Ratio	-	10.4
Gold Grade	g/t Au	5.10
Contained Gold	ounces	758,900
Recovery Gold	%	96.3
Recovered Gold	ounces	730,780
Gold Price	US$ per ounce	900
Revenue from Gold Sales	**US$M**	**657**

Item	US$M	US$/oz Sold	US$/t Processed	US$/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)	249.5	-	-	-
Income Tax Expense	94.8			
Net Profit After Tax	**154.7**			
NPV (5.0%)	**99.0**	**-**	**-**	**-**
IRR	**22%**	**-**	**-**	**-**

Table 19.4 Summary Project Cash Flow

	Year 1 US$'000	Year 2 US$'000	Year 3 US$'000	Year 4 US$'000	Year 5 US$'000	Year 6 US$'000	Year 7 US$'000	Year 8 US$'000
Revenue from operations*	**33,895**	**122,767**	**108,071**	**94,466**	**84,344**	**70,913**	**71,650**	**70,939**
Mining	-15,823	-15,793	-15,852	-15,826	-16,112	-6,924	-5,703	-2,715
Treatment	-151	-546	-481	-420	-375	-315	-319	-316
Processing	-9,709	-15,175	-15,043	-14,936	-16,611	-16,174	-16,125	-6,278
G&A	-4,261	-4,261	-4,261	-4,261	-4,261	-4,261	-4,261	-3,991
Total cash operating costs	**-29,943**	**-35,775**	**-35,636**	**-35,444**	**-37,359**	**-27,674**	**-26,408**	**-13,299**
Royalties @ 5%	-1,695	-6,138	-5,404	-4,723	-4,217	-3,546	-3,582	-3,547
Other expenses	-94	-341	-300	-263	-235	-197	-199	-197
Total cash costs	**-31,732**	**-42,255**	**-41,340**	**-40,430**	**-41,810**	**-31,417**	**-30,190**	**-17,043**
Sustaining capital	-3,521	-	-2,802	-137	-	-1,568	-	-1,310
Net cash flow before tax	**-1,359**	**80,512**	**63,928**	**53,899**	**42,534**	**37,928**	**41,460**	**52,586**
Income tax	-	-	-10,861	-8,809	-16,381	-15,078	-16,090	-21,413
Net cash flow after tax	**-1,359**	**80,512**	**53,067**	**45,089**	**26,153**	**22,850**	**25,370**	**31,173**

The results are highly sensitive to the gold price, as shown in Figure 19.2. Table 19.5 lists the variation in parameters used to assess project sensitivity.

Figure 19.2 Project Sensitivity

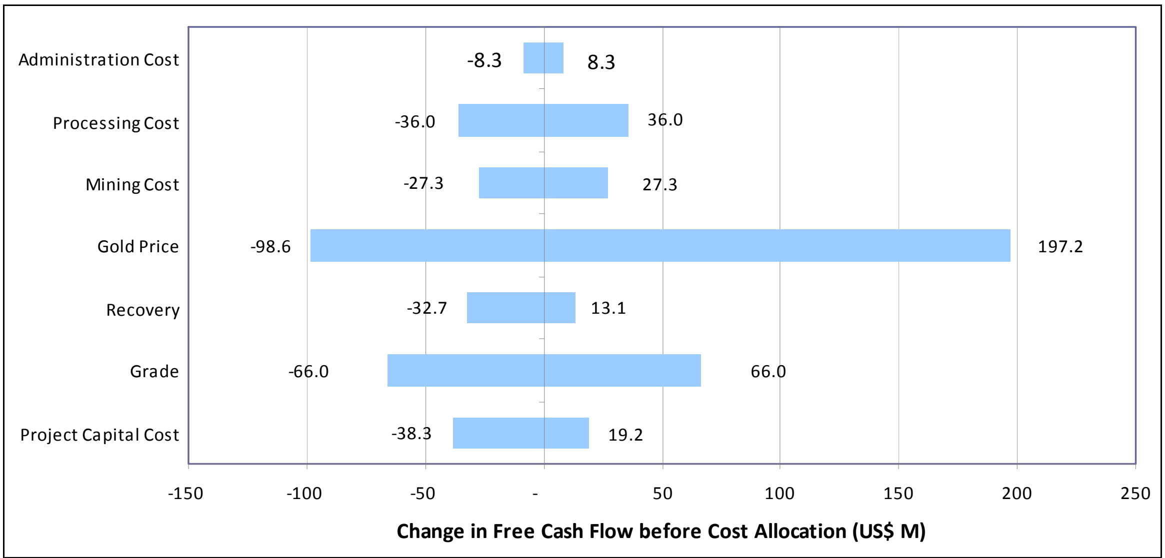


Table 19.5 Project Sensitivity Parameters

Parameter	Unit	Range		
		(-)ve %	Base %	(+)ve %
Project Capital Cost	US$M	-15	-	30
Grade	g/t Au	-10	-	10
Recovery	%	-5	-	2
Gold Price	US$/oz	-15	-	30
Mining Cost	US$/tonne ore	-30	-	30
Processing Cost	US$/tonne ore	-30	-	30
Administration Cost	US$/ tonne ore	-30	-	30

19.8 Risk Analysis

A risk assessment workshop assessed the project 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.

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 government cooperation in improving the standard of roads that will be used by the project. On the basis that this risk already exists prior to the project development, the risk was accepted.

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 ongoing emphasis on safety awareness and training. These aspects will be emphasised in the training programmes to be conducted on site.

20 INTERPRETATION AND CONCLUSIONS

Exploration in a remote area of Eritrea has successfully discovered the Koka gold deposit.

The current exploration and prospecting licences are renewable for one year terms. It is expected that, providing Chalice meets the Eritrean government's requirements for completion of the feasibility study, a mining license will be granted to enable the project to proceed.

Diamond drilling of the deposit on 20m to 40m-spaced sections has lead to the estimation of a Mineral Resource by AMC that has been classified as Indicated Resource in compliance with the JORC Code (Table 20.1). The drilling data are supported by quality control protocol that meets accepted industry practice.

AMC considers that the Mineral Resource estimate has been prepared using common industry practices and has been appropriately classified as Indicated Resource in compliance consideration of the JORC Code. AMC is satisfied that the estimate is of a suitable standard for reporting under NI 43-101.

Table 20.1 Koka Gold Deposit Mineral Resource 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

A feasibility study was completed by Lycopodium in July 2010 with the Mineral Resource and Ore Reserve estimates and geotechnical and mining sections of the study completed by AMC. The feasibility study was based on the June 2010 Indicated Resource and Ore Reserves were estimated and reported to the ASX on 4 June 2010. The Ore Reserve estimate at 1 June 2010 is listed in Table 20.2.

Table 20.2 Koka Gold Deposit Ore Reserve

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

Metallurgical testwork on samples from drillholes completed specifically for this purpose has indicated that overall gold extractions were excellent and ranged from 95.3% to 99.2% for gold head grades ranging from 2.33 g/t Au to 14.51 g/t Au. A conceptual process flowsheet was developed based on industrially proven unit processes that present low technical risk.

The feasibility study demonstrated that the Koka gold deposit could be developed as an economically viable open pit mine with an average annual gold production of 104,000 oz over a mine life of seven years which was the objective of the study.

Limited exploration has been conducted on the tenements beyond the Koka gold deposit. The Proterozoic rocks on the tenements are considered to be prospective for further gold occurrences and a base metal gossan has also been identified. Exploration activities including geological mapping, surface sampling and evaluation of satellite imagery are currently underway.

21 RECOMMENDATIONS

Lycopodium has completed a feasibility study on the Koka gold deposit for Chalice. No further technical work is recommended pending Chalice's consideration of project commitment and finance options.

22 REFERENCES

Coffey Mining Pty Ltd, 2009: Koka Gold Deposit. Geological Modelling and Grade Estimation. Prepared for Sub Sahara Resources NL. 27 May 2009.

Fessahaie Habte, 2009: Legal Opinion regarding Sub-Sahara Limited Exploration License. Prepared for Chalice Gold Mines Limited. 20 July 2009.

Hamer, R D, 2007: Revised Geological Model for Koka Prospect. Unpublished.

Lycopodium Minerals Pty Ltd, 2009: Chalice Gold Mines Limited Zara Project Scoping Study.

Lycopodium Minerals Pty Ltd, 2010: Chalice Gold Mines Limited Zara Project Definitive Feasibility Study.

Maxwell Geosciences, 2010: Chalice Gold Mines Ltd. Zara Project – Eritrea. QAQCR Summary. February 2010.

Sub-Sahara Resources NL, 2009: Zara Gold Project Eritrea – Koka Gold Deposit Resource Update. Australian Securities Exchange Announcement. 1 May 2009.

23 DATE AND SIGNATURE PAGE

This report entitled "Technical Report on the Koka Gold Project, Eritrea" dated 27 July 2010 has been prepared for Chalice Gold Mines Limited by Dean Carville, David Lee and David Gordon each of whom are qualified person(s) as defined by NI 43-101.

All sections except 16, 17.2, 18 and 19 were prepared by Dean Carville.

Dean Carville
Principal Geologist
AMC Consultants Pty Ltd

Section 17.2 was prepared by David Lee.

David Lee
Principal Mining Engineer
AMC Consultants Pty Ltd

Sections 16, 18 and 19 were prepared by David Gordon except for information on taxation that was provided by Chalice and comments on environmental considerations that are based on information provided by consultant Knight Piésold.

David Gordon
Principal Process Engineer
Lycopodium Minerals Pty Ltd

24 CERTIFICATE

I, Dean Paul Carville, of 11 Woodford Wells Way, Kingsley, Western Australia, Australia, 6026, certify as follows concerning the Technical Report for the Koka Gold Project, Eritrea, (the Property) for Chalice Gold Mines Limited (Chalice), dated 27 July 2010.

1) That I have received a BSc (Hons) in Geology from the University of Western Australia 1978 and have practiced my profession continuously since that time. I have carried out and supervised mineral exploration, resource development and resource estimation for a range of mineral commodities in Australia and other countries.

2) I am a Member of the Australasian Institute of Mining and Metallurgy.

3) I am a Qualified Person as defined in National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101).

4) I have prepared the Technical Report and take responsibility for the report with the exception of Sections 16, 17.2, 18 and 19.

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

6) Applying the tests set out in Section 1.5 of NI 43-101, I am independent of Chalice.

7) I have had the following prior involvement with the Property:

    - In September 2009 I visited the Koka site for two days regarding preparation of a scoping study. I reviewed diamond drill core, geological logging, and surface exposure. The sample preparation facility in Asmara was inspected and further discussions were held in Asmara regarding exploration activity. Visits were made to possible equipment suppliers in Asmara.

8) That this report has been prepared in compliance with NI 43-101 and Form 43-101F1 and I have read NI 43-101 and related documents.

9) I consent to the filing of the Technical Report with the Toronto Stock Exchange ("TSX") in relation to a proposed listing by Chalice including filing of the Technical Report on the System for Electronic Document Analysis and Retrieval ("SEDAR").

Dated 27 July 2010

Dean Paul Carville
B.Sc. (Hons), MAusIMM, MGSA

CERTIFICATE

I, David Maxwell Lee, of 30 Hatch Court, High Wycombe, Western Australia, Australia, 6057, certify as follows concerning the Technical Report for the Koka Gold Project, Eritrea, (the Property) for Chalice Gold Mines Limited (Chalice), dated 27 July 2010.

1) That I have received a BE (Hons) in Mining Engineering from the University of Sydney in 1989 and have practiced my profession continuously since that time for a range of mineral commodities in Australia and other countries.
2) I am a Member of the Australasian Institute of Mining and Metallurgy.
3) I am a Qualified Person as defined in National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101).
4) I have prepared Section 17.2 of the Technical Report.
5) As at the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading
6) Applying the tests set out in Section 1.5 of NI 43-101, I am independent of Chalice.
7) I have had the following prior involvement with the Property:
    - In September 2009 I visited the Koka site for two days regarding preparation of a scoping study. I reviewed diamond drill core, geological logging, and surface exposure. The sample preparation facility in Asmara was inspected and further discussions were held in Asmara regarding exploration activity. Visits were made to possible equipment suppliers in Asmara..
8) That this report has been prepared in compliance with NI 43-101 and Form 43-101F1 and I have read NI 43-101 and related documents.
9) I consent to the filing of the Technical Report with the Toronto Stock Exchange ("TSX") in relation to a proposed listing by Chalice including filing of the Technical Report on the System for Electronic Document Analysis and Retrieval ("SEDAR").

Dated 27 July 2010

David Maxwell Lee
B.E. (Hons), MAusIMM

CERTIFICATE

I, David John Gordon, of Lycopodium Minerals Pty Ltd, East Perth, Western Australia, Australia, 6004, certify as follows concerning the Technical Report for the Zara Project, Eritrea, (the Property) for Chalice Gold Mines Limited (Chalice), dated 27 July 2010.

1) That I have received a B. App. Sc in Engineering Metallurgy from the Western Australian Institute of Technology 1983 and have practiced my profession since that time. I have carried out and supervised metallurgical testwork for a range of mineral commodities in Australia and other countries.
2) I am a Member of the Australasian Institute of Mining and Metallurgy.
3) I am a Qualified Person as defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
4) I have prepared Sections 16, 18 and 19 of the Technical Report except for information on taxation that was provided by Chalice and comments on environmental considerations that are based on information provided by consultant Knight Piésold.
5) As at the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
6) Applying the tests set out in Section 1.5 of NI 43-101, I am independent of Chalice.
7) I have had the following prior involvement with the Property:
    - I have supervised the metallurgical testwork program undertaken at an independent laboratory on samples selected and supplied by Chalice and prepared the metallurgical testwork component of the Technical Report based on this independent testwork program.
8) That this report has been prepared in compliance with NI43-101 and Form 43-101F1 and I have read NI 43-101 and related documents.
9) I consent to the filing of the Technical Report with the Toronto Stock Exchange ("TSX") in relation to a proposed listing by Chalice including filing of the Technical Report on the System for Electronic Document Analysis and Retrieval ("SEDAR").

Dated 27 July 2010

David John Gordon
B.App.Sc., MAusIMM