AFRICAN GOLD GROUP

Kobada Gold Project, Mali

Volume 1

Feasibility Study

Prepared by:

International Resource Solutions Pty Ltd;
Obsidian Geological Limited;
Gekko Systems Pty Ltd; and
John Dunlop and Associates Pty Ltd on behalf of:

AFRICAN GOLD GROUP, INC.

Effective Date: 3 February 2016

Qualified Persons: Brian Wolfe, B Sc (Hons), P Grad Cert (Geostats), MAIG

Andrew Chubb, B. Sc (Hons), MAIG

Michael Braaksma, B Eng (Chem), MAusIMM(CP)

John Dunlop, B Eng (Mining), MEngSc (Mining), FAusIMM(CP),

Technical Report NI. 43-101

CERTIFICATE OF QUALIFIED PERSONS

To accompany the technical report entitled:

"Kobada Gold Project, Feasibility Study" dated effective February 3, 2016 (the "Technical Report").

I John Dunlop, FAusIMM(CP) MCIMM, do hereby certify that:

1. I am a Principal Consultant (Mining Engineering) with the firm of John Dunlop and Associates Pty Ltd of El Arish, Queensland 4855, Australia.
2. I am a Chartered Professional Fellow of the Australasian Institute of Mining and Metallurgy. My member number is 100161.
3. I am a graduate of the University of Melbourne, Victoria, Australia and hold a B. Eng (Mining) (1971), MEngSc (Mining) (1979), PCertArb (Adelaide, 2002).
4. I have practiced my profession continuously since 1969. I have 47 years of experience in operating and planning for mines.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I have visited the Kobada Project area.
7. I am responsible for Sections 2 to 6, 15, 16, 18 to 22, and jointly responsible for Sections 1, 21, 25 and 26 of the Technical Report and these sections have been prepared in compliance with the Instrument. I have reviewed and made appropriate cross checks of the remaining sections of the Technical Report.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.
9. I am independent of the Kobada Project and the issuer (African Gold Group, Inc.) pursuant to section 1.5 of the Instrument.
10. I have read the Instrument and Form 43-101F1 (the "Form") and the Technical Report has been prepared in compliance with the Instrument and the Form.
11. I do not have nor do I expect to receive a direct or indirect interest in the Kobada Gold Project, and I do not beneficially own, directly or indirectly, any securities of African Gold Group, Inc or any associate or affiliate of such company.
12. As at the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at El Arish, QLD, Australia on the 29th day of February, 2016.

John Dunlop

John Dunlop
Principal Consultant
John Dunlop and Associates Pty Ltd

CERTIFICATE OF QUALIFIED PERSONS

To accompany the technical report entitled:

"Kobada Gold Project, Feasibility Study" dated effective February 3, 2016 (the "Technical Report").

I Michael Braaksma, MAusIMM CP(Met), do hereby certify that:

1. I am the Consulting Process Engineer with the firm of Gekko Systems Pty Ltd. My business address is 323 Learmonth Road, Ballarat 3350, Victoria, AUSTRALIA.
2. I am a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy. My member number is 112031
3. I am a graduate of the University of New South Wales, Sydney, Australia and hold a B.Eng (Chemical Engineering) (1993)
4. I have practiced my profession continuously since 1994. I have 22 years of experience in operating and designing mineral processing plants.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I have not personally visited the Kobada Project area.
7. I am responsible for Sections 13, 17, and jointly responsible for Sections 1, 21, 24 to 26 of the Technical Report and these sections have been prepared in compliance with the Instrument. I have reviewed and made appropriate cross checks of the remaining sections of the Technical Report.
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12. As at the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Ballarat, Victoria, Australia on the 24th day of February, 2016.

Michael Braaksma
Consulting Process Engineer
Gekko Systems Pty Ltd

CERTIFICATE OF QUALIFIED PERSONS

To accompany the technical report entitled:

"Kobada Gold Project, Feasibility Study" dated effective February 3, 2016 (the "Technical Report").

I Brian Wolfe, MAIG, do hereby certify that:

1. I am a Principal Consultant with the firm of International Resource Solutions Pty Ltd. My business address is 71 Watkins St, White Gum Valley, WA6162, AUSTRALIA.
2. I am a Member of the Australian Institute of Geoscientists (AIG), Member No 4629.
3. I graduated with a BSc Degree (Hons) in Geology in 1992 from the National University of Ireland, Dublin, and hold a Postgraduate Certificate in Geostatistics from Edith Cowan University (2007).
4. I have practiced my profession for a total of 22 years since 1992. I have experience in exploration geology, mining geology and geostatistical modelling and estimation of mineral resources
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I have personally visited the Kobada Project area.
7. I am responsible for Section 14, and jointly responsible for Sections 1, 7, 8, 25 and 26 of the Technical Report and these sections have been prepared in compliance with the Instrument. I have reviewed and made appropriate cross checks of the remaining sections of the Technical Report.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.
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12. As at the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia on the 29th day of February, 2016.

Brian Wolfe
Principal Consultant
International Resource Solutions Pty Ltd

CERTIFICATE OF QUALIFIED PERSONS

To accompany the technical report entitled:

"Kobada Gold Project, Feasibility Study" dated effective February 3, 2016 (the "Technical Report").

I Andrew Chubb, MAIG, do hereby certify that:

1. I am a Principal Consultant with the firm of Obsidian Geological. My business address is P.O Box 58562 Dubai, United Arab Emirates.
2. I am a Member of the Australian Institute of Geoscientists (AIG), Member No. 6421.
3. I graduated with a BSc Degree (Hons) in Geology in 2000 from the University of New England, Armidale, Australia.
4. I have practiced my profession for a total of 15 years since 2001. I have experience in exploration geology and mining geology.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I have personally visited the Kobada Project area.
7. I am responsible for Sections 9 to 12 and jointly responsible for Sections 1, 8, 25 and 26 of the Technical Report and these sections have been prepared in compliance with the Instrument. I have reviewed and made appropriate cross checks of the remaining sections of the Technical Report.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.
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12. As at the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia on the 29th day of February, 2016.

Andrew Chubb
Principal Consultant
Obsidian Geological Ltd.

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1 SUMMARY

1.1 Introduction

This report details a Feasibility Study (FS) for the Kobada Gold Project. Kobada is an advanced gold exploration project. The report sets out the Mineral Resource Estimate, Mineral Reserve Estimate, production schedules and details the capital and operating expenditures over the life of the project. The report culminates in a full economic analysis of the Project's value.

Table 1.1 summarizes the feasibility study findings.

Table 1.1
Feasibility Study Facts and Statistics

Category	Description
Property Name	Kobada
Principal Commodity	Gold
Company Name	African Gold Group, Inc.
Owner/Holder	AGG Mali SARL
Mineral Tenure	80 to 90% ownership of mining license with the balance held by the Government of Mali.
Nearest Population Centre	Kobada Village
Mine Location	Cercle de Kangaba, southern Mali
Topography	Rolling laterite plains
Climate	Dry tropics
Historic Production	Artisanal mining only, no recorded history
Reason for NI 43-101 Technical Report	Feasibility Study
Estimation Type	Multiple Indicator Kriged
Measured Mineral Resource	11.0 Mt at 1.1g/t Au for 380 koz
Indicated Mineral Resource	24.4 Mt at 1.1g/t Au for 835 koz
Inferred Mineral Resource	32.8 Mt at 1.0g/t Au for 1,024 koz
Proved and Probable Reserve	12.7 Mt at 1.25g/t Au for 511 koz
Mine Life	8 Years
Mining Method	Open Pit, 40t Haul Trucks, Wheel Loaders, Excavators
Processing Method	Gravity concentration and intensive leaching
Overall Processing Recovery	80 to 85%
Metal Price	US$1,200 per once
Mining Cost	US$2.35/tonne (average)
Processing Cost	US$6.55 per tonne processed
G&A Cost	$3.54 per tonne processed
Total Pre-production Capital Cost	US$45.5 million
Total Cash Flow after taxation	US$122 million (90% attributable to AGG)
Net Present Value after tax (Discount rate = 5%)	US$86 million
Internal Rate of Return (IRR)	43%
Cash Costs per ounce (LOM average)	US$ 557 per ounce (exclusive of royalties)
All in sustaining cost (LOM average)	US$ 788 per ounce

1.2 Location

The Kobada Gold Project is located in southern Mali, approximately 115km south-west of Bamako and adjacent to the Niger River and the international border with Guinea.

Figure 1.1
Location of Kobada Gold Project

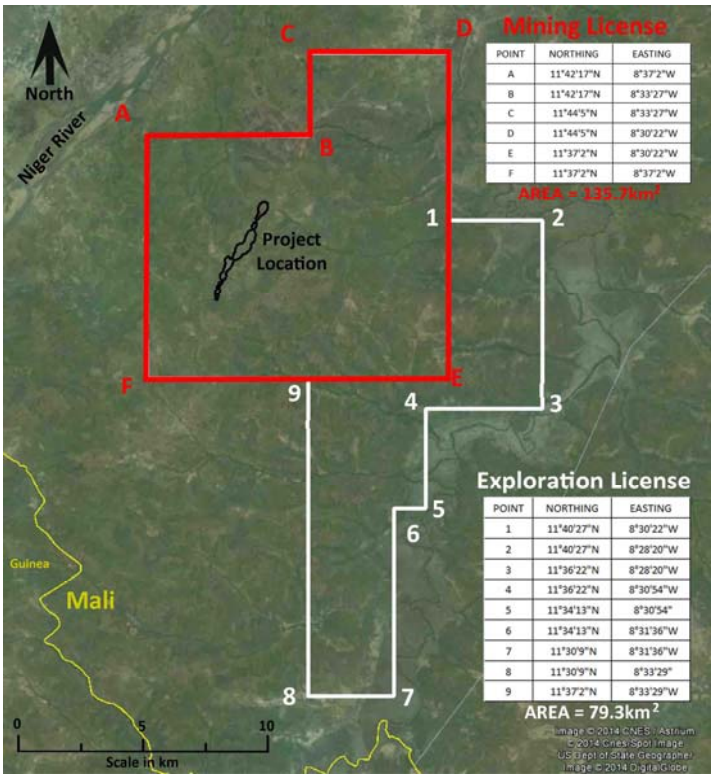
source AGG 2015

1.3 Mineral Tenure

The Project is based on one mining permit and one exploration permit as shown in Figure 1.2.

Figure 1.2

Location of Kobada Project Concessions



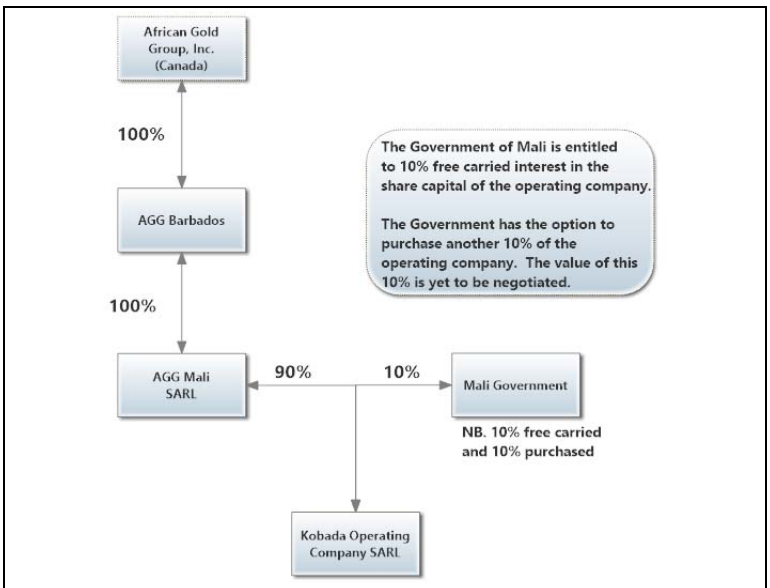
Source: AGG, November 2014

1.4 Ownership

The Project is 100% owned by AGG Mali SARL, the local Malian Company that is a 100% owned subsidiary of African Gold Group, Inc. via AGG (Barbados) Limited. The corporate structure is illustrated in Figure 1.3.

Figure 1.3

Ownership Structure



source AGG 2015

AGG Mali SARL owns both the exploration concession and the mining concession. The Government of Mali has the right to a 10% free carried interest in the Malian Company that will operate the project (Kobada Operating Company SARL). The mining license will ultimately be held in this subsidiary, and this corporate structure is yet to be established.

The Government also has an option to purchase a further 10% of the operation. The consideration for the purchase is yet to be established via negotiations with the Government. The Feasibility Study includes the 10% free carried interest in the economic analysis, but excludes the purchased interest.

The operating company will be established prior to the commencement of production.

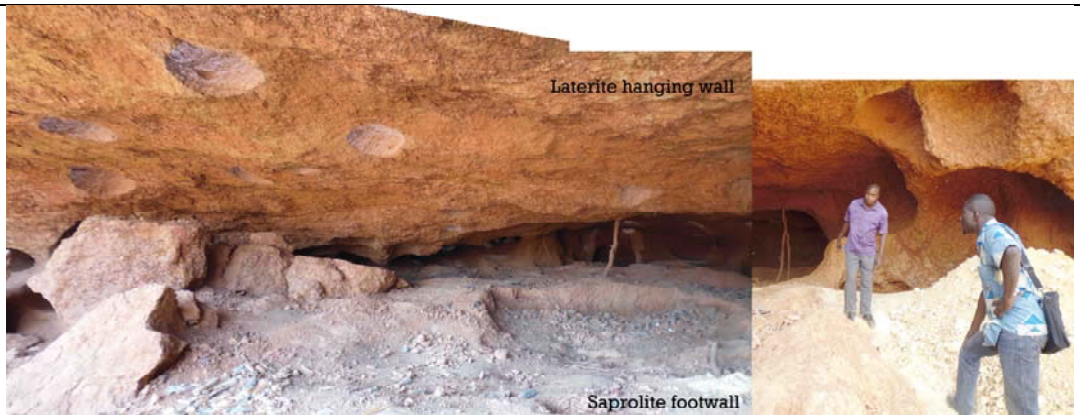
1.5 Geology and Mineralization

The Project is situated on the western flank of the Bougouni Basin composed primarily of sedimentary rocks with minor tholeiitic volcano-sedimentary intercalations. The sediments were deposited in a broad trough during the early Proterozoic Birimian period (2.2 Ga to 1.8 Ga).

A large laterite plateaus cover most of the project area and is underlain by saprolite. The laterite horizon is typically three to four metres thick and generally presents a stark contrast to the saprolite. The saprolite, whilst exceptionally variable in colour from purple to brown, orange, cream and white, shows only very slight variation from what is now a clay (mudstone precursor) to a fine silty clay (fine siltstone precursor).

Gold mineralization is present in the laterite, saprolite and quartz veins that comprise the project. There are also placer style deposits in the region, although these have largely been exploited by artisanal miners. Figure 1.4 illustrates the contact between the laterite and saprolite ore bodies.

Figure 1.4
Mineralized saprolite and laterite



Source: AGG, Nov 2014

1.6 Mineral Resources

Resource estimation was completed by International Resource Solutions Pty Ltd in November 2015. The estimation methodology utilised Multiple Indicator Kriging (MIK)

to estimate gold grade within a mineralized shape that defined the broad zone of mineralization.

The Measured and Indicated Mineral Resource was estimated at 35.4 million tonnes at an average grade of 1.1 g/t Au, and contained 1.215 million ounces. The Inferred Mineral Resource contained a further 1.024 million ounces. The applied lower cutoff grade was 0.3g/t Au.

Table 1.2 summarises the Mineral Resource estimate for the Kobada Gold Project.

Table 1.2
Kobada Gold Project Mineral Resource, November 2015

Mineral Resource Category	Lower Cutoff Grade (g/t Au)	Tonnes (Mt)	Average Grade (g/t Au)	Gold Metal (kozs)
Measured	0.3	11.0	1.1	380
Indicated	0.3	24.4	1.1	835
Total M&I	**0.3**	**35.4**	**1.1**	**1,215**
Inferred	0.3	32.8	1.0	1,024

The Feasibility Study only considers the processing of oxide ore types and Table 1 provides a breakdown of the oxide ore component of the Mineral Resource above a cutoff grade of 0.3g/t.

Table 1.3
Oxide Mineral Resource – November 19, 2015

Mineral Resource Category	Oxide Mineral Resource		
	Mt	Au g/t	koz
Measured and Indicated	18.1	1.1	622
Inferred	10.3	0.9	290

NB: Resource is quoted above 0.3g/t Au cutoff grade

1.7 Mineral Reserve Estimate

The Proved and Probable Mineral Reserve was estimated to be 12.7 million tonnes at 1.25 g/t gold containing 511,000 ounces of gold. The reserve was reported within the optimized pit design and above a cutoff grade of 0.53g/t Au. The final pit physical characteristics are summarized in Table 1.4 and illustrated in Figure 1.6.

Table 1.4
Mineral Reserve Statement – February 3, 2016

Reserve	Ore Tonnes	Grade	Contained	Waste tonnes	Strip Ratio	Total Material	Total Material
Category	Mt	Au g/t	koz	Mt	W:O	Mt	MBCM
Proved	5.7	1.22	225				
Probable	7.0	1.27	286				
Total	12.7	1.25	511	41.8	3.28	54.5	26.8

NB: “Mt” denotes millions of metric tonnes, and “MBCM” means millions of bank cubic metres.

While the Mineral Reserve comprised only material from the Measured and Indicated Resource, there remains an important opportunity to improve the resource category of the large Inferred Mineral Resource immediately to the north and south of the reserve pits. The Company plans to complete this resource upgrade from the cash flow of a producing mine.

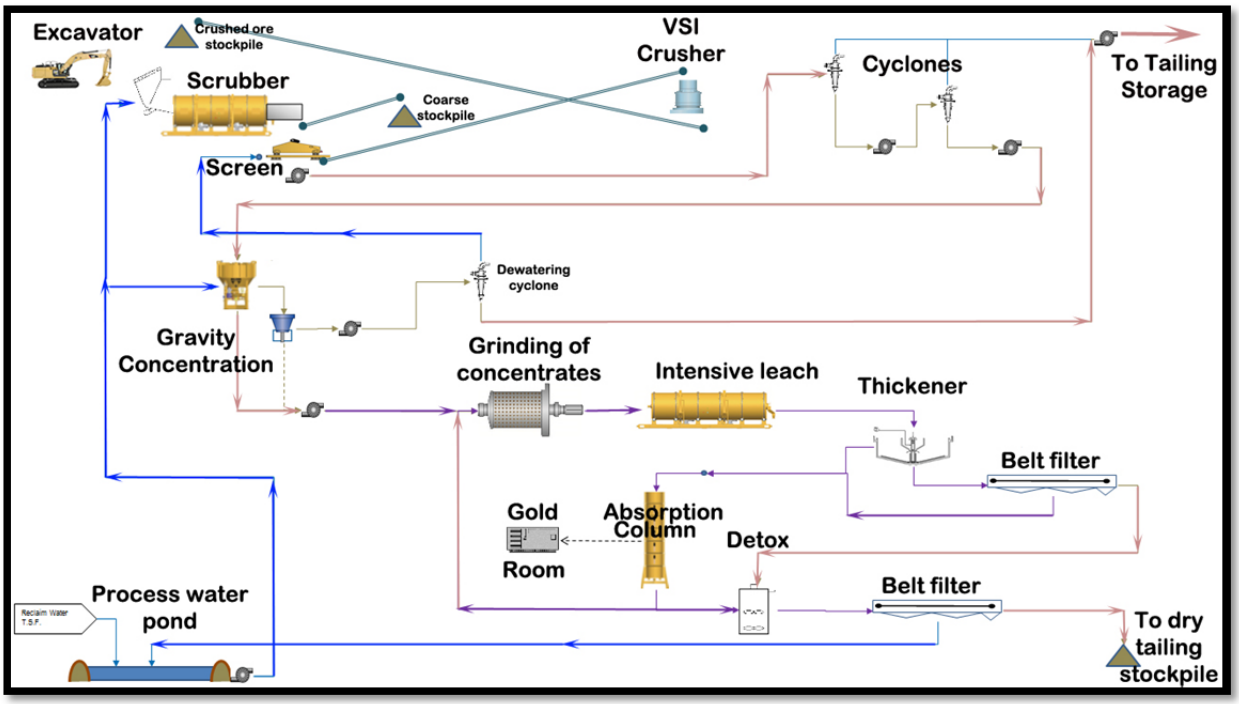
1.8 Mineral Processing

Further test work was completed as part of the Feasibility Study. A 305kg sample of saprolite ore was obtained from 64m below surface to examine the metallurgical response of ore near the base of oxidation. The calculated recovery of gold for this sample was 80.1%. While this was lower than the 2014 testwork program, which recovered 89% of contained gold, the 2014 sample was sourced from the top 20m of the saprolite orebody. The difference between the two is likely indicative of the recovery response throughout the total depth of the saprolite orebody.

Independent test work completed since 2009 now totals 1.2 tonnes of saprolite ore over three separate samples by two different independent consultants. Each provides confirmation of the amenability of the orebody to physical separation and leaching techniques.

The process plant flow sheet is illustrated in Figure 1.5. It consists of a large scrubber to create ore slurry and remove larger rocks (+40mm). Screening of the -40mm from the scrubber separates the fine material (-1.18mm) and directs the coarse ore to further crushing. The -1.18mm slurry is then passed to a bank of hydro-cyclones for the pre-concentration stage. This pre-concentration process rejects up to 70.5% of the feed mass while recovering 94% of the gold.

Figure 1.5
Simplified Processing Flow Diagram



Source: Gekko, Jan 2016

Pre-concentration is to be followed by gravity concentration consisting of InLine Pressure Jigs and Knelson centrifugal concentrators. The gravity circuit is planned to produce a concentrate representing 5% of the run of mine feed. The tailings from the pre-concentration and gravity concentration are pumped directly to the inert Tailings Storage Facility. This tailings material has had no chemical addition and can be disposed of in an unlined impoundment.

The concentrates are then to be passed to the grinding circuit. This comprises a ball mill in closed circuit with cyclones to produce an 80% passing 125 micron grind size. Cyanide is also added to the ball mill to increase leaching residence time. Grinding is followed by intense leach reactors with leaching occurring in a high oxygen environment for around 10 hours.

After leaching, the slurry density is increased in a thickener before the remaining leach solution is recovered from a belt filter. Pregnant solution will be fed directly to an AuRIX resin absorbtion column to extract the gold from the solution and return the barren solution to the process. The solid residue from the belt filter will then be treated in an SO_2 and O_2 cyanide destruction circuit (detox) to reduce the tailings cyanide concentration to below the International Cyanide Management Code (ICMC) requirements. This residue will be stored as dry tailings in a purpose built lined facility that will be encapsulated within the waste rock dumps.

A resin stripping, electro-winning and gold smelting system to produce dorè are located in a secure gold room. The process plant has been designed to treat 1.6 million tonnes of ore per annum, with gold produced as dorè bars ready for shipment to a refinery outside Mali. The process plant selected is expected to recover an average of 82% of the gold contained in ore, and the test work indicated that a constant tailings grade of 0.22 g/t Au can be expected. This means that a higher recovery may be achieved at higher process feed grades.

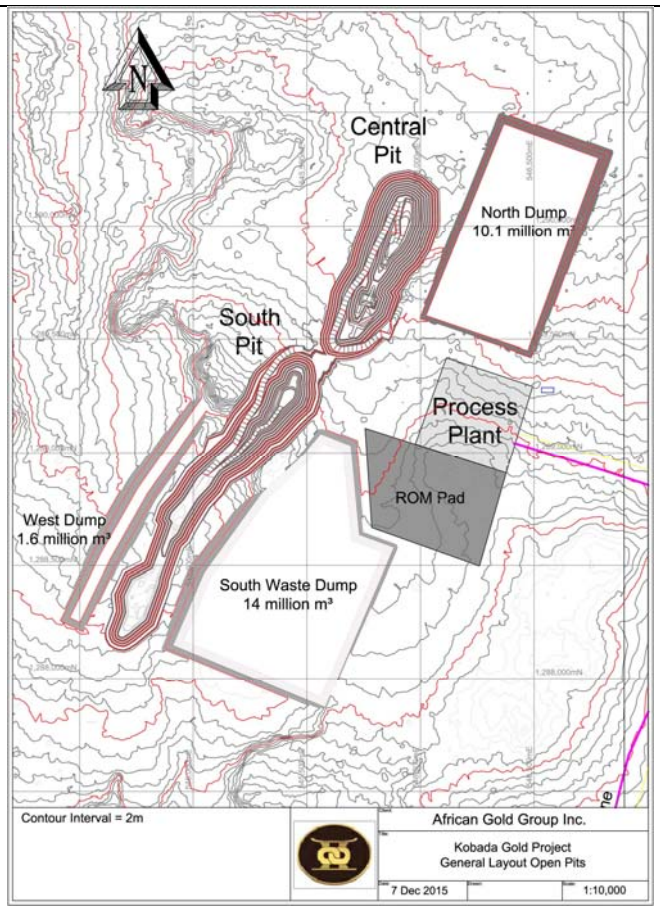
Process plant commissioning is planned to commence in September 2017. Detailed design of the plant and ordering of longer lead time items is planned to commence in July 2016 in order to meet this commissioning target.

1.9 Mining

The Kobada deposit will be mined by conventional open pit mining methods utilizing 40 to 50 tonne articulated dump trucks and $5m^3$ excavators. The saprolite is a free digging rock mass, and there is expected to be only minor use of explosives in blasting some areas of the overlying laterite cap. All mining operations have been assumed to be completed by a mining contractor.

The mine production schedule is based on delivering 1.6 Mt of ore for processing per annum. Due to the relatively small size of the equipment proposed, there is opportunity to tailor the mining program to suit the ore processing and waste stripping schedule.

Figure 1.6
General Mine Layout



Source AGG 2015

1.10 Economic Assessment

Table 1.5 summarises the main economic parameters from the cash flow model. The Project free cash flow was estimated to be US$121.5 million and net present value at 5% discount rate was US$86 million. AGG's 90% equity distribution was US$109.2 million that had a $NPV_{5\%}$ of US$77 million. The Governments free carried distribution was US$12.3 million.

Table 1.5
Financial Model Summary

Item	Project	AGG 90% equity distribution	Gov. 10% equity distribution
Operating Cashflow (US$MM)	$121.5	$109.2	$12.3
NPV @ 5.0% (US$MM)	$86.1	$77.1	$9.0
NPV @ 10.0% (US$MM)	$60.8	$54.1	$6.7
Internal Rate of Return	43%	41%	N/A
Payback Period (years)	2.6		
Maximum negative equity (US$MM)	$43.9		

The payback period for the project was 2.6 years from initial capital expenditures in July 2016, until positive cumulative cash flow in January 2019. This assumes that capital expenditures commence in July 2016 and commissioning is achieved in September 2017 with strong cash flows from this time.

Examination of the Project's cash flow to the key variable of gold price illustrates the robust nature of the project. Table 1.6 details the undiscounted cash flow, discounted cash flow, internal rate of return and payback period at varying gold prices.

At the base gold price of US$1,200 per ounce, the Kobada Project exhibits an NPV of US$86 million, at a discount rate of 5%. The internal rate of return (IRR) was 43% and payback of capital expenditures achieved in 2.6 years. The NPV and IRR are based on the project's free cash flow which is reported net of all costs and Malian taxes.

Table 1.6
Sensitivity to Gold Price

Gold Price	US$/oz	900	1,000	1,100	1,200	1,300	1,400
After tax cashflow	US$M	38	68	95	121	148	174
$NPV_{5.0\%}$ after tax	US$M	22	45	66	86	106	126
Internal Rate of Return	%	17%	28%	36%	43%	49%	55%
Payback Period	years	5.8	3.4	2.8	2.6	2.4	2.3

NB: Undiscounted cash flow, $NPV_{5\%}$, IRR and payback period are all calculated on the Project's free cash flow after taxation at each stated gold price.

1.11 Conclusion

This Project's positive economic outcome is largely a result of the low processing costs associated with the plant design selected, which was estimated at US$6.55 per tonne processed.

The key component in ore processing is the pre-concentration followed by gravity concentration which effectively can achieve up to 95% rejection of the feed mass while recovering more than 82% of the gold. It is this attribute that rejects barren material with relatively low energy inputs. Only the concentrate is ground to less than 180µm and leached, which is the energy intensive part of the ore processing flowsheet. This is the reason the processing cost per tonne is comparatively low compared to traditional CIP/CIL operations.

Mining costs of US$2.35 per tonne mined and G&A costs of US$3.54 per tonne of ore are considered to be of a similar quantum to other oxide projects of similar scale. There is an opportunity to improve the economic outcomes through lowering the waste to ore stripping ratio during the first half of the mine life, although this is contingent on exploration success. It is AGG's stated aim to focus exploration effort on the shallow (less than 25 metres below surface) mineral potential in order to extend the number of years that are maintained with a low strip ratio. Whilst not demonstrable at this time, the strategy is sound.

2 INTRODUCTION

Qualified Persons (QP's) from International Resource Solutions Pty Ltd (IRS), Obsidian Geological Limited (OGL), John Dunlop and Associates Pty Ltd (JDA), and Gekko Systems Pty Ltd. (Gekko) have prepared this technical report (the Report) on the Kobada Gold Project (the Project) in Mali, Africa.

African Gold Group, Inc. (AGG) indirectly owns 100% of AGG Mali SARL (AGG Mali), the Malian holding company for the Kobada Gold Project.

2.1 Scope of the Report

The Report describes the following items for the Kobada Gold Project:

- an updated statement of the Mineral Resource Estimate;
- all processing test work and process design completed to date;
- mining engineering, including pit optimisation and pit designs;
- production scheduling;
- operating and capital cost estimation; and
- cash flow model.

2.2 Qualified Persons

The following serve as the qualified persons for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Mr Brian Wolfe, B Sc (Hons), P Grad Cert (Geostats), MAIG, International Resource Solutions Pty Ltd, Independent Resource Geologist;
- Mr Andrew Chubb, B Sc (Hons), MAIG, Obsidian Geological Ltd, Independent Exploration Geologist;
- Mr Michael Braaksma, MAusIMM(CP), Gekko Systems Pty Ltd, Independent Metallurgist; and
- Mr John Dunlop, FAusIMM(CP), John Dunlop and Associates Pty Ltd, Independent Consulting Mining Engineer.

2.3 Site Visits

Mr. Brian Wolfe conducted a site visit with Mr. Declan Franzmann between 22nd and 27th May 2014. The site visit included review of the resource area, examination of the geology apparent in artisanal working, review of diamond core stored at site and general review of drilling.

Mr Andrew Chubb visited the site on numerous occasions between March and September 2015. Mr Chubb designed and oversaw the execution of the 2015 drilling program.

Mr John Dunlop conducted a site visit between 17 and 19 February 2015. The site visit included review of the resource area, examination of the geology apparent in artisanal working, review of diamond core stored at site, inspection of the tailings storage facility site and general inspection of the project area.

Mr Braaksma did not undertake a site visit to the Kobada Gold Project. It was not considered essential for the determination of the processing methodology, and given the remote nature of the site, the cost and time required for a site visit was better utilised in completing the metallurgical work. It should be noted that the samples used in metallurgical test work were collected by Mr. Andrew Chubb, an independent QP.

2.4 Principal Sources of Information

Reports and documents listed in Section 3 and Section 27 of this Report were used to support preparation of the Report. Additional information was provided by AGG personnel as requested. Supplemental information was also provided to the QPs by third-party consultants retained by AGG in their areas of expertise.

2.5 Previous Technical Reports

AGG has filed the following Technical Reports for the Project:

- Technical Report (NI 43-101), 15 May 2008, "A Review and Resource Estimate for the Kobada Project, Mali, West Africa."
- Technical Report (NI 43-101), 19 August 2011, "Kobada Property – Mali, Preliminary Economic Assessment."
- Technical Report (NI 43-101), 2 August 2013, "Mineral Resource Evaluation."
- Technical Report (NI 43-101), 25 November 2014, "Mineral Resource Estimate and Preliminary Economic Assessment."

2.6 Responsibilities

Specific sections of the report that the Qualified Persons are responsible for are provided in Table 2.1 and are repeated in the attached Qualified Persons certificates.

Table 2.1
Responsibility of Qualified Persons

Qualified Person	Association	Responsible for Sections	Co-Responsible for Sections
John Dunlop	FAusIMM (CP)	2-6, 15-16, 18-22	1,21,25,26
Andrew Chubb	MAIG	9-12	1,8,25,26
Brian Wolfe	MAIG	7, 14	1,8,25,26
Michael Braaksma	MAusIMM (CP)	13,17	1,21,25,26

2.7 Abbreviations

A full listing of abbreviations used in this report is provided in Table 2.2 below.

Table 2.2
List of Abbreviations

Term	Description	Term	Description
$ or US$	United States of America dollars	Kg	kilogram
µm	Micrometres	kg/t	kilogram per tonne
2D	two dimensional	Km	kilometres
3D	three dimensional	km²	square kilometres
AGG	African Gold Group, Inc.	l/hr/m²	litres per hour per square metre
AAS	atomic absorption spectrometer	M	million
Au	Chemical symbol for gold	m	metres
BCM	bank cubic metres	Ma	Million years
BQ	size of diamond drill rod/bit/core	MIK	Multiple Indicator Kriging
ccdf	conditional cumulative distribution function	mL	millilitre
CC	correlation coefficient	mm	millimetres
cfm	cubic feet per minute	MMI	mobile metal ion
CFR	Cost and Freight for the carriage of goods by sea to the port of destination	Mtpa	million tonnes per annum
CIM	The Canadian Institute of Mining, Metallurgy and Petroleum	Mt	Million metric tonnes
cm	Centimetre	N (Y)	Northing
Co	relative nugget	NATA	National Association of Testing Authorities
C1,C2	relative structures	NPV	net present value
cusum	cumulative sum of the deviations	NQ	size of diamond drill rod/bit/core
CV	coefficient of variance	OK	Ordinary Kriging
DDH	diamond drillhole	P80 -75µm	80% passing 75 micrometres
dmt	Dry metric tonne	PAL	pulverize and leach
DTM	digital terrain model	PFS	Prefeasibility Study
E (X)	Easting	ppb	parts per billion
EDM	electronic distance measuring	ppm	parts per million
EGL	Effective grinding length	PVC	poly vinyl chloride
EGRG	Extended Gravity Recoverable Gold	QC	quality control
EV	expected value	Q-Q	quantile-quantile
FEL	front end loader	RAB	rotary air blast drilling
FOB	Free on board for the carriage of goods by sea to the port of destination	RC	reverse circulation drilling
g	Gram	RL (or Z)	reduced level
g/m³	grams per cubic metre	ROM	run of mine
g/t	grams per tonne	RQD	rock quality designation
Ha	Hectares	SAG	semi-autogenous grinding
HQ	size of diamond drill rod/bit/core	SD	standard deviation
h	Hours	SGS	Société Générale de Surveillance
ID	Inverse Distance weighting	SMC	SAG milling comminution
ID²	Inverse Distance Squared	SMU	selective mining unit
ILR	InLine Leach Reactor	T	Metric tonnes
IPJ	InLine Pressure Jig	tonnes	Metric tonnes
IPS	integrated pressure stripping	ton	Short ton
IRR	internal rate of return	t/m³	tonnes per cubic metre
ISO	International Standards Organisation	VSI	Vertical shaft impactor
ITS	Inchcape Testing Services	Y	Year

2.8 Glossary of Terms

A full glossary of terms used in this report is provided in Table 2.3

Table 2.3
Glossary of Terms

Term	Description
cell model	Array of 3D polyhedra that are used to reference the location of stored data such as geological codes or assay data. Also known as a block model.
certified reference material (CRM)	Material of know elemental concentration submitted with drill samples to determine level of accuracy of laboratory results. Also known as a standard.
class	Model field used to store the assigned classification codes
composite	Combination of two or more individual samples, generally to produce a dataset with equal sample support.
estimation variance	Variance of the error made when a value is estimated by another value.
field duplicate	Repeat sample obtained at the time of drilling. Used to test sampling error and laboratory precision.
interpolate	Estimation of a value from a population.
kriging efficiency (KE)	Measure of error estimation when using ordinary kriging (directly linked to kriging variance). Formula: KE = (TBV-KV) / TBV where TBV is the theoretical total block variance of the model cells within the domain and KV is the kriging variance.
lag	Distance between pairs of samples in a variogram.
major direction	Direction with longest spatial variability, as derived from a variogram.
minor direction	Direction with shortest spatial variability, as derived from a variogram.
nugget	Degree of geostatistical variability over a short distance. Usually measured using downhole samples, which represent the closest spaced data.
ordinary kriging (OK)	Linear geostatistical estimation method which aims to minimise error variance.
parent cell	Largest polyhedra used in a cell model. Can be split into subcells to honour interpreted geological boundaries.
quantile-quantile (QQ)	Values extracted at pre-determined positions in the cumulative distribution of two populations to compare the overall population characteristics. Quantile can be set to any interval (10 per population are called deciles; 100 per population are called percentiles.) When plotted as a scatter relative to a 1:1 bisector, easily demonstrates differences in either population (QQ plot is also known as a ranked scatter plot.)
relative percentage difference (RPD)	When plotted, compares numerous paired data by displaying the average value of the pairs on the x-axis and the relative difference as a percentage between the two data on the y-axis.
sample support	The space on which an observation is defined (i.e. length of a sample interval, volume of sampled material, percentage recovery)
semi-major direction	Direction with second longest spatial variability, as derived from a variogram.
sill (of a variogram)	The limit of the variogram, indicating the total variance of a population.
slope of regression	Measure of error estimation when using ordinary kriging. Regression between actual and estimated grades. Formula: R = (BV-KV+μ) / (BV-KV+2 μ) where BV is the theoretical variance of the model cells within the domain, KV is the kriging variance and μ is the Lagrange multiplier.
standard	See certified reference material.
subcell	Smaller polyhedra within a parent cell. Used to optimise geological boundary definition in the cell model.
total block variance	Sill (variance of population) minus F-value.
variogram (semivariogram)	A mathematical model fitted to a function, representing the spatial variability of the component of interest (model of the spatial continuity.) Calculated as the expected squared increment of the values at given lags.
wireframe	Computer generated surface or polyhedral solid made up of interconnecting triangles.

3 RELIANCE ON OTHER EXPERTS

The authors of this report are not qualified to provide extensive comment on legal issues, including status of tenure, exploration concessions, water rights and surface rights associated with the Kobada Gold Project referred to in this report. Assessment of these aspects has relied heavily on information provided by AGG. This report has been prepared on the understanding that the property is, or will be, lawfully accessible for evaluation, development, mining and processing. Mineral title and ownership details are provided in Section 4 and Appendix A of this report.

Unless otherwise stated, all maps and figures have been sourced and prepared by AGG.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Kobada Gold Project is located in southern Mali, approximately 115km south-west of Bamako and adjacent to the Niger River and the international border with Guinea.

Figure 4.1
Location of Kobada Gold Project

Source: AGG, December 2015

4.2 Mineral Tenure

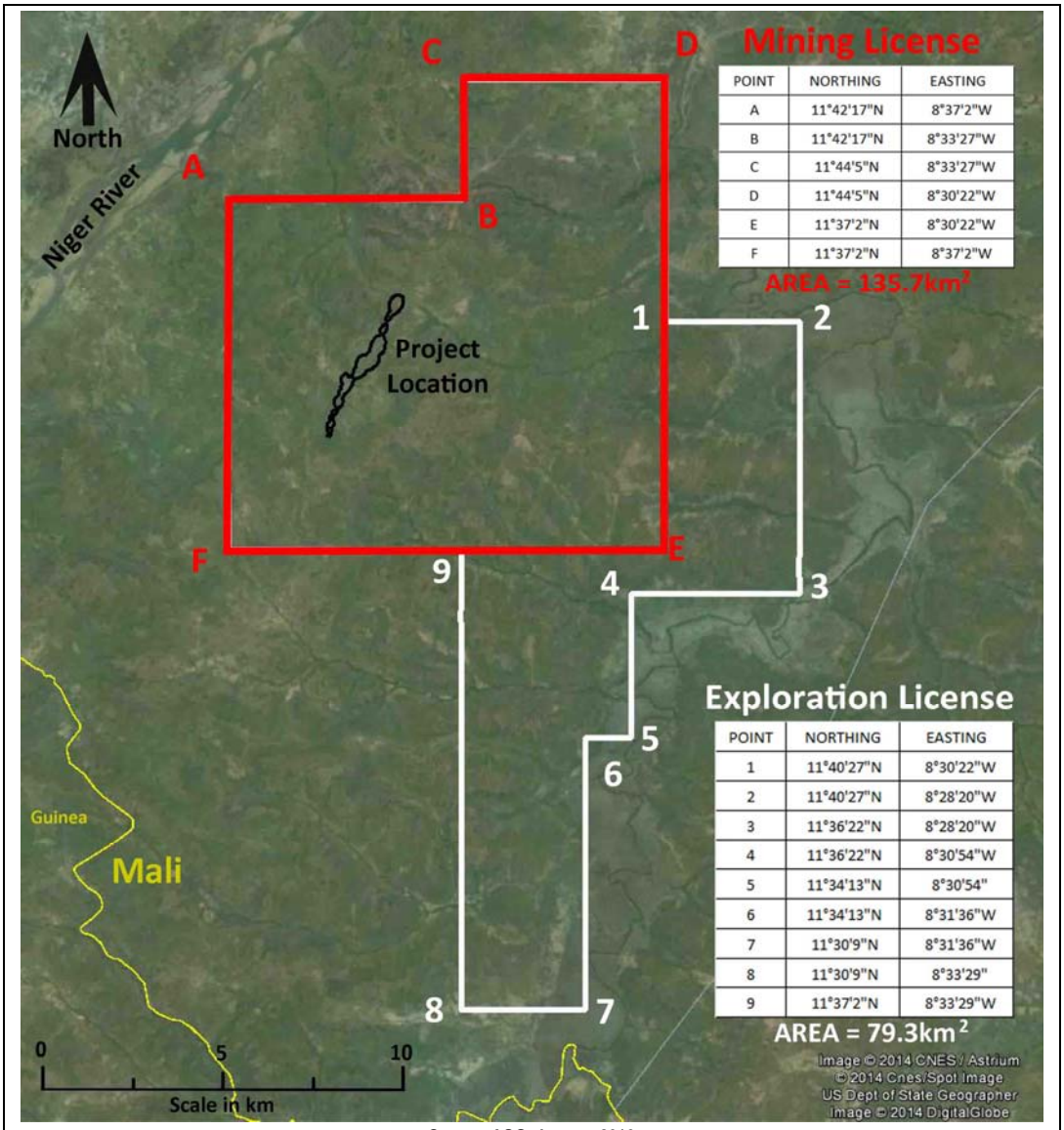
The Project is based on one mining permit and one exploration permit as shown in Figure 4.2.

The mining permit covers an area of 135.7 km^2, as detailed in "Decret No 2015-0528/PM-RM issued on 31 July 2015 and attached in Appendix A.

The remaining area of AGG's concessions is covered by the original exploration permit which was the amalgamation of several permits under "Arrete No 2012-2338/MCMI-SG" on August 9th, 2012. This document is also provided in Appendix A.

Figure 4.2

Location of Kobada Project Concessions



Source: AGG, January 2016

For clarity, the coordinates (UTM) of the mining and exploration licenses are detailed in Table 4.1 and Table 4.2.

Table 4.1
UTM Coordinates of Mining License

POINT	NORTHING	EASTING
A	11°42'17"N	8°37'2"W
B	11°42'17"N	8°33'27"W
C	11°44'5"N	8°33'27"W
D	11°44'5"N	8°30'22"W
E	11°37'2"N	8°30'22"W
F	11°37'2"N	8°37'2"W

Table 4.2
UTM Coordinates of Exploration License

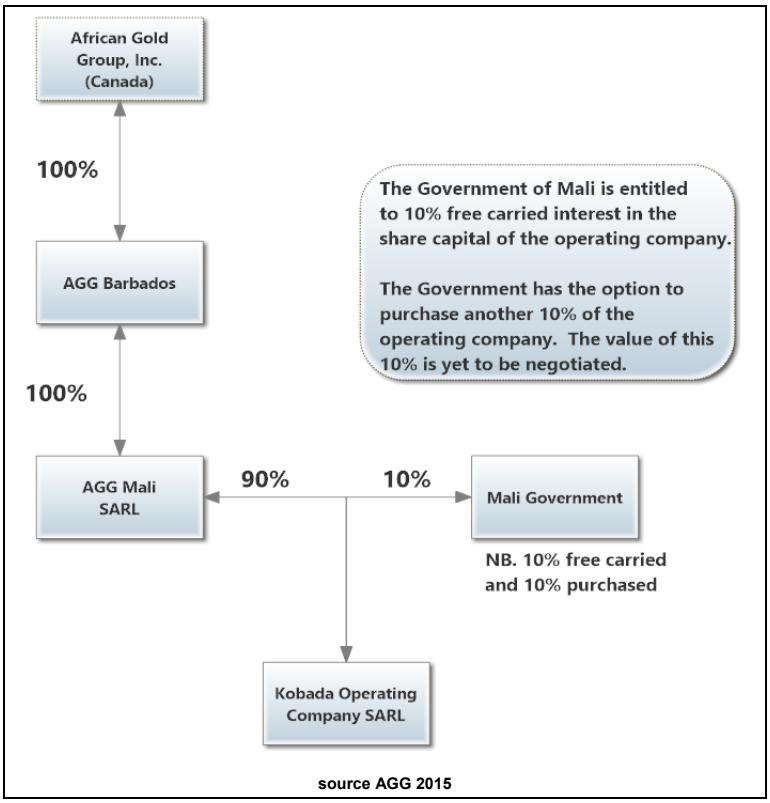
POINT	NORTHING	EASTING
1	11°40'27"N	8°30'22"W
2	11°40'27"N	8°28'20"W
3	11°36'22"N	8°28'20"W
4	11°36'22"N	8°30'54"W
5	11°34'13"N	8°30'54"
6	11°34'13"N	8°31'36"W
7	11°30'9"N	8°31'36"W
8	11°30'9"N	8°33'29"
9	11°37'2"N	8°33'29"W

The Mining License was granted for a period of 30 years, to be renew every ten years. The remaining exploration license requires renewal every three years.

4.3 Project Ownership

The Project is 100% owned by AGG Mali SARL, the local Malian Company that is a 100% owned subsidiary of African Gold Group, Inc. via AGG (Barbados) Limited. The corporate structure is illustrated in Figure 4.3.

Figure 4.3
Ownership Structure



source AGG 2015

AGG Mali SARL owns both the exploration concession and the mining concession. The Government of Mali has the right to a 10% free carried interest in the Malian Company that will operate the project (Kobada Operating Company SARL). The mining license will ultimately be held in this subsidiary, and this corporate structure is yet to be established.

The Government also has an option to purchase a further 10% of the operation. The consideration for the purchase is yet to be established via negotiations with the Government. The Feasibility Study includes the 10% free carried interest in the economic analysis, but excludes the purchased interest.

The operating company will be established prior to the commencement of production.

4.4 Royalties and Encumbrances

The mining permit has been awarded under the 1999 Malian Mining Code. Under this code there is one applicable royalty payable to the Government of Mali. The "Impot Special sur Certain Produits (ISCP)" is a tax levied on gold production at a rate of 3% of gold sales.

4.5 Environmental Liabilities

AGG have advised that there are no current environmental liabilities associated with the project.

4.6 Surface Rights

There is no private ownership of land in the area the Project is located. All surface rights are owned by the Government of Mali and the company has unfettered rights of access.

4.7 Other Obligations

Mineral rights granted by virtue of the 1999 Malian Mining Code can be cancelled or withdrawn by the Government for the following reasons:

- when a written warning has not been acted upon for a period of ninety (90) days for mining permit, and sixty (60) days in the case of all other mineral rights.
- non-compliance with exploration budgets and programs;
- delay or suspension of activities in exploration without a valid reason for more than one year;
- delay or suspension of mining activities for more than two years after the establishment of the mining company without authorization from the Malian Government;
- non-payment of taxes, royalties and duties related to mining activities; and failure to comply with obligations regarding environmental protection and conservation, and reclamation of sites that have been disturbed.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography

The topography is generally flat, and the average altitude is 350m above sea level. Only a few lateritic plateaus with steep rise 50m above the surrounding erosion plain. Drainage is predominately to the east towards the sinuous northerly running Fie River which defines the eastern edge of the Project's boundary. The Fie River is a tributary of the Niger River which rises in the Fouta Djallon in Guinea, which drains much of north-western Africa.

5.2 Project Access and Proximity to Population Centres

The Project is located approximately 125km in a straight line south-west by south of Bamako. By road from Bamako it is approximately a four hour drive to Kobada. The route is 86km via RN7 and then another 53km from RN7 to the Selingue Dam, all on good quality sealed roads. From Selingue the Project site is a further 48km on an un-sealed road.

The closest population centre of any size is Selingue, 48km to the east of the project site.

5.3 Climate

The climate is characteristic of the Sahelian climatic zone with a dry season from November to May and relatively abundant wet season, with annual rainfall greater than

1,100 mm, from June to September. Temperatures range between 24°C at night and 40°C during the day from March to May and between 15°C at night and 32°C during the day the remaining months.

5.4 Vegetation

The vegetation is characteristic of the Sahelian savannah, with acacia, shea, ficus, baobab, and large trees in the flood plains (bombax, mango trees) and gallery forests of palm trees and liana along oxbow lakes.

During the wet season, the countryside becomes lush and green with grass attaining 2m in height. By December, the vegetation dries up and the villagers burn the grass. The land returns to its brown and yellow semi-desert state.

There is little local wildlife and the region has warthogs, monkeys, antelopes, birds and small variety of snakes (vipers, mambas).

5.5 Workforce

The Project area is populated with people concentrated in several small hamlets, with the closest being Kobada and Foroko. Kobada is around 700m to the west of the planned pit crest, while Foroko is 1.5km from the closest planned open pit working.

The population is made of different ethnic groups namely: Malinké, Fulani and Bambara. They are typically farmers, ranchers, traders, fishermen or artisanal miners.

Many of the less skilled occupations required in a mining operation will be sourced from the local communities. Skilled artisans and professionals will need to be sourced from further away, although the Company plans to provide significant opportunities and education to local communities.

5.6 Infrastructure

The area is accessible via an unsealed track, and travel times from Selingue can vary depending on road conditions. There is a bridge across the Fie River constructed from sea containers, although during the wet season access is restricted to boats ferrying goods and people across the Fie River.

6 HISTORY

6.1 Historical Production

There has been no documented historical production at Kobada, although the artisanal (orpaillage) mining has probably occurred in the area for many decades if not centuries.

6.2 Exploration

The first reported work on exploration in the region and visits of several “orpaillage” (artisanal mining) areas occurred between 1935 and 1937 (E. Julien).

There is no recorded work on the project until the 1980’s. Work was carried out by the Bureau de Recherches Geologiques et Minieres ("BRGM"), the Geological Survey of France. The Kobada Shear Zone was initially identified by geochemistry surveys in 1982. The first drilling was completed in 1988.

In 1995-1996, La Source (a joint venture between Normandy Mining NL and BRGM) conducted a 50-hole, 4,803 m reverse circulation ("RC") drilling program (holes KBD01- KBD50).

COMINOR acquired the property in 2000 and continued to explore the project until Kobada was sold to African Gold Group in 2005. AGG completed 116,870 metres of diamond, RC, air core and augur drilling between 2005 and 2012.

In 2012 the four separate exploration tenements were amalgamated.

A more detailed exploration history is provided in Section 9.1.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located in the Bagoe formation in the north central border of the Birimian rocks units that are part of the Leo Rise in the south part of the North African Craton.

North West African geology is characterized by the Precambrian West-African craton that stabilized about 1800 Ma. It comprises the south Leo (Man) and the north Reguibat Shields (Rises) and the significantly smaller inliers of Kenieba and Kayes. These basement exposures are bounded and separated by northerly trending Pan-African fold belts to the west and by an extensive cover of Late Proterozoic to Phanerozoic sedimentary basins, namely the central Taoudeni basin.

The Leo shield is composed in the southwest, of the Keneman-Man domain (2.7 Ga), an Archean nucleus comprising metamorphic rocks, granitic rocks and catazonal to epizonal metamorphic supracrustal greenstone belts and the Proterozoic Baoule-Moussi domain in the remainder. The Baoule-Moussi contains relics of Archean rocks and the Paleoproterozoic Birimian formations (2.2 Ga). The Birimian consists of narrow elongated belts of mainly epi-metamorphosed volcano-sedimentary formations deformed by:

- D1, the Tangaean Event (2.15 Ga) with NNW trending folds, reverse shear zones (steeply NE dipping and SW verging) and NNW plunging folds and boudins;

- The emplacement of pre to syn tectonic granitic, granodioritic and tonalititic plutons (2.1 Ga);

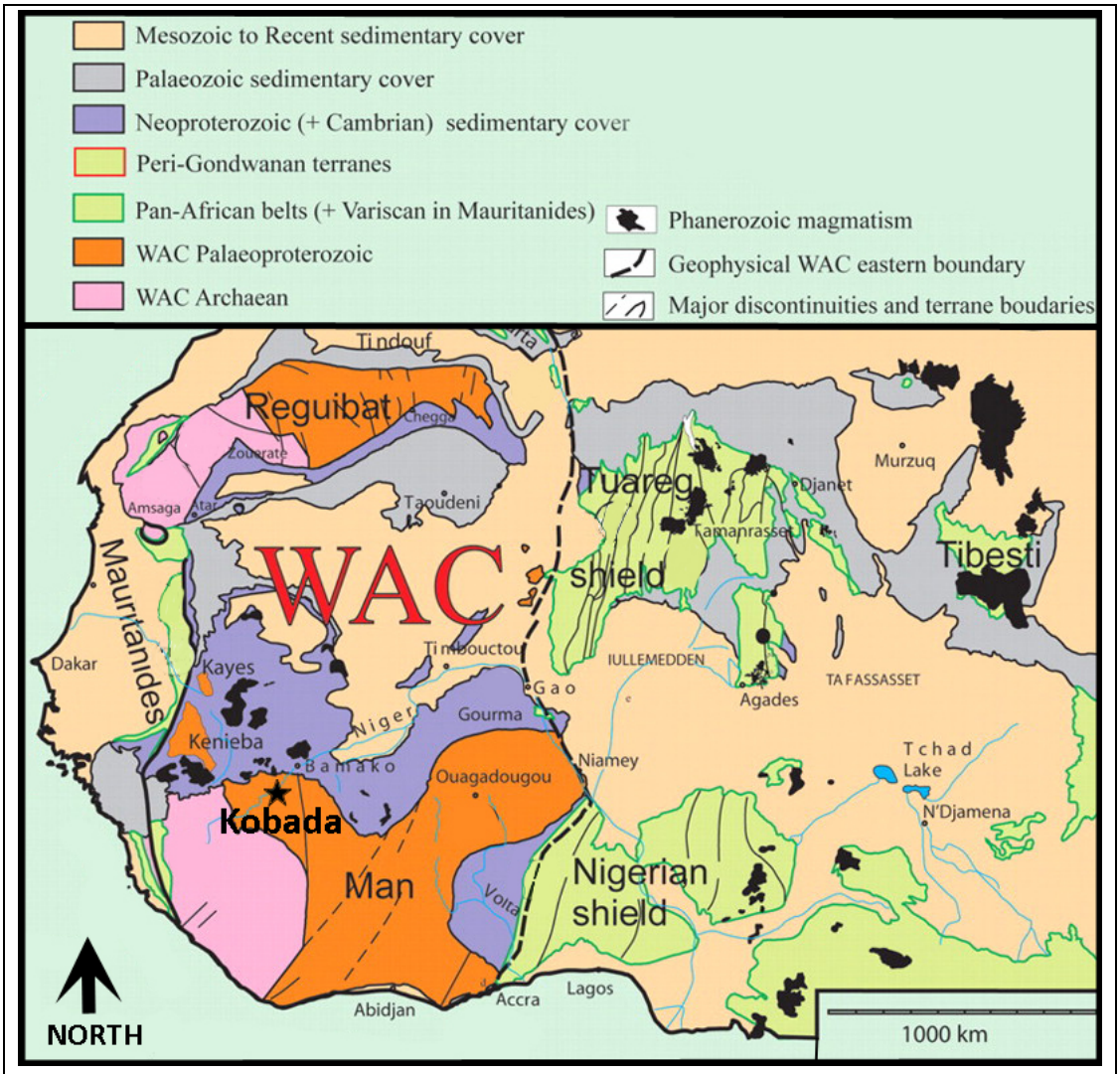
- D2, the Ebumean Orogeny (2,0 Ga) with development of NNE to NE trending, dextral and sinistral reverse shears zones and folds, with a prolonged back arc volcanism during the late stages;

- D3, the Wabo-Tampelse Event, which is a period of N-S shortening with development of WNW dextral reverse thrusts (transport to the north) and E-W folds.

The regional metamorphism reach the greenstone facies with amphibolite facies restricted to the intrusive granitoids contact.

Figure 7.1

West African Craton



Source: AGG, November 2014

The last event in the region (±250 Ma) is the intrusion of NNE trending mafic dykes (dolerite/diorite).

The Massigui Yanfolia region has two volcano-sedimentary series separated by a granitic intrusive unit (Massigui Batholith) oriented northerly and the NNE trending Banifing Shear system (Sassandra);

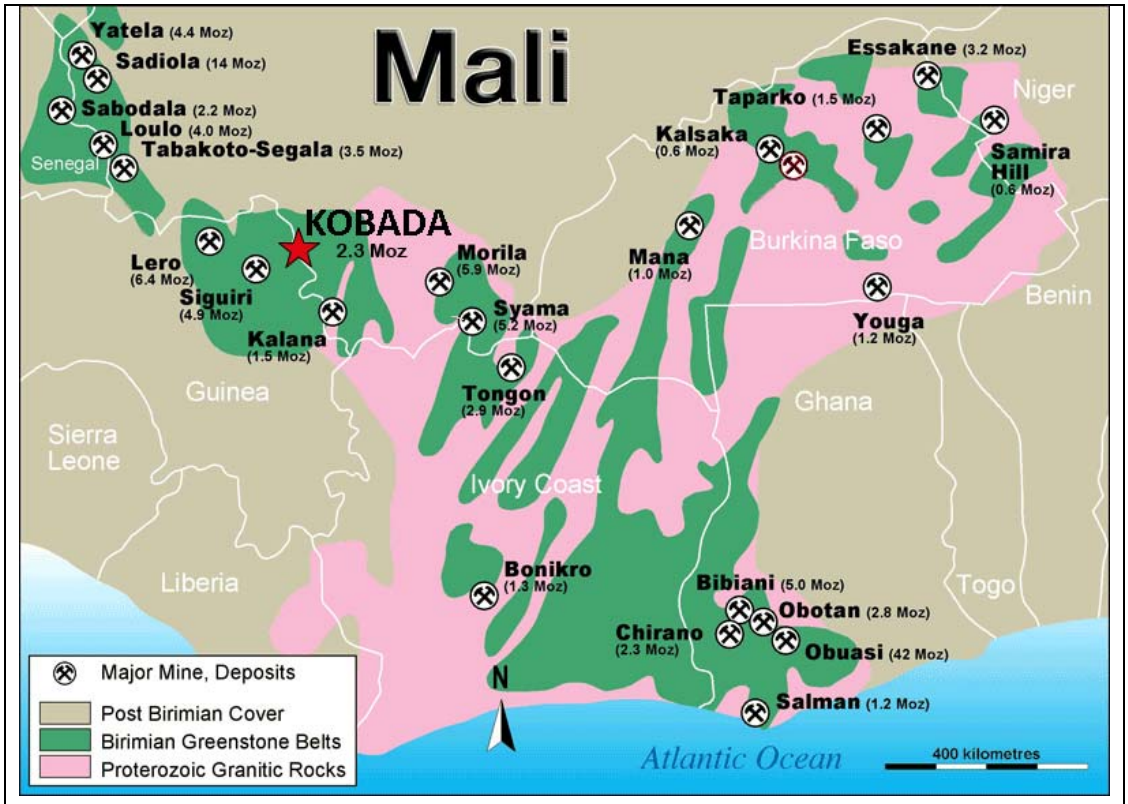
- to the west, the Bougouni-Kekoro Formation is a volcano-sedimentary unit composed of orthoquarzite;
- to the east, the northern portion of the Bagoe Formation is composed of intermediate felsic volcanics with a few rare interlayers of basalt and metasediments.

This formation is divided into three distinct lithological units and the transition between these members is considered gradual:

- the east member is composed of flyschoide unit composed of sandstone to argillite with graphitic and conglomeratic bands;
- the centre member, is composed of quartz litharenite;
- the west member, mostly felsic volcaniclastite comporting chert and manganese units;

Figure 7.2

Regional Geology



Source: AGG, Nov 2014

7.2 Project Geology

The Project is situated on the western flank of the Bougouni Basin composed primarily of sedimentary rocks with minor tholeiitic volcano-sedimentary intercalations. The sediments were deposited in a broad trough during the early Proterozoic Birimian period (2.2 Ga to 1.8 Ga). The Bougouni Batholith, a large felsic intrusion, occupies the central part of the basin and is known to appear some 25 km NE and SW of the Project area. Small intermediate intrusions, mainly diorite to granodiorite, also occur within the basin.

The terrane is intensely lateritized, and with the exceptions of the granitic rocks, the protolith is rarely identifiable in outcrop. Large laterite plateaus cover most of the area. The underlying saprolite is exposed below the plateau boundaries and is generally of a yellowish ochre col-

our, whitened in numerous places by intense kaolinization, likely of hydrothermal origin. Rare outcrops occur where resistive quartz veins protected their host rocks, and the schistose nature of the sediments can then be observed. Drilling intersected interbedded greywackes, siltstones and mudstones.

The saprolite, whilst exceptionally variable in colour from purple to brown, orange, cream and white, shows only very slight variation from what is now a clay (mudstone precursor) to a fine silty clay (fine siltstone precursor). A thin section study of 9 core samples show the lithology of the rock types found in the core to be slates and phyllites. There are no marker horizons and no sedimentary features are preserved. The deformation intensity of these metasediments is moderate. The regional foliation is not intense, and is often not recognized in the saprolite and while shear zones occur at Kobada, these tend to be discrete structures 5cm to 50cm wide. These discrete shears often contain limonite rinds parallel to the foliation and the mottle zone supergene alteration extends down these structures probably indicating increased ground water movement through these natural pathways.

Stereogram plots of several hundred measured quartz veins show strikes and dips of all orientations and angles, however three broad populations can be determined.

- A N20°E population which is parallel to the regional foliation which is remarkably consistent at between N10°E and N35°E dipping between 60° and 90° to the E. These veins range from 5mm to 1m in width and are often sheared, strongly brecciated and cemented with iron and manganese oxides. In the area of the fold outcrop these veins appear to be mylonatised in places. Quartz veins of this orientation are mineralised but tend to be of lower grade, 0 to 1ppm Au.

- An E-W population which includes strikes between N45°E and N135°E but with a concentration between N80°E and N110°E. Probably about 65% of these veins dip to the N at between 60° and 90°, the rest dip steeply to the S. Veins of this population are between 1mm and 50cm in width, pinch and swell, are quite discontinuous and can be sigmoidal. Occasionally they form stock work zones up to 3m in width. Almost invariably these veins display a fracture cleavage, and when the regional foliation is visible in the saprolite, this cleavage can be seen to be the same fabric refracted by the quartz vein. The fracture cleavage is commonly stained with red iron oxides, and often the white to translucent grey of the quartz is not visible until the fractured fragments of the vein are broken open. These veins are also often surrounded by limonite rinds which vary from 5mm to 10 cm in width and are often wider than the veins themselves. Sometimes there is no oxide staining and the veins are white-grey in colour. In a few locations the E-W veins are seen to be folded in open folds. The E-W veins clearly cross cut the foliation parallel veins and are not as intensely deformed as the latter. As a population, these veins appear to be well mineralised with grades ranging from 1 to 17ppm Au (within the mineralised envelope), and it is these veins that are the target of the orpailleur workings. It is thought that these veins may have formed as extensional fractures in the Kobada shear zone, which from geometrical evidence of E-W extensional fractures in a N20°E trending shear zone would imply a right lateral or dextral sense of shear but this is by no means certain.

- A sub-horizontal population of veins. Due to the nature of low angle planes and the difficulty of measuring them in a trench environment, this population of 0° to 30° dipping structures has strike directions at all points of the compass, and as yet no preferential orientation has been determined, but mapping and analysis is continuing. These veins vary from 1mm to 10 cm in thickness and often form short and sigmoidal features which occur in stock works and ladder vein systems. They can form long continuous features which clearly cross cut all other structures seen in the project to only be moderately mineralised, 1 ppm to 2 ppm Au within the mineralised envelope, and the stock work zones have proven to be barren of gold.

A structural model has been proposed (Downing 2011) and is outlined below.

After deposition and burial of the Kobada sediments (which are assumed to be distal turbidites typical of the Birimian sedimentary facies), folding of these metasediments occurred at the ductile/brittle deformation crustal level. At this juncture, quartz vein introduction occurred. Deformation continued, and the early quartz veins were folded and the limbs of the folds sheared in the Kobada shear which is now oriented N20°E to form the sheared foliation parallel early veins described above.

Progressive strike slip deformation (thought to be right lateral in nature) with a sub horizontal principal extension direction resulted in the high angle E-W striking quartz filled extension fractures, which accompany the introduction of significant quantities of gold, and also pyrite and arsenopyrite, though there is no direct correlation between gold and the sulphides.

Deformation continued with a similar E-W principal compression direction, but now the principal extension direction rotated into the vertical or sub vertical plane to produce reverse faulting or thrusting. This progressive deformation continued to cause movement on the earlier formed shear zones, the same penetrative foliation in the metasediments and the open folding and fracture cleavage formation in the E-W high angle vein set. Also at this stage, further hydrothermal fluids were introduced into the system and tension gash and extensional fracture veins formed with a sub horizontal orientation. Only weak gold mineralization was introduced at this late stage, and sub horizontal stock works and tension gashes are barren.

7.3 Mineralization

Gold mineralization at Kobada was coeval with the hydrothermal events that introduced the quartz veins that are common throughout the area. The N20°E structures are the only regional structures that have been identified on the property in the field, the E-W and low angle features seem to be confined to the mineralised zone in-between discrete shear zones. It has to be noted that only very scant data is available outside the Kobada mineralised area so it is not possible to be certain that the E -W structures are not regional.

Mineralization at Kobada extends for a minimum strike of 4 km. Gold mineralization is associated with narrow, irregular, high-angle quartz veins and with disseminated sulphides in the wall rock and vein selvages. Mineralization occurs as free gold, sulphides present include arsenopyrite, pyrite and very rare chalcopyrite. Visible gold is not common in the Kobada

deposit. Arsenopyrite (up to 5 mm) is localized near vein selvages and also as fine-grained disseminated patches within the host rock. Pyrite was noted in finely disseminated patches within the host rocks and as euhedral crystals in the black shale, generally as trace to 3% by volume with up to 10% locally in the wall rock over centimetre scale intervals adjacent to quartz veins.

Veins have a milky white colour, are generally discordant with a thickness ranging from milli-metric to sub metric. Mineralized veins were usually described as narrow, high-angle quartz veins, either cross-cutting another vein or the main fabric. This indicates more than one generation of quartz veining is present, with a later phase resulting from remobilization of gold mineralization from an earlier hydrothermal event.

8 DEPOSIT TYPES

Within the Kobada project area, three types of gold occurrence may be expected:

- Primary lode gold mineralization associated with quartz veins and fault zones related to one or several Eburnean deformation phases. Deposits of this type occur in the surrounding area (and shown in Figure 7.2):
    - Siguiri, 80 km to the west;
    - Kiniero, 190 km to the southwest in Guinea;
    - Kalana, 125 km to the southeast;
    - Morilla, 180 km to the east.
- Lateritic deposits resulting from the climatic weathering and alteration of primary deposits with diffusion of the gold into mushroom -shaped red oxidation zones (saprolite);
- In placer deposits, where the gold is associated with large angular fragments of quartz and intensely silicified pyritic rocks in alluvial sand and gravel horizons usually several metres in thickness. These fragments are derived from proximal lode deposits.

On the Project, placers are present and have long been worked by “orpaillage” (artisanal gold surface exploitation).

The Kobada gold deposit appears to be a quartz-carbonate veined “mesothermal” gold deposit. It is located in arenites affected by a geological structure oriented NE along the border of an intermediate intrusive that has basic components.

Mesothermal veins are formed at moderate temperature and pressure, in and along fissures or fractures in rocks. They are known for their large size and continuation to depth, and therefore, are a major source of the world’s gold production. Veins are usually less than two meters wide and often occur in parallel sets. Typical mineralization includes the sulphides chalcopyrite, sphalerite, galena, tetrahedrite, bornite and chalcocite. Gangue includes quartz, carbonates and pyrite. Classic mesothermal vein deposits include: the Motherlode District,

California; Coeur d'Alene District, Idaho; Cassiar District, B.C. Archean lode gold deposits are found in Ontario, Quebec and Manitoba, and the Golden Mile Kalgoorlie in Australia.

Mesothermal vein deposits may also be described as "orogenic" gold deposits. The term "orogenic" recognises the fact that quartz-carbonate vein gold deposits in greenstone and slate belts, including those in BIF, have similar characteristics and have formed by similar processes. The term orogenic is defined to only include the syn-tectonic quartz- carbonate vein-type deposits and their equivalents formed at mid-crustal levels.

Specific deposit types in this clan include the turbidite-hosted and greenstone-hosted vein deposits, as well as the BIF-hosted veins and sulfidic replacement deposits.

Greenstone-hosted orogenic deposits are the most important of the clan and the best represented type among the >10 Moz deposits, including Hollinger- McIntyre, Dome, Sigma-Lamaque, Victory-Defiance, Norseman, and Mt Charlotte. The quartz-carbonate veins in these deposits typically combine laminated veins in moderately to steeply dipping reverse shear zones with arrays of shallow-dipping extensional veins in adjacent competent and lower strain rocks. The reverse character of the shear-zone-hosted veins and shallow-dips of extensional veins attest to their formation during crustal shortening.

In greenstone belts, the significant vein deposits are typically distributed along specific regional compressional to transpressional structures. By virtue of their association with regional structures, these camps are also located at the boundaries between contrasted lithologic or age domains within the belts. Along these structures, the deposits commonly cluster in specific camps, localized at bends or major splay intersections, and where deposits typically occur in associated higher-order structures. The larger camps and deposits are commonly spatially associated with late conglomeratic sequences as exemplified by the Timiskaming polymict conglomerates in the Abitibi greenstone belt and the Tarkwaian quartz pebble conglomerates in the Birimian Shield. The deposits occur in any type of supracrustal rocks within a greenstone belt and, covering stratigraphic positions from lower mafic-ultramafic volcanic to upper clastic sedimentary stratigraphic levels. However, large deposits tend to occur stratigraphically near the unconformity at the base of conglomeratic sequences, especially if developed above underlying mafic -ultramafic volcanic rocks.

At the local scale, favourable settings for these deposits represent a combination of structural and lithologic factors. Favourable structural settings are linked mainly to the rheologic heterogeneities in the host sequences. Shear zones and faults, universally present in these deposits, are developed along lithologic contacts between units of contrasting competencies and along thin incompetent lithologic units. Along these contacts and along in competent rocks, deposits will preferentially develop at bends and structural intersections. Competent rock units enclosed in less competent favour fracturing and veining. Common lithologic associations include Fe-rich rocks such as tholeiitic basalts, differentiated dolerite sills and BIFs, and with competent porphyry stocks of intermediate to felsic composition, whether they intrude mafic -ultramafic volcanic or clastic sedimentary rocks.

9 EXPLORATION

9.1 Exploration History

9.1.1 2005

Exploration was managed by CME and Company ("CME") of Vancouver under contract to AGG. Work completed included:

- Data compilation of all historical work on the Kobada Project.
- Verification of historical data by COMINOR through re-sampling rejects and re-analysis for gold and multi-elements.
- Differential GPS survey to accurately locate the historical drill holes on the property and confirm reported locations, as well as to tie in other features on the property (i.e. roads). The results of this survey verified the drill hole locations.
- 6 holes and 1,033m of diamond drilling by BLY-Mali SARL (Boart-Longyear) to test Kobada Zone 1 at depth (KBO5-3, 5 and 6), and to twin 3 old holes KBRC056, 126 and 127 (KBO5 -1, 2 and 4). 743 samples are analysed at the Eco Tech Lab in Kamloops, B.C., Canada.

This work confirmed the validity of Cominor's databases and demonstrated continuity of the mineralization in Kobada Zone 1 to vertical depths up to 125m.

A 486m drill program in the north area was completed by PDRM (Programme de Développement des Ressources Minérales) a Malian governmental service division of the DNGM. Four holes were drilled (NE05-1 to 4). Hole NE05-4 was abandoned before completion at 83m. A total of 370 samples collected and sent to Abilab in Bamako, Mali, for gold assay analyses.

9.1.2 2006

Exploration was again managed by CME of Vancouver under contract to AGG. Work completed included:

- A 126 line-km magnetometer and induced Polarization/Resistivity survey to establish the relationship between IP anomalies and known mineralized zones and to delineate discrete targets potentially associated with gold mineralization outside of Zone 1.
- A diamond drilling program was completed totalling 99 drillholes for 23,741.7m to provide the database necessary to support the initial mineral resources estimation. The program in-filled the previous widely spaced drilling with 50m spaced sections and a similar hole spacing. HQ core was drilled through the duricrust and saprolite until bedrock, and then the core size was reduced to NQ. A total of 16,131 samples

were taken. A Reflex EZ-Shot instrument was used to collect down-hole survey data. Density measurements were undertaken and core orientation was performed.

9.1.3 2007

Exploration was again managed by CME of Vancouver under contract to AGG. Work completed in 2007 comprised reverse circulation (RC) drilling by WADS Drilling (total 110 holes for 9851m) to test the IP resistivity highs. Drill cuttings were sampled at 1m intervals after being riffle split to 1/8 in the field with about 3kg of material sent to Abilab (Bamako) for fire assay (FA50). The holes were drilled in the Kobada mineralized zones and elsewhere on geochemical anomalies that are coincident with artisanal mining.

9.1.4 2008

AGG mandated Watts, Griffis and McOuat Limited ("WGM") to assist the Company with the implementation and execution of its expanded Kobada exploration and to complete a 43-101 compliant Mineral Resources estimate for the so called "Zone 1". The WGM report made a number of recommendations for work follow-up.

9.1.5 2009

Work completed in 2009 included reverse circulation drilling (2256.7m in 22 holes) and diamond drilling (200m in 2 holes) on Kobada zone.

Objectives for the programme included:

- Showing the effect of bigger sample supports (RC vs DD and PQ vs NQ core) and aliquots for assaying (50g fire assay vs 2kg Leachwell.
- Testing the Kobada structure 300m north of the zone used for the 2008 resources estimate;
- Metallurgical testing at Lakefield, Ontario, Canada of saprolite composite (290 kg) representative of the oxide resource. Results of the testing indicate a Leachwell grade of 1.4g/t Au using desliming followed by gravity separation of the coarse fraction and by bottle rolls cyanidation of the whole mineralized material.

The testing at Lakefield of the 282kg saprolite composite shows that desliming using an hydrocyclone removed 56% of the fines (-11 microns) while losing only 4% of the gold. The 44% hydrocyclone underflow (122kg) subjected to gravity concentration using bench-scale Knelson concentrator yielded a concentrate containing 96% of the gold with an overall gravity recovery of 89.5%. Three bottle roll tests were conducted with recovery better than 95% at an initial P80 grind of 89 microns. Finally, scrubbing and desliming tests were conducted on 10kg RC cutting samples, fines and coarse material were analysed by Leachwell on 2 kg aliquot. The desliming removes an average of 85% of the fines and the remaining coarse fraction contained 85% of the gold.

9.1.6 2010

The drilling program focussed on resource definition with RC drilling (93 holes for 11,286m) at 50m centres and with twin control diamond holes (8 holes for 827 m). Exploration drilling (70 holes for 7,847m) was also conducted on the Northern and Southern extensions of Zone 1, at Foroko North and the Termite Zone.

A trenching program (8 trenches for a total of 3980m) was completed in the Kobada Zone 1 south area. They vary from 0.5 to 4m deep. 2 trenches totalling 96m are hand dug and 2 trenches totalling 247m are dug using a Cat D7 dozer. All of these trenches are mapped and sampled on 1m intervals. Following assaying, gold anomalous sections were revisited and resampled to test what structures is carrying mineralization. A total of 4,323 channel samples were analysed by 50gm fire assay at ALS in Bamako.

Xcalibur Airborne Geophysics of South Africa conducted a high resolution aeromagnetic and radiometric airborne survey over the entire Kobada project footprint (215 km^2). A fixed wing aircraft flew a total of 4,700 line-km on EW lines spaced 50 metres with NS tie lines spaced 500m. Ground clearance was 30m. The airborne data was interpreted by Paterson, Grant & Watson Limited, Toronto, Canada which indicated the data being of high quality and the interpretation resulted in the selection of seven (7) new prospective targets with signatures similar to that of the Kobada Zone 1 mineralization.

A reconnaissance termite mound geochemical survey was undertaken covering 218.5km^2 on a grid of 400m N-S by 80m E-W. Failing the occurrence of any termite activity within the 80m x 40m area around the idealized location, a soil sample was taken at that location. A total of 6,685 samples were collected. Assay results indicated a more anomalous area in the west and an infill sampling programme was completed to a grid dimension of 200m N-S by 40m E-W. Anomalism in the eastern area was also infill sampled to the same grid resulting in 529 additional samples. A total of 418 soil samples were also collected in lieu of termite sampling. If orpaillage, quartz veins or other anomalous features were encountered during this program, a grab sample was made of the anomalous material, 240 grab samples were thus collected. The samples were analysed for gold at the ALS lab in Bamako by 50gm fire assay. Sample quality is tracked with blanks or standards every 20th sample.

Auger drilling was completed to a total of 5,475m in 386 holes and 1,825 samples were sent to ALS Ouagadougou for gold analysis using the Leachwell technique. Auger drilling was used in areas where it is determined that trenching would not be able to penetrate the saprolite horizon. A total of 235 holes were drilled along the trend of the Kobada shear in 6 lines trending 290°. A further 83 holes were drilled on 4 E-W lines on the northern extension of the Kobada shear, at the Chakabougou anomaly. At the Diaban anomaly, a total of 68 holes were drilled. All of these holes are drilled on nominal 25m centres along the lines.

9.1.7 2011

The program concentrated on drilling exploration targets (258 holes for 26,571m) to outline the potential for strike extensions of Zone 1 and away from Zone 1.

A Mobile Metal Ion (MMI) soil orientation survey was carried over 2 lines (1200S in Zone 1 Northern Extension and 2300N in Foroko North). Sampling was conducted in 20cm increments to a depth of 1m. A total of 495 samples were collected and analysed for gold and 42 elements.

A soil geochemical survey (1,377 sites) was conducted the Chakabougou and Gossokorodji (Gosso) targets.

A total of 377 mechanical auger holes for 3,770m were drilled to a maximum of 27m and 1,827 samples are collected generally at 3m intervals and analysed for gold.

Ten mechanical and hand dug trenches (4,049 m) were dug within Zone 1 and at Chakabougou mainly for mapping purposes.

Bumigeme Inc completed a preliminary economic assessment including an oxide resource estimate (Baril et al 2011).

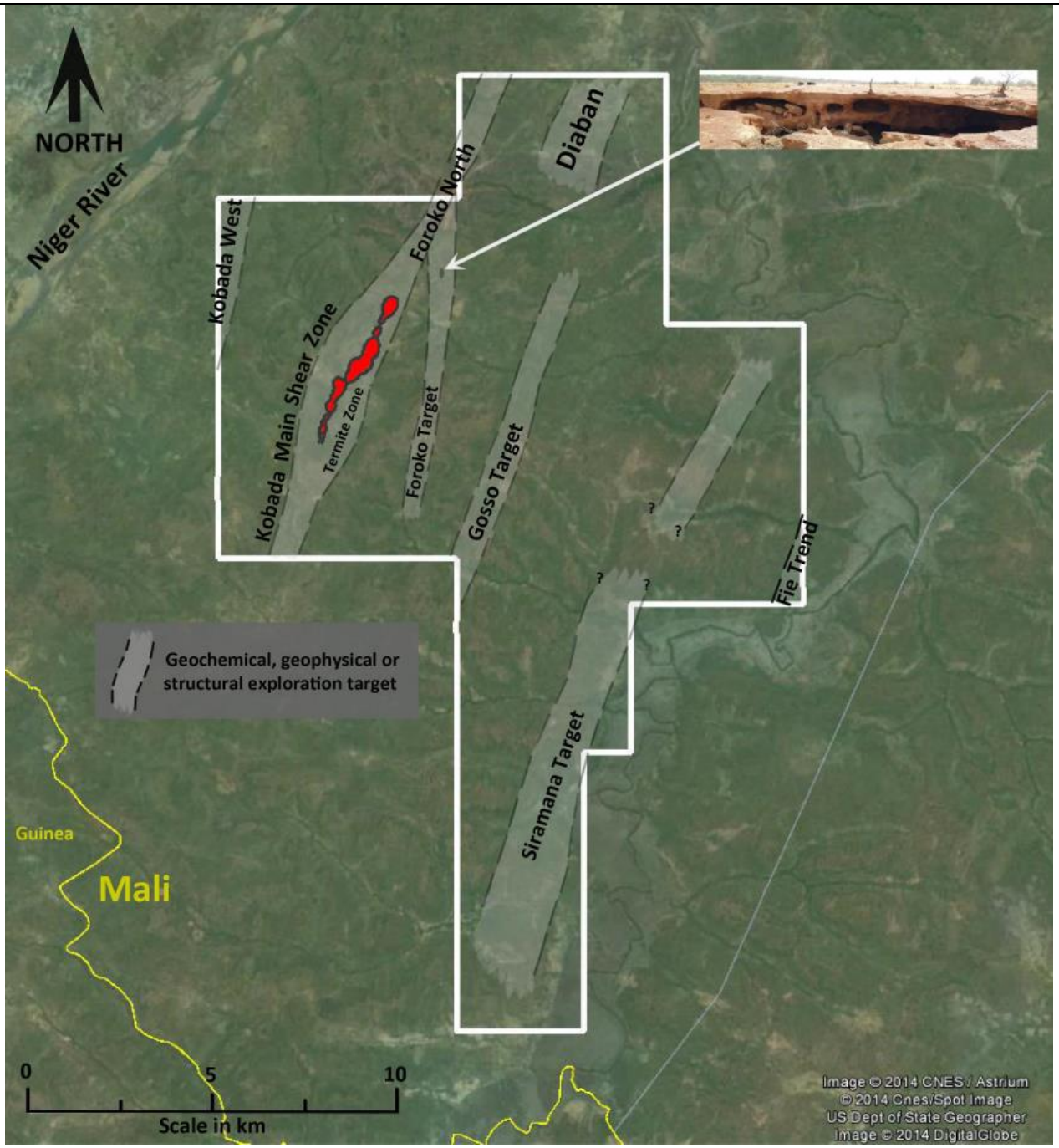
9.1.8 2012

The exploration programs priority during 2012 was on resource definition with infill RC drilling (228 holes for 23,944m) at 25m centre and with twin control diamond holes (10 holes for 1,300 m).

An MMI geochemical survey covered 12 km^2 with 1,777 samples analysed for Au, As, Ba, Cu, K, Mg, Mn and Sn.

Fugro NPA Ltd. of UK processed satellite stereographic imagery to provide a sub meter topographic elevation map (digital elevation model). The survey was ground controlled with high precision differential GPS.

Figure 9.1
Kobada Exploration Targets



Source: AGG, Nov 2014

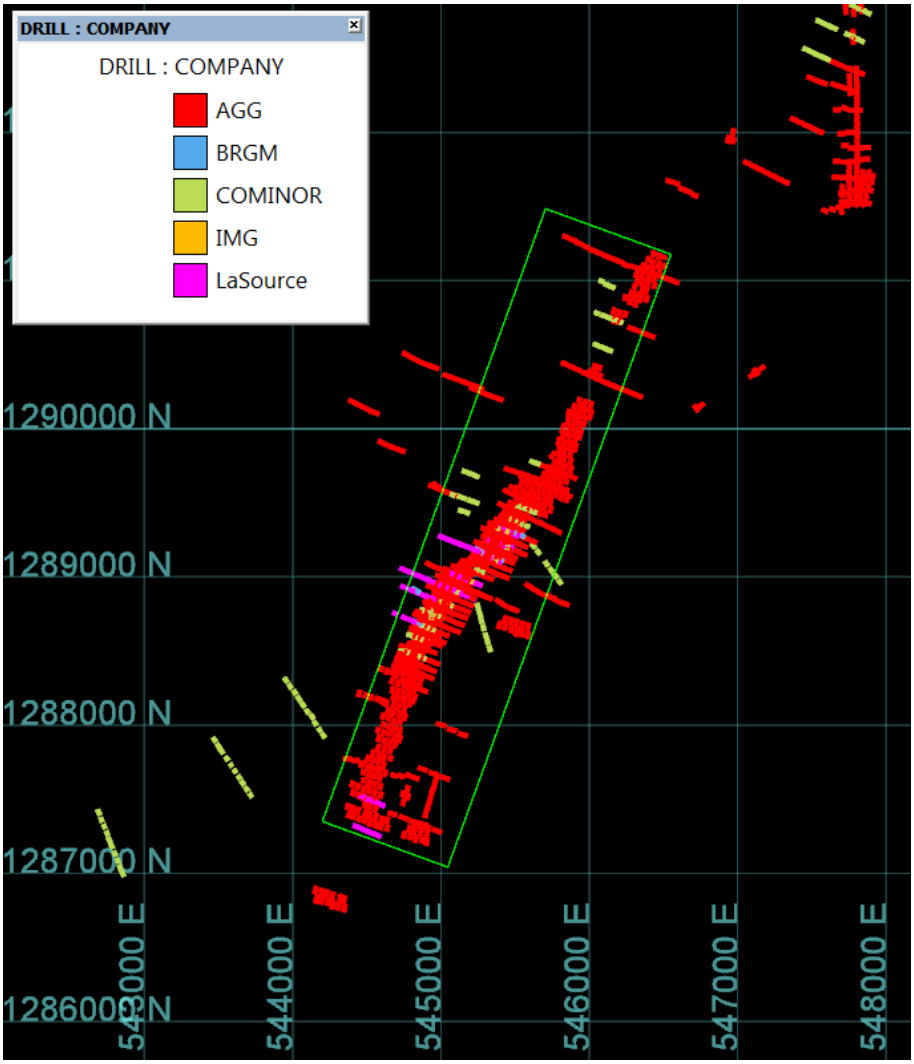
10 DRILLING

10.1 Introduction

Several Diamond, RC and air core drilling programmes have been completed at the Kobada project concentrating on resource definition. Data collection can be subdivided into two distinct periods of exploration; prior to commencement of AGG's involvement in 2005 and then 2005 onwards. The first period relates to data collected as part of BRGM, La Source and COMINOR's exploration management and drilling programmes. The second period relates to data collected under work programmes conducted by AGG.

The final drilling database statistics grouped by company and type are presented in Table 10.1. A summary of all known drilling completed at the Kobada gold project where analytical data exists are included in Figure 10.1 presents drilling on the property coloured by company.

Figure 10.1
Plan View of Drilling coloured by company



Source IRS – November 2015

Table 10.1 Drilling Statistics Grouped by Company and Drilling/Sampling Method					
Company	**Number**	**Total metres**	**Diamond Core (m)**	**Reverse Circulation (m)**	**Air Core (m)**
BRGM (1988)	7	913	913	-	-
La Source (1996)	50	4,825	-	4,825	-
COMINOR (2002 and 2004)	132	10,113	-	8,377	1,736
IAMgold (2009)	2	200	200	-	-
AGG (2005 – 2012)	904	108,886	26,901	81,985	-
AGG (2015)	13	1,398	1,398	-	-
Totals	1,108	126,335	29,412	95,187	1,736

10.2 Drilling Methods

10.2.1 Historical

Drilling methods and sampling techniques have been previously reported in detail in Watts, Griffis and McOuat's report of May, 2008 (Alexander & Lafleur, 2008). As almost 90% of the drilling has been completed by AGG, the detail of drilling completed prior to their involvement has been omitted and a brief summary presented below.

10.2.2 BRGM

A resistivity anomaly outlined during the 1987 geophysical surveys was tested in 1988 with all diamond drillholes intersecting mineralization. Drilling was orientated towards 290° or 300° with a westerly dip of 55°.

10.2.3 La Source

RC drilling completed in 1996 was designed with the objective of extending the mineralization intersected by previous diamond drilling. RC drilling was orientated at ~290° and drilled with a westerly dip of 55° with the exception of three holes which were drilled towards the east.

10.2.4 Cominor

La Source drilling was followed up by Cominor in 2002 by reconnaissance air core and RC drilling programs. Air core drilling totaled 25 drillholes and was not used for the resource estimate. RC drilling was orientated at ~290° and drilled with a westerly dip of 55°. The program continued in 2004 with the majority of drillholes at the same orientation with a small number oriented at ~345°.

10.2.5 IAMGold

IAMGold Corp. (2009) drilled 10 RC holes (1136 m) and 2 DD holes (200) m halfway between previously drill holes in Zone 1 to test the effects of a larger sample support and analytical aliquot and collect a representative bulk sample for metallurgical testing at SGS Lakefield, Canada. In addition, IAMGold drilled 12 RC holes for (1321 m) to test the northern extension of Zone 1 over 400 m of strike length.

10.2.6 AGG

Drillhole Location and Survey

The proposed RC and DDH drill sites are located using a Garmin hand held GPS (accuracy of ±3 meters). A compass is then used to measure the azimuth of the holes. After drilling, the actual locations of all of the boreholes are surveyed with a differential GPS (Novatel Flex Pack 6 DGPS using the Omni STAR XP-G2 service), with ±20 cm accuracy. 2015 drilling was located and surveyed with a Garmin handheld GPS with ±5 metre accuracy.

Drillholes are surveyed with a Flexit tool that measures the azimuth, magnetic field, inclination and temperature at each depth selected. These measurements are taken at three planned depths: 6m, 65m and 130m for DDH. For RC holes, surveys are taken at 6m and at the EOH (max depth).

Reverse Circulation Drilling

A total of 781 RC drillholes for 81,984.7m have been completed by AGG at Kobada. Drilling campaigns were undertaken in 2007 (110 drillholes), 2009 (22 drillholes), 2010 (163 drillholes), 2011 (258 drillholes) and 2012 (228 drillholes).

Drillhole orientations were generally approximately 290° azimuth with approximately 55° westerly dip which in general is orthogonal to the orientation of the shear zone mineralising structure and is optimal for defining margins to the mineralised system. In 2010, as a result of a geological review, the decision was made to re-orient the drilling with a 200° azimuth and a southerly dip. The rationale was to intersect E-W striking veins with a northerly shallow dip that are postulated to carry most of the gold at Kobada. Regardless of the orientation of these veins, this drilling orientation makes the definition of a hangingwall and a footwall to the mineralization (mineralised envelope) difficult at best in the areas where only N-S orientated drilling exists. This has a knock on effect when resource categorisation is considered and this is discussed in Section 14 in more detail.

Two drilling contractors have been used: Longyear for diamond drilling, and WADS (now Layne Drilling Company) for reverse circulation and Air Core.

The 2007 RC programme was subcontracted to WADS. This rig used a (4.5 inch) face sampling hammer. Samples were taken every 1m. Sampling was done manually using a Jones Riffle Splitter.

Diamond Drilling

A total of 136 diamond drillholes for 28,299.05m have been completed by AGG at Kobada. Drillholes have generally been collared with HQ size and this is continued to the base of the oxide material. Drill hole size is then reduced to NQ and drilling continued to planned depth. A Reflex EZ-Shot instrument was used to collect down-hole survey data with an EZY mark tool used to orientate core from mid-2006 onwards. Drilling campaigns were undertaken in 2005 (6 drillholes), 2006 (13 drillholes), 2007 (86 drillholes), 2010 (8 drillholes), 2012 (9 drillholes), and 2015 (13 drillholes). A variety of drilling contractors have been used and these include Boart-Longyear. Drillhole orientations were generally approximately 290° azimuth with approximately 55° westerly dip which in general is orthogonal to the orientation of the shear zone mineralising

structure and is optimal for defining margins to the mineralised system. A small number of diamond drillholes have been drilled at different orientations.

The 2005 diamond drilling programme was subcontracted to Longyear. This rig used a HQ core in the oxide zone and NQ core in fresh core. Samples were taken every 1m (choose). Sampling was done manually after core was sawed in half.

10.3 Drilling Results

A significant quantity of drilling has been completed and it is not practical to include a listing of all significant drilling intersects. Additionally, the property can now be classified as advanced and a drill plan and representative sectional view is presented in Figures 13.3.2_1 and 13.3.2_2.

To summarise, the drilling at Kobada has outlined a large mineralised system within a NE striking shear zone that dips moderately/steeply to the east. Widths of mineralization vary however true widths of mineralization commonly vary between 25m to 60m and occasionally even wider to 100m. Strike length of the mineralization is approximately 4km and mineralization has been intersected in drilling to approximately 300m vertically below surface.

11 SAMPLING PREPARATION, ANALYSES AND SECURITY

11.1 Introduction

Sample preparation, analysis and security information is not generally available prior to AGG's involvement in the project. Some detail is available in the WGM report of 2008 (Alexander& Lafleur, 2008) but it is minimal in nature. Other information made available by AGG staff is summarised in Table 11.1 below. Quality control information is discussed in the relevant section.

Table 11.1

Analysis summary

Company	Year	Comment
BRGM	1988	Analytical procedure uncertain: during the 1980s field crews did colorimetric test and if above certain colour sample sent to Dakar or Orleans for fire assay on 10 or 15-g aliquot
La Source	1996	Analytical procedure unknown: assumed to be fire assay (30g) with AAS finish
COMINOR	2002, 2004	Analytical procedure unknown: assumed to be fire assay (30g) with AAS finish
IAMgold	2009	Leachwell on 2kg at ALS Ouagadougou
AGG	2005	FA AAS at either Abilab, Bamako or Eco Tech Lab, Kamloops, BC, Canada
AGG	2006, 2007 (DD)	FA AAS at either Abilab, Bamako or ALS Bamako
AGG	2007, 2008 (RC)	FA AAS at Abilab, Bamako
AGG	2010, 2011 (RC, DD)	Leachwell on 2kg at ALS Ouagadougou
AGG	2012 (RC, DD)	Leachwell on 2kg at ALS Ouagadougou and Fire assay 50g at ALS Bamako
AGG	2015 (DD)	Leachwell on 2kg at SGS Ouagadougou – Sample Preparation completed by SGS Bamako Laboratory. SG, Metallurgical and Geotechnical studies also completed.

Collection and processing of all exploration samples (RC and DDH) is carried out by employees of AGG. The quality control procedures are carried out by geologists on site and supervised by the project senior geologist.

A record for each sample is generated and includes the following;

- Type of sample (rock, soil, dump, etc.)
- Location, which includes the hole number, footage.
- Brief description.
- Date sample collected.
- Person responsible for collecting sample.
- Elements analysed including; method used (fire assay, Leachwell, etc.), laboratory, etc. NB: Although this data has clearly been collected assay method is not always clear in the AGG database.
- Geochemical or assay results for each element.

A portion of assay certificates from the appropriate laboratories are kept on file for all analysis completed. Housekeeping on data storage has not been ideal, and has been discussed with AGG personnel in detail, subsequent to these discussions and analysis of all available data the database is considered to be satisfactory. However it is highly recommended that AGG obtains the original data and certificates from the Laboratory and re merges all data into a complete database that clearly annotates, dates sampled and analysed, AGG submission numbers, lab batch numbers and assay method/s used. Where multiple analysis types are used for the same sample the database should also clearly show analysis method used for reporting and resource estimation. Raw and split sample weights should also be recorded in the database as well as DDH recoveries.

A minimum of 3kg of sample is collected for each 1m drill interval if circumstances allow. Exceptions may be RC sample where subsurface voids or high water content preclude this weight of sample or core intervals where poor recovery precludes the collection of sample.

Since 2009, each sample was analysed by Leachwell and/or fire assay for gold content from a subsample of 2000g (Leachwell), or 50 grams (fire assay) in the ALS Chemex Lab, 2015 samples were submitted to SGS Labs(in each case Ouagadougou labs completed Leachwell, and Bamako for fire assay). Splitting to these subsample sizes is done at the lab after sample pulverization.

In addition to the protocols of quality assurance and quality control (QA/QC) of ALS Chemex and SGS Lab, an internal QA/QC is implemented during the sampling program, including the use duplicates, blanks and standards (see below for details).

Transportation of sample from the Kobada camp to Bamako ALS Laboratory was carried out by ALS. In the case of the DDH twinning campaign, the shipping of all samples is planned at the end of campaign. All sample submissions for the recent 2015 DDH program were delivered by AGG personnel to the SGS Bamako laboratory where samples were recorded using the SGS LIMS system.

11.2 RC Sampling procedures

Before the commencement of each drill hole, the cyclone is cleaned of debris from the previous hole. After each metre drilled, the driller lifts the feed to flush out ant any remaining rock chips and so prevent contamination of the next sample and smearing of any contained gold. As a drill rod is added, compressed air flushes any water or any remaining material form the string avoiding sample contamination and keeping the sample dry.

Samples were collected from the cyclone directly using a polythene bag large enough to collect the one metre sample. Sample bags were tied and transported to the on-site sample preparation shed. Once there, samples are weighed to calculate recovery. Each sample is poured into a large basin and poured from there into a riffle splitter and recombined. This is done twice. They are then poured back into the splitter where 1/8 (approx. 3kg) is split off and sent to the lab. The basin and splitter are cleaned with compressed air between samples. The remaining portion of the sample is poured back into the polythene bag for storage on site.

Alternatively, a second method has been used utilising a roller mixer. The roller mixing device is comprised of two metal H-beams (height 2 meters) clamped together with freely moving cylindrical metal tubes (ball bearings at each end) mounted on top of the H-beam. Two similar beams are mounted opposite each other about 0.8 to 1m apart. A linoleum mat is spread over these two cylindrical metal tubes to make a U-shape between the two H-beams. The mat also goes half way the height of the H-beams on the outsides.

The samples are poured on the U-shaped area of the linoleum mat. There are two sets of people positioned at both ends to pull the mat up and down in an alternating manner. Another set of people are also stationed on the sides of the mat to stop the samples from falling by raising the sides when the need arises.

After fifteen strokes the sample is thoroughly mixed and everything is poured back into a big basin which is then split through a Jones riffle splitter to get the required 3 kg of sample for the laboratory for analysis.

All samples are placed in bags with a sample ticket inside and a sample ticket stapled into the rolled and staple sealed top of the bag. The identification number is clearly written on the bag in addition to the internal and external tickets.

A duplicate sample is taken every 20th sample and a standard reference material is included every 25th and 75th sample in each drillhole. Blank material is inserted every 50th sample. An external umpire laboratory sample is taken every 30th sample.

11.3 Diamond Core Sampling procedures

The metal core boxes are well identified with the hole and the box number. The core is placed in the core boxes with the depth intervals marked with plywood blocks under the supervision of an AGG geologist or technician. The core boxes are transported, under supervision of a geologist, to the sample prep facility situated in the AGG camp. During logging the sample intervals are selected by the geologist over about 1m intervals, honouring geological boundaries with regard mainly to quartz veining. The core is then halved using a diamond saw, a manual hydraulic splitter or a strong knife depending on the weathering state of the material. Half core is sampled and sent to lab, the other remains in the core box, which is stored in secure core shack in the camp.

The same QAQC is used for the core drilling if there are enough samples. However core samples are notably smaller than RC and often there is not enough sample for field and laboratory duplicates. When this is the case, these duplicates are not taken.

11.4 Sample Analysis

11.4.1 AGG 2005 Diamond drilling

For holes drilled in so called Zone 1 mineralization, samples were assayed by 30g fire assay with AAS finish at Eco Tech Laboratory in Kamloops, BC, Canada. Other drillholes (regional) were assayed by 50g fire assay with AAS finish at Abilab, Bamako (now owned by Groupe de Laboratoire ALS Mali SARL).

11.4.2 AGG 2006 – 2007 Diamond drilling and 2007 – 2008 RC drilling

During these campaigns, samples were assayed by 50g fire assay with AAS finish at Abilab, Bamako, later Groupe de Laboratoire ALS Mali SARL.

11.4.3 AGG 2009 - 2012 DD and RC drilling

Assaying by Leachwell technique was initiated during the 2009 season. Assaying by Leachwell was completed on a 2kg aliquot of sample at ALS Chemex, Ouagadougou, Burkina Faso. Additionally, all samples collected during 2012 were assayed by Leachwell and 50g fire assay at ALS Bamako (Groupe de Laboratoire ALS Mali SARL).

11.4.4 AGG 2015 DD drilling

Diamond drilling program completed dominantly for geotechnical, metallurgical and density studies, remainder of samples were analysed for gold using the Leachwell technique. One (1) metre half core samples were collected from PQ and HQ diameter drill core at all times respecting lithological contacts. Sample preparation and analysis was completed by SGS Bamako and SGS Ouagadougou respectively. Metallurgy studies were completed by Gekko Systems, Australia.

11.5 Logging

All logging is undertaken using a standardized system designed to facilitate ease of computer data capture and manipulation without any loss of geological detail. The system allows for consistency between the large numbers of geological personnel responsible for the collection of these data. The entire database, including RC holes and diamond drill holes are in this format. Logging codes provide for description of sample weight, lithology, alteration, mineralization, structure, and weathering styles.

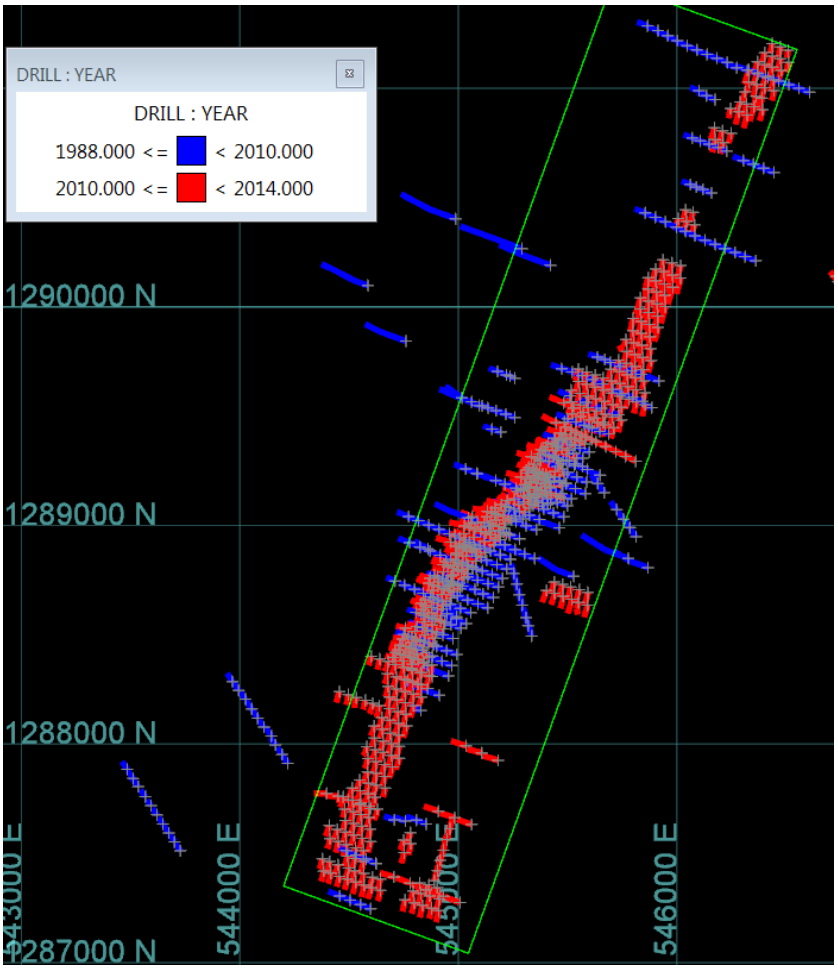
The core is logged in detail including lithology, structure, alteration, mineralization and other notable characteristics. Percentages of core recovery and rock quality designation (RQD) are measured and included in the log. The core recovery is calculated based on each drill run (interval). The RQD calculations are based on the total length of core sections recovered that are greater than 2.0 times the core diameter for each drill run or interval. The core is photographed in batches of three boxes per photo, with the beginning and end depths and the hole-ID shown.

11.6 Assay Quality Assurance and Quality Control (QA/QC)

The supplied database contains limited QAQC data prior to 2010. This includes data collected by AGG as well as the previous operators. The distribution of drillholes with QAQC data vs those without is presented in Figure 11.7_1. Drillholes with QAQC are presented in red, of note is the even distribution of these drillholes throughout the strike length of the deposit. This provides some mitigation for the lack of QAQC data prior to 2010.

The WGM report of 2008 only includes a description of the laboratory QAQC protocol and no detail is included of the AGG implemented field QAQC protocol. Also of note that Marchand (2013) only discusses results of the QAQC field protocol from 2011 onwards.

Figure 11.1
Pre 2010 vs post 2010 drilling



Source IRS – November 2015

11.6.1 AGG 2005 Diamond drilling

During the 2005 drilling campaign one standard reference sample and one blank sample were inserted into the sample stream every 20th sample. The standard reference material used was CDN-GS-3B from CDN Resource Labs of Delta, BC, Canada. This standards and blank data would not appear to be present in the database. The blank used was described as from a barren granitic rock found in Selingue area, Mali (no coordinates given). A limited number of

field duplicates (9) were taken and are represented in the supplied database. This is insufficient to provide a meaningful conclusion.

11.6.2 AGG 2006 – 2007 Diamond drilling

During the 2006 – 2007 diamond drilling campaign one of three standard reference samples and one blank sample were inserted into the sample stream. The frequency of insertion varied and is described as either every 20th, 25th, 30th or 50th sample. The standard reference materials used were CDN-GS-3B, CDN-GS-13 or CDN-GS-2B from CDN Resource Labs of Delta, BC, Canada. The results of the standards are not present in the database, 433 analyses of blanks are attributed to this program. The blank used was described as from a barren granitic rock found in Selingue area, Mali (no coordinates given). Field duplicates were also taken, however only 27 are recorded in the database. This is insufficient to provide a meaningful conclusion.

11.6.3 AGG 2007 – 2008 RC drilling

During this phase of drilling field duplicates were taken at a rate of 1 per 20 samples. A total of 440 field duplicates have been attributed to this program in the database. 115 standards and 106 blanks analysis are present in the database attributed to RC drilling in 2007. Standards used include CDN-GS-2B, OXF 53 and OFG 60. Two standards have also been used (S125 and ST259, total 55) about which no information can be determined with accuracy.

11.6.4 AGG 2009 onwards - DD and RC drilling

In 2009 Leachwell was utilized as the main assaying technique. Assaying was carried out at ALS Ouagadougou. One of two standards or a blank was inserted every 20th sample. One in 20 leached tails was assayed by screen fire assay on a 1kg aliquot. One field duplicate was taken every 20 samples. The duplicate was analyzed by Leachwell, screen fire assay and fire assay.

From 2010 every 30th sample was taken as a duplicate and sent to the main lab and the umpire lab. Additionally a 3kg field duplicate was taken every 100th sample for 50g fire assay and screen fire assay.

The blank standards used for the Leachwell program consists of a certified blank from ALS Lab of Bamako. The material consists of crushed sandstone (Taoudeni Basin deposits) from a quarry near Bamako. The standards inserted into the Leachwell assay stream were constructed on-site in the following manner: two standard reference materials were sourced from Geostats Pty Ltd laboratory of Perth Australia. The two standards used were G306-4 and G901-8. On site, AGG adds one 100g bag of either one to 3kgs of blank material from ALS Bamako. This material is homogenized and in the case of FA analysis 50g from the resultant 3,100g is sent to the lab and 2kg is sent for the Leachwell analysis.

For the Leachwell analysis, tailings verification (residual gold) was undertaken on every 25th sample by screen fire assay.

In 2012, the first 10,000 samples were analyzed by both Leachwell and fire assay. Normal QAQC procedures as described above apply.

Every 30 samples a field duplicate is obtained and sent for Leachwell analysis at SGS Ouagadougou. It should be noted that SGS Ouagadougou undertake Leachwell analysis on only 1kg compared to ALS Ouagadougou 2kg.

11.6.5 AGG QAQC – 2015 Diamond Drilling

During the 2015 diamond drilling campaign AGG staff inserted blanks, standards, and field duplicates into the sample stream according to table 8 below. Field duplicates were taken as ¼ core, and all core drilled during this campaign was either PQ3 or HQ3 diameter. All analysis completed was Leachwell at SGS Ouagadougou laboratories.

Table 11.2
AGG 2015 Diamond Drilling - QC insertions

Sample No.	Type of QC Sample	Selected QC materials to insert
10	Std 1	Std Number OxG103 (1.109ppm)
19/20	Field Dup	parent/daughter
30	Std 2	Std Number OxD87 (0.417ppm)
39/40	Field Dup	parent/daughter
50	Blank	Waste material
59/60	Field Dup	parent/daughter
70	Std 3	Std Number G901-7 (1.52ppm)
79/80	Field Dup	parent/daughter
90	Std 1	Std Number OxG103 (1.109ppm)
100	Blank	Waste material

Blanks were inserted at a rate of 2 in 100 samples. Blank material was source from the SGS laboratory in Bamako. The material was a barren Quartz /Feldspar pegmatite. Blanks performed well and all were at or below detection limits for the 2015 program.

Three (3) Standards were used by AGG during the 2015 diamond drilling campaign (table 8), all standards performed well and reported within 1 or 2 standard deviations from their expected value, and are shown in the charts below. The standards were not diluted and submitted a 100g of sample. Shewhart control charts for each of the AGG standards are shown in figures 30 to 32 below.

11.6.6 Other QAQC measures

In 2008 WGM completed an assay check program on 35 randomly selected check samples from three drillholes, KB05, KB06 and KB07 (Alexander & Lafleur, 2008). WGM sampled quarter core compared to AGG's half core and assayed for gold via conventional fire assay and screen fire assay methods. In their discussion, WGM found that a bias existed between the two datasets with the WGM assays lower for both FA and SFA. At the time they recommended a program to re-assay 10% of the assays contributing to the resource estimate at that time however noted also the potential effects on the variance of the smaller sized sample. WGM therefore concluded that the resource database was not fatally flawed.

11.7 QA/QC Results

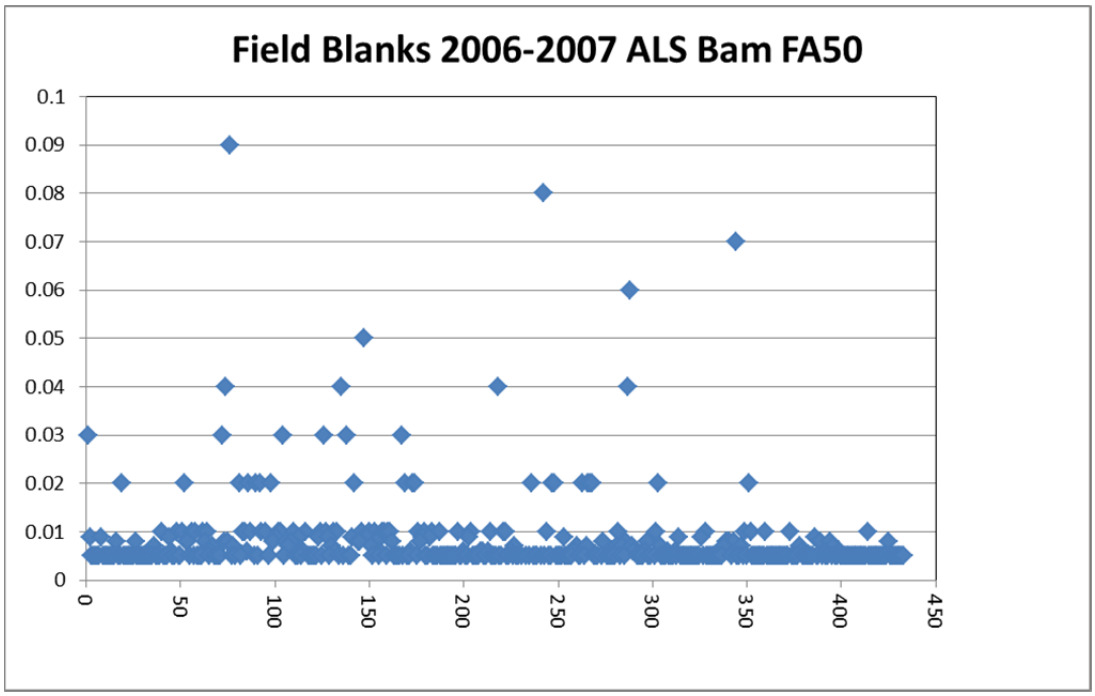
Marchand (2013) presented a detailed review of the available QAQC data at the time. This work is not repeated here, however some of the items have been reviewed and checked as outlined below.

11.7.1 Blanks Analysis

An analysis of the performance of inserted blanks is presented in Table 11.8_1 and the relevant plots are presented below. Data is available from 2006 through to 2012. Values over 0.1g/t Au have been generally excluded from the review. The expected value for the blanks prior to 2010 is unknown. The EV for blanks from 2010 is 0.0017g/t Au. In most cases it can be seen that the blank is performing as expected and returns very low values around the detection limit of the methods employed. The accuracy and precision relating to analysis of blanks are normal for the analytical methods.

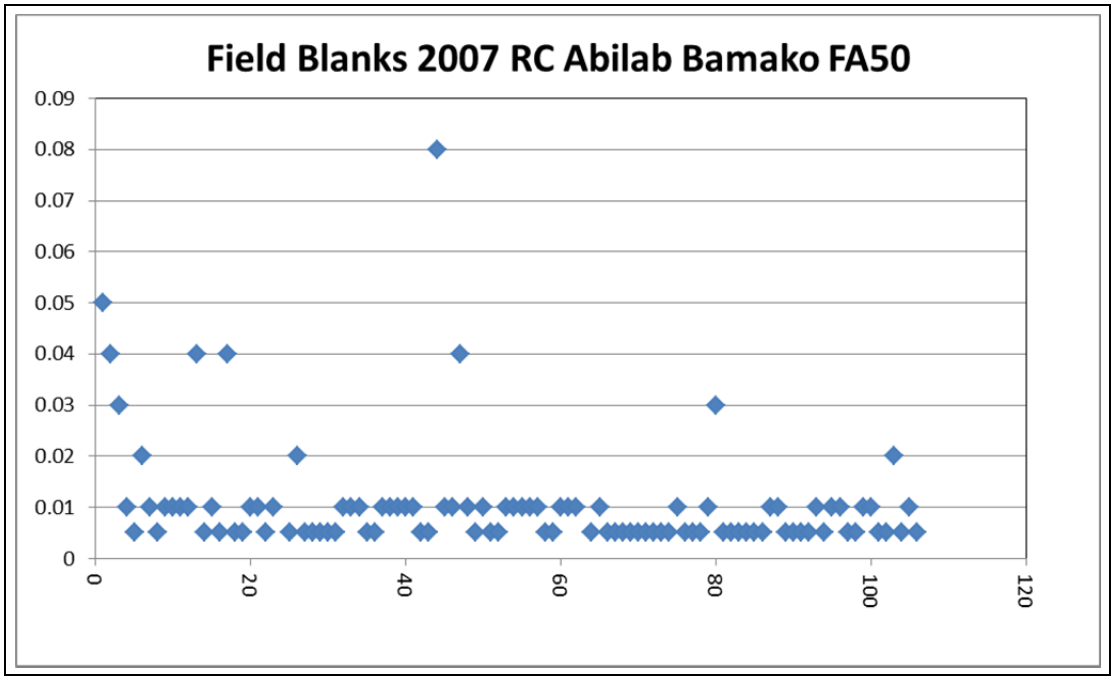
2006-2007 DD Blanks at Abilab, Bamako (later ALS)

Figure 11.2
Field Blanks 2006-2007 ALS Bam FA50



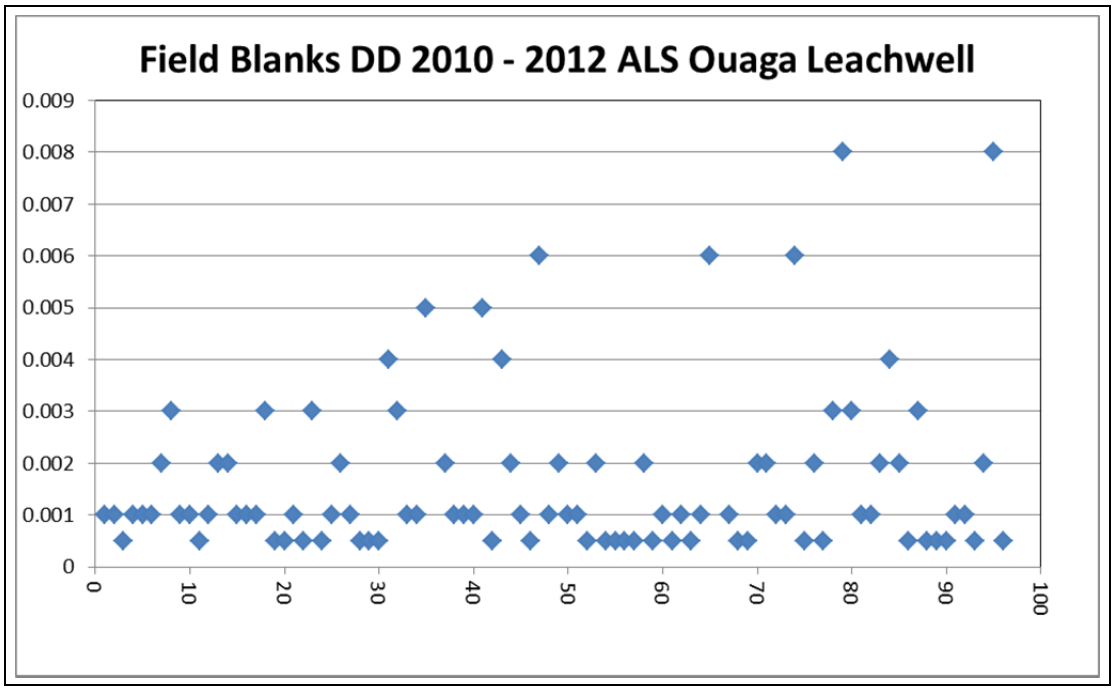
2007 – 2008 RC Blanks at ALS Bamako

Figure 11.3
Field Blanks 2007 RC Abilab Bamako FA50



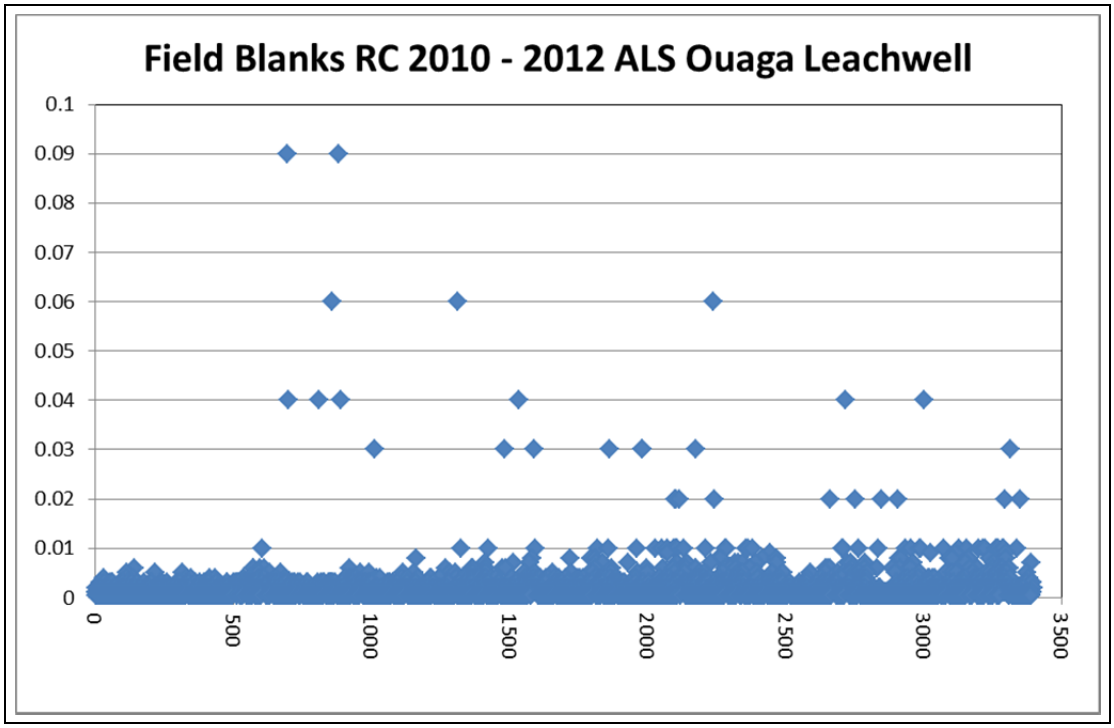
2010 – 2012 DD Blanks at ALS Ouagadougou

Figure 11.4
Field Blanks DD 2010 - 2012 ALS Ouaga Leachwell



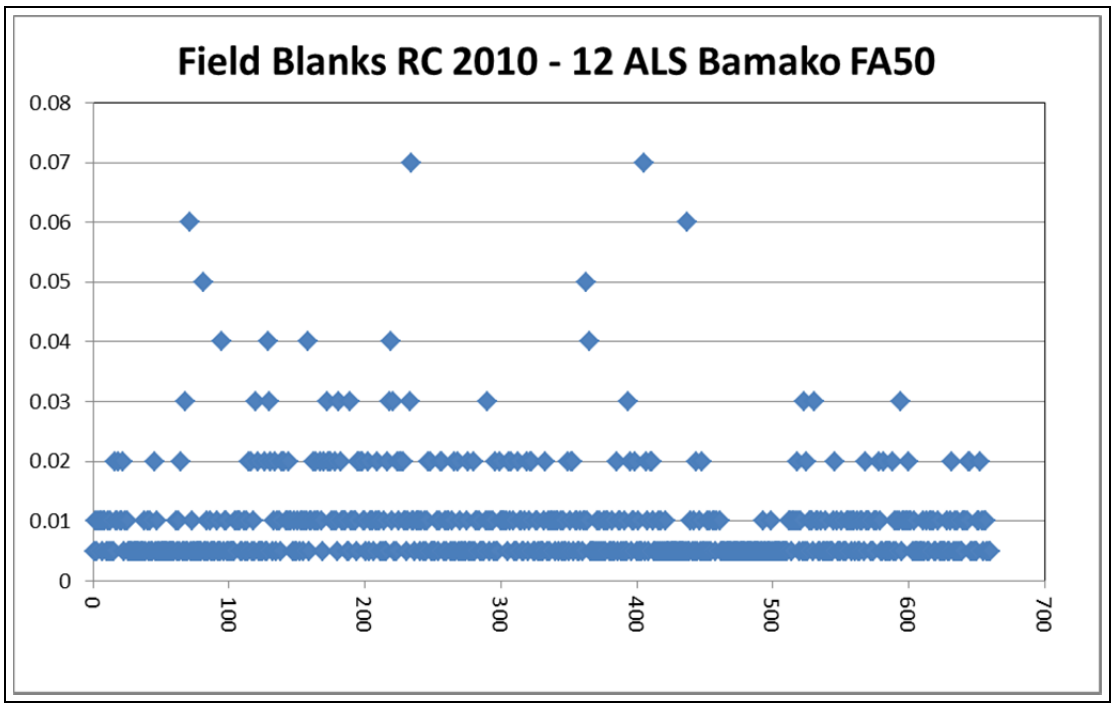
2010 – 2012 RC Blanks at ALS Ouagadougou Leachwell

Figure 11.5
Field Blanks RC 2010 - 2012 ALS Ouaga Leachwell



2010 – 2012 RC Blanks at ALS Ouagadougou FA50

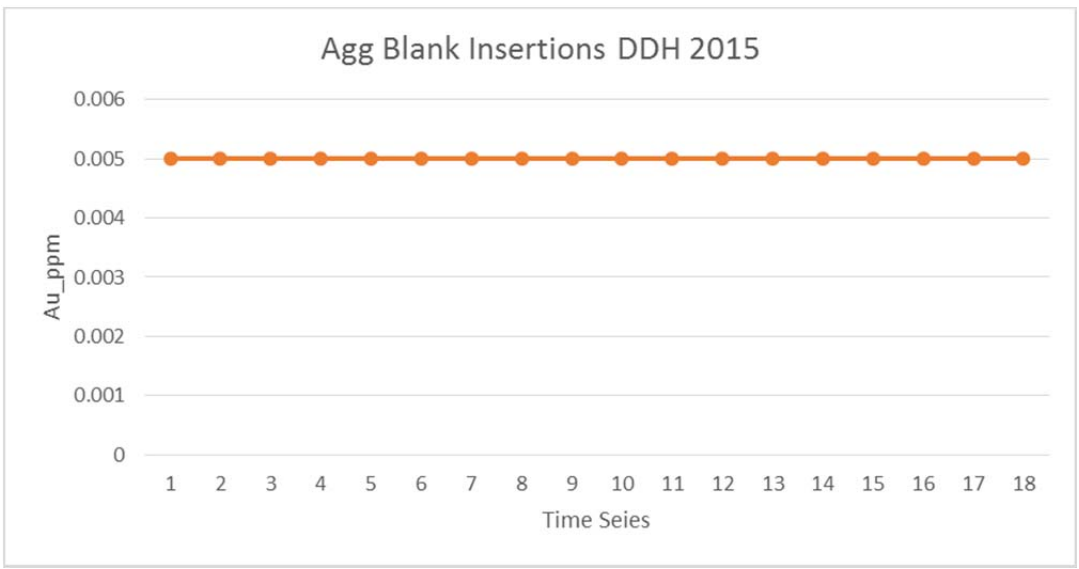
Figure 11.6
Field Blanks RC 2010 - 12 ALS Bamako FA50



2015 DDH Blanks at SGS Ouagadougou LWL

Figure 11.7

SGS Ouagadougou Field Blanks 2015 DDH – Leachwell



11.7.2 Standards Analysis pre 2010

Very low numbers of standards analysis are available in the database prior to 2010. CDN GS 2B, OXG 60 and OXF 53 were inserted into the sample stream during the RC program in 2007. Analysis was undertaken at Abilabs Bamako (later to be ALS Bamako). Assay method was fire assay. All 3 are plotted below and tabulated in Table 11.8_1. When obviously erroneously attributed samples are excluded, all 3 perform well with >90% falling within ±10% of the EV and no indication of bias.

Figure 11.8
Abilab RC 2007 FA50 OXG60

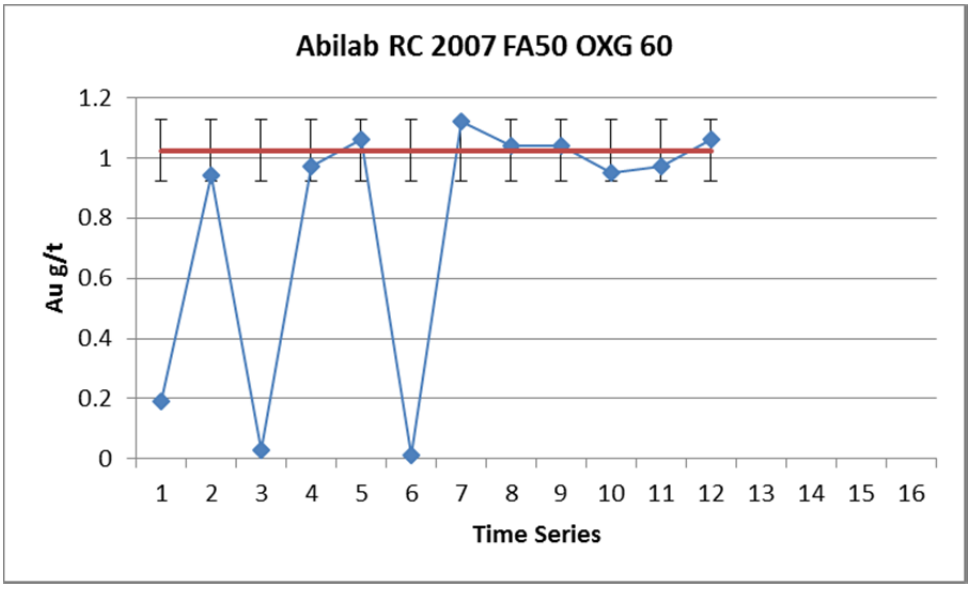


Figure 11.9
Abilab RC 2007 FA50 OXF53

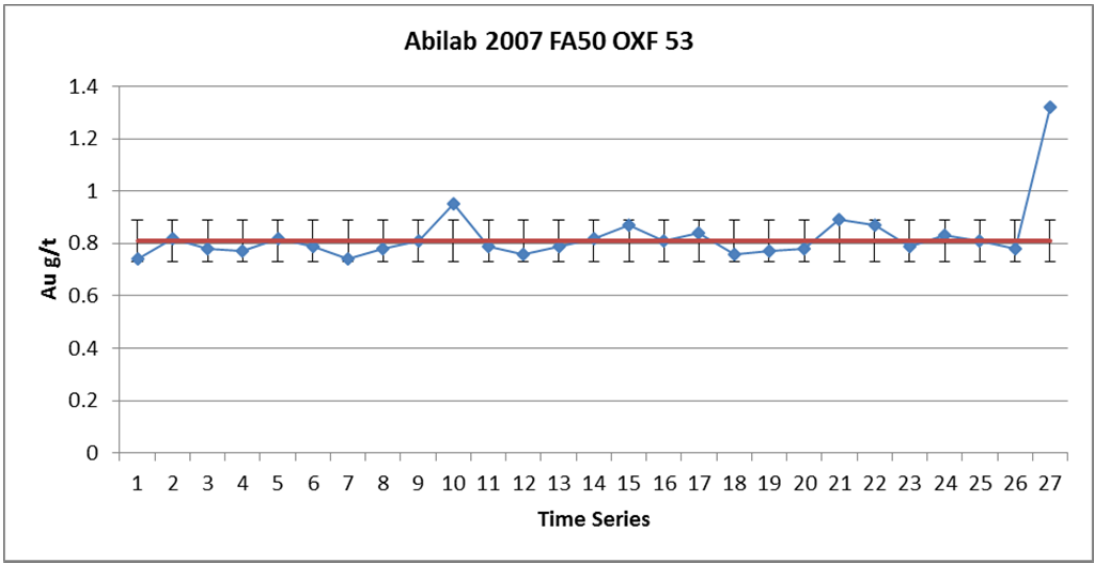
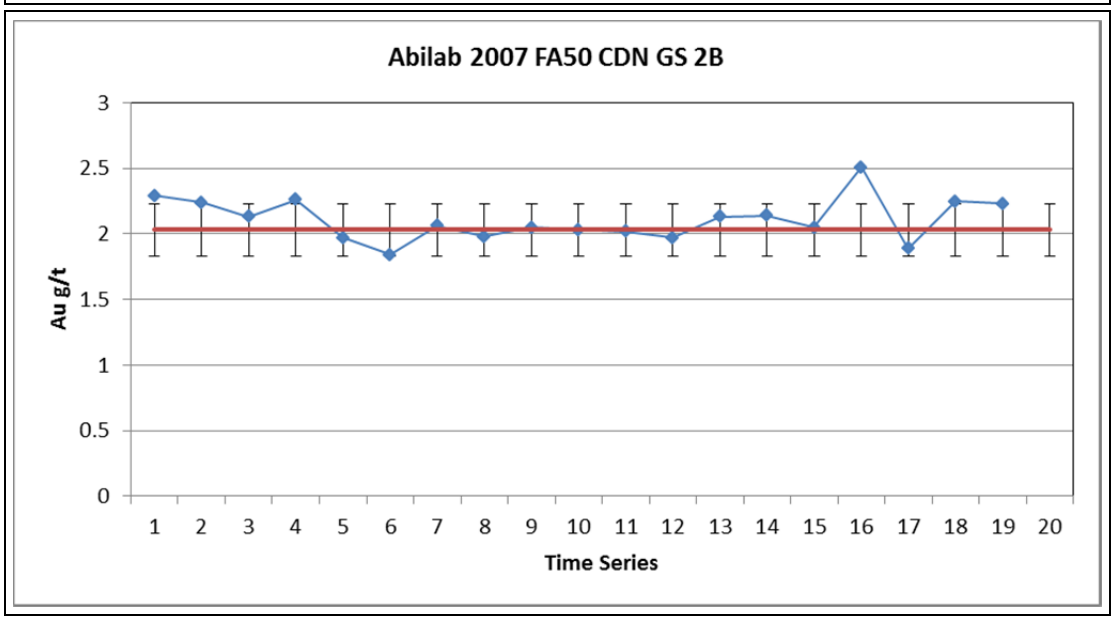


Figure 11.10
Abilab 2007 FA50 CDN GS 2B

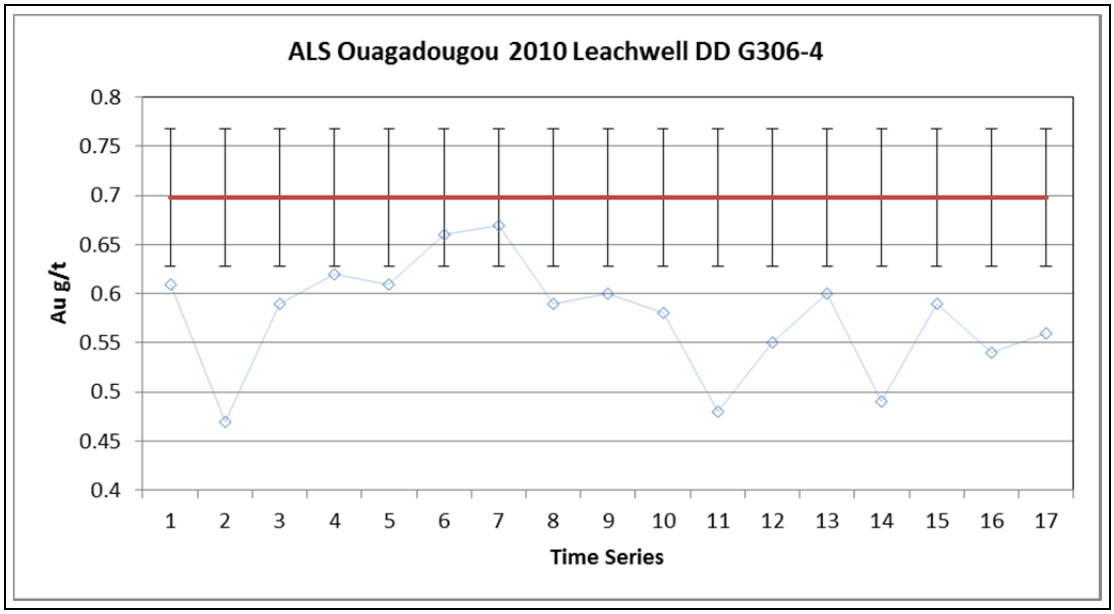


11.7.3 Standards Analysis G306-4 (2010 to 2012)

Standard G306-4 is blended on-site as described in Section 12.4.4. It has an expected value of 0.698 g/t Au. A summary is presented in table 11.8_1. Analysis for this standard was subdivided into sample type and analysis method. Plots are presented below. In summary, all types of samples and different analysis methods show the same features. The mean assayed value demonstrates a low bias in all cases and there is a large spread about the mean assayed value. Bias varies from -4.7% to -13.5%. Very low numbers of values fall within ±10% of the EV ranging from 37% to 48%. The causes for such result may be an issue with lab calibration, however the fact that such consistently poor results span a number of years, different sampling types and different analysis methods indicate that the fault is likely to be inherent in the method of preparation of the standard. The balance used for weighing the blank material is likely to have not been correctly calibrated and the homogenization of the blended material is likely to be poor, also no weighted average calculation was used when the standard was blended with blank material. The possibility should not be excluded that the low mean assay value is related to the fact that Leachwell is a partial analysis and fire assay is considered total.

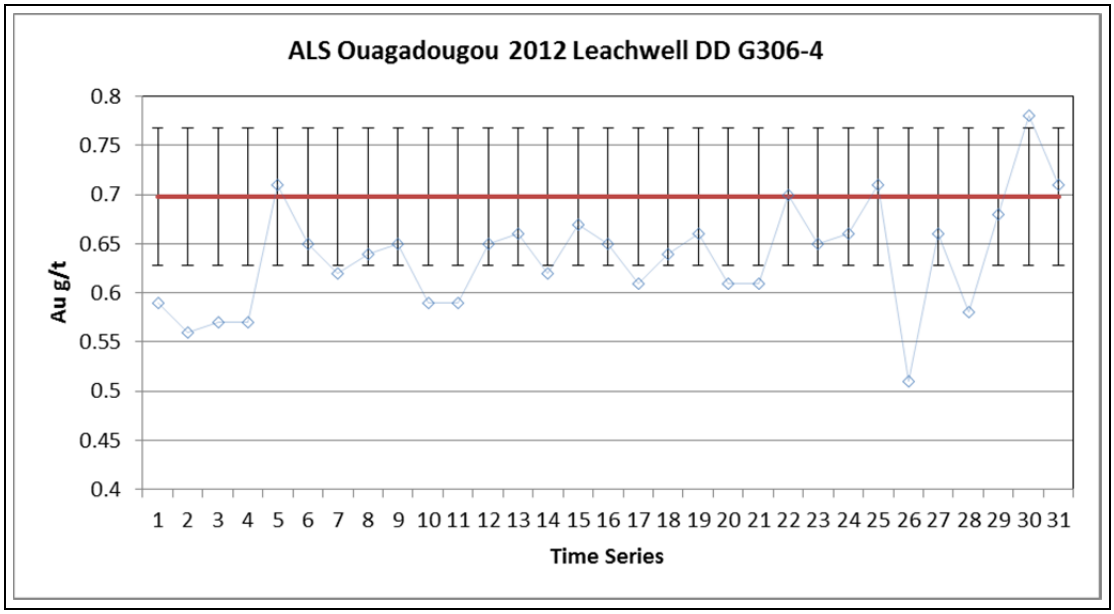
2010 DD at ALS Ouagadougou Leachwell

Figure 11.11
ALS Ouagadougou 2010 Leachwell DD G306-4



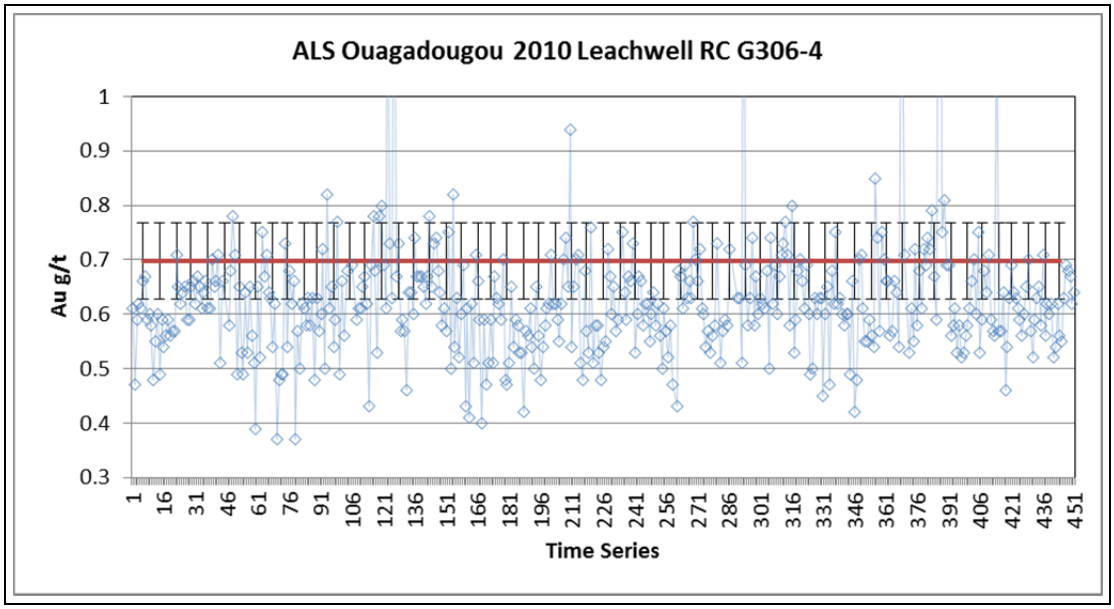
2012 DD Standards at ALS Ouagadougou Leachwell

Figure 11.12
ALS Ouagadougou 2012 Leachwell DD G306-4



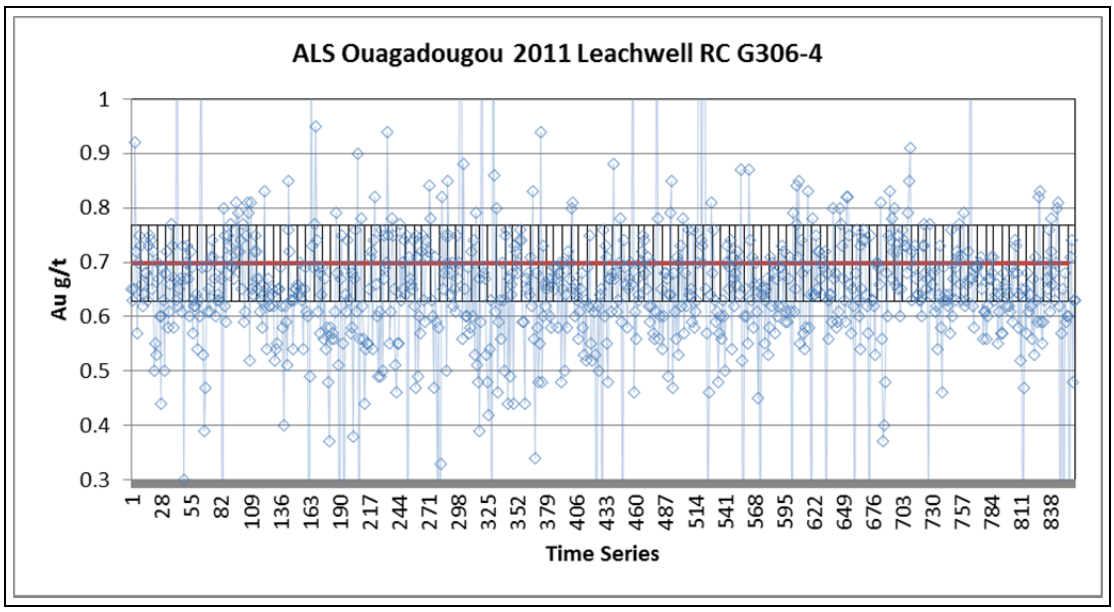
2010 RC Standards at ALS Ouagadougou Leachwell

Figure 11.13
ALS Ouagadougou 2010 Leachwell RC G306-4



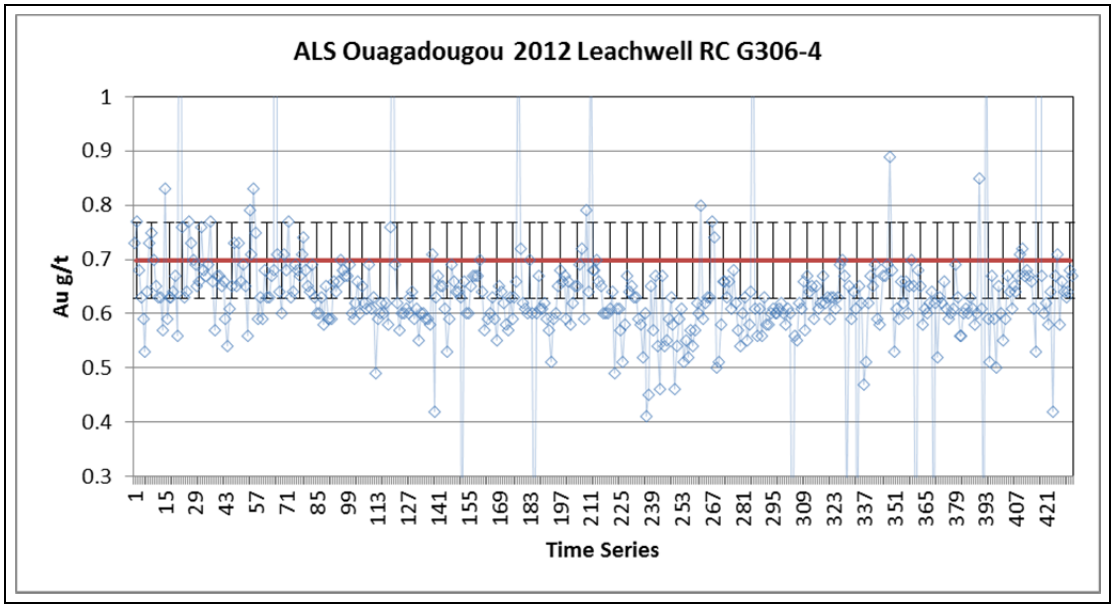
2011 RC Standards at ALS Ouagadougou Leachwell

Figure 11.14
ALS Ouagadougou 2011 Leachwell RC G306-4



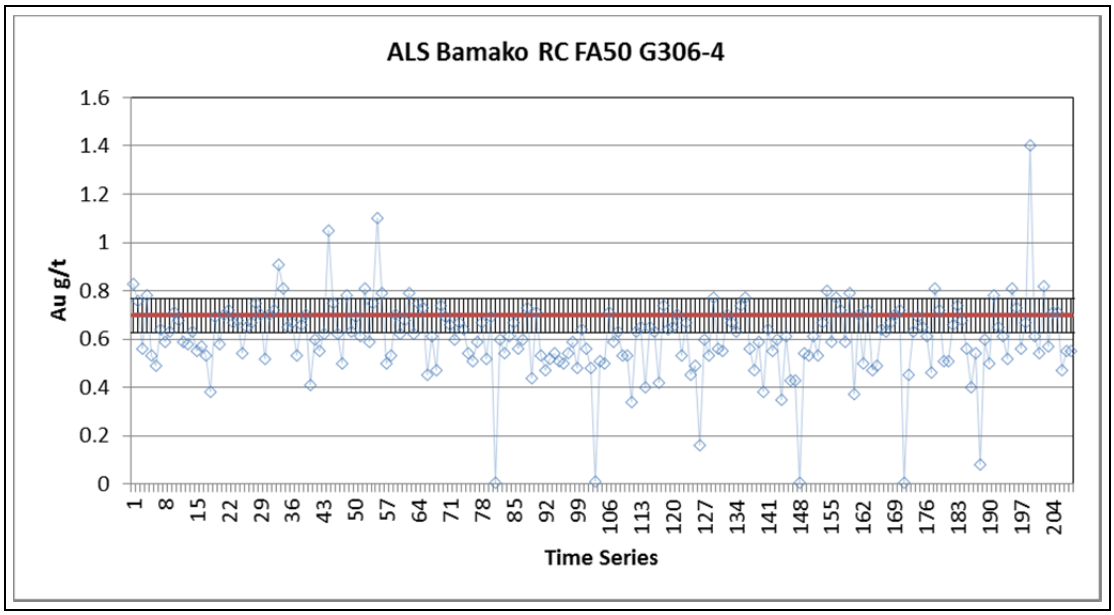
2012 RC Standards at ALS Ouagadougou Leachwell

Figure 11.15
ALS Ouagadougou 2012 Leachwell RC G306-4



2012 RC Standards at ALS Bamako FA50

Figure 11.16
ALS Bamako RC FA50 G306-4



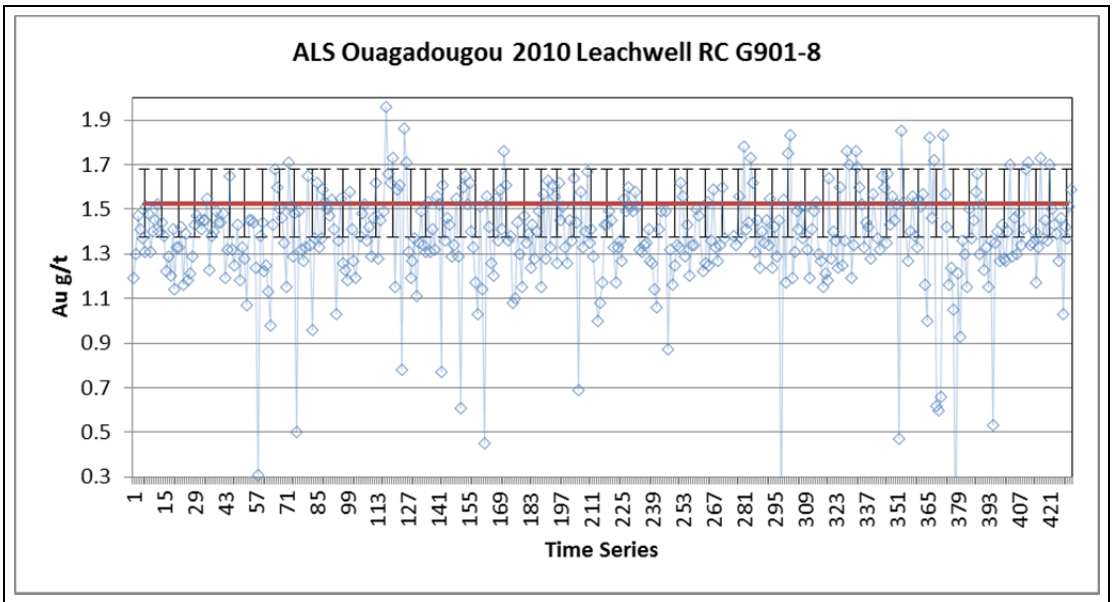
11.7.4 Standards Analysis G901-8 (2010 to 2012)

Standard G901-8 is blended on-site as described in Section 12.4.4. It has an expected value of 1.526 g/t Au. A summary is presented in table 11.8_1. Analysis for this standard was subdivided into sample type and analysis method. Plots are presented below. In summary, all types of samples and different analysis methods show the same features. The mean assayed value demonstrates a low bias in all cases and there is a large spread about the mean assayed

value. Bias varies from -8.4% to -19%. Very low numbers of values fall within ±10% of the EV ranging from 29% to 55%. The causes for such result may be an issue with lab calibration, however the fact that such consistently poor results span a number of years, different sampling types and different analysis methods indicate that the fault is likely to be inherent in the method of preparation of the standard. The balance used for weighing the blank material is likely to have not been correctly calibrated and the homogenization of the blended material is likely to be poor, also no weighted average calculation was used when the standard was blended with blank material.

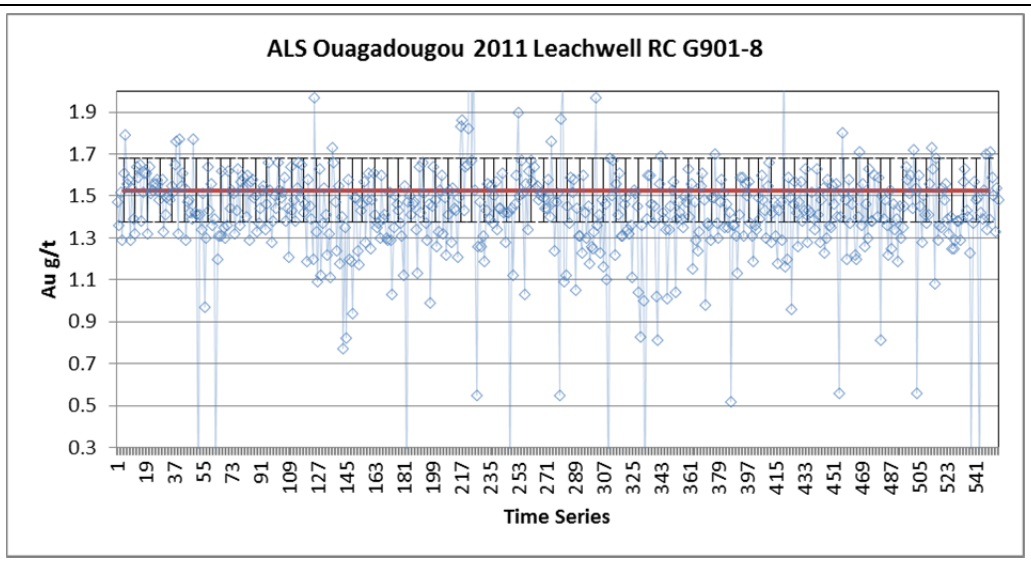
2010 –RC at ALS Ouagadougou Leachwell

Figure 11.17
ALS Ouagadougou 2010 Leachwell RC G901-8



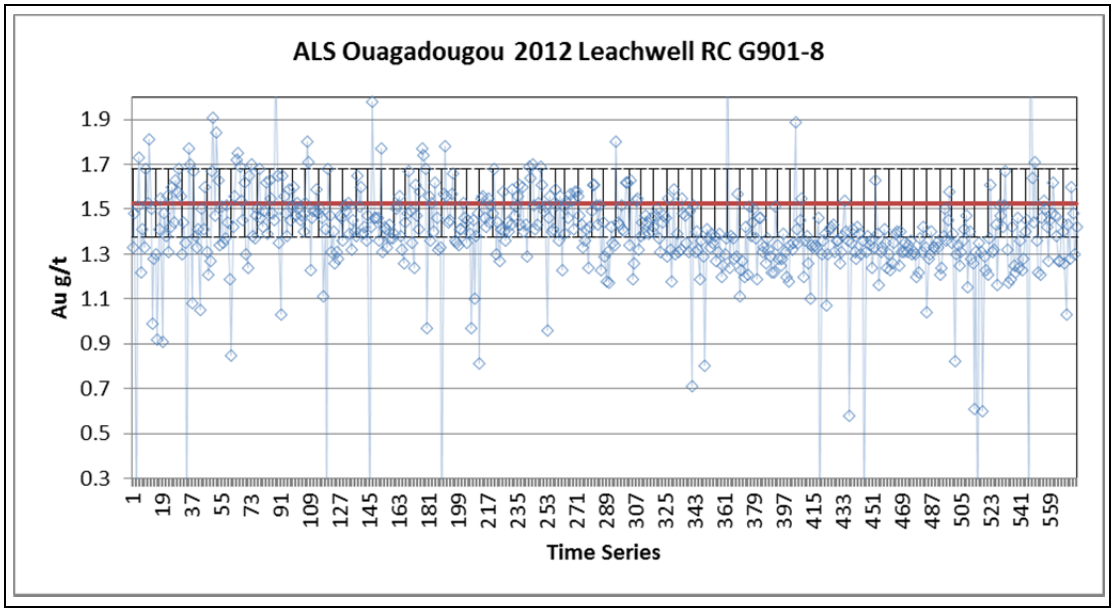
2011 –RC at ALS Ouagadougou Leachwell

Figure 11.18
ALS Ouagadougou 2011 Leachwell RC G901-8



2012 –RC at ALS Ouagadougou Leachwell

Figure 11.19
ALS Ouagadougou 2012 Leachwell RC G901-8



2015 –DDH at SGS Ouagadougou Leachwell - AGG Standard insertions

Three standards were used by AGG during the 2015 campaign, and inserted as described in table 11.2 above. All Standards inserted performed as expected and lie within their expected value range.

Figure 11.20
SGS Ouagadougou 2015 Leachwell CRM G901-7 (Geostats)

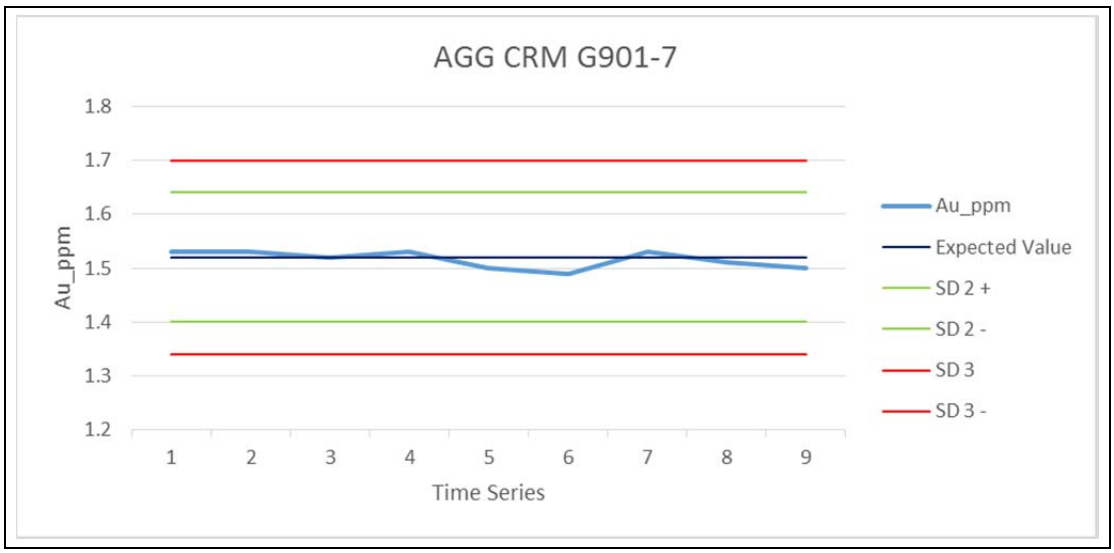


Figure 11.21
SGS Ouagadougou 2015 Leachwell CRM OXG103 (Rocklabs)

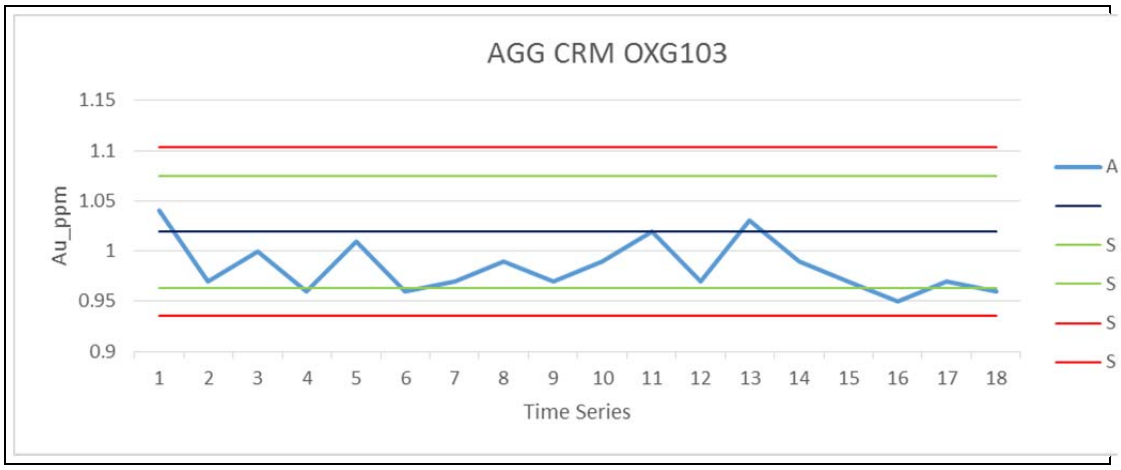
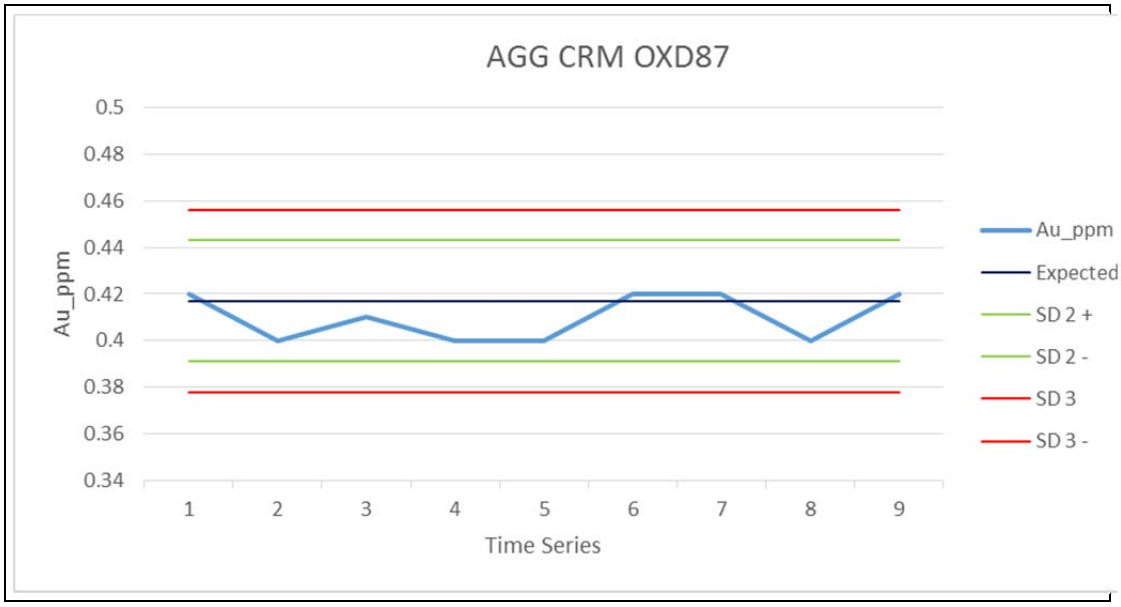


Figure 11.22
SGS Ouagadougou 2015 Leachwell CRM OXD87 (Rocklabs)



2015 –DDH at SGS Ouagadougou Leachwell - AGG Field Duplicates

Field duplicates performed as expected with near perfect repletion of assay values at low grades and greater variation at higher grades which is likely due to nugget effects and small sample size that results from DDH drilling – there is one particular outlier where the duplicate sample assayed c. 9x. The correlation and QQ plots below show the poorer performance of field duplicates at higher grades.

Figure 11.23
Correlation plot – AGG 2015 Field duplicates

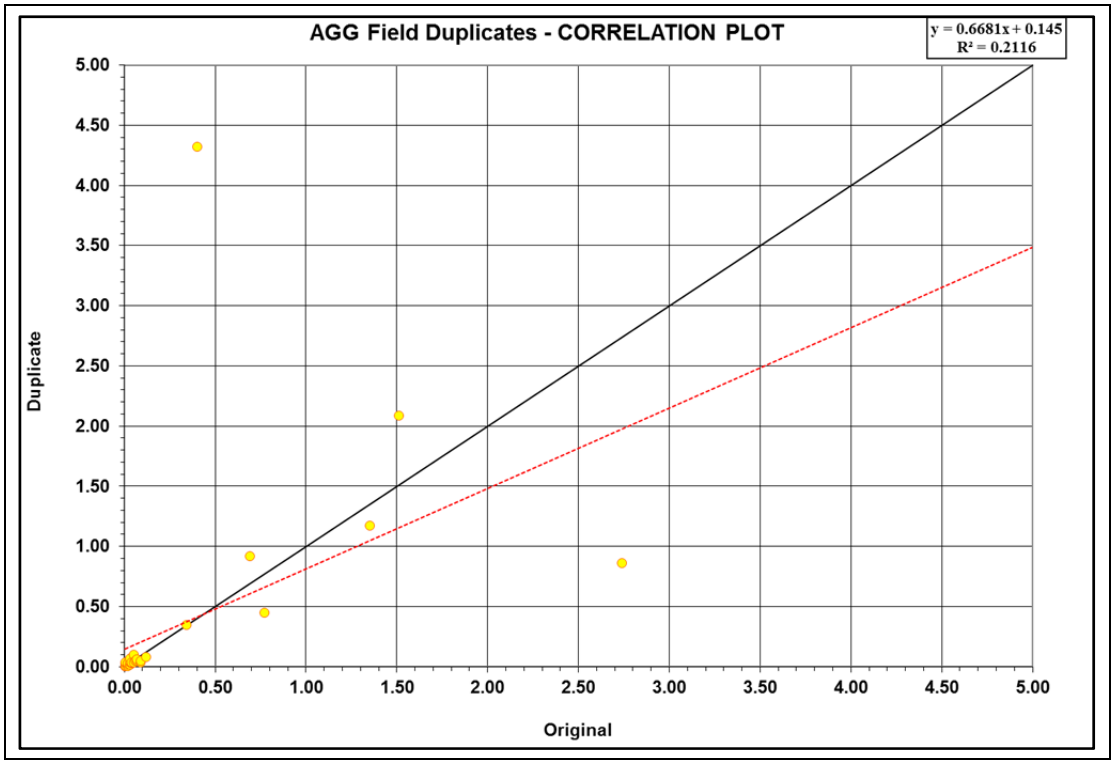
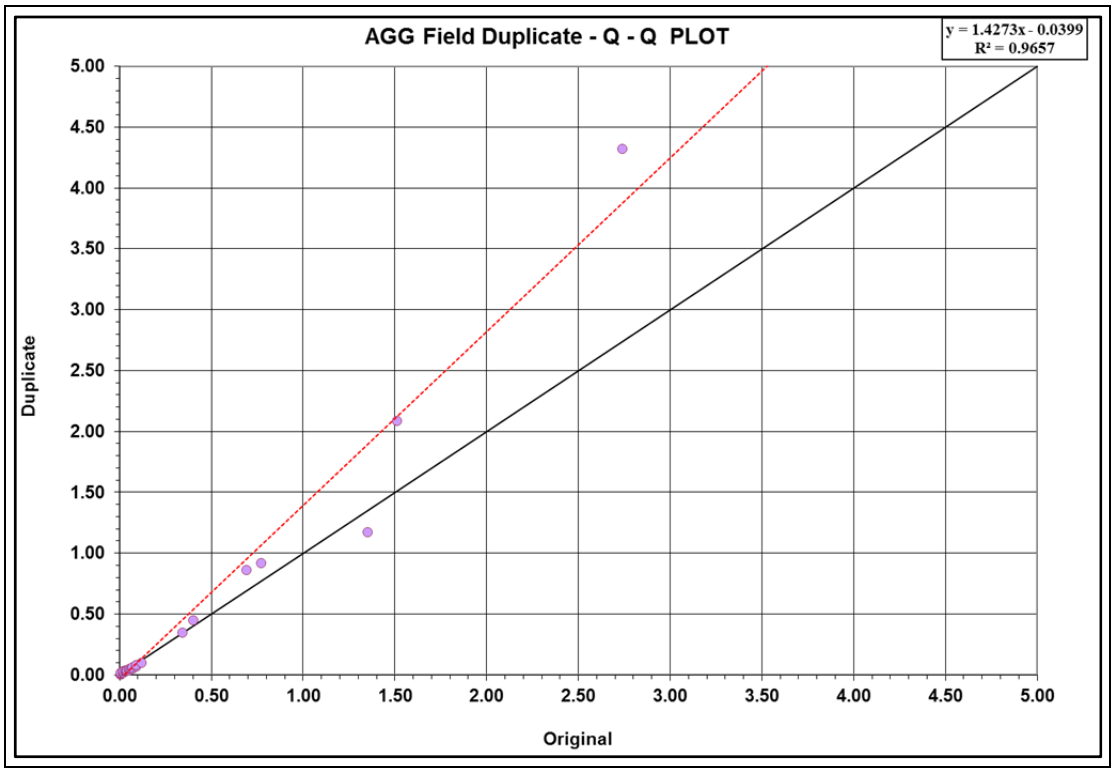


Figure 11.24
AGG 2015 Field Duplicates Quantile-Quantile plot



11.7.5 SGS Ouagadougou Laboratory Standard and Blank Insertions

SGS inserted standards and blanks into the sample stream, three differing Rocklabs standards were used. All SGS standards performed well and are tabulated in figures 37 to 39 below. All blanks were at or below detection limits.

Figure 11.25
SGS Ouagadougou 2015 Leachwell SGS internal Standards Insertions - Rocklabs CRM OXF100

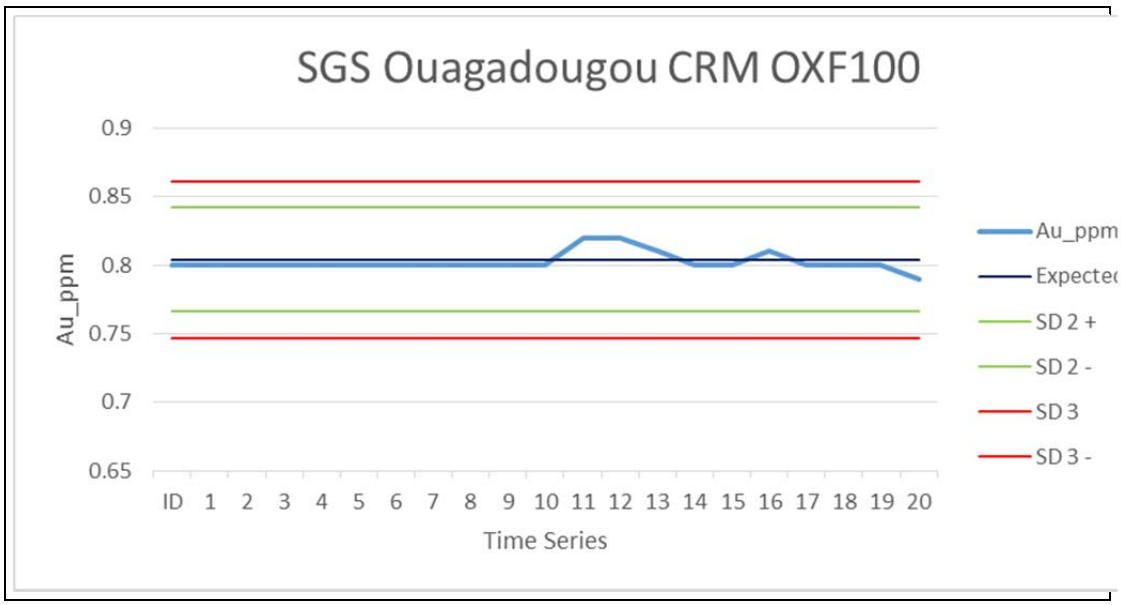


Figure 11.26
SGS Ouagadougou 2015 Leachwell SGS internal Standards Insertions - Rocklabs CRM OXL118

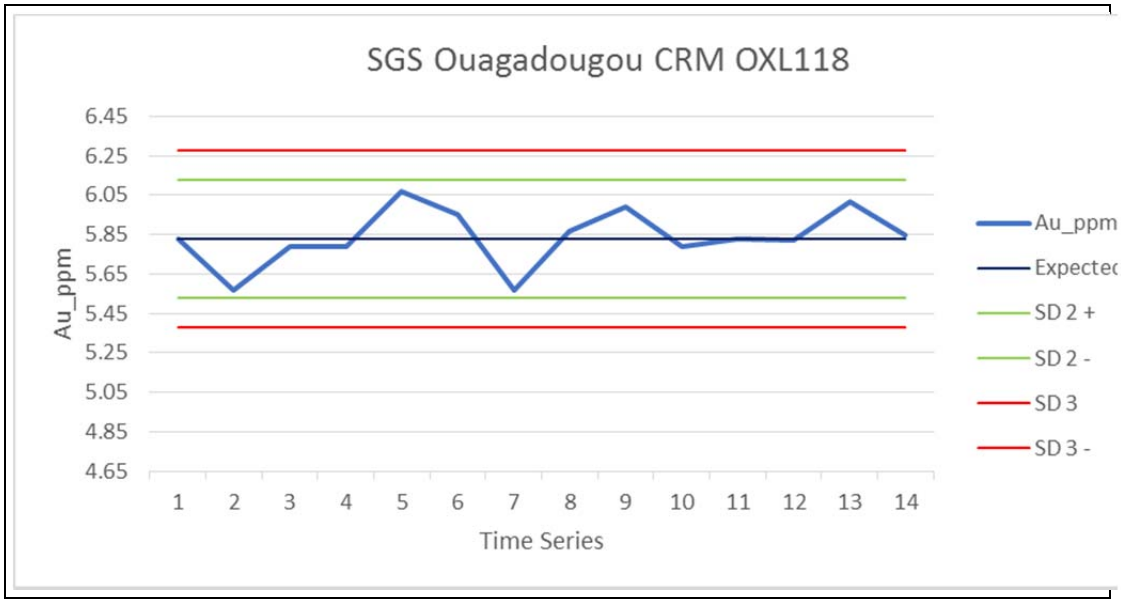


Figure 11.27
SGS Ouagadougou 2015 Leachwell SGS internal Standards Insertions – Rocklabs CRM OXI121

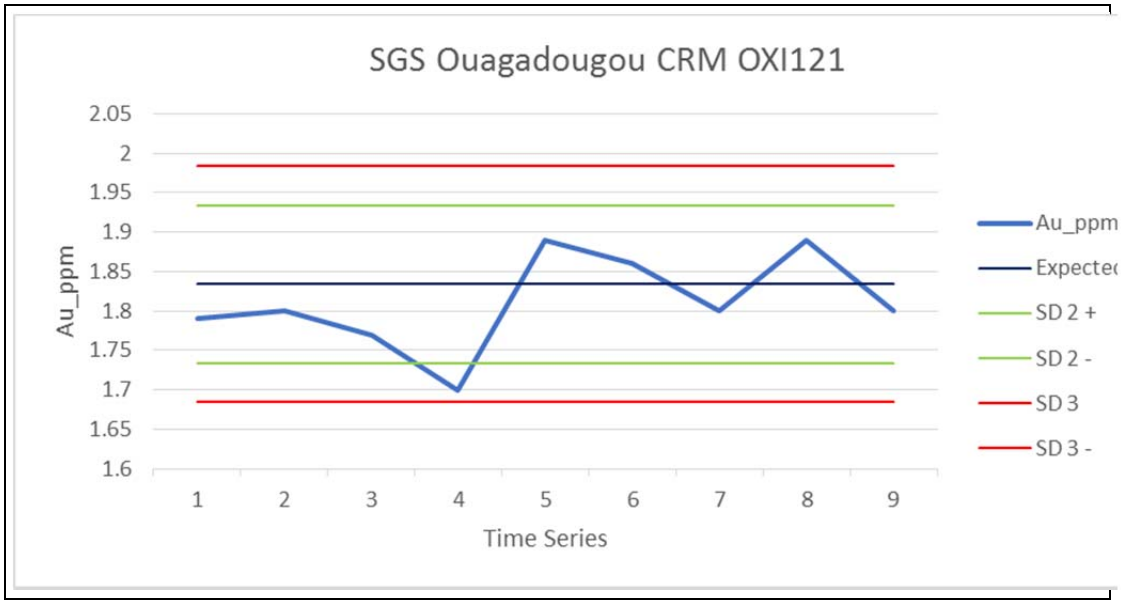


Table 11.3

Field Submitted Blanks and Standards

Standard Name	Type	Year	Expected Value (EV)	+/-10% (EV) (g/t)	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% Bias (from EV)
AGG Submitted Blanks										
Sample Blank	DD ,FA	2006/2007	0.001	-	433	0.005	0.4	0.0176	-	-
Sample Blank	RC, FA	2006/2007	0.001	-	106	0.005	0.55	0.0192	-	-
Sample Blank	DD, LW	2010/2012	0.0017	-	97	0.0005	0.27	0.0061	-	-
Sample Blank	RC, LW	2010/2012	0.0017	-	3,359	0.0005	0.09	0.0019	-	-
Sample Blank	RC, FA	2010/2012	0.0017	-	657	0.0005	0.07	0.0097	-	-
Sample Blank	DD	2015	0.001	-	18	<0.001	0.001	0.001	-	-
AGG Submitted Standards										
G3064RC	LW	2010/2012	0.698	0.628 to 0.768	1729	0.000	21.71	0.67	48	-4.7%
G3064DD	LW	2010/2012	0.698	0.628 to 0.768	48	0.470	0.78	0.62	40	-11.7%
G3064RC	FA	2010/2012	0.698	0.628 to 0.768	208	0.490	1.4	0.60	37	-13.5%
G9018RC	LW	2010/2012	1.526	1.373 to 1.679	1549	1.140	1.1	1.40	53	-8.4%
G9018DD	LW	2010/2012	1.526	1.373 to 1.679	45	0.870	1.62	1.29	55	-15.3%
G9018DD	FA	2010/2012	1.526	1.373 to 1.679	189	0.005	1.88	1.24	29	-19.0%
OXG 60	FA	2007	1.025	0.923 to 1.128	12	0.010	1.12	0.78	100	-23.7%
OXF 53	FA	2007	0.81	0.729 to 0.891	27	0.740	1.32	0.83	93	1.9%
CDN GS 2B	FA	2007	2.03	1.827 to 2.233	20	0.040	2.51	2.00	90	-1.3%
G901-7	LW	2015	1.52	1.368 to 1.672	9	1.5	1.53	1.51	100	-0.7%
OXG103	LW	2015	1.019	0.917 to 1.209	18	0.96	1.04	0.98	100	-3.9%
OXD87	LW	2015	0.417	0.459 to 0.375	9	0.4	0.42	0.41	100	-1.7%

11.8 Comparative Analysis – Leachwell and Fire Assay

During the course of the 2012 RC and DD drilling programs, 10,498 samples and were analysed by both Leachwell and Fire Assay to ascertain the most effective method for gold analysis for the Kobada project.

Figure 11.28 is an Anova Scatterplot of the two analysis types (with linear regression line and R squared of 0.517). The total leach method (Leachwell) generally returns higher grades than the 50g fire assay method. The vertical axis shows the Leachwell grade, while the horizontal is the Fire Assay grade.

Figure 11.29 further divides the analysis type into grade bins and shows that above 1 gram/tonne the Leachwell method returns higher grades than fire assay. Table 11.4 shows the average grade in each bin and the percentage distribution of each bin in the dataset.

Figure 11.28
Two Way Anova Scatterplot with Linear Regression Line of Leachwell vs Fire assay

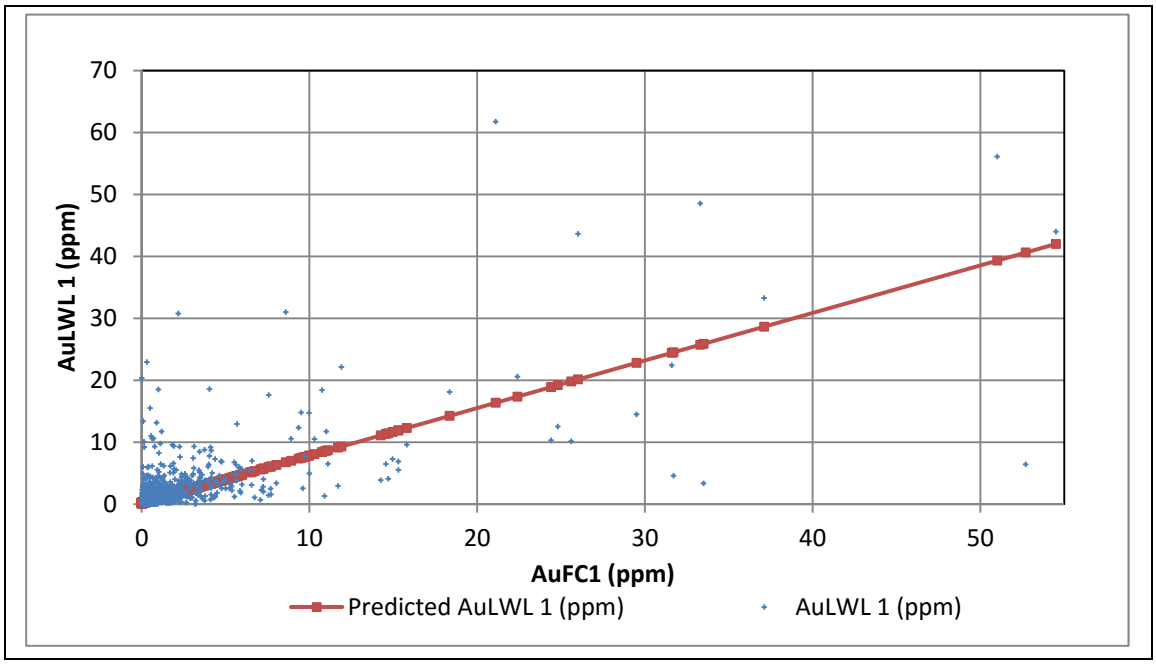
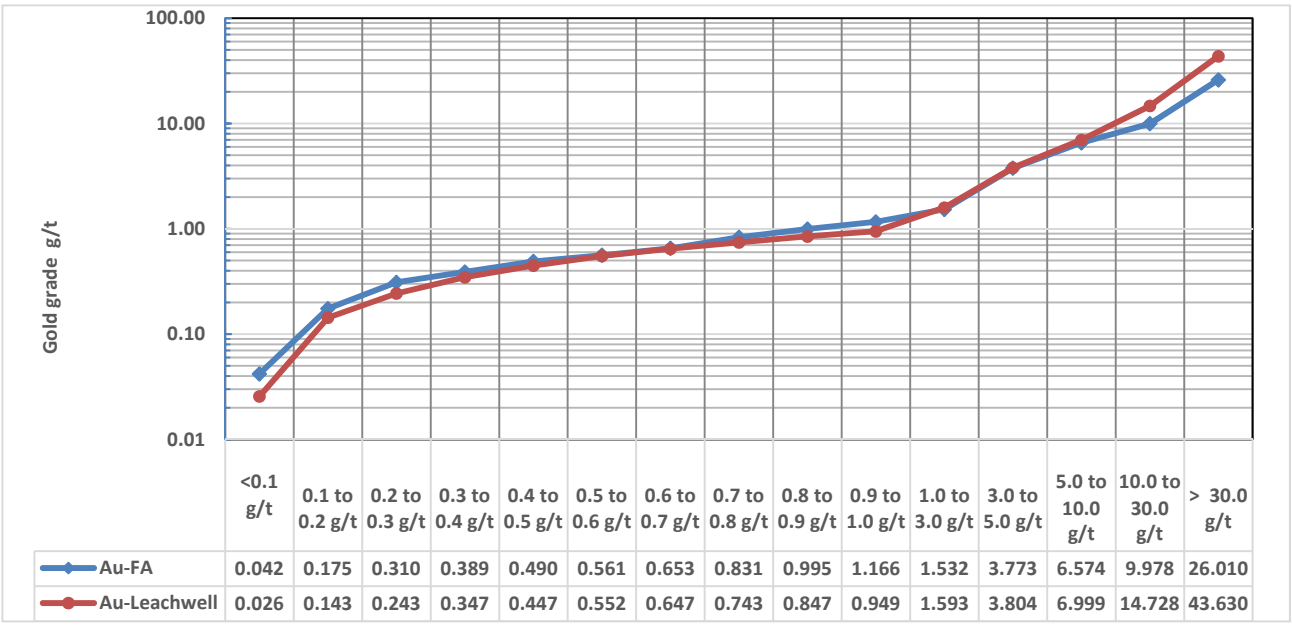


Table 11.4
Leachwell v Fire Assay

	Fire Assay 50-g aliquot			Leachwell 1000-g aliquot		
	Count	Pct population	Au ppm	Count	Pct population	Au ppm
Total Samples	10498			10498		
Below Detection	1122	10.7%	<0.01	268	2.6%	<0.001
<0.1 g/t	6447	61.4%	0.042	5868	55.9%	0.026
0.1 to 0.2 g/t	1351	12.9%	0.175	1267	12.1%	0.143
0.2 to 0.3 g/t	640	6.1%	0.310	674	6.4%	0.243
0.3 to 0.4 g/t	378	3.6%	0.389	456	4.3%	0.347
0.4 to 0.5 g/t	271	2.6%	0.490	281	2.7%	0.447
0.5 to 0.6 g/t	260	2.5%	0.561	267	2.5%	0.552
0.6 to 0.7 g/t	200	1.9%	0.653	297	2.8%	0.647
0.7 to 0.8 g/t	151	1.4%	0.831	196	1.9%	0.743
0.8 to 0.9 g/t	105	1.0%	0.995	119	1.1%	0.847
0.9 to 1.0 g/t	73	0.7%	1.166	88	0.8%	0.949
1.0 to 3.0 g/t	489	4.7%	1.532	802	7.6%	1.593
3.0 to 5.0 g/t	61	0.6%	3.773	97	0.9%	3.804
5.0 to 10.0 g/t	43	0.4%	6.574	50	0.5%	6.999
10.0 to 30.0 g/t	21	0.2%	9.978	29	0.3%	14.728
> 30.0 g/t	8	0.1%	26.010	7	0.1%	43.630

Figure 11.29 illustrates the data from Table 11.4.

Figure 11.29
Log Normal Plot of Gold Grade for both Leachwell and Fire Assay analysis



The analysis comparison shows that both methods are effective for analysis at the Kobada project, and there is a bias toward higher grade results from the total leach where samples have a higher grade. This is to be expected as the method has a higher sample size (1 to 2 kilograms) compared to the 50 gram charge of the Fire Assay. At lower grades the detection limits of Fire Assay are marginally superior to Leachwell, which may explain the method reporting higher. Care also must be taken with the Leachwell method where sulphides are present as they ad-

versely affect the pH of the leach (reducing) which in turn reduces the effectiveness of the leach and amount of gold dissolved in solution.

11.9 Drillhole twinning

A drill-hole twinning program was completed during February and March 2012. Ten drill-hole twins were completed by diamond drilling. KBRC122, KBRC052, KBD19, KBRC060, and KBRC7-6 were analysed by fire assay on 50g and all other holes by Leachwell. Results are presented in Figure 11.30 to Figure 11.39 below.

Graphically presented, it can be seen that twin holes generally reproduced the intercepts of anomalous gold mineralization. This relationship becomes less clear the further apart the drillholes become. Variations in grade are noticeable and this can be explained by differences in assay technique, sample size and local variability (nugget effect) of the mineralization. The drillhole twinning program can be considered successful in the objective of reproducing the mineralised intercept.

Figure 11.30
Drillhole Twins KBD19 and KBD12-115

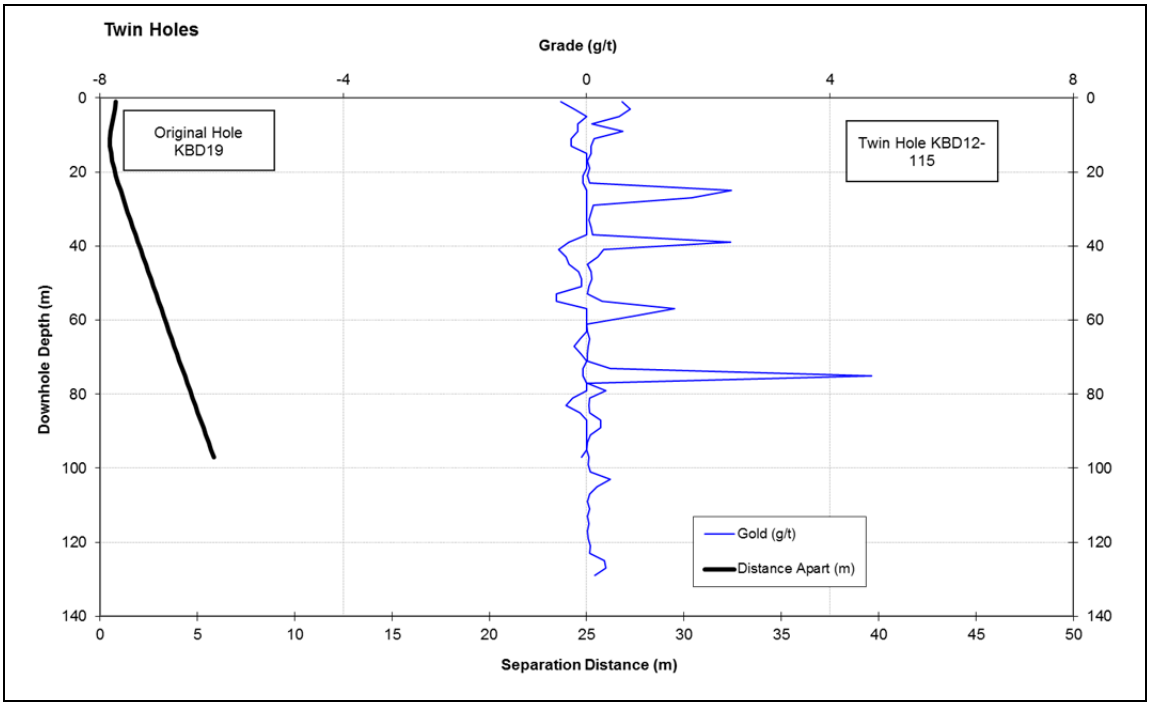


Figure 11.31
Drillhole Twins KBRC7-6 and KBD12-119

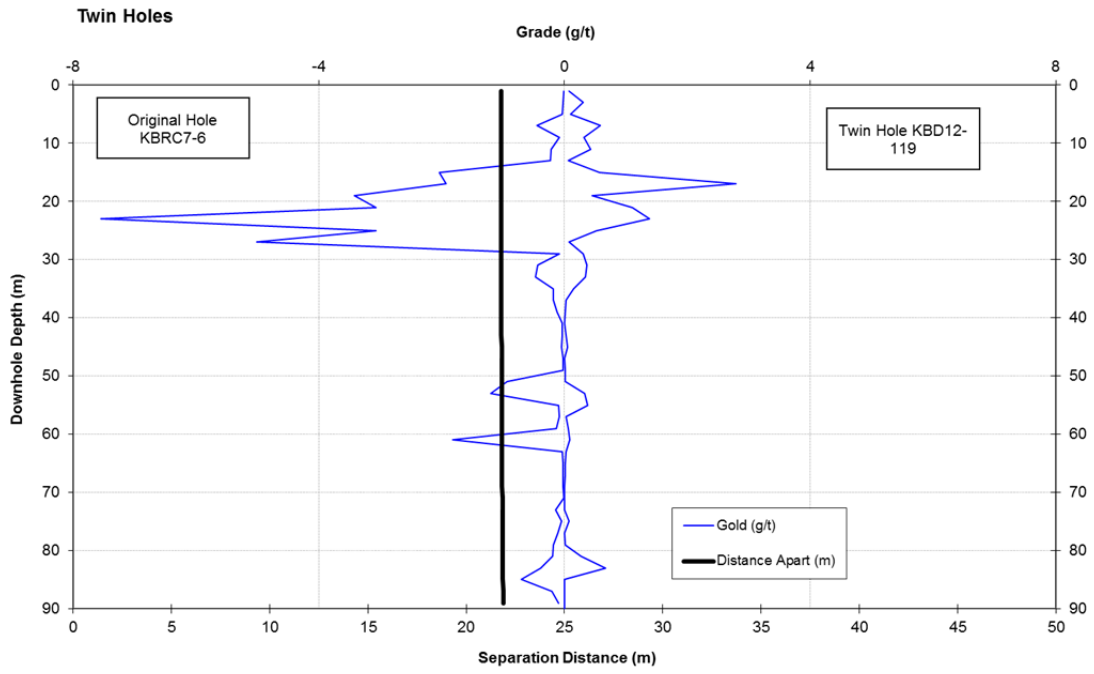


Figure 11.32
Drillhole Twins KBRC9-08 and KBD12-116

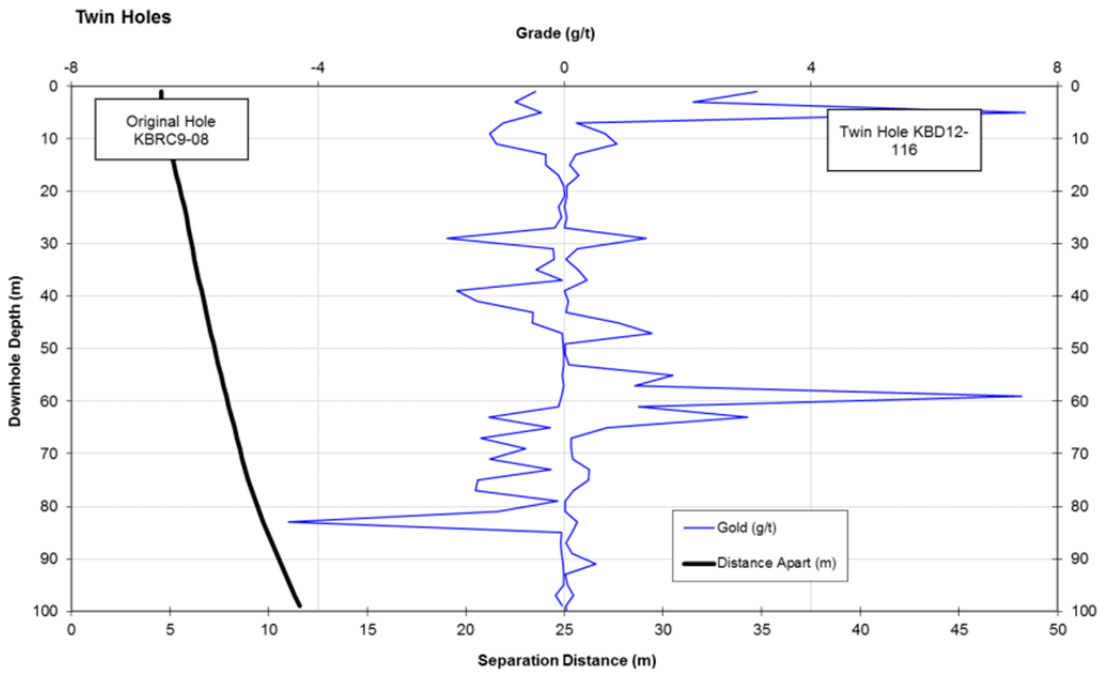


Figure 11.33

Drillhole Twins KBRC10-014 and KBD12-118

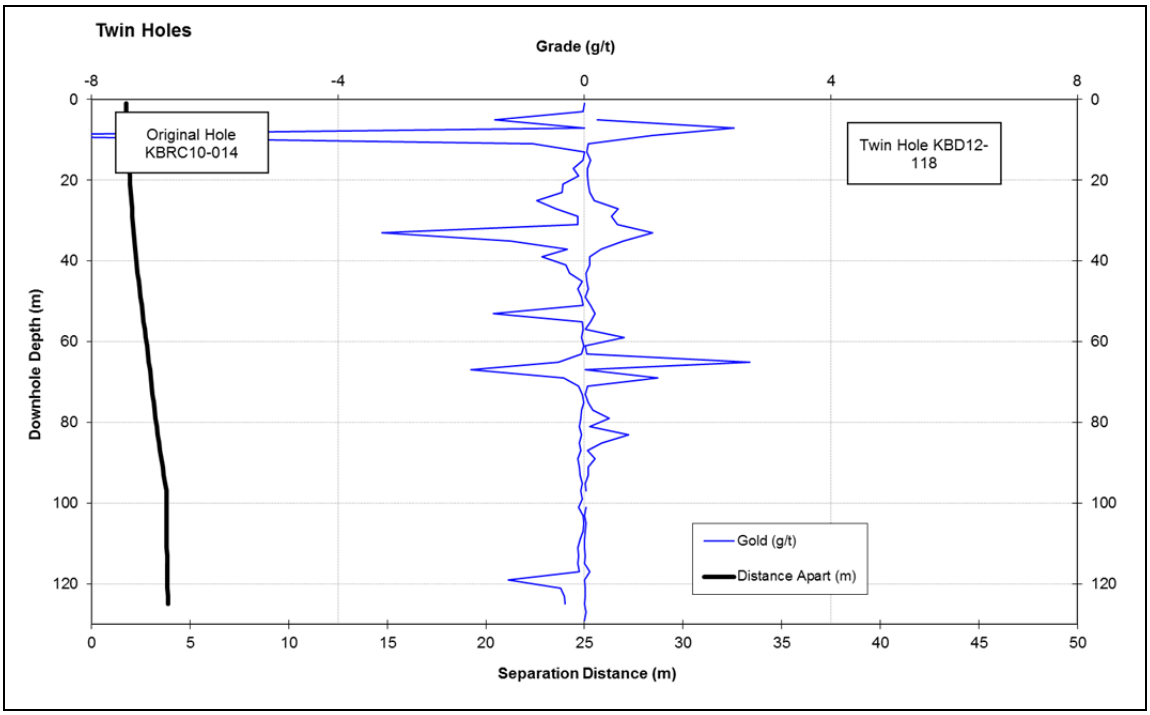


Figure 11.34

Drillhole Twins KBRC052 and KBD12-113

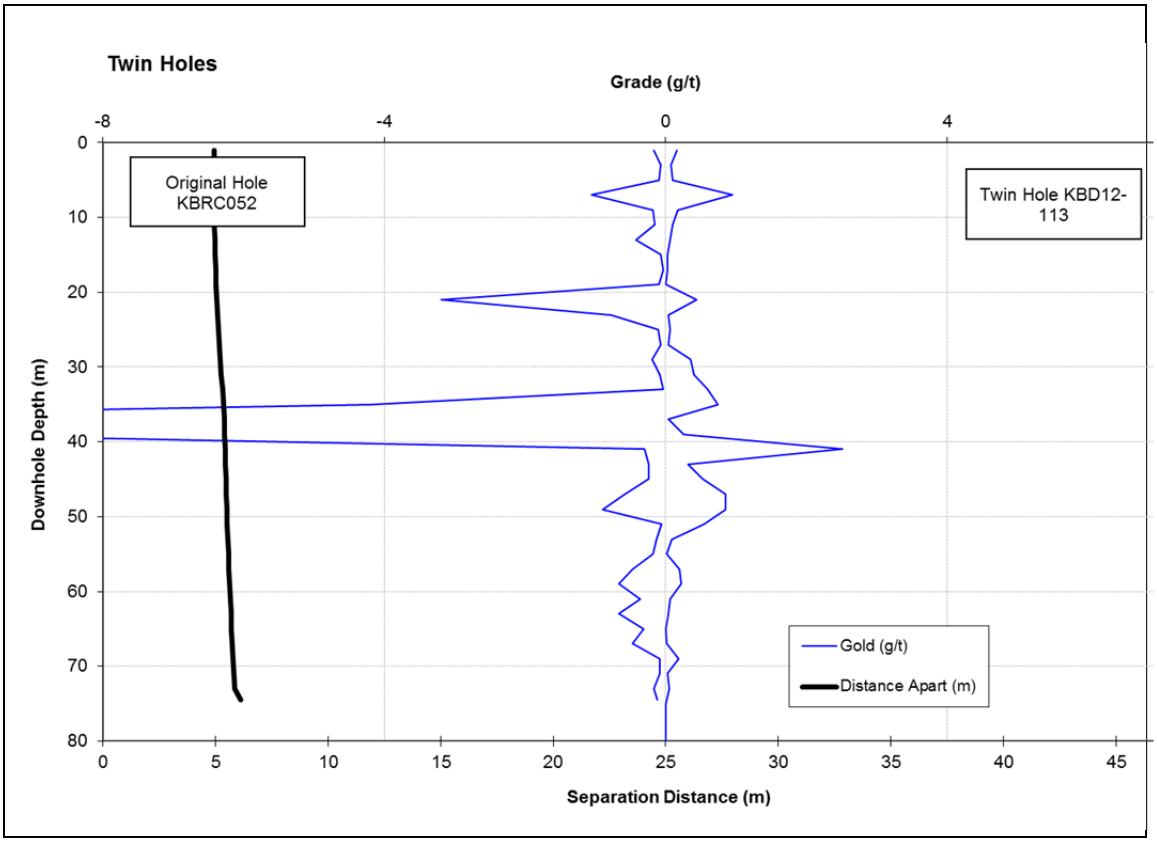


Figure 11.35

Drillhole Twins KBRC060 and KBD12-117

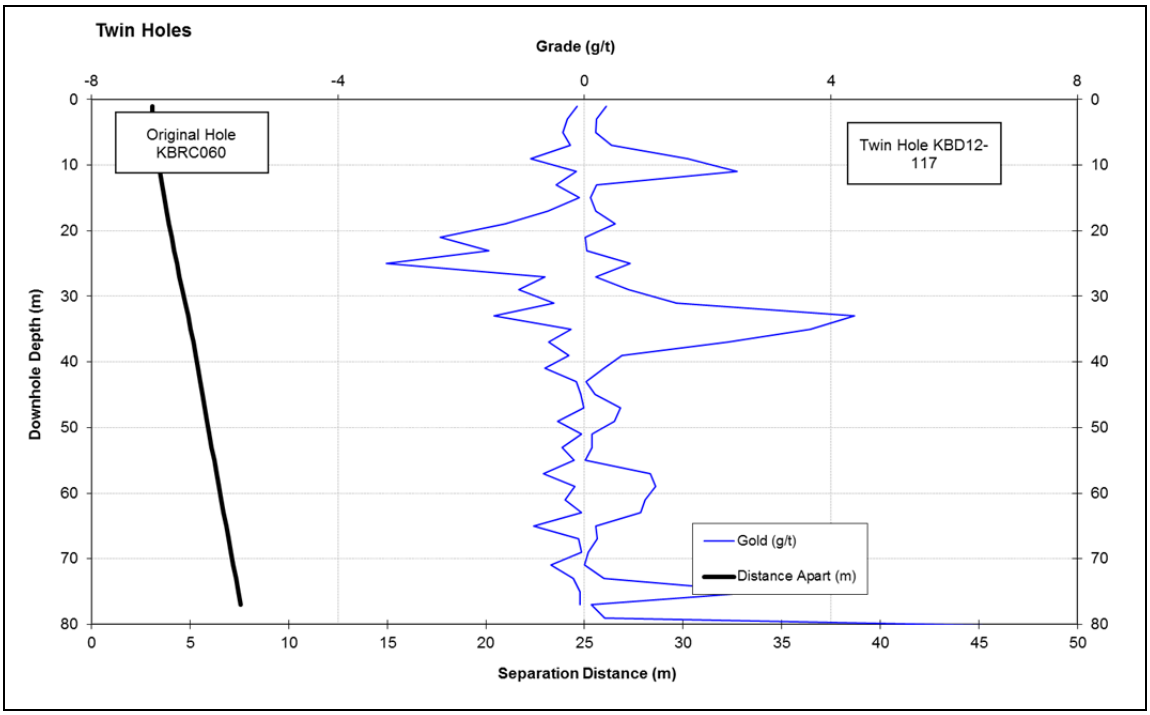


Figure 11.36

Drillhole Twins KBRC122 and KBD12-111

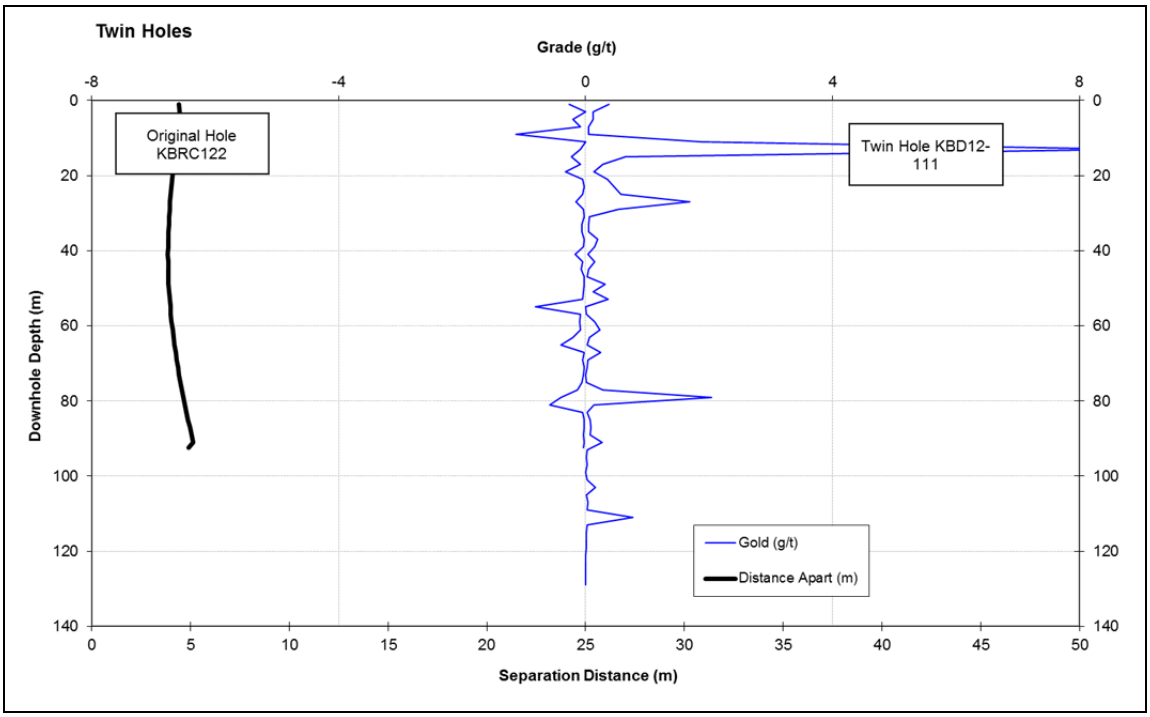


Figure 11.37
Drillhole Twins RPA10-006 and KBD12-110

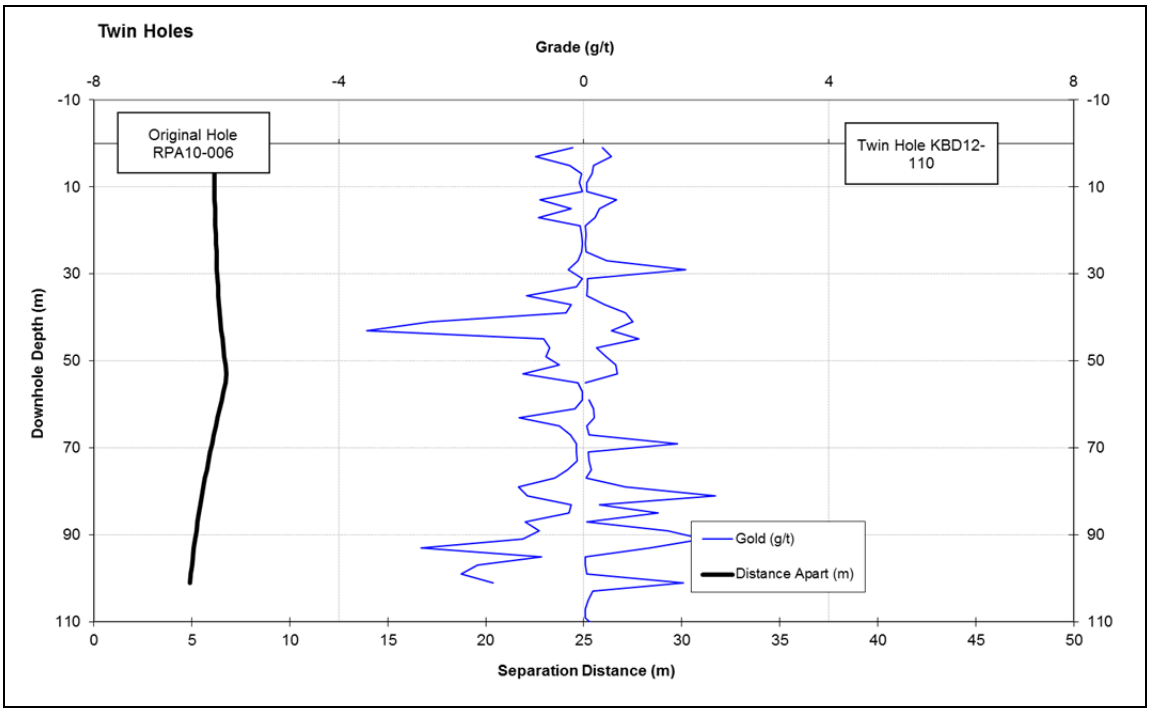


Figure 11.38
Drillhole Twins RPA10-009 and KBD12-112

Figure 11.9_9

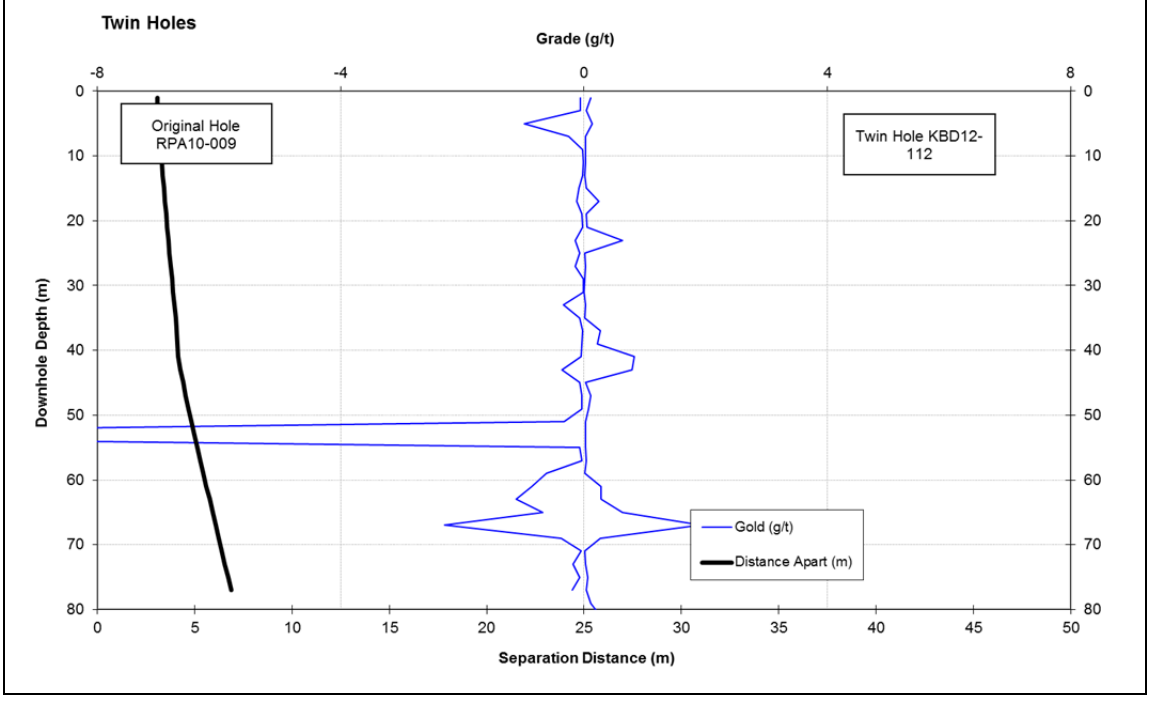
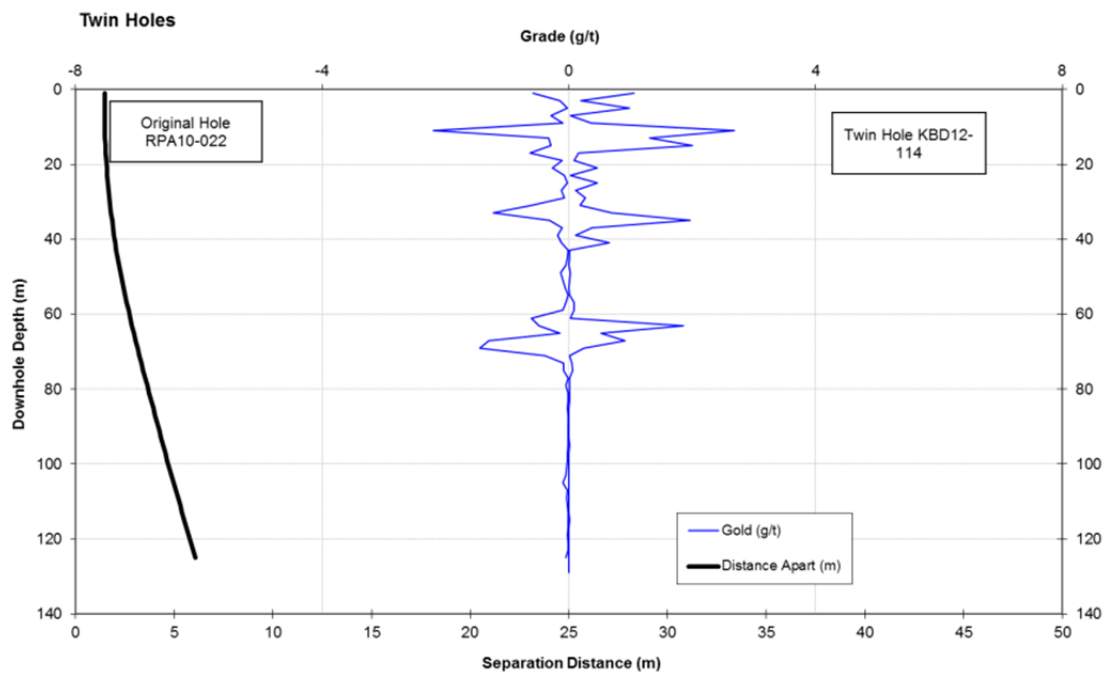


Figure 11.39
Drillhole Twins RPA10-022 and KBD12-114



11.10 Conclusions and Recommendations

The main AGG drill hole database contains drill hole and assay data dating back to 1988. The drillhole database has been provided as a series of MS Excel spreadsheets and also a Master MS Access database. This has been maintained internally by AGG staff, however, by current standards (Acquire, Datashed, etc.), the database has not been maintained in a form or manner that allows for easy and intuitive interrogation of the data. This includes extraction of QAQC data in a manner that allows the user to easily understand and interpret this information.

Very limited if any assay QAQC data exists prior to AGG's involvement in the project. QAQC data in the drillhole database is also somewhat limited prior to the beginning of the 2010 drill programme. Since the beginning of 2010 AGG have undertaken a thorough QAQC program as relates to laboratory analysis that includes standards and blanks analysis, field duplicate sampling and umpire assaying at a second laboratory.

Previous interpretation of existing QAQC data exists (Alexander & Lefleur, 2008 and Marchand 2013) and this has been reviewed and some of the work has been duplicated as a check on this work. The following conclusions are drawn:-

- Standards analysis available from 2007 shows no bias and indicates no issues with the analysis undertaken at that time.
- Available blanks analysis does not indicate a problem with any undertaken analysis.

- Standards analysis available from the period 2010 to 2012 indicates a likely problem with the standard itself as evidenced by poor performance. No conclusion can therefore be drawn regarding assay accuracy and precision. This problem is likely to relate to the method of production of the standard. Marchand (2013) draws the same conclusion.
- Notwithstanding the above point, the conclusion is not automatically drawn that the overall low mean grades obtained during the standards analysis indicate an overall bias during the assaying.
- The QAQC analysis and discussion presented in Marchand (2013) have been reviewed and it is considered that many comprehensive and adequate QAQC procedures have been undertaken since 2010. The analysis presented does not appear to indicate any systematic problems with the data.
- Drillhole twinning has been successful in reproducing mineralised intercepts albeit with positional differences relating to the pierce point separation. Grades however, appear highly variable on relatively short scales.
- It is strongly recommended that all available QAQC data be compiled into the drillhole database in a format that allows intuitive interrogation of results. This includes the internal laboratory QAQC protocol, none of which has been made available at this time.
- It is also recommended to compile a complete re-analysis of all QAQC data available to assist in future decisions regarding drilling and sampling. Currently much of this data is not available in the database, only spreadsheets and this is currently not industry standard.
- To this end, it is recommended to recompile the drillhole database into a more industry standard format. As the project progresses to feasibility stage this becomes more critical.

Despite shortcomings in the performance of the standards from 2010 onwards, it is not considered likely that significant bias exists in the database relating to its use for resource estimation.

It is considered that AGG company procedures are sufficient to ensure the integrity and validity of the samples taken and that sample preparation and analytical procedures employed are industry standard.

It is therefore considered that the data supplied is suitable and adequate for the purposes of resource estimation.

12 DATA VERIFICATION

It was considered that AGG's company procedures are sufficient to ensure the integrity and validity of the samples taken and that sample preparation and analytical procedures employed are industry standard. It was also considered that the data supplied is suitable and adequate for the purposes of resource estimation.

12.1 Introduction

Data verification steps undertaken have included:

- Reviews and checks of the drillhole database have been generated by AGG.
- Checking of available original assay certificates against the generated database.
- Independent verification of drillhole collar locations by GPS during site visit.
- Visual verification of altered and mineralised drill core during the site visit.
- Review of QAQC results analysis to identify any irregularities (previous section).

12.2 Database review and checks

The exploration database is maintained as an Access database generated by AGG internal staff. Data has been exported as csv's and validated prior to use for resource estimation purposes. Additionally some QAQC data has been supplied by AGG staff as excel spreadsheets. At least some of this QAQC data would not appear to be present in the MS Access database. Further to this, in 2012 more than 10,000 samples were used for comparative analysis between fire assay and Leachwell analysis methods. This comparative analysis was completed using the same sample numbers with differing analysis batches, again this is not clear in the Access Database and it is not immediately clear which method is used for reporting and resource estimates, this can also be rectified as part of a database reconstruction exercise.

Original assay certificates (PDF format) for analysis undertaken as part of AGG's work programmes have been provided as part of the data verification process. Original assay certificates have been randomly checked against the original Access database and no discrepancies have been noted, the certificate set was near complete. No assay certificates have been provided for work undertaken prior to AGG's involvement in the project. However it should be noted that AGG's drilling comprises 87% of the total metres drilled at Kobada and that, statistically, there is no difference in the tenor of grades obtained prior to their involvement. Additionally, a number of drillholes have been twinned and no significant discrepancies noted.

12.3 Site Visit Review and Checks

Initial site visits were undertaken prior to the 2014 resource estimation, subsequently a second visit was undertaken during the 2015 drilling program and a limited relogging exercise was undertaken by Andrew Chubb. During the initial site visit numerous drillhole collars were located and coordinates checked by handheld Garmin GPS. Collars were randomly chosen throughout the strike length of the deposit. Checked coordinates correlated with the recorded coordinates in the drillhole database within the tolerance of error of the handheld GPS.

Representative drill core has been inspected during the site visit. Altered and mineralised drill core was well represented in the suite of drillholes presented. Visible gold is not common at Kobada and was not noted during the visit.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

All work completed to date has focussed on the oxide ore types found at the Kobada Project. Test work and metallurgical concepts have been developed since 2009. In total 1.2 tonnes of saprolite ore and 1.2 tonnes of laterite ore has been exported to Canada and Australia for independent testing by SGS Canada Inc. and Gekko Systems Pty Ltd. These samples were sourced from different locations and depths on the Kobada Mineral Resource area.

Additionally, AGG completed a bulk sample program on site in 2012 which treated a total of 100 tonnes of RC drill chips. This work specifically examined the volume variance effect that increasing sample size had on sample grade. This work was discussed in the Company's press release of June 18, 2014.

13.2 Ore Characteristics

13.2.1 Mineralogy

Mineralogical examinations were conducted on a sample from the 2009 program, which consisted of a de-slimed head sample (ground to P80 150 µm) that was pre-concentrated using heavy liquid separation at a specific gravity of 3.1 g/cm3 and super-panning of the sinks fractions. Approximately 86% of the gold was concentrated into the sinks, which accounted for 11% of the mass. 70% of the gold was concentrated in the super-panner into 0.15% of the mass, indicating that the gold is liberated and of a significant size.

Mineralogy showed that the sample consisted mainly of quartz, with moderate amounts of goethite and minor to trace amounts of mica, kaolinite, rutile, zircon and maghemite. The gold present is near fully liberated and has a composition of 90.6%Au, 8.9% Ag and 0.6% Fe. The visible gold present is near to fully liberated, with an average size of 12 µm, which is very small given the high gravity gold recovery observed in other testwork. An explanation for the high gravity recovery is that fine gold is occluded within heavy particles such as goethite/iron oxides, or simply not enough gold grains were observed (just 26) to be representative of the sample.

A whole rock analysis (by XRD) determined the main chemical composition as: 68.6% SiO_2; 11.1% Al_2O_3; and 12.1% Fe_2O_3.

13.2.2 Head Grades

The head assay grades from the test programs are summarised in Table 13.1. In all test programs the calculated head assay was significantly higher than the head assay result, which is a clear indication of the presence of coarse gold in the deposit, which is more accurately measured by taking larger samples and concentrating the gold into a smaller mass.

Table 13.1 Sample Characteristics							
Test Program	Head assay Au (g/t)	Calc. assay Au (g/t)	Fe (%)	Cu (ppm)	Zn (ppm)	As (ppm)	Specific Gravity (g/cm^3)
2009 - Saprolite	1.07	1.92	~8.5*	~73*	-56*	~1500 *0	~2.82*
2014 - Saprolite	0.88	1.20	5.8	66	42	880	
2014 - Laterite	0.78	0.94	>1.5	66	32	2051	
2015 - Saprolite	0.84	0.92	-	-	-	-	2.7
2015 - Laterite	0.78	1.3	-	-	-	-	3.0

*Deslimed sample

Screen analyses of the head samples were completed during the 2014 program and results are shown in Table 3-3. The Saprolite gold distribution is much coarser than the particle size distribution, showing that there is potential for preconcentration of the gold value by desliming using cyclones. The characteristic is not shown by the Laterite sample tested, as the gold distribution is slightly finer than the particle size distribution.

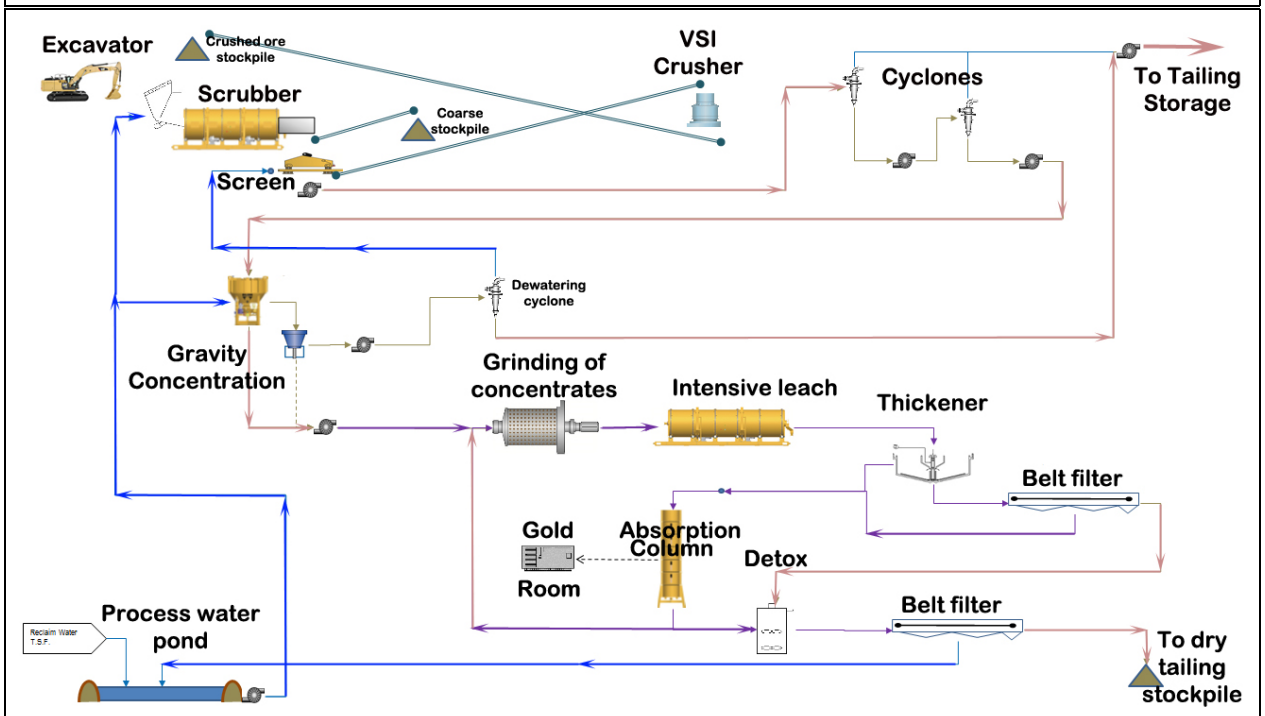
Table 13.2 Sizing Analysis				
Test Program	Particle dist'b P80 (µm)	Particle dist'b P50 (µm)	Gold dist'b P80 (µm)	Gold dist'b P50 (µm)
2014 - Saprolite	230	28	1687	116
2014 - Laterite	3163	790	1058	288

13.3 Minerals Processing

This feasibility study considers the treatment of oxide ore types only. The ore at Kobada is amenable to physical concentration by taking advantage of differences in material size and bulk density. The Kobada gold project process plant will consist of scrubbing, oversize crushing and recycling of crushed product to scrubber, screening, a hydrocyclone pre-concentration stage, gravity concentration, intensive leaching, leach discharge thickening and filtration, resin absorption, resin stripping and elctrowinning, cyanide detoxification circuit, leach tails filtration, with associated services and ancillaries. The process plant has been designed to treat 1.6 million tonnes per annum, with gold produced as dorè bars ready for shipment to a refinery. The proposed flow sheet is detailed in Figure 13.1.

Figure 13.1

Simplified Process Flow Sheet



Source Gekkos January 2016

It consists of a large scrubber (7.9m long x 3m diameter) to create ore slurry and remove larger rocks (+40mm). Screening of the -40mm from the scrubber separates the fine material (-1.18mm) and directs the coarse ore to further crushing. The -1.18mm slurry is then passed to a bank of hydro-cyclones for the pre-concentration stage. This pre-concentration process rejects up to 70.5% of the feed mass while recovering 94% of the gold.

Pre-concentration is to be followed by gravity concentration consisting of InLine Pressure Jigs and Knelson centrifugal concentrators. The gravity circuit is planned to produce a concentrate representing 5% of the feed to the jigs. The tailings from the pre-concentration and gravity concentration are pumped directly to the inert Tailings Storage Facility. This tailings material has had no chemical addition and can be disposed of in an unlined impoundment.

The concentrates are then to be passed to the grinding circuit. This comprises a ball mill in closed circuit with cyclones to produce an 80% passing 125 micron grind size. Cyanide is also added to the ball mill to increase leaching residence time. Grinding is followed by intense leach reactors with leaching occurring in a high oxygen environment for around 10 hours.

After leaching, the slurry density is increased in a thickener before the remaining leach solution is recovered from a belt filter. Pregnant solution will be fed directly to an AuRIX resin absorption column to extract the gold from the solution and return the barren solution to the process. The solid residue from the belt filter will then be treated in an SO_2 and O_2 cyanide destruction circuit (detox) to reduce the tailings cyanide concentration to below the International Cyanide Management Code (ICMC) requirements. This residue will be stored as dry tailings in a purpose built lined facility that will be encapsulated within the waste rock dumps.

A resin stripping, electro-winning and gold smelting system to produce dorè are located in a secure gold room. The process plant has been designed to treat 1.6 million tonnes of ore per annum, with gold produced as dorè bars ready for shipment to a refinery outside Mali. The process is expected to recover an average of 82% of the gold contained in ore, and the test work indicated that a constant tailings grade of 0.23 g/t Au can be expected. This means that a higher recovery should be achieved at higher process feed grade.

13.4 Metallurgical Testing

A review of all testwork on Saprolite and Laterite from the Kobada deposit was completed. The key findings were:

- The Kobada deposit consists of two main mineralization types: laterite and saprolite. A surface layer of laterite is generally between one and 8 meters thick, although in some areas the laterite is not present. The underlying saprolite is consistent to the base of oxidation. The base of oxidation is highly variable, being between 50m and 120m below surface. The Mineral Reserve contains 6% laterite by tonnage;
- The main constituent minerals are: SiO_2 (69%), Al_2O_3 (11%) and Fe_2O_3 (12%);
- Gold is generally relatively coarse and fully or partially liberated which allows for high gravity recovery potential;
- The Saprolite gold distribution is much coarser than the particle size distribution, showing strong potential for preconcentration of the gold value by desliming using cyclones. Testwork has demonstrated hydrocycloning can reject up to 70% of the Saprolite ore feed mass following the scrubbing stage. Gold recoveries to the product stream (cyclone underflow) averaged 94%, leading to average slimes grades of 0.09g/t. Preconcentration only applies to the saprolite ores, with laterite not exhibiting the same characteristics;
- As a result of saprolite preconcentration, on average, 70% of total ore mass can be sent directly to tails with an expected loss of 0.09g/t. At the reserve grade of 1.25 g/t, this leads to material grading 2.85 g/t on average passing to the gravity concentration circuit;
- Gravity concentration has been shown to be effective in recovering gold from ore after preconcentration. Gravity tails grades of between 0.15 to 0.23 g/t gold were recorded across all testwork on saprolite ore types, at a mass recovery of 5%. The average grade of saprolite mineral reserve was 1.25g/t, and average gold recovery is expected to be between 81.6% and 88%;
- Testwork on the laterite ore type indicated that the tail grade was 0.25g/t Au at a 1.18mm crush. The average grade of laterite mineral reserve was 1.26g/t, and average gold recovery is expected to be 80%;
- An aggregate gold recovery of 82.1% is expected to be achieved on average from the mineral reserve over the life of mine;
- Screening following scrubbing (at 1-2mm) will result in the concentration of 50% of the gold values into the oversize fraction, which made up 10 – 30% of the feed

mass. This characteristic can be exploited, by crushing and then bypassing this fraction to the gravity concentration circuit;

- Gold leaching recoveries to solution were high, and leach times can be reduced to <10 hours by grinding down to a P80 of 125 µm; and
- The ground leached residue proved to be very amenable to belt filtering and washing.

13.4.1 Testwork History

Samples from the Kobada mineral resource were subject to testwork programs which are summarised in Table 13.3. The location of samples sourced for these programs are displayed in Figure 13.2. The initial program in 2009 was completed by SGS, while all subsequent programs have been completed at the Gekko Systems Metallurgical laboratory (GML).

Table 13.3
Sample Source and Test Program Summary

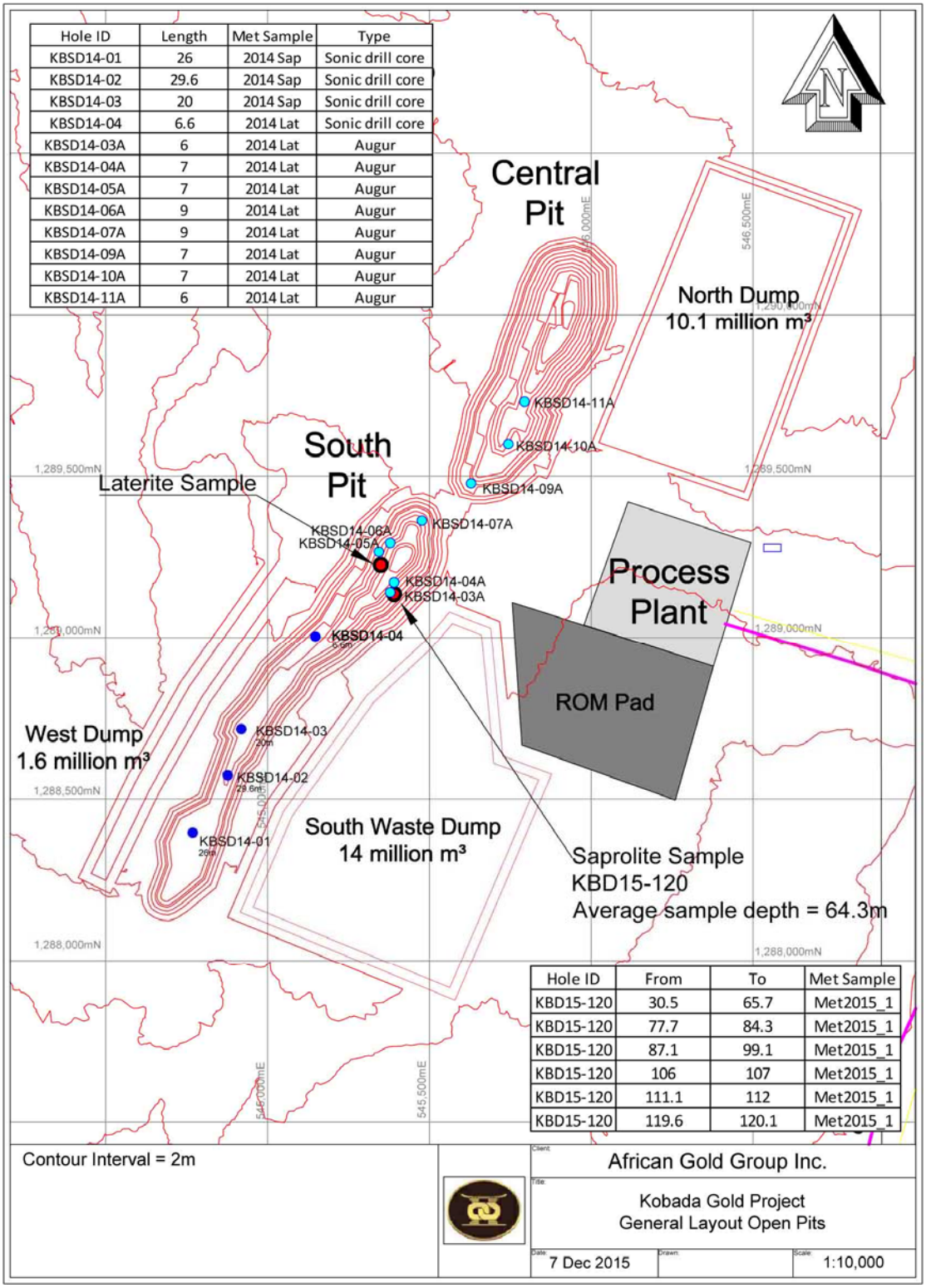
Test Program	Sample Name and Size	Source	Tests
2009 SGS	287 kg Saprolite composite	Collected from 127 separate intervals in 8 drill holes. The holes were located in the northern part of the south pit to southern part of central pit.	Mineralogy, Cyclone desliming, Gravity (Knelson) and Leaching of cyclone underflow.
2014 GML	594 kg Saprolite composite	Core samples via sonic drill. Holes 01a (6-25.7m), 02A (3-29.7m), 03A (3-21m), 04A (2.5-6.6m), 07A (7-9m), 10A (6-7m), 11A (4-6m).	Dry scrubbing and screening, Hydrocycloning, VSI crushability, Gravity tabling
2014 GML	1108 kg Laterite composite	Samples collected by augur. Holes 01A to 11A. Depths augured varied between 0 to a maximum of 9m.	Dry scrubbing and screening, VSI Crushing, Gravity tabling, Gravity batch centrifugal concentrator (BCC)
2015 GML	305 kg Saprolite composite	A single diamond drill KBD15-120. Average depth 64 m.	Wet scrubbing and screening, Hydrocycloning, Gravity tabling and Gravity BCC, Concentrate Leaching, Thickening and Filtration
2015 GML	110 kg Laterite composite	A single auger drill hole	Wet scrubbing and screening, Gravity tabling and Gravity BCC

All test programs on Saprolite material, have focused on rejecting mass by screening and desliming using hydrocyclones, followed by gravity concentration and cyanide leaching. The first SGS program also included whole ore leaching of the cyclone underflow. Tests on the Laterite material at GML, have focused on a gravity only treatment route to produced concentrate for leaching.

Full details of the test work completed by Gekko Systems are provided in Appendix D.

Figure 13.2

Source of samples 2014 and 2015 Testwork



Hole ID	Length	Met Sample	Type
KBSD14-01	26	2014 Sap	Sonic drill core
KBSD14-02	29.6	2014 Sap	Sonic drill core
KBSD14-03	20	2014 Sap	Sonic drill core
KBSD14-04	6.6	2014 Lat	Sonic drill core
KBSD14-03A	6	2014 Lat	Augur
KBSD14-04A	7	2014 Lat	Augur
KBSD14-05A	7	2014 Lat	Augur
KBSD14-06A	9	2014 Lat	Augur
KBSD14-07A	9	2014 Lat	Augur
KBSD14-09A	7	2014 Lat	Augur
KBSD14-10A	7	2014 Lat	Augur
KBSD14-11A	6	2014 Lat	Augur

Hole ID	From	To	Met Sample
KBD15-120	30.5	65.7	Met2015_1
KBD15-120	77.7	84.3	Met2015_1
KBD15-120	87.1	99.1	Met2015_1
KBD15-120	106	107	Met2015_1
KBD15-120	111.1	112	Met2015_1
KBD15-120	119.6	120.1	Met2015_1

Source AGG 2016

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource for the Kobada Gold Project has been estimated as at 19 November 2015. All grade estimation was completed using Multiple Indicator Kriging ('MIK') for gold. This estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralization, and the style and geometry of mineralization. The estimation was constrained with geological and mineralization interpretations.

14.2 Database Validation

The resource estimation was based on the available exploration drillhole database which was compiled in-house by AGG. The database has been reviewed and validated prior to commencing the resource estimation study.

The database includes samples from trenching, air core, RC and diamond drilling. Trenching and air core drillhole data have been excluded from the database for the purposes of modelling and grade estimation. The resultant database was validated and the checks made to the database prior to use included:

- Check for overlapping intervals.
- Downhole surveys at 0m depth.
- Consistency of depths between different data tables.
- Check gaps in the data.
- Replacing less than detection samples with half detection.
- Replacing intervals with no sample with -999.
- Replacing intervals with assays not received with -999.

14.3 Interpretation and Modelling

14.3.1 Geological interpretation

Based on grade information and geological observations, oxidation boundaries have been interpreted and wireframes modelled to constrain resource estimation for the Kobada deposit. Interpretation and digitising of all constraining boundaries has been undertaken on cross sections orientated at 110º (drill line orientation). The resultant digitised boundaries have been used to construct wireframe surfaces or solids defining the three-dimensional geometry of each interpreted feature.

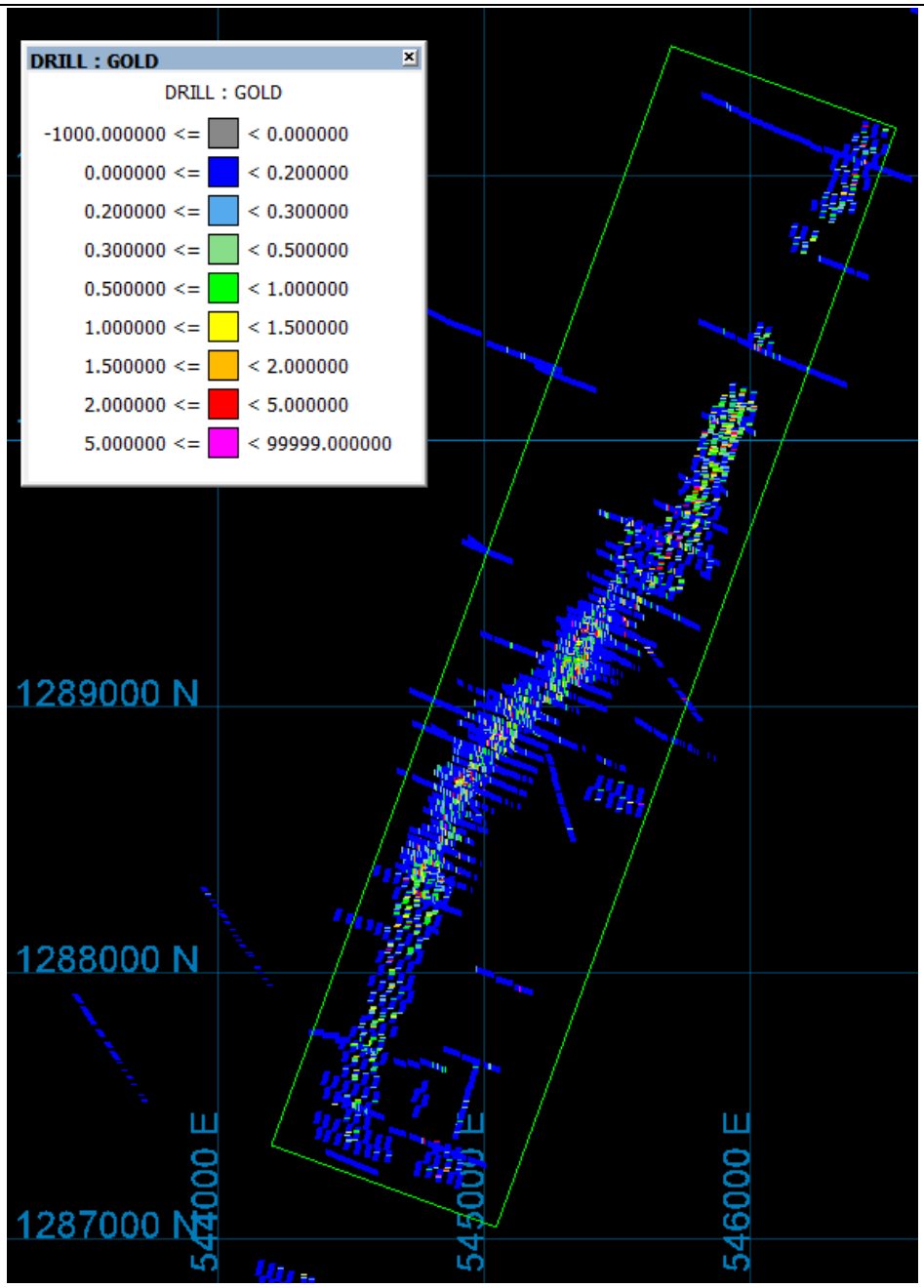
14.3.2 Mineralization Interpretation

The Kobada gold deposit is a quartz carbonated hosted mesothermal gold deposit and is entirely hosted within deformed sediments and gold mineralization is associated with quartz veins within a NE-SW trending shear zone defined over at least 4km of strike that dips con-

sistently east, with a steep (60°-80°) dip (Figure 14.2). Mineralization also occurs is the laterite hardcap that overlies the weathered sediments and this mineralization is spatially related to the bedrock mineralization and is interpreted to be derived thereof. Typically for many deposits of its type, the mineralization presents as generally somewhat discontinuous and irregularly distributed on the scale of the sectional drilling spacing (25m to 50m).

Figure 14.1

Plan View of Drilling



Source IRS November 2015

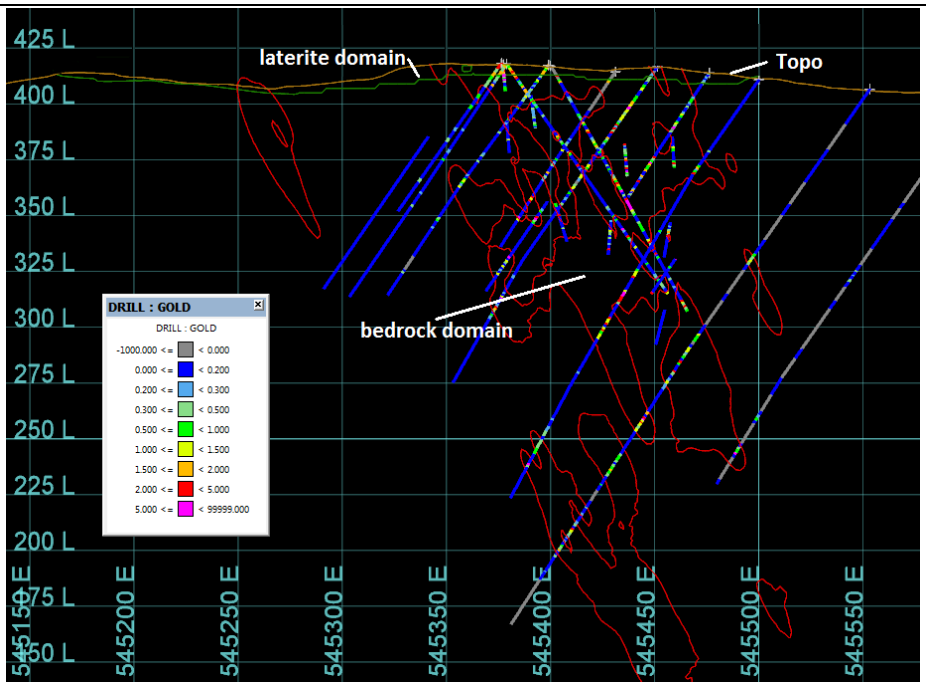
To establish appropriate grade continuity, the mineralization model was therefore based upon a nominal 0.3ppm Au indicator mineralization shell estimated using 5m unconstrained down-hole composites. This interpretation is designed to capture the broad mineralization halo that encompasses the geological vein system and is not intended to constrain individual veins or vein clusters. As the grade estimation technique is MIK with change of support technique, this type of mineralization constraint is deemed appropriate.

Of note is the drill orientation in the north and south where drilling was orientated at approximately 200° to target postulated shallow north dipping vein hosted mineralization. This orientation is effectively sub parallel to the orientation of the overall shear zone and can be considered to be sub-optimal from the perspective of determining foot-wall and hanging-wall margins to the mineralization as it does not effectively cross-cut these margins. As such, this has ramifications when considering confidence criteria related to resource classification as it becomes more difficult to accurately define volume.

The mineralization grade shell was generated by grade estimation via indicator kriging at a single cutoff, 0.3 g/t Au. Grade estimation was into a block model with cell dimensions of 2m E × 2m N × 2 m RL. Grade shell triangulations were then generated by constraining the block model at a 40% probability cut-off for both the laterite and the underlying bedrock (saprolite, transition and fresh ore types). Grade shells were subsequently reviewed in multiple orientations and in plan and section view prior to being accepted for grade estimation and block modelling purposes. Three mineralization estimation domains have been thus defined; a sub horizontal laterite domain and two steeply dipping bedrock domains (north domain and south domain separated at the flexure in the shear at 1,289,000mN (Figure 14.3).

Figure 14.2

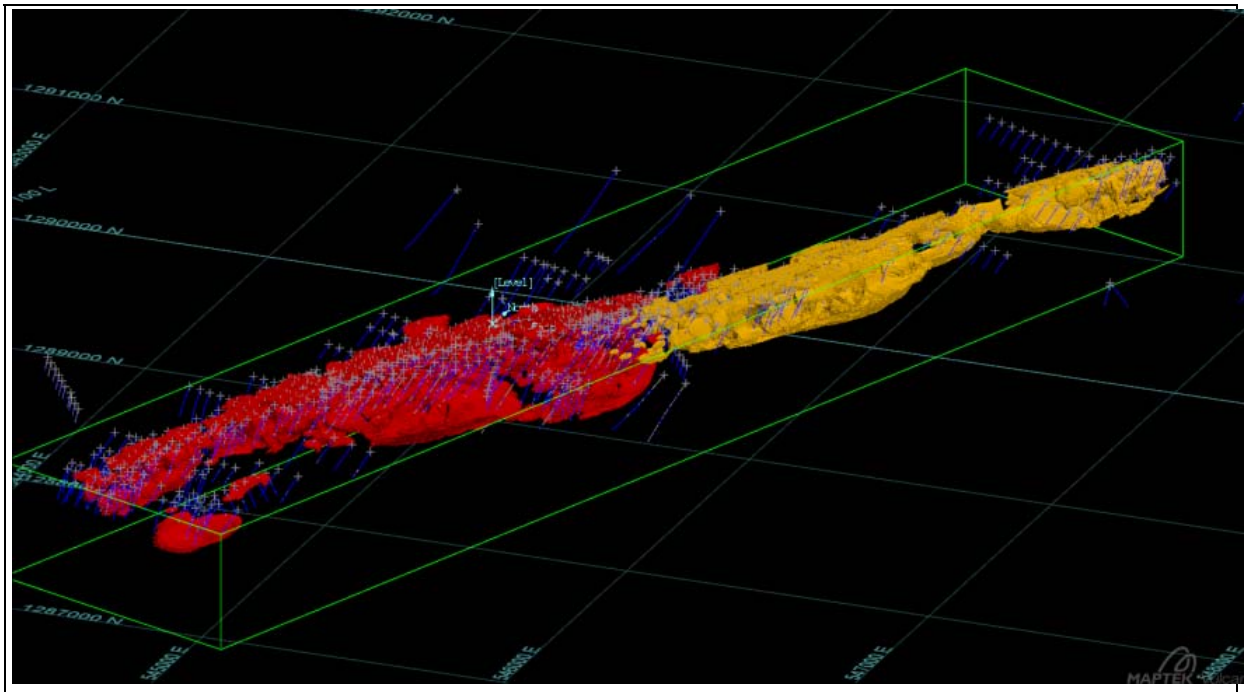
Typical Cross Section (1,800S local grid) of mineralization Domains



Source IRS – November 2015

Figure 14.3

Isometric SE view of mineralization domains



Source IRS – November 2015

14.4 Data Flagging and Compositing

Drillhole samples were flagged with the three indicator grade shells and the oxidation wireframes described in the previous section. Coding was undertaken on the basis that if the individual sample centroid fell within the grade shell boundary it was coded as within the grade shell. Each domain has been assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation. Laterite has been designated Zone 100, the northern portion of the bedrock Zone 201 and the southern portion Zone 202.

The drillhole database coded within each grade shell was then composited as a means of achieving a uniform sample support. It should be noted, however, that equalising sample length is not the only criteria for standardising sample support. Factors such as angle of intersection of the sampling to mineralization, sample type and diameters, drilling conditions, recovery, sampling/sub-sampling practices and laboratory practices all affect the 'support' of a sample. Exploration/mining databases which contain multiple sample types and/or sources of data provide challenges in generating composite data with equalised sample support, and uniform support is frequently difficult to achieve.

The lengths of the samples were statistically assessed prior to selecting an appropriate composite length for undertaking statistical analyses, variography and grade estimation. Summary statistics of the sample length indicates that approximately 87% of the samples were

collected at 1m intervals, 2% were collected at 2m intervals and 9% have been sampled at intervals between 1m and 2m.

After consideration of relevant factors relating to geological setting and mining, including likely mining selectivity and bench/flitch height, a regular 3m run length (down hole) composite was selected as the most appropriate composite interval to equalise the sample support at Kobada. Compositing was broken when the routine encountered a change in flagging (grade shell boundary) and composites with residual intervals of less than 1.2m were retained by addition to the previous composite resulting in a composite file containing composites between 1.2m and 4.2m in length.

14.5 Statistical Analysis

The composites flagged as described in the previous section were used for subsequent statistical, geostatistical and grade estimation investigations.

Summary descriptive statistics were generated for all domains (Table 14.1). A high maximum grade of 48.76 g/t Au is noted for Domain 202. The grade distributions are typical for gold deposits of this style and show a positive skew or near lognormal behaviour (Figure 13.2.4_1). The coefficient of variation (CV - calculated by dividing the standard deviation by the mean grade) is moderately high, consistent with the presence of high outlier grades that potentially require cutting (capping) for grade estimation.

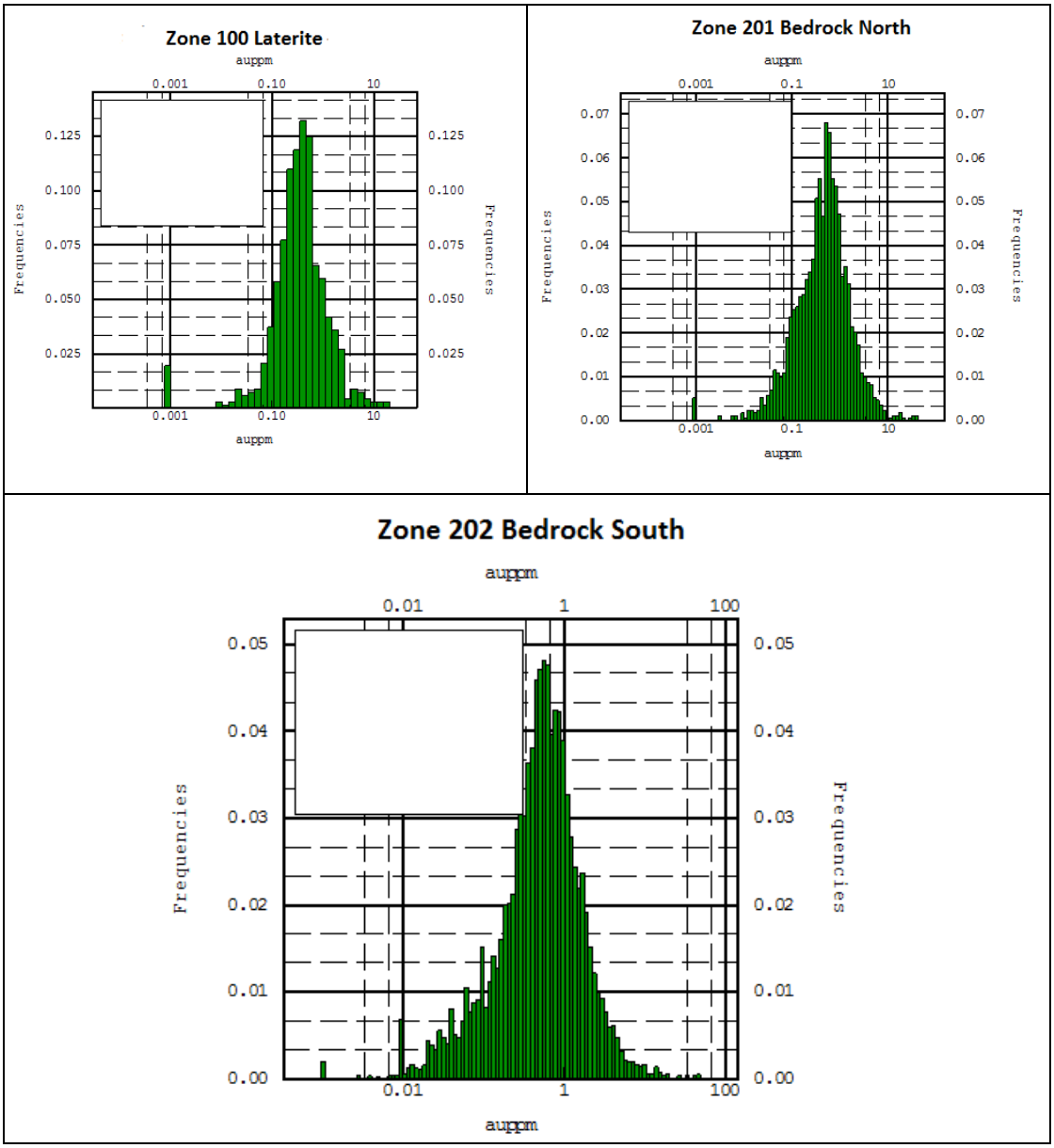
Table 14.1

Summary Statistics Grouped by Domain for 3m Composites of Uncut Gold Grade (g/t)

	Domain 100	Domain 201	Domain 202
Count	818	1,751	6,271
Minimum	0.007	0.001	0.001
Maximum	39.880	41.84	48.763
Mean	0.845	0.998	0.950
Std. Dev.	2.308	2.327	2.076
Variance	5.328	5.415	4.311
CV	2.730	2.332	2.188

Figure 14.4

Log Histograms of uncut gold grade by domain



14.5.1 High Grade Outlier Analysis

MIK is an appropriate method to estimate the gold grades for the Kobada Gold deposit as previously described. However, the grade datasets for the various estimation domains are characterised by moderately high CV values, indicating that high-grade values may contribute significantly to the mean grades reported for the various datasets.

The effects of the highest grade composites on the mean grade and standard deviation of the gold dataset for each of the estimation domains have been investigated by compiling and re-viewing statistical plots (histograms and probability plots). The resultant plots were reviewed

together with probability plots of the sample populations and an upper cut for each dataset was chosen coinciding with a pronounced inflection or increase in the variance of the data. Composite data was viewed in 3D to determine the clustering or otherwise of these highest grades observed in each domain to assess the appropriateness of the high grade cut. Clustering of the highest grades in one or more particular areas may indicate that the grades do not require to be cut and need to be dealt with in a different way. A list of the determined upper cuts applied and their impact on the mean grades of the datasets is provided in Table 14..

It should be noted that while gold grades are not cut or capped for the purposes of MIK estimation the use of cut grades is often employed for variography and the change of support process. As MIK estimates are essentially a series of OK estimates applied to the binary transformation of a series of indicator cutoffs, high grade cutting will have no effect on the resultant MIK estimate unless the high grade cut is lower than the chosen upper indicator cutoff and this scenario would be considered highly sub-optimal in the context of MIK estimation. A full description of the MIK estimation method with change of support is provided in Section14.9.

Table 14.2

Top cut Summary Statistics (g/t)

	Domain 100	**Domain 201**	**Domain 202**
Count	818	1,751	6,271
Minimum	0.007	0.001	0.001
Maximum	10.00	20.000	20.000
Mean	0.752	0.963	0.92
Std. Dev.	1.350	1.873	1.600
Variance	1.823	3.507	2.550
CV	1.796	1.945	1.735

14.5.2 Multiple Indicator Kriging Cutoffs

Indicator Kriging cutoffs or indicator bins were selected for each domain to be estimated by MIK. Cutoffs were based upon population distributions and metal proportions above and below the mean composite value of the proposed cutoff bins. Conditional statistics for data within each domain to be estimated by Multiple Indicator Kriging are listed in Table 14.2. A total of 11 cutoffs were applied to Domain 100 and 17 each to Domains 201 and 202. Top cuts have not been applied for the purposes of conditional statistics calculation.

Table 14.2

Indicator Class Statistics

Domain					
Zone 100			**Zone 201**		
Probability Threshold	**Grade Threshold (Au g/t)**	**Class Mean (Au g/t)**	**Probability Threshold**	**Grade Threshold (Au g/t)**	**Class Mean (Au g/t)**
0.107	0.1	0.064	0.122	0.1	0.049
0.222	0.18	0.140	0.244	0.2	0.149
0.336	0.25	0.213	0.337	0.3	0.249
0.446	0.33	0.290	0.429	0.4	0.351
0.550	0.42	0.371	0.510	0.5	0.451
0.646	0.5	0.454	0.588	0.6	0.546
0.744	0.65	0.566	0.646	0.7	0.645
0.838	1	0.785	0.700	0.8	0.747
0.926	1.8	1.339	0.751	0.95	0.874
0.987	7	3.191	0.795	1.1	1.018
Max	Max	15.928	0.830	1.3	1.196
-	-	-	0.860	1.5	1.388
-	-	-	0.886	1.7	1.588
-	-	-	0.911	2.05	1.887
-	-	-	0.938	2.6	2.330
-	-	-	0.966	4	3.166
-	-	-	0.986	6.5	4.865
-	-	-	Max	Max	15.398
Zone 202					
Probability Threshold	**Grade Threshold (Au g/t)**	**Class Mean (Au g/t)**			
0.116	0.1	0.048			
0.202	0.2	0.149			
0.293	0.3	0.251			
0.383	0.4	0.350			
0.473	0.5	0.449			
0.551	0.6	0.549			
0.623	0.72	0.653			
0.686	0.85	0.785			
0.732	0.95	0.899			
0.779	1.1	1.021			
0.822	1.3	1.196			
0.866	1.6	1.444			
0.901	1.9	1.729			
0.933	2.4	2.121			
0.964	3.5	2.878			
0.982	5	4.141			
0.116	9	6.622			
Max	Max	18.587			

14.6 Variography

14.6.1 Introduction

Variography is used to describe the spatial variability or correlation of an attribute (gold, silver etc.). The spatial variability is traditionally measured by means of a variogram, which is gen-

erated by determining the averaged squared difference of data points at a nominated distance (h), or lag (Srivastava and Isaacs, 1989). The averaged squared difference (variogram or γ(h)) for each lag distance is plotted on a bivariate plot, where the X-axis is the lag distance and the Y-axis represents the average squared differences (γ(h)) for the nominated lag distance.

Several types of variogram calculations are employed to determine the directions of the continuity of the mineralization:

- Traditional variograms are calculated from the raw assay values.
- Log-transformed variography involves a logarithmic transformation of the assay data.
- Gaussian variograms are based on the results after declustering and a transformation to a Normal distribution.
- Pairwise-relative variograms attempt to 'normalise' the variogram by dividing the variogram value for each pair by their squared mean value.
- Correlograms are 'standardized' by the variance calculated from the sample values that contribute to each lag.

Fan variography involves the graphical representation of spatial trends by calculating a range of variograms in a selected plane and contouring the variogram values. The result is a contour map of the grade continuity within the domain.

The variography was calculated and modelled in the geostatistical software, Isatis. The rotations are tabulated as dip and dip direction of major, semi-major and minor axes of continuity. Modelled variograms were generally shown to have good structure and were used throughout the MIK estimation and also were used for the change of support process.

14.6.2 Kobada Gold Deposit Variography

Grade and indicator variography was generated to enable grade estimation via MIK and change of support analysis to be completed. In addition, Gaussian variograms were also examined as part of the change of support process. Indicator thresholds for Zones 100, Zones 201 and Zone 202 had variograms modelled with every second variogram typically modelled. Variograms not modelled have had their parameters interpolated based on the bounding modelled variograms. Interpreted anisotropy directions correspond well with the modelled geology and overall geometry of the interpreted domains.

<u>Zone 100</u>

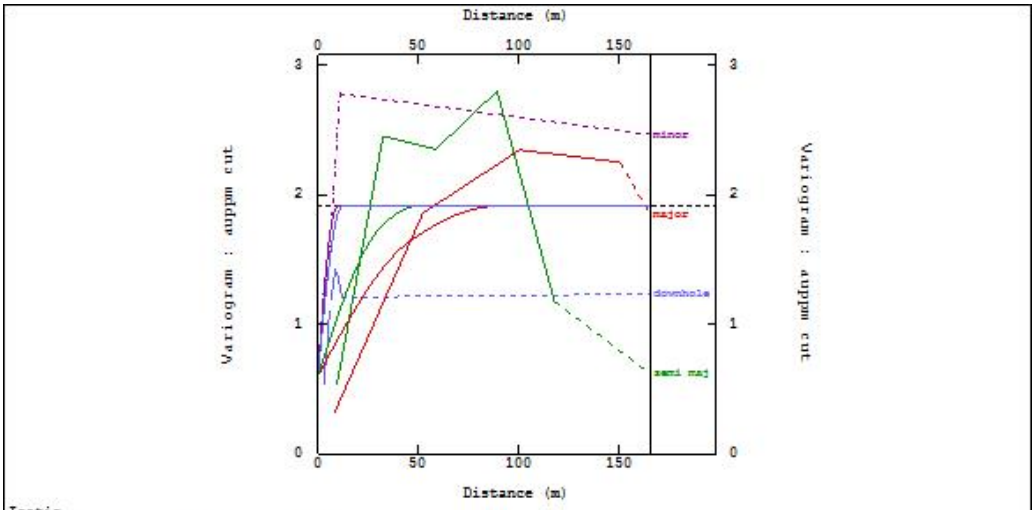
Grade variography shows good structure and displays moderate anisotropy between the major and semi-major axes. Two spherical models have been fitted to the experimental variogram, with the variogram exhibiting a moderate relative nugget effect (calculated by dividing the nugget variance by the sill variance) of 31%. The short-range structure, which has been modelled with ranges of 45m, 30m and 5m for the major, semi-major and minor axis respec-

tively, accounts for 21.5% of the non-nugget variance. The overall ranges fitted are 125m, 70m and 10m for the major, semi-major, and minor axis respectively.

The interpreted major direction of continuity dips at 0º towards 025º with the semi major direction dipping at 0º towards 115º (essentially a horizontal plane). The modelled grade variogram plot is provided in Figure 14.5.

Modelled indicator variograms display a range of relative nugget values from 30% to 50%. Table 14.3 presents the fitted grade and indicator variogram models for Zone 100.

Figure 14.5
Zone 100 – Grade Variogram




```
Isatis
composites/3m comps(zone(geocod=100))                                  Brian Wolfe
- Variable #1 : auppm cut
Variogram : in 4 direction(s)                                   Dec 03 2014   10:39:09
D1  : major                                                              Kobada
      Angular tolerance =  30.00
      Lag =  50.000m, Count = 4 lags, Tolerance = 30.00%
      Horizontal Slicing =  30.000m
      Vertical Slicing =  5.000m
D2  : semi major
      Angular tolerance =  20.00
      Lag =  30.000m, Count = 5 lags, Tolerance = 50.00%
      Vertical Slicing =  7.000m
D3  : minor
      Angular tolerance =  45.00
      Lag =  6.000m, Count = 3 lags, Tolerance = 50.00%
D4  : downhole
      Angular tolerance =  45.00
      Lag =  3.000m, Count = 5 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N25.00 (Geologist Plane)
S1  - Nugget effect, Sill =        0.6
S2  - Spherical - Range = 8.000m, Sill =       0.41
      Directional Scales = (         45.000m,         30.000m,          8.000m)
S3  - Spherical - Range = 10.000m, Sill =        0.9
      Directional Scales = (         90.000m,         50.000m,         10.000m)
```

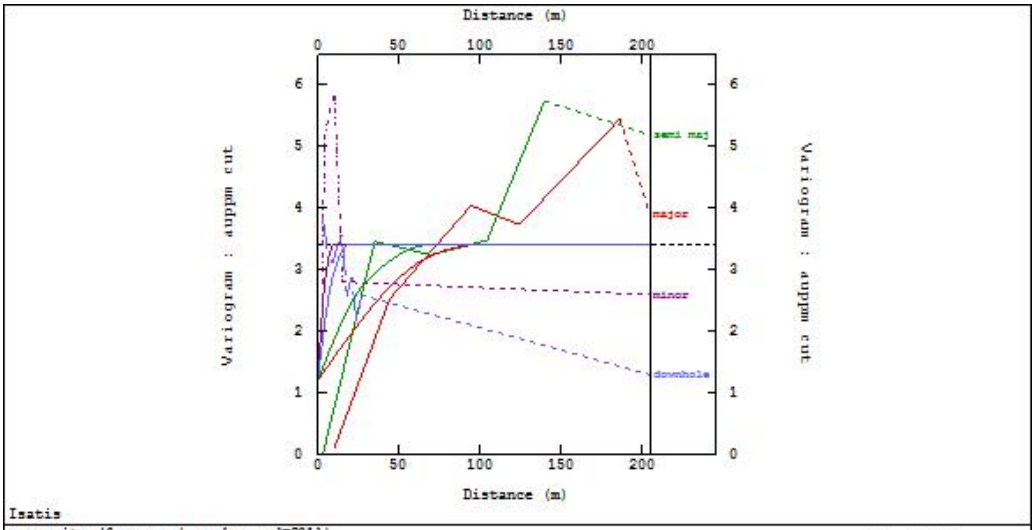
Zone 201

Grade variography again shows good structure and displays moderate to strong anisotropy between the major and semi-major axes. Two spherical models have been fitted to the experimental variogram, with the variogram exhibiting a moderate relative nugget effect (calculated by dividing the nugget variance by the sill variance) of 24%. The short-range structure, which

has been modelled with ranges of 70m, 30m and 5m for the major, semi-major and minor axis respectively, accounts for 18% of the non-nugget variance. The overall ranges fitted to the Zone 201 variogram are 135m, 60m and 10m for the major, semi-major, and minor axis respectively.

The interpreted major direction of continuity dips at 10º towards 20º with the semi major direction dipping at 30º towards 310º. Table 14.4 presents the fitted grade variogram and indicator variogram models for Zone 201 while the grade variogram plot is provided in Figure 14.6.

Figure 14.6
Zone 201 – Grade Variogram




```
composites/3m comps(zone{geocod=201})                                   Brian Wolfe
- Variable #1 : auppm cut
Variogram : in 4 direction(s)                                      Dec 03 2014   10:38:08
D1  : major                                                                 Kobada
      Angular tolerance =  20.00
      Lag =  45.000m, Count = 5 lags, Tolerance = 40.00%
      Vertical Slicing =  4.000m
D2  : semi major
      Angular tolerance =  30.00
      Lag =  35.000m, Count = 5 lags, Tolerance = 10.00%
D3  : minor
      Angular tolerance =  45.00
      Lag =  5.000m, Count = 4 lags, Tolerance = 50.00%
D4  : downhole
      Angular tolerance =  45.00
      Lag =  3.000m, Count = 10 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N25.00 Dip=60.00 Pitch=150.00 (Geologist Plane)
S1  - Nugget effect, Sill =        1.2
S2  - Spherical - Range = 5.000m, Sill =        0.7
      Directional Scales = (         70.000m,          30.000m,          5.000m)
S3  - Spherical - Range = 10.000m, Sill =        1.5
      Directional Scales = (        100.000m,          70.000m,         10.000m)
```

Zone 202

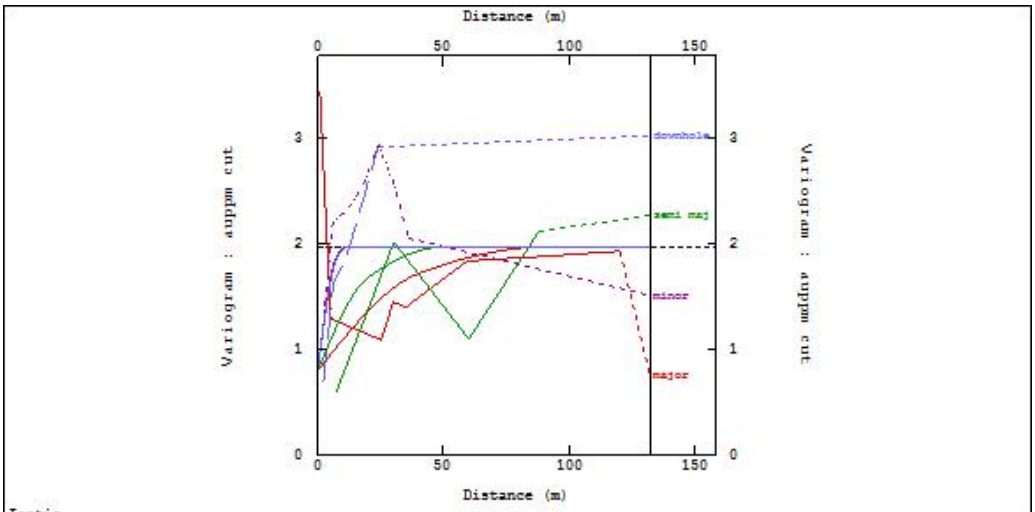
Grade variography again shows good structure and displays moderate anisotropy between the major and semi-major axes. Two spherical models have been fitted to the experimental variogram, with the variogram exhibiting a moderate relative nugget effect (calculated by dividing the nugget variance by the sill variance) of 23%. The short-range structure, which has been modelled with ranges of 60m, 55m and 5m for the major, semi-major and minor axis re-

spectively, accounts for 20% of the non-nugget variance. The overall ranges fitted to the Zone 202 variogram are 120m, 70m and 10m for the major, semi-major, and minor axis respectively.

The interpreted major direction of continuity dips at 0º towards 35º with the semi major direction dipping at 50º towards 125º. Table 14.5 presents the fitted grade variogram and indicator variogram models for Zone 202 while the grade variogram plot is provided in Figure 14.7.

Figure 14.7

Zone 202 – Grade Variogram




```
Isatis
composites/3m comps(zone{geocod=202})                                   Brian Wolfe
- Variable #1 : auppm cut
Variogram : in 4 direction(s)                                      Dec 03 2014   10:40:05
D1  : major                                                                 Kobada
      Angular tolerance =  30.00
      Lag =  30.000m, Count = 11 lags, Tolerance = 25.00%
      Horizontal Slicing =  30.000m
      Vertical Slicing =  3.000m
D2  : semi major
      Angular tolerance =   5.00
      Lag =  30.000m, Count = 4 lags, Tolerance = 30.00%
      Horizontal Slicing =  5.000m
      Vertical Slicing =  3.000m
D3  : minor
      Angular tolerance =  35.00
      Lag =  6.000m, Count = 7 lags, Tolerance = 50.00%
D4  : downhole
      Angular tolerance =  45.00
      Lag =  6.000m, Count = 5 lags, Tolerance = 40.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N35.00 Dip=50.00 (Geologist Plane)
S1  - Nugget effect, Sill =        0.8
S2  - Spherical - Range = 7.000m, Sill =        0.5
      Directional Scales = (         40.000m,          20.000m,          7.000m)
S3  - Spherical - Range = 11.000m, Sill =        0.67
      Directional Scales = (         90.000m,          50.000m,         11.000m)
```

Table 14.3

Zone 100 Variogram Models

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip/dip dir)			Structure 1				Structure 2			
					Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
		Major	Semi Major	Minor		Major	Semi Major	Minor		Major	Semi Major	Minor
Grade Variography												
Gold (Au g/t)	0.6	0/25	0/115	-90/0	0.41	45	30	5	0.9	125	70	10
Indicator Variography												
0.10[1]	0.0288	0/25	0/115	-90/0	0.0336	35	30	6	0.0336	160	60	10
0.18[1]	0.0519	0/25	0/115	-90/0	0.0606	35	30	6	0.0606	155	60	10
0.25	0.0650	0/25	0/115	-90/0	0.0650	30	25	5	0.0930	150	55	8
0.33[2]	0.0736	0/25	0/115	-90/0	0.0711	27.5	20	4.5	0.1032	125	47.5	7
0.42	0.0750	0/25	0/115	-90/0	0.0700	25	15	4	0.1030	100	40	6
0.50 [3]	0.0768	0/25	0/115	-90/0	0.0625	22.5	12.5	3.5	0.0897	80	32.5	5.5
0.65	0.0700	0/25	0/115	-90/0	0.0500	20	10	3	0.0700	60	25	5
1.00[4]	0.0556	0/25	0/115	-90/0	0.0327	17.5	8.5	3	0.0477	50	22.5	4.5
1.80	0.0310	0/25	0/115	-90/0	0.0150	15	7	3	0.0230	40	20	4
7.00[5]	0.0065	0/25	0/115	-90/0	0.0025	12	6	3	0.0040	35	15	4

Note: 1) Assumed model based on 0.25 Au g/t variogram model
2) Assumed model based on 0.25 Au g/t and 0.42 Au g/t variogram models
3) Assumed model based on 0.42 Au g/t and 0.65 Au g/t variogram models
4) Assumed model based on 0.65 Au g/t and 1.80 Au g/t variogram model
5) Assumed model based on 1.8 Au g/t variogram model

Table 14.4

Zone 201 Variogram Models

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip/dip dir)			Structure 1				Structure 2			
					Sill 1 (C1)	Range (m)			Sill 2 (C2)	Range (m)		
		Major	Semi Major	Minor		Major	Semi Major	Minor		Major	Semi Major	Minor
Grade Variography												
Gold (Au g/t)	0.55	0/70	-30/160	-60/340	0.4	70	30	5	1.3	135	60	10
Indicator Variography (Correlograms)												
0.10[1]	0.37	0/70	-30/160	-60/340	0.0535	40	30	15	0.0321	200	100	25
0.20[1]	0.37	0/70	-30/160	-60/340	0.0920	37	25	12	0.0552	190	95	23
0.30	0.37	0/70	-30/160	-60/340	0.1100	35	20	10	0.0640	180	90	20
0.40[2]	0.37	0/70	-30/160	-60/340	0.1131	32.5	17.5	9	0.0654	165	85	19
0.50	0.39	0/70	-30/160	-60/340	0.1080	30	15	8	0.0620	150	80	18
0.60 [3]	0.39	0/70	-30/160	-60/340	0.1027	27.5	14	7.5	0.0597	130	65	16.5
0.70	0.41	0/70	-30/160	-60/340	0.0950	25	13	7	0.0560	110	50	15
0.80[4]	0.41	0/70	-30/160	-60/340	0.0858	25	14	6.5	0.0530	100	50	13.5
0.95	0.44	0/70	-30/160	-60/340	0.0740	25	15	6	0.0480	90	50	12
1.10[5]	0.44	0/70	-30/160	-60/340	0.0650	25	12.5	6	0.0408	85	50	12
1.30	0.44	0/70	-30/160	-60/340	0.0560	25	10	6	0.0340	80	50	12
1.50[6]	0.49	0/70	-30/160	-60/340	0.0460	25	10	5.5	0.0283	75	47.5	11
1.70	0.49	0/70	-30/160	-60/340	0.0370	25	10	5	0.0230	70	45	10
2.05[7]	0.49	0/70	-30/160	-60/340	0.0260	25	10	5	0.0201	67.5	42.5	9
2.6	0.55	0/70	-30/160	-60/340	0.0047	25	10	5	0.0046	65	40	8
4[8]	0.55	0/70	-30/160	-60/340	0.0018	30	30	5	0.0017	60	40	10
6.5[8]	0.55	0/70	-30/160	-60/340	0.0006	30	30	5	0.0006	60	40	10

Note:
1) Assumed model based on 0.30 Au g/t variogram model
2) Assumed model based on 0.30 Au g/t and 0.50 Au g/t variogram models
3) Assumed model based on 0.50 Au g/t and 0.70 Au g/t variogram models
4) Assumed model based on 0.70 Au g/t and 0.95 Au g/t variogram model
5) Assumed model based on 0.95 Au g/t and 1.30 Au g/t variogram model
6) Assumed model based on 1.30 Au g/t and 1.70 Au g/t variogram models
7) Assumed model based on 1.70 Au g/t and 2.60 Au g/t variogram models
8) Assumed model based on 2.60 Au g/t variogram model

Table 14.5
Zone 202 Variogram Models

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip/dip dir)			Structure 1				Structure 2			
					Sill 1 (C1)	Range (m)			Sill 2 (C2)	Range (m)		
		Major	Semi Major	Minor		Major	Semi Major	Minor		Major	Semi Major	Minor
Grade Variography												
Gold (Au g/t)	0.7	0/35	-50/125	-40/205	0.6	60	55	5	1.78	120	70	10
Indicator Variography (Correlograms)												
0.10[1]	0.0170	0/35	-50/125	-40/205	0.0445	30	20	6	0.0445	60	42	10
0.20[1]	0.0297	0/35	-50/125	-40/205	0.0677	27	17	5	0.0677	55	37	9
0.30	0.0420	0/35	-50/125	-40/205	0.0850	25	15	4	0.0850	50	35	8
0.40[2]	0.0545	0/35	-50/125	-40/205	0.0929	22.5	12.5	3.5	0.0906	50	28.5	7.5
0.50	0.0650	0/35	-50/125	-40/205	0.0950	20	10	3	0.0900	50	22	7
0.60[3]	0.0664	0/35	-50/125	-40/205	0.0934	20	10	3	0.0862	50	21	6.5
0.72	0.0650	0/35	-50/125	-40/205	0.0880	20	10	3	0.0790	50	20	6
0.85[4]	0.0622	0/35	-50/125	-40/205	0.0781	20	10	3	0.0697	47.5	20	6
0.95	0.0600	0/35	-50/125	-40/205	0.0700	20	10	3	0.0620	45	20	6
1.10[5]	0.0573	0/35	-50/125	-40/205	0.0599	20	9	3	0.0528	45	20	6
1.30	0.0520	0/35	-50/125	-40/205	0.0490	20	8	3	0.0430	45	20	6
1.60[6]	0.0448	0/35	-50/125	-40/205	0.0378	20	8	3	0.0334	45	20	6
1.90	0.0370	0/35	-50/125	-40/205	0.0280	20	8	3	0.0250	45	20	6
2.40[7]	0.0277	0/35	-50/125	-40/205	0.0201	19	8	3	0.0172	42.5	18.5	6
3.50	0.0150	0/35	-50/125	-40/205	0.0105	18	8	3	0.0085	40	17	6
5.00[8]	0.0083	0/35	-50/125	-40/205	0.0050	17	8	3	0.0047	38	16	5
9.00[8]	0.0040	0/35	-50/125	-40/205	0.0020	16	7	3	0.0020	35	15	4

Note:
1) Assumed model based on 0.30 Au g/t variogram model
2) Assumed model based on 0.30 Au g/t and 0.50 Au g/t variogram models
3) Assumed model based on 0.50 Au g/t and 0.72 Au g/t variogram models
4) Assumed model based on 0.72 Au g/t and 0.95 Au g/t variogram model
5) Assumed model based on 0.95 Au g/t and 1.30 Au g/t variogram model
6) Assumed model based on 1.30 Au g/t and 1.90 Au g/t variogram models
7) Assumed model based on 1.90 Au g/t and 3.50 Au g/t variogram models
8) Assumed model based on 3.50 Au g/t variogram model

14.7 Block Modelling

A 3-D block model was created in the National grid (UTM WGS84 Zone 29N) using Vulcan mining software. The parent block size was selected on the basis of the average drill spacing with a parent cell size of 25m E by 25mN by 10m RL which was sub-blocked down to 5m E by 5m N by 2.5m RL (to ensure adequate volume representation). The model covered all the interpreted mineralization zones and included suitable additional waste material to allow later pit optimisation studies. Block coding was completed on the basis of the block centroid, wherein a centroid falling within any wireframe was coded with the wireframe solid attribute. A clockwise rotation of 20° was applied to the block model.

The main block model parameters are summarised below in Table 14.6. Variables were coded into the block model to enable multiple indicator kriging estimation and subsequent change of support and grade tonnage reporting. A visual review of the wireframe solids and the block model indicated correct flagging of the block model. Additionally a check was made of coded volume versus wireframe volume which confirmed the above.

Table 14.6
Block Model Parameters

	Northing (Y)	Easting (X)	RL (Z)
Min. Coordinates	1,287,350	544,200	50
Max Coordinates	1,291,750	545,100	430
Block size (m)	25	25	10
Sub Block size (m)	5	5	2.5
Rotation (° around axis)	0°	0°	20°

14.8 Bulk Density Data

A bulk density database has been supplied containing a total of 1,907 data. The database can be subdivided on the basis of flagging by the various lithological and oxidation modelling wireframes. There have been a total of 1,795 samples taken in fresh rock, 24 in transition material, 24 taken from the oxide and 7 taken from the laterite material. Additionally, most samples have been taken between coordinate 1,288,300mN and 1,289,350mN and it can be clearly seen that the southern and northern parts of the deposit have had relatively few samples taken for density determination. All of the samples taken from the laterite and oxide were obtained from a metallurgical drilling program in 2015. Although the relatively few samples taken from oxide and lateritic material represent an improvement from previously, the bulk density database still requires additional samples to provide a better geographical spread of data for the laterite and oxide material.

Bulk densities have been estimated into material designated as fresh rock only in the model via inverse distance squared estimation method. Other material designation categories were deemed to not have enough data to warrant assignment via estimation. One estimation pass only was used and the estimation parameters are tabulated in Table 14.7 below. Approxi-

mately 8% of all material classified as fresh material have been estimated in this manner. For all other blocks not estimated a default value of 2.65t/m^3 was applied.

Table 14.7
Bulk Density Sample Search Criteria

Domain	Sample Search Orientation (dip/dip direction°)			Sample Search Distance (m)			Numbers of Samples		
	Major	Semi-Major	Minor	Major	Semi-Major	Minor	Min.	Max.	Maximum Per Drillhole
Fresh	0/35	65/305	25/125	50	50	25	4	8	-

For the other categories, the average bulk densities as per the subdivisions outlined above have been applied to the model as presented in Table 14.8 below.

Table 14.8
Bulk Density Values

Lithology	Assigned Bulk Density (t/m³)
Laterite	2.02
Strongly Oxidised	1.85
Transition	2.10
Fresh	2.65

14.9 Grade Estimation

14.9.1 Introduction

Multiple Indicator Kriging (MIK) was applied to grade estimation at Kobada within the defined indicator mineralization shell. Estimation was completed in the mining package Vulcan using the GSLib geostatistical software. MIK is considered a robust estimation methodology for grade estimates for gold deposits such as Kobada when adequate consideration is given to restricting the influence of high grade data. MIK grade estimation with change of support has been applied to produce 'recoverable' gold estimates targeting a selective mining unit (SMU) of 6.25mE x 12.5mN x 5mRL.

14.9.2 The Multiple Indicator Kriging Method

The MIK technique is implemented by completing a series of Ordinary Kriging ("OK") estimates of binary transformed data. A composite sample, which is equal to or above a nominated cutoff or threshold, is assigned a value of 1, with those below the nominated indicator threshold being assigned a value of 0. The indicator estimates, with a range between 0 and 1, represent the probability the point will exceed the indicator cutoff grade. The probability of the points exceeding a cutoff can also be considered broadly equivalent to the proportion of a nominated block that will exceed the nominated cutoff grade.

The estimation of a complete series of indicator cutoffs allows the reconstitution of the local histogram or conditional cumulative distribution function (ccdf) for the estimated point. Based on the ccdf, local or block properties, such as the block mean and proportion (tonnes) above or below a nominated cutoff grade can be investigated.

Post MIK Processing - E-Type Estimates

The E-type estimate provides an estimate for the grade of the total block or bulk-mining scenario. This is achieved by discretising the calculated ccdf for each block into a nominated number of intervals and interpolating between the given points with a selected function (e.g. the linear, power or hyperbolic model) or by applying intra-class mean grades. The sum of all these weighted interpolated points or mean grades enables an average whole block grade to be determined.

The following example shows the determination of an E-type estimate for a block containing three indicator cutoffs.

The indicator cutoffs and associated probabilities calculated are shown in Table 14.9.

Table 14.9
Indicator cut off and probability

Indicator	Cutoff Grade Aug/t	Indicator Probability (cumulative)
minimum grade *	0	0.00 **
indicator 1	1	0.40
indicator 2	2	0.65
indicator 3	3	0.85
maximum grade *	4	1.00 **

Note: * Cutoff grades determined by the user.
** Indicator probability is assumed at the minimum and maximum cutoff.

The whole block grade can now be determined in this block with the following parameters used for the purposes of the interpolation:

- Number of discretisation intervals: 4.
- Linear extrapolation between all points (median grade between nominated cutoffs).

The worked example is then calculated with the following steps:

- Interval 1 (0-1g/t Au) median grade x probability/proportion attributed to the interval (0.5g/t Au x 0.40 = 0.200).
- Interval 2 (1 - 2g/t Au) median grade x proportion (1.5g/t Au x 0.25 = 0.375).
- Interval 3 (2 - 3g/t Au) median grade x proportion (2.5g/t Au x 0.20 = 0.500).
- Interval 4 (3 - 4g/t Au) median grade x proportion (3.5g/t Au x 0.15 = 0.525).
- Calculate total grade average all calculated intervals ((0.2+0.375+0.500+0.525)/1) = 1.60g/t Au.

It is also possible from this example to calculate the proportion and grade above a nominated cutoff (e.g. 2g/t - at sample support or complete selectivity). The following steps would be undertaken to calculate the tonnes and grade at sample selectivity using a 2g/t cutoff:

- Interval 3 (2 - 3g/t Au) median grade x proportion (2.5g/t Au x 0.20 = 0.500).
- Interval 4 (3 - 4g/t Au) median grade x proportion (3.5g/t Au x 0.15 = 0.525).
- Calculate total grade average all calculated intervals ((0.500+0.525)/0.35) = 2.93g/t Au with 0.35% of the block above the cutoff.

The effect of using a non-linear model to interpolate between cutoffs is to shift the grade weighting associated with that cutoff away from the median. For Kobada, the intra-class means based on the cut composite data have been used to reconstitute the ccdf and produce block statistics.

It is noted, however, that the calculation of the E-type estimate and complete selectivity often does not allow mine planning to the level of selectivity which is proposed for production. To achieve an estimate which reflects the levels of mining selectivity envisaged, a selective mining unit ("SMU") correction is often applied to the calculated ccdf.

Support Correction (Selective Mining Unit Estimation)

A range of techniques are known to produce a support correction and therefore allow for selective mining unit emulation. The common features of the support correction are:

- Maintenance of the mean grade of the histogram (E-type mean).
- Adjustment of the histogram variance by a variance adjustment factor (the 'f' factor).

The variance adjustment factor, used to reduce the histogram or ccdf variance, can be calculated using the variogram model. The variance adjustment factor is often modified to account for the likely grade control approach or 'information effect'.

In simplest terms, the variance adjustment factor takes into account the known relationship derived from the dispersion variance.

Total variance = variance of samples within blocks + variance between blocks.

The variance adjustment factor is calculated as the ratio of the variance between the blocks and the variance of the samples within the blocks, with a small ratio (e.g. 0.10) indicating a large adjustment of the ccdf variance and large ratio (e.g. 0.80) representing a small shift in the ccdf.

Two simple support corrections that are available include the Affine and Indirect Lognormal correction, which are both based on the permanence of distribution. The discrete Gaussian model is often applied to global change of support studies and has been generated on the composite dataset as a comparison. The indirect lognormal correction was applied to the Kobada MIK grade estimates.

Indirect Lognormal Correction

The indirect lognormal correction can be implemented by adjusting the quantiles (indicator cutoffs) of the ccdf with the variance adjustment factor so that the adjusted ccdf represents the statistical characteristics of the block volume of interest.

This is implemented with the following formula:

$$q' = a \times q^b$$

q = quantile of distribution.

q' = quantile of the variance-reduced distribution.

where the coefficients a and b, are given by the following formula:

$$a = \sqrt{\frac{m}{f \cdot CV^2 + 1}} \left[\sqrt{\frac{CV^2 + 1}{M}} \right]$$

$$b = \sqrt{\frac{\ln (f \cdot CV^2 + 1)}{\ln (CV^2 + 1)}}$$

m = mean of distribution.
f = variance adjustment factor .
CV = coefficient of variation.

At the completion of the quantile adjustments, grades and tonnages (probabilities are then considered a pseudo tonnage proportion of the blocks) at a nominated cutoff grade can be calculated using the methodology described above (E-type). The indirect lognormal correction, as applied to the Julie Gold Project, is the best suited of the common adjustments applied to MIK to produce selective mining estimates for positively skewed distributions.

14.9.3 Multiple Indicator Kriging Parameters

MIK estimates were completed using the indicator variogram models (Section 14.5.2), and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests performed in Isatis geostatistical software. A total of 12 indicator thresholds were estimated for Zones 100 and 17 for Zones 201 and 202 (see Tables 13.6.2_1 to 13.6.2_3).

The sample search parameters are provided in Table 14.10. Hard domain boundaries were used for the estimation throughout. A two-pass estimation strategy was applied to each domain, applying a progressively expanded and less restrictive sample search to the successive estimation pass, and only considering blocks not previously assigned an estimate. Parent cell estimations (25m E by 25mN by 10m RL) were applied throughout and discretisation was applied on the basis of 3X by 3Y by 2RL for 18 discretisation points per block.

Table 14.10 MIK Sample Search Criteria										
Domain	Pass	Sample Search Orientation (dip/dip direction˚)			Sample Search Distance (m)			Numbers of Samples		
		Major	Semi-Major	Minor	Major	Semi-Major	Minor	Min.	Max.	Maximum Per Drillhole
100	Pass 1	0/25	0/115	-90/0	100	100	50	24	32	10
	Pass 2	0/25	0/115	-90/0	1000	1000	100	18	32	-
201	Pass 1	-9/195	-59/90	30/110	100	100	30	24	32	10
	Pass 2	-9/195	-59/90	30/110	400	250	80	18	32	-
202	Pass 1	0/35	50/305	40/125	100	100	30	24	32	10
	Pass 2	0/35	50/305	40/125	400	250	80	18	32	-

14.9.4 Estimate Validation

All relevant statistical information was recorded to enable validation and review of the MIK estimates. The recorded information included:

- Number of samples used per block estimate.
- Average distance to samples per block estimate.
- Estimation flag to determine in which estimation pass a block was estimated.
- Number of drillholes from which composite data were used to complete the block estimate.

The estimates were reviewed visually and statistically prior to being accepted. The review included the following activities:

- Comparison of the E-type estimate versus the mean of the composite dataset, including weighting where appropriate to account for data clustering.
- Comparison of the reconstituted cumulative conditional distribution functions of the estimated blocks versus the input composite data.
- Visual checks of cross sections, long sections, and plans.

Alternative estimates were also completed to test the sensitivity of the reported model to the selected MIK interpolation parameters. An insignificant amount of variation in overall grade was noted in the alternate estimations.

14.9.5 Change of Support

Applying the modelled variography, variance adjustment factors were calculated for to emulate a 6.25mE x 12.5mN x 5mRL selective mining unit ("SMU") via the indirect lognormal change of support. The intra-class composite mean grades were used in calculating the whole block and SMU grades. The change of support study also included the calculation of the theoretical global change of support via the discrete Gaussian change of support model.

An 'information effect' factor is commonly applied to the originally derived panel-to-block variance ratios to determine the final variance adjustment ratio. The goal of incorporating infor-

mation effect is to calculate results taking into account that mining takes place based on grade control information. There will still be a quantifiable error associated with this data and it is this error we want to incorporate. This is achieved in practice by running a test kriging estimation of an SMU using grade control data (the results required to incorporate this option in the change of support do not depend on the assay data so the grade control data can be hypothetical). The incorporation of the information effect is commonly found to be negligible, however can have a significant effect in some cases. In this case, the information effect factor was found to have a minor effect and has been incorporated in the calculation.

The variance adjustment ratios are provided in Table 14.11.

Table 14.11
Variance Adjustment Ratios (6.25mE x 12.5mN x 5mRL SMU)

Zone	100	201	202
Variance adjustment factor (f)	0.3	0.3	0.3

14.10 Depletion for mining activity

Depletion to account for artisanal mining, mainly within the laterite horizon, has been applied to the model. Artisanal mining of varying intensity has been identified in the area of Kobada. Artisanal open pits exist and depletion within these areas is assumed to be 100% to the base of the laterite. Outside the areas of open pitting, artisanal workings consist of shafts into the hardcap where the gold bearing material within and at the base of the hardcap has been removed. In these areas where there is an intense concentration of shafts, depletion has been assumed to be 50% to the base of the laterite. In other areas with less intense workings depletion is assumed to be 25% to the base of the laterite. Depletion has been applied by block model calculations to adjust the density by the appropriate amount.

14.11 Resource Classification

The resource categorisation has been based on the robustness of the various data sources available, including:

- Geological knowledge and interpretation;
- Variogram models and the ranges of the first structure in multi-structure models;
- Drilling density and orientation; and
- Estimation statistics.

The resource estimate for the Kobada deposit has been classified as Measured, Indicated and Inferred Resources based on the confidence levels of the key criteria as presented in Table 14.12. Applying these confidence levels, resource classification codes were assigned to the block model using the following criteria:

- Measured Resources
    - Blocks are predominately estimation pass 1

- Distance to nearest data of 25m or less (drill section spacing 25m)
- Minimum of 24 data used (from at least 3 drillholes)
- Only in areas of optimal drilling orientation (orientation approx. 290°)

▪ Indicated Resources

- Blocks are predominately estimation pass 1
- Distance to nearest data of 25m or less
- Minimum of 18 data used (from at least 3 drillholes)
- Drillhole section spacing predominantly 25m.
- Only in areas of optimal drilling orientation (orientation approx. 290°)

▪ Inferred Resources

- Blocks are predominantly estimation pass 2.
- In areas of sub-optimal drilling orientation (orientation approx. 200°)

Table 14.12
Confidence Levels by Key Criteria

Items	Discussion	Confidence
Drilling Techniques	Diamond/RC - Industry Standard approach.	Moderate/High
Logging	Standard nomenclature has been adopted but not used in entire database.	Moderate
Drill Sample Recovery	Recoveries are not recorded in entire database. Review of current drilling suggests RC recoveries are of acceptable standard.	Moderate/High
Sub-sampling Techniques and Sample Preparation	Diamond and RC sampling conducted by industry standard techniques.	Moderate/High
Quality of Assay Data	Appropriate quality control procedures are available. They were reviewed on site and considered to be of industry standard.	Moderate/High
Verification of Sampling and Assaying	Sampling and assaying procedures have been assessed and are considered of appropriate industry standards.	Moderate
Location of Sampling Points	Survey of all collars conducted with accurate survey equipment. Investigation of downhole survey indicates appropriate behaviours.	Moderate/High
Data Density and Distribution	Majority of regions defined on a notional 25mE x 25mN drill spacing.	Moderate
Audits or Reviews	Data collection assessed during site review.	N/A
Database Integrity	Assay certificates have been verified and no issues were identified.	Moderate
Geological Interpretation	Mineralization controls are well understood. The mineralization constraints are robust but relatively broad and therefore of moderate confidence.	Moderate
Estimation and Modelling Techniques	Multiple Indicator Kriging is considered to be appropriate given the geological setting and grade distribution.	High
Cutoff Grades	MIK is independent of cutoff grade although the mineralization constraints were based on a notional 0.3g/t Au lower cutoff grade. A 0.3g/t lower cutoff grade is considered appropriate for reporting.	Moderate/High
Mining Factors or Assumptions	A 6.25mE x 12.5mN x 5mRL SMU emulated for gold assuming a bulk open pit mining scenario. Change of support for Inferred component has higher degree of uncertainty.	Moderate
Metallurgical Factors or Assumptions	Not applied or available.	N/A

Tonnage Factors (In-situ Bulk Densities)	Localised data collected diamond core in transition and fresh only. Density database is deficient and requires remediation	Low

14.12 Resource Reporting

The summary total resource for the Kobada Gold Project is provided in Table 14.13 below. The preferred lower cutoff grade for reporting is 0.3g/t Au. In view of the nature and style of the mineralization and potential mining approach and method, this is considered an appropriate cut-off grade. It should be noted that mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.13

Mineral Resource Report
Summary Grade Tonnage Report, Multiple Indicator Kriged Estimate
Parent Cell Dimensions of 25mN by 25mE by 10mRL
SMU correction using 6.25mE x 12.5mN x 5mRL

	Lower Cutoff Grade (g/t Au)	Tonnes (Mt)	Average Grade (g/t Au)	Gold Metal (kozs)
Measured	**0.3**	**11.00**	**1.1**	**380**
	0.5	8.97	1.2	354
	0.7	6.70	1.4	310
	0.9	4.91	1.7	265
	1	4.15	1.8	242
Indicated	**0.3**	**24.38**	**1.1**	**835**
	0.5	19.49	1.2	771
	0.7	14.15	1.5	668
	0.9	9.75	1.8	556
	1	8.13	1.9	507
Inferred	**0.3**	**32.82**	**1.0**	**1,024**
	0.5	25.15	1.1	924
	0.7	16.90	1.4	767
	0.9	11.03	1.7	618
	1	9.02	1.9	556

Note: Appropriate rounding has been applied.

15 MINERAL RESERVE ESTIMATE

The open-pit Mineral Reserve estimate is based upon the 2015 resource model prepared by IRS and is reported as at 30 December 2015.

The total open pit Mineral Reserve estimate is tabulated in Table 15.1.

Table 15.1
Mineral Reserve Estimate – February 2016

Category	Tonnes (Mt)	Average Grade (g/t Au)	Gold Metal (kozs)
Measured	5.7	1.22	225
Indicated	7.0	1.27	286
Total Proved and Probable	**12.7**	**1.25**	**511**

Note: Appropriate rounding has been applied.

The total Mineral Reserve at Sukari was estimated at 12.7 Mt of ore at an average grade of 1.25 g/t Au for 511 koz of contained gold.

The Mineral Reserve is based on a gold price of US$1,200/oz and open-pit gold cut-off grades of 0.51 g/t Au for oxide ore types.

This Mineral Reserve estimate is classified and reported in accordance to National Instrument43-101 (NI 43-101) Standards of Disclosure for Mineral Projects, Form 43-101F1 Technical Report, and Companion Policy 43-101CP which came into force on 30 June 2011.

Furthermore, the reserve classifications are also consistent with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves of 2012 (the JORC Code) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia JORC (2012), with the minor exception that the Code refers to Ore Reserves while NI 43-101 refers to Mineral Reserves.

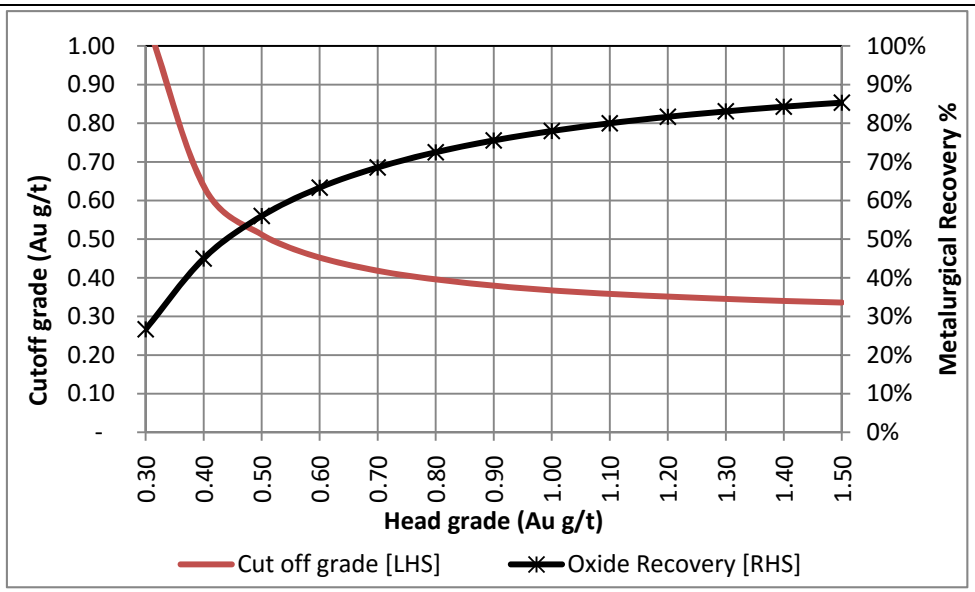
The Qualified Person responsible for the open-pit Mineral Reserves estimate is John Dunlop, a full-time employee of John Dunlop and Associates. Mr. Dunlop has the appropriate relevant qualifications and experience to be considered a Qualified Person as defined in NI 43-101.

15.1 Economic Cut Off Grade

The calculation of the economic cutoff grade is complicated by the metallurgical recovery being a function of tailings grade. As discussed in Section 13, a testwork has indicated that a constant tailings grade of 0.23 g/t Au is expected. Consequently, the economic cutoff grade is directly dependent on the head grade of ore processed.

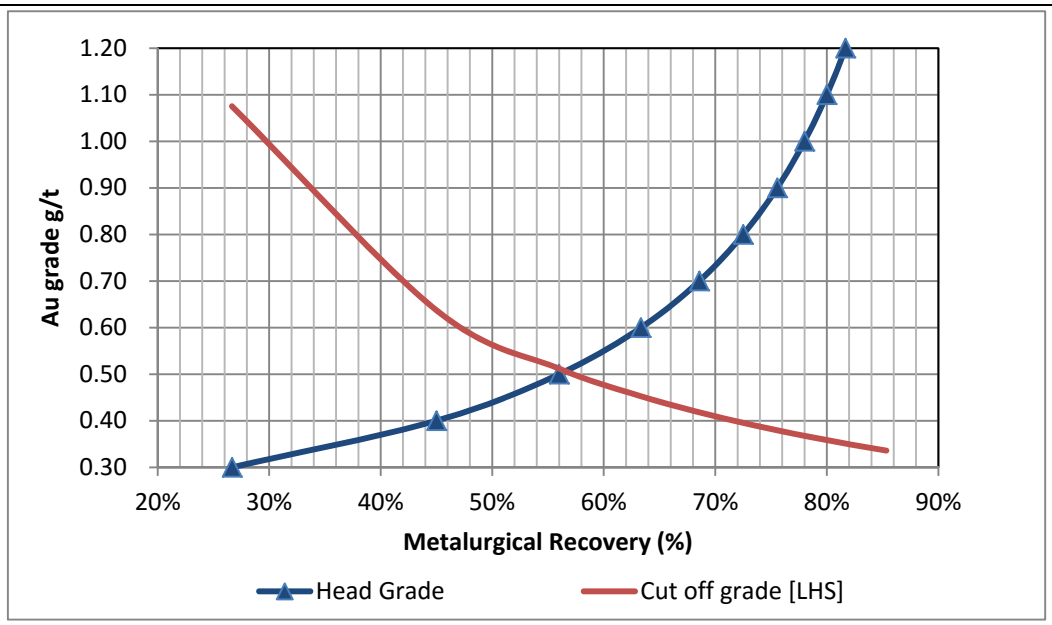
Figure 15.1 and Figure 15.2 illustrates the effect of head grade has on metallurgical recovery and cut-off grade, while Table 15.2 shows the calculation of the cut-off grade at the average grade of the Mineral Reserve (1.25 g/t Au).

Figure 15.1
Head grade, Recovery and Cut-off grade



In Figure 15.1, a head grade of 1.1 g/t (x-axis) will achieve a recovery of 80% (right y-axis). The corresponding economic cut-off grade is 0.36 g/t (left y-axis). The same data is shown in Figure 15.2, although cut off and head grade are charted on the same axis...

Figure 15.2
Head grade, Recovery and Cut-off grade



The important concept in this is that as the head grade decreases, so does the recovery, and the corresponding cut-off grade increases. The intersection of the head and cut-off grade curves in Figure 15.2 indicates the minimum economic cut-off grade. That is the point when the head grade equals the cut-off grade.

The minimum economic cut-off grade for the Project is 0.51g/t Au.

Table 15.2

Calculation of cut-off grade with average reserve grade and 0.23g/t tail grade

Item	Unit	Quantity
Mining Parameters		
Mining dilution	%	2%
Mining recovery	%	100%
Base Mining Cost	**$/t**	**2.24**
Ore Costs		
Processing cost	$/t ore	6.55
G&A cost	$/t ore	3.54
Additional Ore mining costs	$/t ore	0.43
Total Oxide Ore Cost	**$/t ore**	**10.52**
Head Grade	**g/t Au**	**1.25**
Oxide Recovery	%	82%
Royalties	%	3.0%
Effective Recovery	%	79.9%
Gold Price	**A$/oz**	**1,200.00**
Cut-off grade	g/t Au	0.35

15.2 Dilution

The Mineral Reserve estimate included a dilution allowance of 2%.

16 MINING METHODS

16.1 Introduction

The Kobada deposit will be mined by conventional open pit mining methods utilizing 30 to 40 t articulated dump trucks and 4.5m^3 excavators. The mine production schedule is based on delivering 1.6 Mt of mill feed material per year. It is anticipated that a mining contractor will be utilized to perform all mining functions consisting of mostly loading and hauling, as well as road and pit maintenance functions. There may be a need for some blasting, but this requirement is considered non-routine as the saprolite rock types are free digging.

16.2 Geotechnical Review

During 2015 AGG collected PQ diamond core from none holes adjacent to the location of the preliminary design pit walls. Logging of core and strength testing was carried out and used in slope design undertaken by Ground Control Engineering Pty Ltd (GCE).

Preliminary slope design parameters for each geotechnical unit for pit walls up to 120 m high are given in Table 16.1. They have been reached using a combination of results derived from geotechnical assessment based on the following:

1. Empirical assessment;

2. Numerical stability modelling; and

3. Engineering judgment and past experience from similar open pits.

Table 16.1
Slope Design Parameters

Geotechnical Unit	Maximum IRSA*	Batter Height (m)	Batter Angle	Berm Width (m)
Clay Zone, Laterite	40°	10	60°	6
Saprolite	40°	10	60°	6
Bedrock	49°	10	70°	5

*Inter-ramp slope angle

Full details of the report by GCE are provided in Appendix B.

16.3 Hydrogeological Review

In-pit water management will primarily consist of runoff control and sumps. Given that the pit will be operating at depths of up to 100m below crest, high-lift pontoon- mounted pumps will be necessary to draw from sumps. This ensures the pumps are not submerged as sump water levels rise rapidly in response to a heavy rainfall event.

The pit location coincides with the water shed of the local area and no stream diversions are required. This is illustrated in Figure 16.2.

A hydrogeological review was carried out by ''l'entreprise Sahélienne pour l'Hydraulique et le Génie Civil'' (ESHYGEC) and consisted of a geophysical study and the drilling of five boreholes and conducting pumping tests.

The depth of the water table is highly dependent on the season, with the depth of the water table in the wet season visually seen at around 5m below surface in artisanal workings. Drilling data indicated that water bore production was variable with the most productive bore producing 3.5L/second from and aquifer at 62m depth.

The steady-state groundwater inflow rates into the pit were assumed to be around 15 L/s. However, the inflow rates are highly sensitive to assigned hydraulic conductivity values and to a lesser degree recharge rates. Groundwater inflows into the pit will result in a partial dewatering of the aquifer in the vicinity of the mine.

16.4 Pit Optimisation

Whittle optimization software was used to develop a guide for the mine designs. Whittle enables the generation of a series of nested optimal pits where each successive outline is for a slightly higher product price than the previous one. This is done for a range of prices, from the lowest price that produces the smallest economic pit, to the highest price. These pits are then interrogated at the base case costs and prices to establish their relative values.

16.4.1 Pit Optimization Assumptions

The pit optimization input parameters used in the optimization process were supplied by AGG. The optimization parameters are summarized in Table 16.2.

Table 16.2
Pit Optimisation Parameters

Parameter	Unit	Deposit	
Gold Price	US$/oz	1,200	
Royalty	%	3.0	
Process Recovery - Oxide	%	Based on constant tail grade of 0.22g/t	
Process Recovery – transition/Fresh	%	0	
Mining Dilution	%	2	
Mining recovery	%	100	
Base Mining Costs	US$/t mined	2.24	
Ore Mining Cost Premium	US$/t ore	0.43	
Processing Cost	US$/t ore	6.55	
G&A Costs	US$/t ore	3.54	
Total Ore Cost	**US$/t ore**	**10.52**	
Pit Slopes		No ramp	With ramp
Pit Slope – overall slope angle (weathered rock)	degree	40	35

Resource Models

The resource models used for the analysis were prepared by International Resource Solutions Pty Ltd. The model was Multiple Indicator Kriged (MIK) estimation as described in Section 14.

Topography

A high quality topography based on an aerial survey was used for all pit optimizations. This data was collected as part of the 2010 geophysical survey completed by Xcalibur Airborne Geophysics, and consisted of high resolution LIDAR.

Mining Dilution and Recovery

Dilution of 2% was included in the pit optimization with a 100% recovery of ore blocks in the mining operation. This was considered to be appropriate given the style of mineralization. The resource exhibits a large scale and relatively homogenous gold distribution, and the proposed mining fleet consists of small equipment (50t excavator and 40t trucks), allowing selectivity if required.

Operating Costs

Operating costs are summarized in Table 16.2. The mining cost used in pit optimization was US$2.24 per tonne of waste mined. This was the base mining cost.

An additional ore mining costs of US$0.43 per tonne of ore was allowed for the provision of technical services and grade control, US$6.55 per tonne of ore for mineral processing and US$3.54 per tonne of ore for general and administrative expenditure. Total ore cost was $10.52 per tonne of ore processed.

Processing Recovery

The processing recovery used was based on a fixed tail grade of 0.22g/t Au. The recovered grade for each block was calculated and used in the optimization process (i.e. [block grade g/t] minus 0.22). Thus a block with a grade of 1.25g/t would achieve a processing recovery of 82.4%.

Given the use of a recovered grade, rather than the insitu grade from the resource model, a recovery of 97% was used in the Whittle software, which was the allowance for the 3% Malian Government Royalty.

Discount Rate

A discount rate of 10% was used in the calculation of NPV.

Mining Rate

A steady state ore mining rate of 1.6 Mtpa was assumed, and was based on the throughput of the Gekko processing plant.

Capital and Other Costs

No allowance was made for capital costs, taxation or project finance charges during the optimisation process, as is normal. These costs should be subtracted from the value of the optimal pits obtained, and have been included in cash flow models for the project.

16.4.2 Pit Optimization Results

A summary of the optimisation results for the Kobada Project is presented in Table 16.3 and illustrated in Figure 16.1.

The highest cash flow was recorded for Pit 16, which was selected for detailed mine design. This pit contained 12.3 million tonnes of ore at an average grade of 1.27 g/t Au, with an NPV_{10} estimated at US$161 million, excluding pre-production and sustaining capital expenditures.

The cash cost for Pit 16 was US$612 per ounce of gold mined which returned an incremental operating cost of US$1,137 per Au ounce. The incremental cost is the cost of mining the increment between pit 15 and pit 16 as detailed in Table 16.3.

Figure 16.1

Pit Optimisation Results - Discounted Cash Flow and Incremental Operating Cost

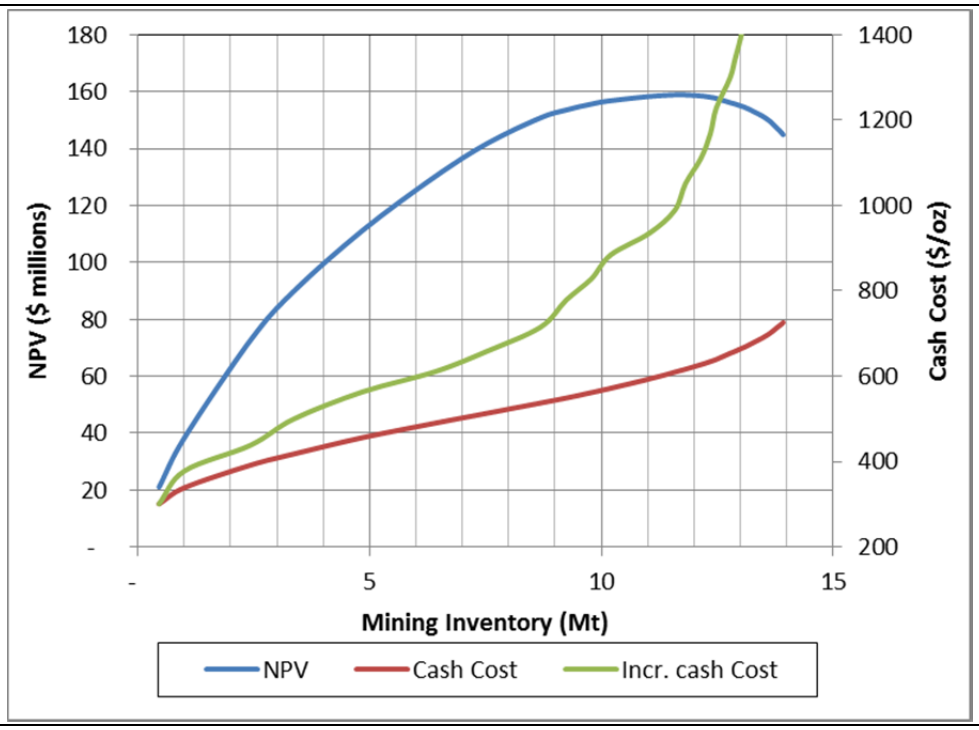


Table 16.3

Pit Optimisation Results

	PHYSICALS											FINANCIALS							
Pit No	Laterite Inventory			Saprolite Inventory			Oxide Inventory			Waste	Strip	Mining Cost	Process Cost	Total Cost	Cash Cost	Inc. Cost	Process Revenue	Cashflow	Average DCF
	dmt	Au g/t	Au oz	dmt	Au g/t	Au oz	dmt	Au g/t	Au oz	t	Ratio	$ '000	$ '000	$ '000	$/oz Au	$/oz Au	$ '000	$ '000	$ '000
1	219,352	1.75	12,352	371,713	1.86	22,257	591,065	1.82	34,609	594,433	1.0	2,594	6,218	8,812	299	299	35,106	26,294	25,384
2	348,780	1.57	17,618	826,531	1.72	45,731	1,175,311	1.68	63,349	1,304,756	1.1	5,436	12,364	17,800	333	377	63,496	45,696	42,606
3	470,566	1.45	21,950	2,131,609	1.53	104,960	2,602,175	1.52	126,911	3,072,286	1.2	12,503	27,375	39,878	379	426	125,184	85,306	75,801
4	552,323	1.39	24,637	3,057,585	1.46	143,951	3,609,908	1.45	168,588	4,677,116	1.3	18,274	37,976	56,251	405	489	165,044	108,794	93,680
5	628,638	1.33	26,890	4,602,184	1.39	205,539	5,230,822	1.38	232,429	7,966,917	1.5	29,115	55,028	84,144	444	549	225,472	141,328	116,167
6	657,322	1.32	27,849	5,892,721	1.36	258,349	6,550,043	1.36	286,199	11,894,987	1.8	40,738	68,906	109,644	471	592	276,741	167,097	132,080
7	676,791	1.31	28,427	7,001,034	1.35	303,528	7,677,825	1.34	331,955	15,921,701	2.1	52,199	80,771	132,970	494	637	320,328	187,358	143,655
8	689,885	1.30	28,806	8,206,399	1.33	350,537	8,896,284	1.33	379,343	20,638,335	2.3	65,447	93,589	159,036	518	693	365,056	206,021	153,147
9	701,492	1.29	29,155	8,778,090	1.31	370,844	9,479,582	1.31	399,999	22,737,620	2.4	71,454	99,725	171,179	530	757	384,128	212,949	155,904
10	714,589	1.28	29,502	9,289,635	1.30	389,042	10,004,224	1.30	418,544	24,944,756	2.5	77,584	105,244	182,828	542	810	401,243	218,415	158,080
11	722,420	1.28	29,727	9,705,361	1.29	403,211	10,427,781	1.29	432,938	26,788,668	2.6	82,647	109,700	192,347	552	861	414,392	222,046	159,247
12	725,614	1.28	29,819	10,555,225	1.27	432,055	11,280,839	1.27	461,873	30,922,404	2.7	93,878	118,674	212,553	574	910	440,814	228,261	160,827
13	732,697	1.27	30,027	11,024,083	1.27	449,301	11,756,780	1.27	479,328	34,071,632	2.9	102,028	123,681	225,709	587	963	457,068	231,359	161,509
14	735,445	1.27	30,103	11,271,532	1.26	457,834	12,006,977	1.26	487,937	35,660,940	3.0	106,168	126,313	232,482	595	1,021	464,959	232,478	161,529
15	737,849	1.27	30,165	11,609,819	1.26	469,086	12,347,668	1.26	499,252	37,865,194	3.1	111,912	129,897	241,810	605	1,080	475,232	233,423	161,245
16	741,197	1.27	30,252	11,808,894	1.25	475,822	12,550,091	1.25	506,074	39,365,636	3.1	115,730	132,027	247,757	612	1,137	481,452	233,695	160,824
17	744,865	1.27	30,348	11,946,681	1.25	480,595	12,691,546	1.25	510,943	40,563,246	3.2	118,743	133,515	252,258	617	1,200	485,915	233,657	160,400
18	746,441	1.27	30,389	12,075,031	1.25	485,093	12,821,472	1.25	515,482	41,778,345	3.3	121,768	134,882	256,650	623	1,251	490,092	233,442	159,914
19	749,292	1.26	30,463	12,343,862	1.25	495,151	13,093,154	1.25	525,614	44,818,620	3.4	129,209	137,740	266,949	636	1,293	499,563	232,615	158,699
20	749,966	1.26	30,480	12,503,043	1.25	500,717	13,253,009	1.25	531,198	46,545,847	3.5	133,469	139,422	272,890	643	1,375	504,701	231,811	157,886
21	750,042	1.26	30,483	12,566,006	1.24	502,799	13,316,048	1.25	533,282	47,203,731	3.5	135,090	140,085	275,175	646	1,438	506,591	231,417	157,510
22	750,261	1.26	30,489	12,643,583	1.24	505,293	13,393,844	1.24	535,782	48,002,526	3.6	137,078	140,903	277,981	650	1,484	508,841	230,860	157,035
23	750,484	1.26	30,496	12,762,269	1.24	509,305	13,512,753	1.24	539,801	49,434,316	3.7	140,580	142,154	282,734	656	1,542	512,507	229,772	156,143
24	750,649	1.26	30,500	12,862,536	1.24	512,693	13,613,185	1.24	543,193	50,714,630	3.7	143,710	143,211	286,921	662	1,609	515,602	228,680	155,295
25	750,756	1.26	30,503	12,947,269	1.24	515,399	13,698,025	1.24	545,903	51,737,693	3.8	146,234	144,103	290,337	667	1,670	518,035	227,698	154,600
26	751,128	1.26	30,512	13,001,516	1.24	517,084	13,752,644	1.24	547,596	52,408,302	3.8	147,869	144,678	292,547	670	1,742	519,543	226,997	154,063
27	751,193	1.26	30,514	13,088,737	1.24	520,034	13,839,930	1.24	550,548	53,702,487	3.9	151,011	145,596	296,607	676	1,793	522,237	225,629	153,145
28	751,193	1.26	30,514	13,145,743	1.24	522,034	13,896,936	1.24	552,548	54,645,186	3.9	153,287	146,196	299,483	680	1,857	524,079	224,596	152,428
29	751,193	1.26	30,514	13,174,823	1.23	523,113	13,926,016	1.24	553,627	55,195,580	4.0	154,597	146,502	301,099	682	1,907	525,086	223,988	152,008
30	751,193	1.26	30,514	13,202,738	1.23	524,207	13,953,931	1.24	554,721	55,792,881	4.0	156,003	146,795	302,799	685	1,954	526,121	223,322	151,547

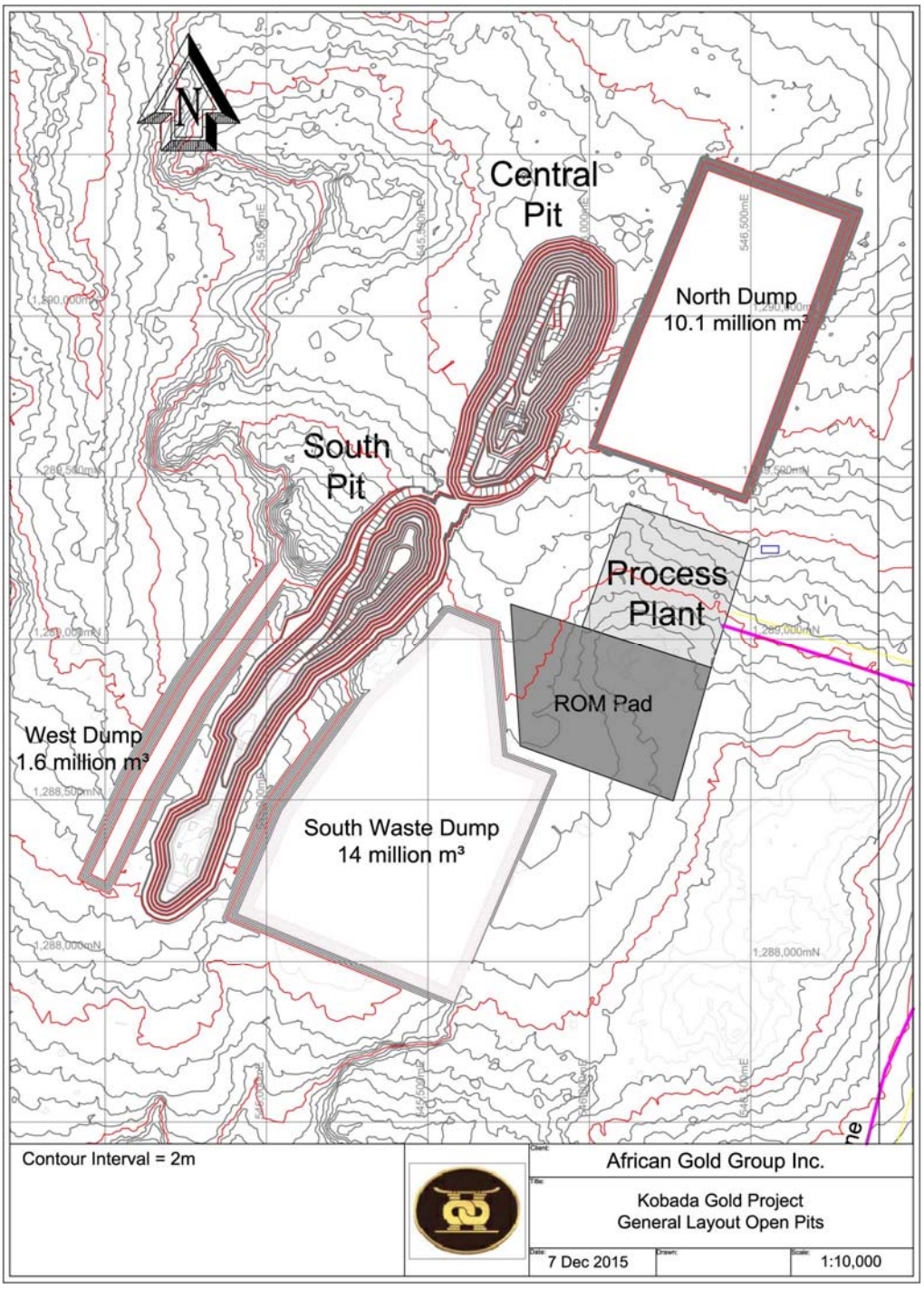
NB Au grade detailed in Table 16.3 are insitu grades.

16.5 Pit Designs

Pit designs were completed based on the pit shell selected in Section 16.4. The Mineral Reserve was estimated to be **12.7Mt at 1.25g/t Au, containing 511,000 ounces of gold**. In addition, the pit designed also contained inferred resources of 0.86 Mt at 1.19g/t, containing 33,000 ounces of gold...

Figure 16.2

Final Pit Design



16.5.1 Comparison of Design Pits to Optimized Pit Shells

Table 16.4 summarizes the difference between pit optimization results and final pit designs. The detailed design was around 5% larger than the optimized pit, mostly as a result of detailed design of ramps for access to the base of the pit.

Table 16.4
Comparison of Final Design With Optimised Pits

Pit	Parameter	Unit	Optimised Pit	Final Pit Design	Variance (%)
Total	Ore	Mt	12.6	12.7	1%
	Grade	g/t Au	1.25	1.25	0%
	Metal	Au oz	506	511	1%
	Waste	Mt	39.4	41.8	6%
	Total Material	Mt	51.9	54.5	5%
	Strip Ratio		3.1	3.3	5%

16.6 Production Schedules

Production schedules were prepared using the Mineral Reserve. In general the schedule aim was to maximize grade in the first two years of operations whilst deferring the waste stripping requirements. Figure 16.3 illustrates the waste and ore mining requirements along with the annual stripping ratio over the life of the project.

The key scheduling priorities were the following:

- Maximise grade early in the production profile to improve early cash flow;
- Minimise waste to ore ratio in first two years to improve early cash flow; and

The production schedule was split into five separate pit stages:

Stage 1 – starter pit on Central Pit;

Stage 2 – final cut back of Central Pit;

Stage 3 – starter on South Pit; and

Stage 4 – final cut back on South Pit.

Figure 16.3
Production Schedule

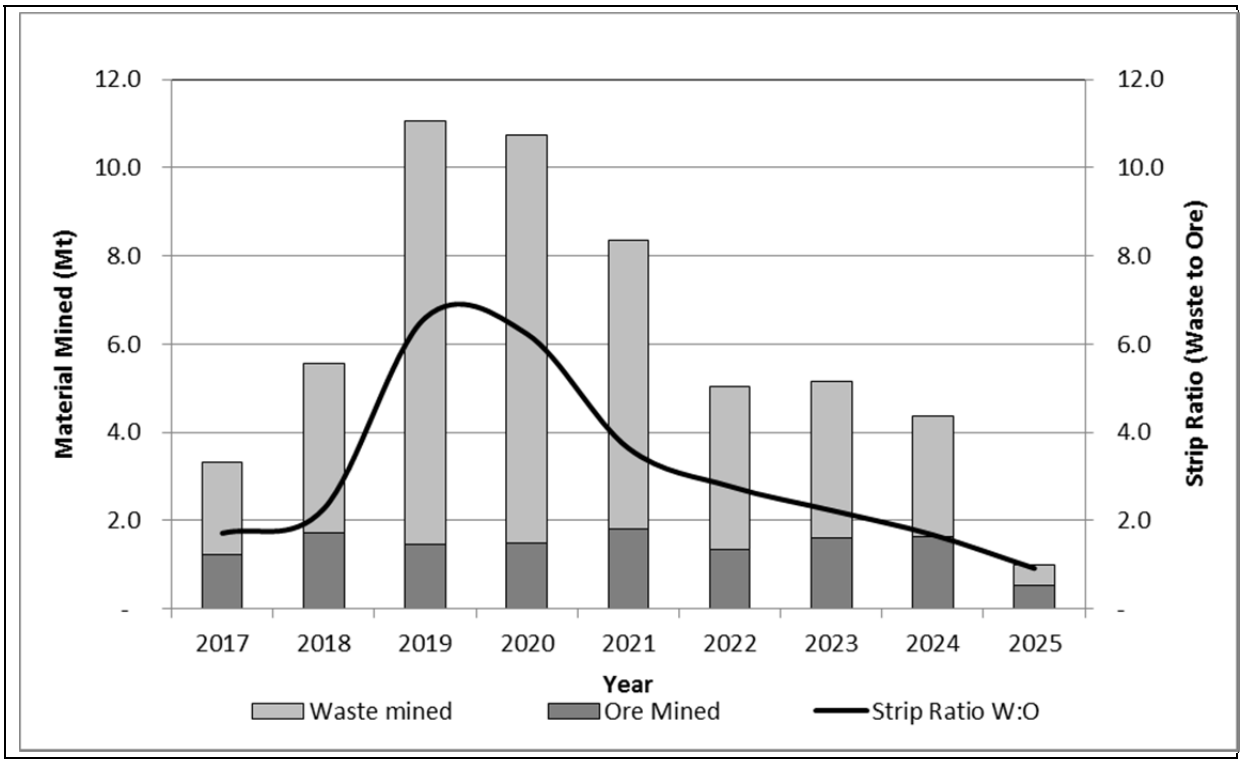


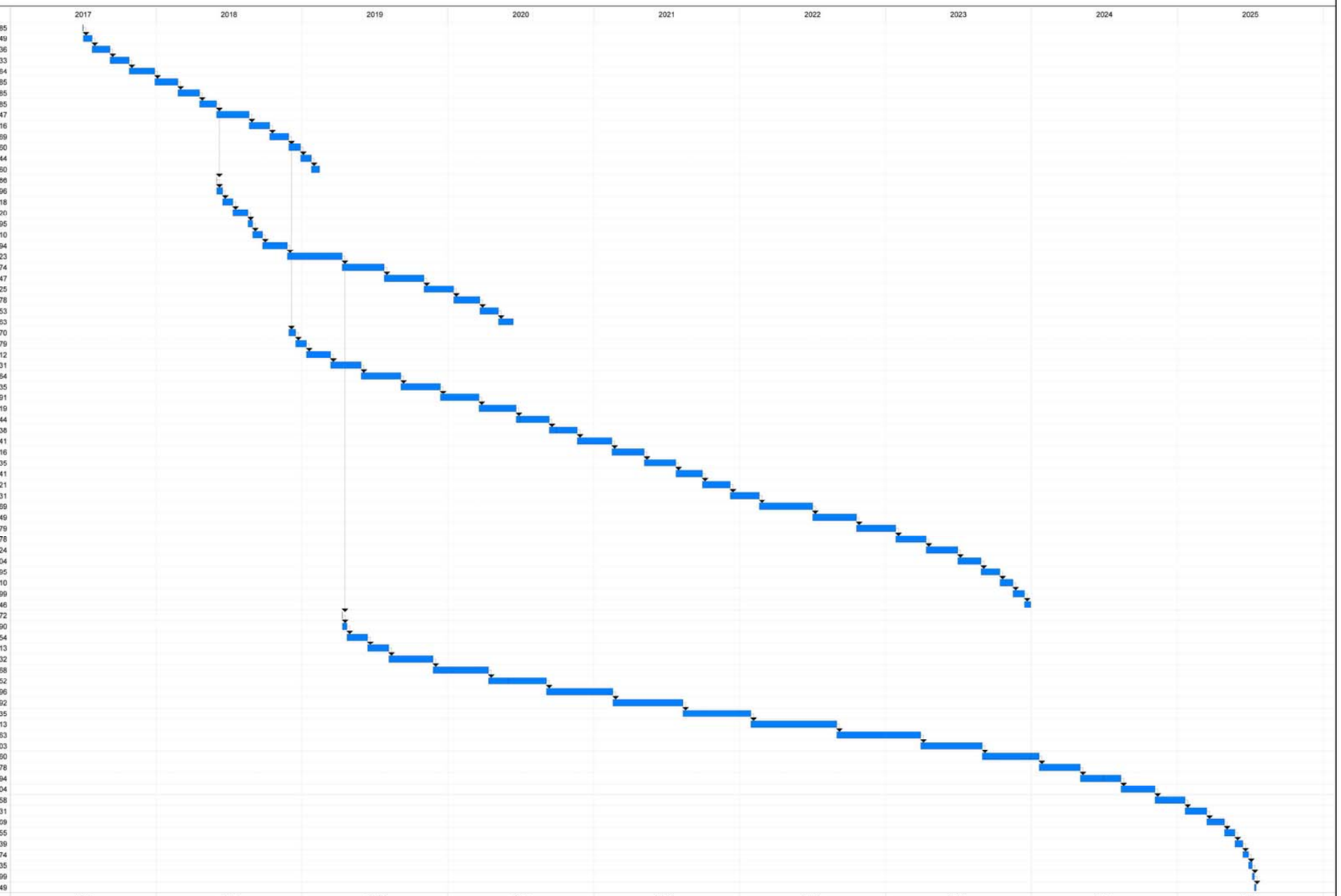
Figure 16.4

Production Schedule Gantt

AFRICAN GOLD GROUP, INC.

Kobada Gold Project
Life of Mine Production Schedule

Pit	mRL	Start	Total Ore t	Total Ore g/t	Total Ore oz	Waste t	Material t	BCM
1	390	01 Jul 17	4,085	1.03	135	17,478	21,563	8,985
1	385	02 Jul 17	68,099	1.64	3,626	217,607	285,406	119,949
1	380	24 Jul 17	189,841	1.45	8,845	393,206	583,047	245,436
1	375	07 Sep 17	224,442	1.48	10,644	365,402	589,844	262,933
1	370	26 Oct 17	281,708	1.49	13,636	431,496	713,203	351,664
1	365	28 Dec 17	258,685	1.52	12,655	374,284	632,969	316,485
1	360	24 Feb 18	238,104	1.53	11,708	354,865	592,969	296,485
1	355	19 Apr 18	201,062	1.47	9,532	266,907	467,969	233,985
1	350	01 Jun 18	202,910	1.47	9,619	243,184	446,094	223,047
1	345	21 Aug 18	161,606	1.40	7,284	120,425	282,031	141,016
1	340	12 Oct 18	162,058	1.41	7,325	98,880	260,938	130,469
1	335	28 Nov 18	102,180	1.43	4,699	59,539	161,719	80,860
1	330	28 Dec 18	82,647	1.47	3,895	47,041	129,688	64,844
1	325	24 Jan 19	71,186	1.30	2,971	40,533	111,719	55,860
2	415	01 Jun 18	3,227	1.15	119	714	3,941	586
2	410	01 Jun 18	38,272	1.08	1,324	49,697	87,969	40,396
2	405	16 Jun 18	67,347	1.07	2,306	83,512	150,859	70,218
2	400	11 Jul 18	87,563	1.02	2,870	128,296	215,859	103,420
2	395	18 Aug 18	15,575	1.02	512	49,738	65,313	32,295
2	390	30 Aug 18	51,852	0.99	1,650	90,570	142,422	68,010
2	385	24 Sep 18	110,532	1.01	3,602	246,031	356,563	169,694
2	380	25 Nov 18	196,449	1.17	7,381	568,317	764,766	376,423
2	375	11 Apr 19	179,725	1.30	7,529	395,353	575,078	287,374
2	370	25 Jul 19	177,915	1.30	7,449	368,178	546,094	273,047
2	365	02 Nov 19	150,428	1.38	6,665	255,822	406,250	203,125
2	360	15 Jan 20	140,244	1.41	6,379	219,912	360,156	180,078
2	355	21 Mar 20	114,639	1.44	5,325	139,266	253,906	126,953
2	350	06 May 20	103,422	1.48	4,934	99,703	203,125	101,563
3	390	28 Nov 18	8,072	0.84	217	104,897	112,969	47,070
3	385	16 Dec 18	18,891	1.00	608	352,047	370,938	163,779
3	380	12 Jan 19	56,794	1.23	2,250	890,861	947,656	413,312
3	375	13 Mar 19	68,642	1.19	2,629	1,040,967	1,109,609	522,531
3	370	29 May 19	78,333	1.25	3,153	1,283,073	1,361,406	680,664
3	365	05 Sep 19	76,567	1.16	2,862	1,278,902	1,355,469	677,735
3	360	13 Dec 19	75,273	1.15	2,771	1,250,508	1,325,781	662,891
3	355	19 Mar 20	108,371	1.10	3,836	1,165,067	1,273,438	636,719
3	350	20 Jun 20	111,356	1.10	3,952	1,118,332	1,229,688	614,844
3	345	11 Sep 20	163,514	1.18	6,190	1,058,361	1,221,875	610,938
3	340	20 Nov 20	173,924	1.18	6,606	1,014,357	1,188,281	594,141
3	335	15 Feb 21	215,348	1.24	8,566	891,684	1,107,031	553,516
3	330	06 May 21	232,601	1.24	9,292	849,722	1,082,324	540,935
3	325	24 Jul 21	225,432	1.21	8,802	685,486	910,918	455,241
3	320	29 Sep 21	289,637	1.22	11,398	665,794	955,430	477,221
3	315	08 Dec 21	264,681	1.18	10,034	532,995	797,676	398,031
3	310	18 Feb 22	259,808	1.18	9,876	474,782	734,590	365,569
3	305	02 Jul 22	240,920	1.18	9,146	364,959	605,879	300,549
3	300	20 Oct 22	229,444	1.19	8,791	314,637	544,082	269,479
3	295	26 Jan 23	192,695	1.18	7,298	223,595	416,289	207,678
3	290	12 Apr 23	172,779	1.18	6,580	174,056	346,836	173,224
3	285	30 Jun 23	144,381	1.26	5,858	112,396	256,777	128,204
3	280	28 Aug 23	124,090	1.26	5,076	83,566	208,457	103,595
3	275	14 Oct 23	97,099	1.25	3,911	47,353	144,453	71,710
3	270	16 Nov 23	84,800	1.24	3,376	42,250	127,051	62,799
3	265	14 Dec 23	42,662	1.28	1,760	26,030	68,691	34,246
4	415	11 Apr 19	4,174	1.15	154	166	4,340	1,172
4	410	12 Apr 19	29,638	1.08	1,025	44,972	74,609	31,290
4	405	23 Apr 19	44,910	1.15	1,667	274,933	319,844	141,154
4	400	14 Jun 19	34,155	1.01	1,106	650,923	685,078	305,413
4	395	06 Aug 19	144,390	1.04	4,825	1,181,314	1,325,703	636,732
4	390	25 Nov 19	209,137	1.06	7,142	1,444,769	1,653,906	799,368
4	385	12 Apr 20	265,029	1.08	9,186	1,825,675	2,090,703	1,030,352
4	380	04 Sep 20	275,875	1.07	9,493	2,019,203	2,295,078	1,139,796
4	375	17 Feb 21	307,408	1.09	10,782	1,706,576	2,013,984	1,006,992
4	370	11 Aug 21	303,483	1.10	10,685	1,651,986	1,955,469	977,735
4	365	28 Jan 22	301,661	1.10	10,655	1,463,964	1,765,625	882,813
4	360	31 Aug 22	290,610	1.10	10,315	1,437,516	1,728,125	864,063
4	355	30 Mar 23	302,514	1.15	11,186	1,213,892	1,516,406	758,203
4	350	30 Aug 23	325,450	1.15	12,065	1,161,269	1,486,719	743,360
4	345	20 Jan 24	408,649	1.24	16,277	1,001,507	1,410,156	705,078
4	340	02 May 24	404,929	1.26	16,443	891,262	1,296,191	647,894
4	335	12 Aug 24	373,674	1.24	14,882	608,611	982,285	489,904
4	330	05 Nov 24	346,302	1.26	13,979	523,073	869,375	433,058
4	325	19 Jan 25	298,995	1.29	12,383	331,591	630,586	312,831
4	320	15 Mar 25	193,012	1.34	8,330	244,005	437,017	216,609
4	315	28 Apr 25	133,737	1.36	5,858	129,661	263,398	130,955
4	310	24 May 25	95,002	1.39	4,248	101,053	196,055	97,239
4	305	13 Jun 25	71,475	1.44	3,300	68,759	140,234	69,274
4	300	27 Jun 25	55,177	1.42	2,521	37,480	92,656	45,935
4	295	07 Jul 25	39,669	1.45	1,846	13,808	53,477	26,599
4	290	12 Jul 25	32,409	1.47	1,529	10,541	42,949	21,349



Indicators	Name	2017	2018	2019	2020	2021	2022	2023	2024	2025
	Laterite t	405,323	124,883	202,389	52,485	2,675				
	Lat g/t	1.46	0.98	1.12	1.01	0.92				
	Lat oz	19,033	3,946	7,268	1,699	79				
	Saprolite t	378,723	1,643,357	1,178,308	1,478,992	1,769,113	1,337,614	1,588,176	1,479,099	924,894
	Sap g/t	1.52	1.39	1.25	1.20	1.17	1.14	1.18	1.24	1.33
	Sap oz	18,494	73,573	47,172	56,865	66,703	49,135	60,356	59,099	39,413
	Other t					8,314	30,961	21,926	26,097	82,152
	Other g/t					1.59	1.54	1.69	1.36	1.56
	Other oz					425	1,533	1,191	1,144	4,118
	Total Ore t	784,046	1,768,241	1,380,697	1,531,477	1,780,102	1,368,576	1,610,102	1,505,196	1,007,046
	Total Ore g/t	1.49	1.36	1.23	1.19	1.17	1.15	1.19	1.24	1.34
	Total Ore oz	37,527	77,518	54,440	58,563	67,206	50,668	61,548	60,243	43,532
	Waste t	1,447,138	2,613,298	8,882,048	9,679,518	7,459,433	4,000,031	3,559,187	3,102,226	1,068,262
	Material t	2,231,184	4,381,539	10,262,746	11,210,995	9,239,534	5,368,606	5,169,290	4,607,422	2,075,306
	BCM	1,007,527	2,140,243	4,928,646	5,564,712	4,616,354	2,677,750	2,581,158	2,301,051	1,029,809
	Ramp tkm	424,582	1,534,528	2,957,849	4,682,639	5,256,803	3,714,561	4,131,847	3,702,873	2,120,539
	Horizontal tkm	4,104,752	6,978,088	11,796,602	13,092,033	10,699,307	6,478,918	6,417,044	6,019,394	2,556,638

% Complete Milestone Summary Bar Default Bar

Project: Kobada Gold Project

Table 16.5
Production Schedule

Year		Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Production												
Ore Mined	000t	12,735		784	1,757	1,392	1,528	1,784	1,369	1,605	1,497	1,020
Gold Grade	g/t	1.25		1.49	1.36	1.23	1.19	1.17	1.15	1.19	1.24	1.34
Contained Gold	000oz	511		38	77	55	58	67	51	61	60	44
Waste Mined	000t	41,811		1,447	2,607	8,888	9,656	7,483	4,000	3,540	3,102	1,088
Total Tonnes Mined	000t	54,547		2,231	4,364	10,280	11,184	9,267	5,369	5,144	4,599	2,109
Total BCM Mined	000 BCM	26,847		1,008	2,132	4,937	5,551	4,630	2,678	2,569	2,297	1,046
Strip Ratio	W:O	3.28		1.85	1.48	6.39	6.32	4.20	2.92	2.21	2.07	1.07
Tonnes Processed	000t	12,735		569	1,577	1,577	1,581	1,577	1,577	1,547	1,563	1,168
Contained Gold	000oz	511		27	71	63	61	59	58	59	62	50
Recovery	%	82%		85%	84%	82%	82%	81%	81%	81%	82%	83%
Gold Recovered	000oz	420		23	60	51	50	48	47	48	51	41

16.7 Waste Storage Facilities

There are three waste rock facilities planned, which are shown in Figure 16.2. The design capacity of each is detailed in Table 16.6.

Table 16.6
Waste dump design

Dump Location	Design Volume (million m^3)	Design height (m)
North	10.1	37m to 410mRL
South	14.0	44m to 440mRL
West	1.6	18m to 400mRL

The west dump is planned to attenuate nose from the operation and acts as a barrier between the mine and the Kobada village. It is planned as a narrow waste rock facility around 1,100m long along the western edge of the planned open pit.

The north and south waste dumps have been located to minimise the haulage distance from the central and south pits.

All waste rock facilities have been designed to meet the requirements set forth by the Malian Mining Regulations. While operating, waste facilities are allowed to be sloped at an angle of repose of 37º; however, final slopes will be graded to 3 (horizontal) to 1 (vertical) to allow for slope stability and re-vegetation.

Waste rock will also be used for the construction of mill run of mine (ROM) stockpile pad, tailings storage facility and infrastructure facilities as necessary during the site construction phase.

16.8 Mining Equipment

It is planned to utilise a Malian mining contractor in order to minimise the pre-production capital requirements of the project. AGG has developed relationships with several Malian companies that have the equipment and skills to undertake the mining project.

The mine plan calls for the use of relatively small trucks, and this is likely to be a mix of 6 x 6 50 tonne capacity articulated dump trucks and 30 to 40 tonne capacity road dump trucks. This size of equipment is readily available in Mali, and will provide significant flexibility to the operation. Trucks and excavators may be added or removed to adjust production capacity without a significant cost penalty.

Based on total annual material movement, equipment productivities, a 24 hour working day and a 365 day per year mining schedule, the Project will require at a minimum the equipment listed in Table 16.7.

Table 16.7 Equipment Fleet Estimate									
Item	2017	2018	2019	2020	2021	2022	2023	2024	2025
LH R966 excavator	2	2	3	3	2	2	2	1	1
Bell B50D trucks	3	6	8	9	9	7	7	6	3
Water truck	1	1	1	1	1	1	1	1	1
Bell 872G Grader	1	1	1	1	1	1	1	1	1
Cat 536E Roller	1	1	1	1	1	1	1	1	1
LH PR 764 dozer	1	1	1	1	1	1	1	1	1
I5600 Service Truck	1	1	1	1	1	1	1	1	1
Toyota Landcruiser Troop Carrier	2	2	2	2	2	2	2	2	2
Toyota Landcruiser Utility	4	4	4	4	4	4	4	4	4
Isuzu NPS300 Dual Cab Utility + Hiab	1	1	1	1	1	1	1	1	1
Terex Lighting Plants	6	6	6	6	6	6	6	6	6
Dewatering Diesel pumps	2	2	2	2	2	2	2	2	2
RC drill rig	1	1	1	1	1	1	1	1	1

A single drill rig was included for grade control drilling, and assumed to operate a continuous roster, along with the rest of the mining fleet.

16.9 Mining Personnel

Mine operations personnel from the owner's team were estimated to be 12 people consisting of a Mining Manager, one mining engineer, three geologists, four technical assistants, and three surveyors.

Mining is planned to be undertaken using a contractor. Schedules and equipment estimates have been based on mining occurring in 12 hour shifts, seven days per week. A three panel roster is planned with crews working a two week work cycle followed by a one week break. The main mine contractor personnel numbers were estimated at between 59 and 77, depending on the quantity of equipment required to meet material movements. Additional personnel will be provided by

16.10 Comment on Section 16

Areas that additional work is required include:

- Sterilisation drilling of the waste dumps, processing and ore stockpile areas is required prior to the commencement of construction.

17 RECOVERY METHODS

17.1 Introduction

Gekko System Pty Ltd (Gekkos) were engaged by AGG to prepare this study to identify and develop the scope of facilities, capital costs, operating costs and schedule for the development of the processing plant for the project. The flowsheet was further developed from the Preliminary Economic Assessment, published on 23 December 2014, and was based on additional testwork completed in 2015. The level of engineering definition and any limitations of the available data are reflected in the quoted accuracy of the cost estimates.

Full details of the test work completed by Gekko systems are provided in Appendix D.

17.2 Study Contributors

In addition to Gekko Systems staff and its contractors, Table 17.1 details the contributors to various aspects of the metallurgical studies and work.

Table 17.1
Contributors to Metallurgical Studies

Metallurgical interpretation	Gekko Systems
Process plant design	Gekko Systems
- Tailings pipeline and reclaim water pump and pipeline - Site Layout	Gekko Systems SF Design (Perth)
Tailings storage facility and site water balance	SRK Consulting / AGG
Infrastructure and services	Gekko Systems / AGG
Implementation plan and preliminary schedule	Gekko Systems
Operating cost estimate	Gekko Systems / AGG
Capital cost estimate	Gekko Systems / AGG
Compilation of study document	Gekko Systems

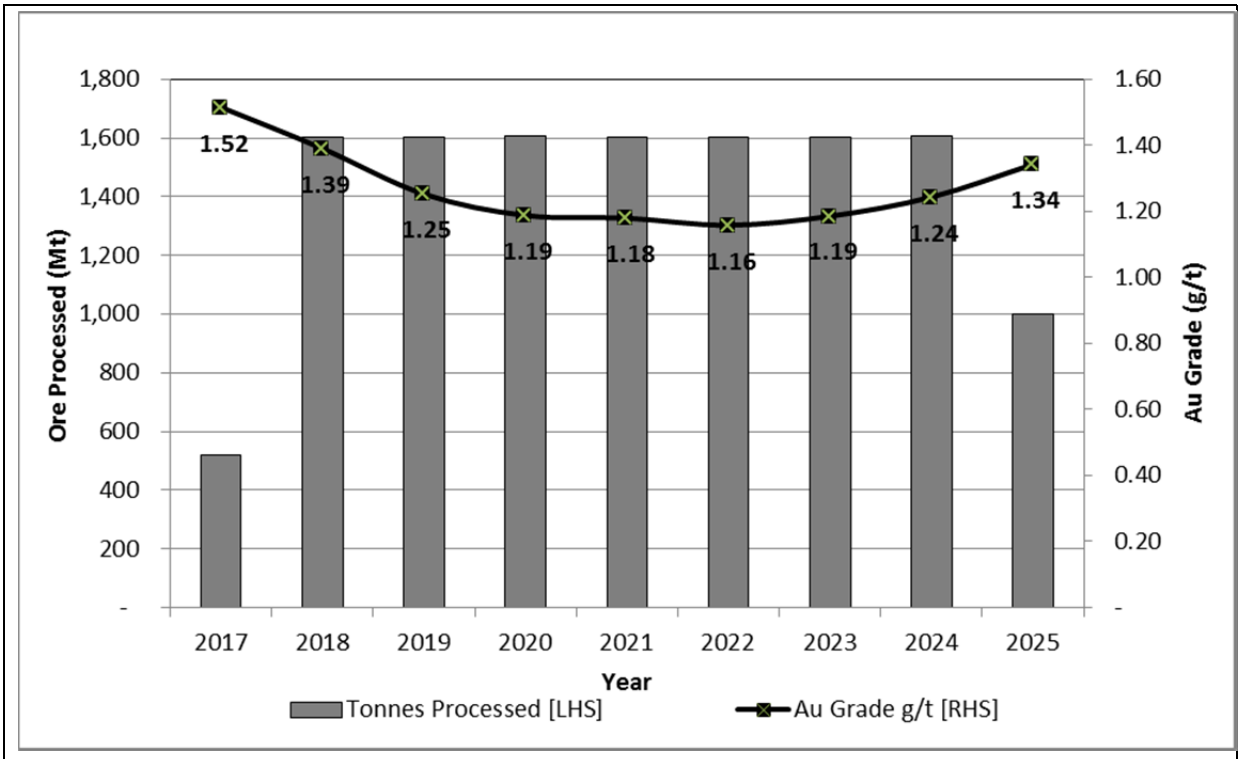
17.3 Plant Throughput and Ore Types

The flow sheet has been developed based on the results of the test work conducted by Gekko Systems Pty Ltd. The unabridged report by Gekkos is included in Appendix D.

The process is a combination of scrubbing, slimes rejection, gravity recovery and intensive cyanidation of the concentrates. Nominal feed grade to the mill will be 1.25 g/t Au, although this is expected to vary over the course of the mine life due to variability of the mineralized body. Figure 17.1 shows the throughput and head grade over the life of mine.

The plant design has been based on a nominal capacity of 1.6 Mt per annum of blended ore made up of from Saprolite and Laterite ores. The ratio of ore types per annum is given in Figure 17.2. The process plant will operate continuously with shutdowns as required for routine maintenance.

Figure 17.1
Plant throughput and head grade



Source AGG

Table 17.2
Ore type feed percentage

Ore Type	2017	2018	2019	2020	2021	2022	2023	2024	2025
Laterite Ore (%)	10	10	10	10	10	4			
Saprolite Ore (%)	90	90	90	90	90	96	100	100	100

17.4 Process Plant Description

A simplified flowsheet for the Kobada Gold project is presented in Figure 17.2. Preliminary engineering on the plant design has been completed and the proposed layout is shown in Figure 17.3. The detailed process plant description can be found in Appendix D.

The treatment plant design incorporates the following unit process operations:

- Ore scrubbing and rejection of +40mm oversize;
- Screening and crushing (-40 to +1.18mm) with recycling of product to scrubber;
- A rougher and cleaner cycloning circuit to reject fines and pre-concentrate the feed;
- A coarse gravity circuit utilising Rougher, Scavenger and Cleaner InLine Pressure Jigs (IPJ);
- A fine gravity circuit utilising two Knelson batch centrifugal concentrators;

- A gravity tails dewatering cyclone circuit – for recycling process water;
- A tails storage facility (TSF) pumping and piping system, along with a TSF reclaim water pump and piping system;
- A gravity concentrate regrind circuit, comprising a ball mill in closed circuit with a cyclone to produce an 80% passing 125 micron grind size. Leaching in cyanide solution to increase leaching residence time;
- An intensive leaching reactor train processing the cyclone overflow, utilising InLine leach reactor drums, to maximise the gold leaching rate in high oxygen environment, providing a total of 10 hours leaching;
- A post leaching thickener to increase the slurry density feeding the leach residue belt filter;
- An AuRIX resin counter-current absorption column to extract the gold from the solution and return of barren solution to the process;
- A resin stripping, electrowinning and gold smelting system to produce dorè; and
- SO_2 / Oxygen cyanide destruction circuit to reduce the tailings cyanide concentration to below the International Cyanide Management Code (ICMC) requirement.

17.5 Plant Layout

The layouts provided are conceptual and are typical for the flowsheet adopted for the process plant and reflect the level of engineering completed for this study. The layouts support the capital cost estimation.

Materials handling, containment and bunding in all areas will meet the requirements of the International Cyanide Management Code (ICMC).

17.6 Process Design Criteria

The plant design is based on an annual ore treatment of 1.6 million dry tonnes per annum. The key design criteria is summarised in Table 17.3 below.

Table 17.3
Process Design Criteria Summary

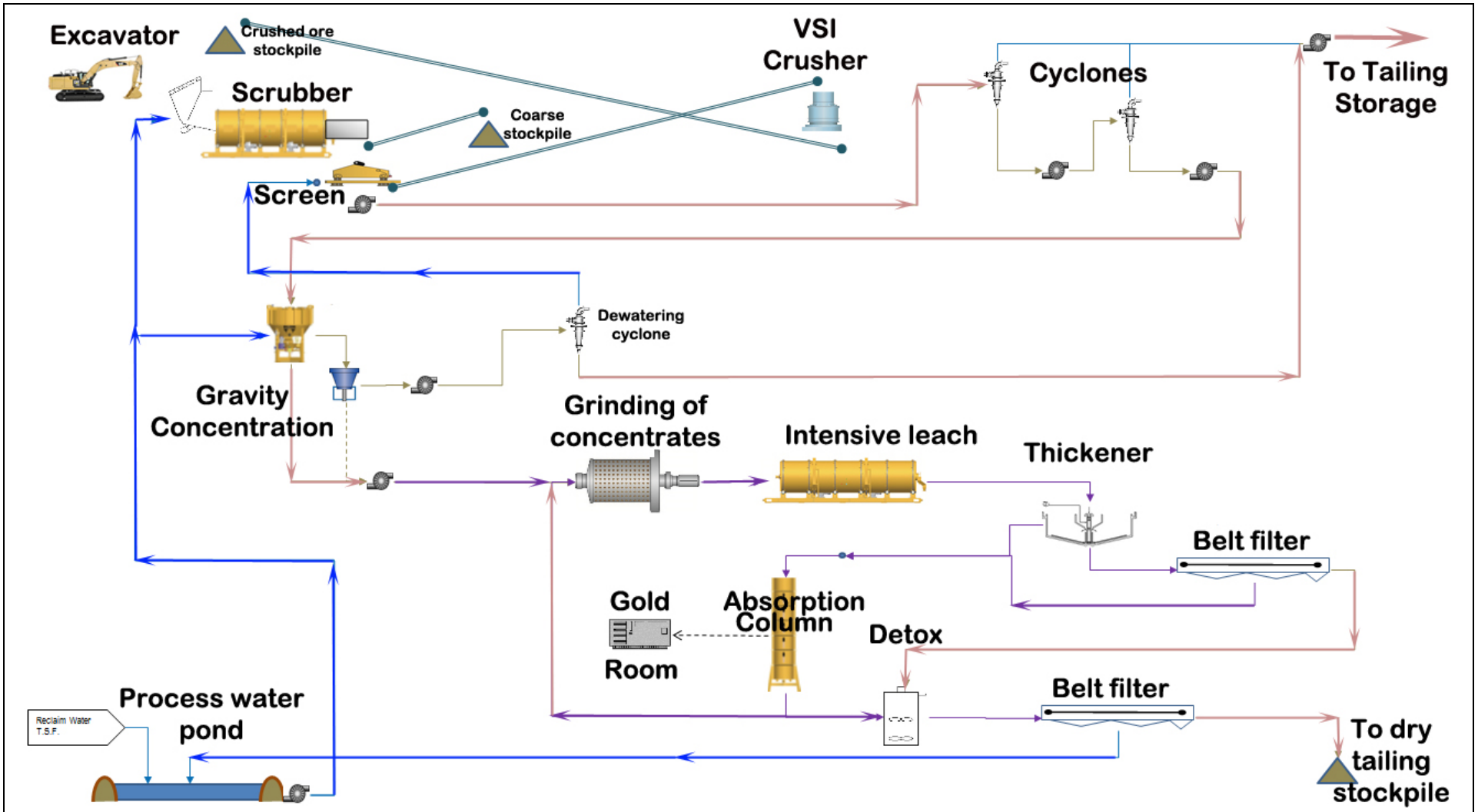
Parameter	Units	Value
Plant Throughput		
Ore moisture	%	12
Ore gold grade	g/t	1.27
Ore specific gravity (average)	t/m3	2.8
Plant feed	t/h (dry)	215
	t/h (wet)	241
	t/a (dry)	1,600,000
	t/a (wet)	1,792,000
Plant Availability	%	85

Table 17.3 Process Design Criteria Summary		
Parameter	**Units**	**Value**
Gold Recovery (based on 1.27 g/t head grade)		
Hydrocyclone recovery	%	94.0
Gravity recovery	%	88.9
Total Pre-concentration recovery	%	**83.5**
Leaching recovery	%	99.0
Resin	%	99.3
Overall recovery	%	**82.1**
Overall tails grade	g/t	**0.227**
Scrubber		
Type	Girth gear driven, bearing mounted, rubber lined	
Dimensions	m	3.0 x 7.86
Residence time	min	2.0
Critical Speed	Nc %	60
Scrubbed particles above 1.18mm	%	20
Vibrating Screen		
Type	Double deck horizontal	
Dimensions	m	3.7 x 7.0
Vertical Shaft Impact Crusher		
Model	REMco 320 oil lubricated	
Circulating load	%	250
Throughput	dry tph	108
Pre-concentration Hydrocyclone Circuit		
Optimum cyclone feed density	%w/w	20
Rougher cyclones	24 x 150-CVX-06 Cluster	
Cleaner cyclones	8 x 150-CVX-06 Cluster	
Solids recovery to underflow	%	**29.6**
	dry tph	63.7
Inert Tailings (Cyclone o/f + gravity tails)	g/t	0.220
	dry tph	204.7
	%w/w	16.4
Gravity Recovery Circuit - Coarse		
Coarse rougher gravity units	2 x InLine Pressure Jigs (IPJ2400)	
Coarse cleaner gravity unit	1 x InLine Pressure Jigs (IPJ1500)	
Fine gravity units	2 x Knelson Concentrators (KC-XD40)	
Concentrate production	tph	10.8
	g/t	21.2
Concentrate Grinding		
Type	Overflow ball mill	
Dimensions	m	2.1 x 3.6 m EGL
Power	kW	180
Grind size – P80	µm	125
Classification cyclone	1 x 250CVX20 at 60 degrees	
Concentrate Leaching		
Type	Intensive leaching reactor drums	
Model	6 x ILR5000CA	
Residence time (incl. feed and discharge tanks)	h	10

Table 17.3 Process Design Criteria Summary		
Parameter	**Units**	**Value**
Concentration of lixiviant	%NaCN w/v	0.25
Leach solids density	%w/w	37.5
Consumption – NaCN	kg/t	1.0
Consumption – NaOH	kg/t	1.5
Leach residue thickener		
Type	6m diameter high rate	
Consumption – High pH flocculant	g/t	120
Leach residue filter		
Type	Vacuum Belt Filter, RB-SV 1.6m x 16m	
Design flux rate	kg DS/(m^2.h)	600
Design wash efficiency	%	97
Detoxification		
Type	SMBS into agitated and oxygenated tank	
Reactor size	m^3	43
Detox Tails filter		
Type	Vacuum Belt Filter, RB-SV 1.6m x 8m	
Design flux rate	kg DS/(m^2.h)	750
Resin Absorption		
Contactor	Upflow counter-current resin column	
Number of stages		6
Column dimensions (height-diameter)	m	7.5 x 1.5
Resin Type	Alkaline resin functionalised with guinidine	
Maximum gold loading	g/m^3	8,000
Utilities		
Process Water Pond	HDPE lined pond	
Volume	m^3	8,750
Residence time (max'm)	h	8
Pump capacity	m^3/h	1,350
TSF Reclaim Water Pump	Electric floating pontoon	
Pipeline length	m	5,625
Pipeline diameter	mm	500
Reagents		
High pH flocculant	Packaged powder preparation plant	
Consumption	kg/d	31.0
	t/y	11.3
Sodium Cyanide	Bulk-a-bag mix and holding tanks	
Consumption	kg/d	19.3
	t/a	143.4
Sodium Hydroxide	Bulk-a-bag mix and holding tanks	
Consumption	kg/d	36.5
	t/a	271
Copper Sulphate	25 kg bag mix and holding tanks	
Consumption	kg/d	0.9
	t/a	7.0
Oxygen	VSA Oxygen Generators (duty/standby)	
Consumption	Nm^3/h	30

Figure 17.2

Simplified Process Flow Sheet



Source: Gekko, January 2016

Figure 17.3

Proposed plant layout

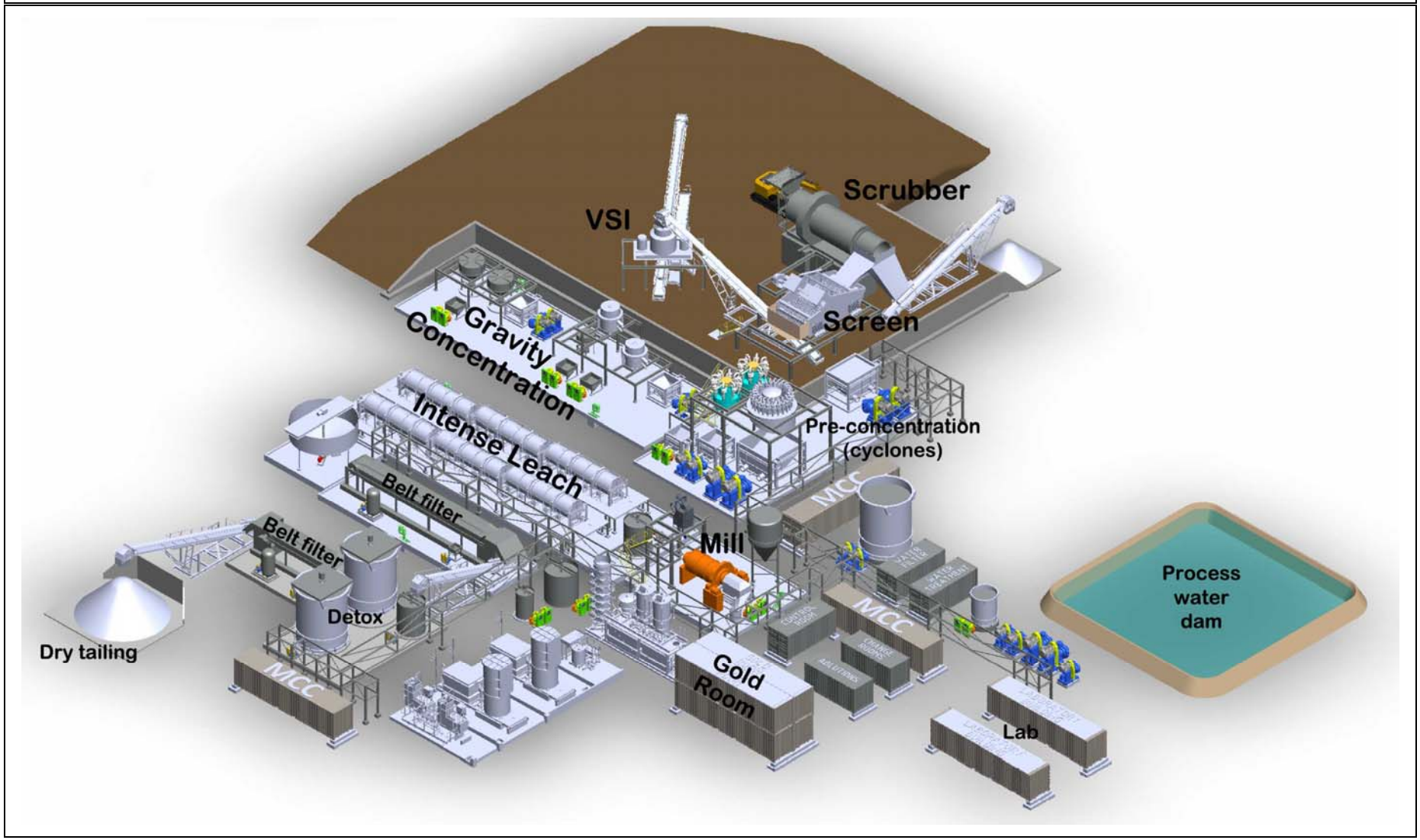
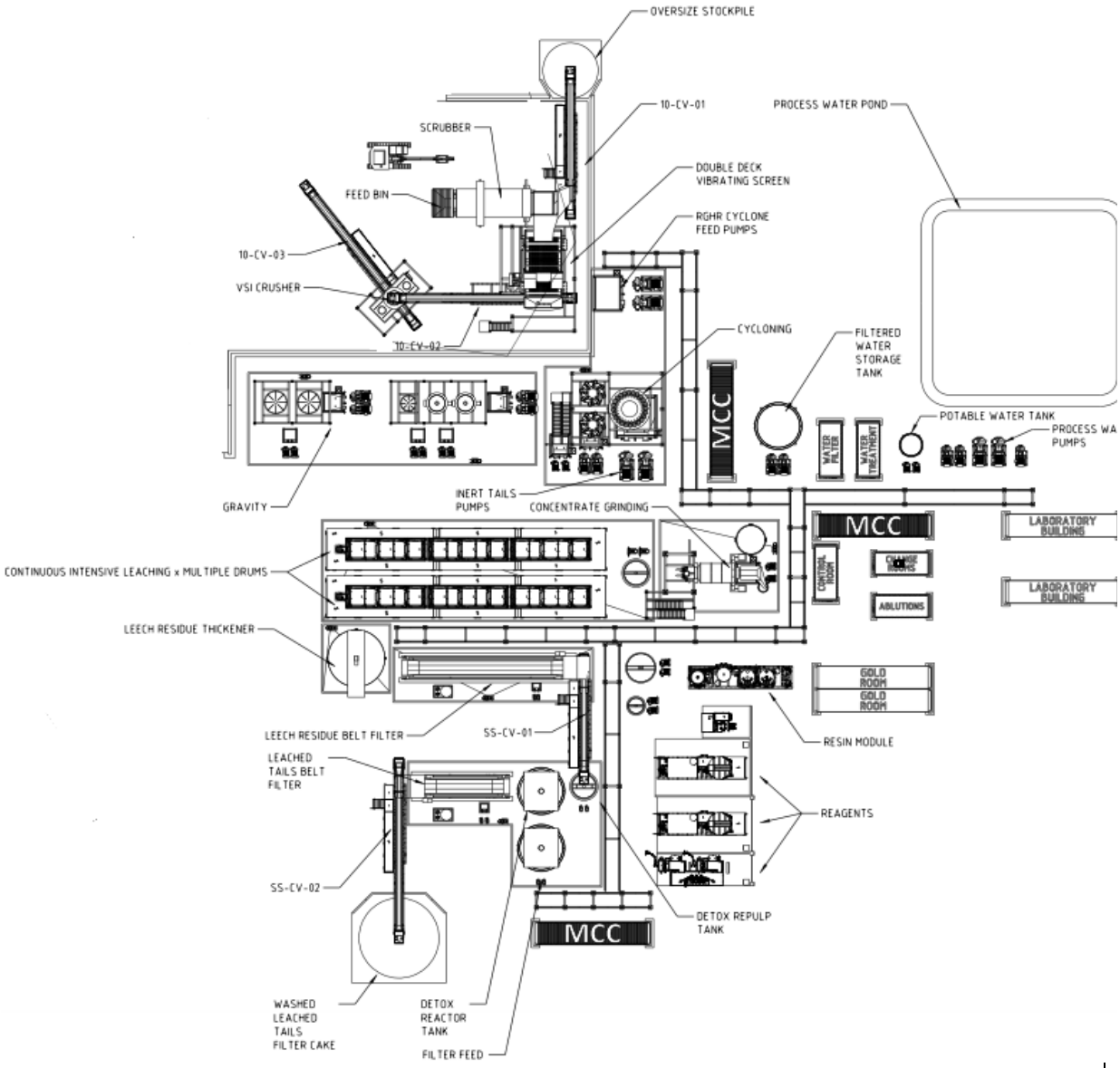


Figure 17.4

Plan of Process Plant



Source Gekkos January 2016

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The key Project infrastructure will include:

- Access roads;
- Bridge crossing of the Fié River;
- Two open pits;
- Waste rock facilities for mining spoil;
- Processing plant and power generation;
- Tailings storage facilities, with separate facilities for inert gravity tailings and dry tailings storage for material that has been leached;
- Plant and mine infrastructure;
- Accommodation camp with services; and
- Office and medical facilities.

The layout of the infrastructure proposed is included in Figure 18.4.

18.2 Road and Logistics

Material and consumables will be transported to site via the existing road that links Selingue to the Kobada Village. The road will be upgraded to accommodate larger vehicles and the river crossings will be upgraded to ensure that the site is accessible at all times. The road designs have observed the requirements of the relevant authorities, and AGG has the necessary permits to undertake all aspects of road upgrades and bridge construction.

The road will have an improved gravel surface finish with proper and adequate drainage structures, to ensure reliable passage throughout the year. Cost estimates of US$1.9 million were received from road construction contractors for the road upgrade.

An important part of the access upgrade is the construction of a new bridge across the Fie River. The alignment of the new bridge is adjacent to the old bridge and construction quotes have been received indicating at build cost of $1.2 million. The new alignment just to the south of the existing “container” bridge, which will be decommissioned.

Haul roads will be constructed across the site to connect the pits with waste rock facilities, the plant area and tailings storage facilities.

18.3 Accommodation Facilities

The accommodation facilities will be located at the entrance to the mine, adjacent the Foroko-Kobada Road, as shown in Figure 18.4.

The camp is planned to accommodate approximately 100 persons and will include catering and messing facilities and all the necessary ablution, sewage, water treatment and recreational facilities.

Construction costs of these facilities have been estimated at $1.38 million, including the administration office and security and medical facilities.

Many of unskilled and semi-skilled labour will be sourced from the local communities, reducing the accommodation needs significantly for the Project.

18.3.1 Administration Building

The main administration building will be located inside the security fence at the main accommodation camp and accessed through the security gates. It will be a building of approximately 400 m^2, providing offices for the Management, Mining and Processing staff.

The administration building will also have a spare office for visiting managers and a meeting room. The building would have a veranda on the north and south sides. The administration offices are to be fitted with split system refrigerated air conditioners throughout.

The building will include the main communications room for the site. This will include phone, V-sat and internet servers for the entire site.

18.3.2 Main Security and First Aid Building

The security building, which is approximately 50m^2 in internal area, will house the security personnel and the first aider or nurse. The building will straddle the security fence and be adjacent to the main boom gates for vehicle entry. The building will provide controlled access to the mine site.

One end of the building will have a veranda for an ambulance (not provided). The other end will be a breezeway for pedestrian access with turnstiles controlling access and egress. The centre of the building will house the security staff and the medical clinic.

The medical clinic includes a bed, toilet and storage areas. The security offices and clinic will be fitted with split system refrigerated air conditioners throughout.

18.4 Workshop and warehousing

The mining heavy vehicle workshop will be the main shop for maintenance and re- builds of mining equipment. The building design is capable of handling maintenance work for Volvo A40 trucks and support equipment.

A light vehicle service bays will be used for maintenance of all mine light vehicles, including those operating in the processing plant areas. The general engineering workshop will be located adjacent to the light vehicle service bays and will hold stores for servicing of the light vehicles and small mechanical equipment at the mine.

The tyre change workshop will include space for inflation equipment to inflate and repair heavy vehicle tyres.

A heavy vehicle wash bays will be provided for the maintenance of the mining fleet. One light vehicle wash bay will be provided for the maintenance of the light vehicle fleet.

A warehouse facility consisting of converted sea containers will be adjacent the workshop.

18.5 Power and Electrical

The total power requirements for the project are estimated at approximately 1.9 MW. It is planned to utilise diesel powered generators for power production and have a power station located at the process plant. The accommodation camp will have an independent power supply.

The cost of power was estimated to be approximately US$0.263 per kWhr.

18.6 Fuel

There will be diesel fuel facilities at each power station and a dedicated facility for the mining contractor, process plant and light vehicles. All fuel storage facilities will consist of self bunded tanks. Discussions with fuel suppliers have indicated that fuel storage will be provided by the fuel supplier as part of a supply contract.

18.7 Waste Rock Facilities

The waste rock facilities are discussed in Section 16.7.

18.8 ROM Stockpiles

A run of mine stockpile area of 70,000 m2 has been identified on the southern and western sides of the proposed processing plant location. Whilst trucks will dump directly in to the ROM bin to be fed directly to the plant, it was estimated that 30% or 3.7 million tonnes of ore will be rehandled from the ROM stockpiles.

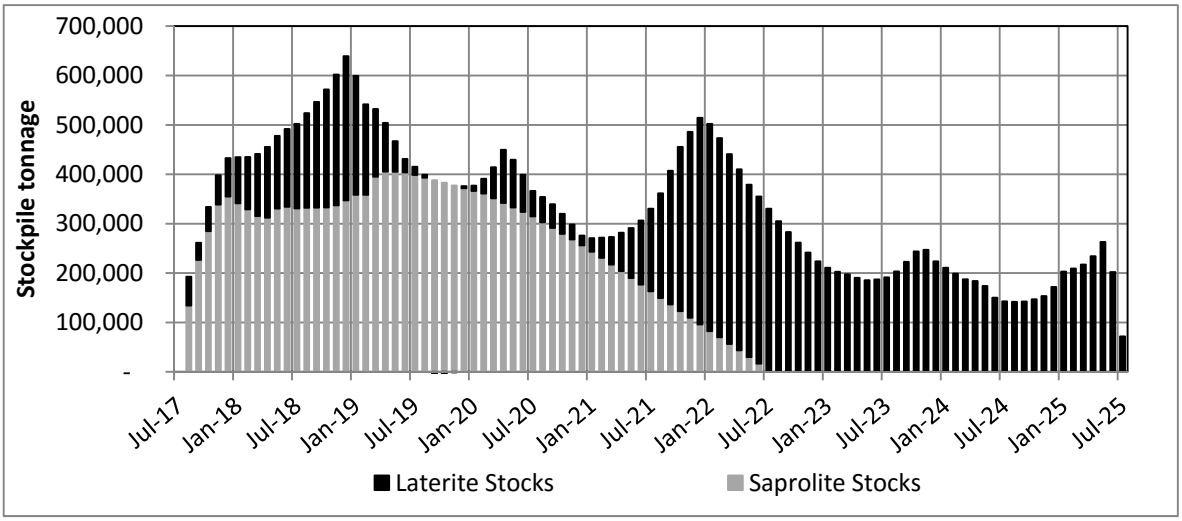
Stockpiling has only been used to manage the over production or underproduction of ore from the open pits, and to smooth ore supply to the processing plant. Blending of laterite material has also been included, so that laterite makes up a maximum of the process ore feed.

Figure 18.1 shows the total laterite and saprolite stockpiles, month on month, over the life of the project. Maximum stockpile capacity is around 639,000 tonnes in December 2018, comprising 347kt of laterite ore and 292kt of saprolite ore.

All laterite is ore mined prior to mid-2019, when this stockpile peaks at 400kt, from which time it is consistently depleted as 10% of the ore feed. The saprolite stockpile varies as the various pit stages change the quantity of waste required to be stripped.

Figure 18.1

Plan of Process Plant



18.9 Tailings Storage Facilities

It is proposed to construct two separate tailings storage facilities (TSF) for the project. One for inert tailings product from the gravity circuit and a second much smaller facility for the residue of the leaching process.

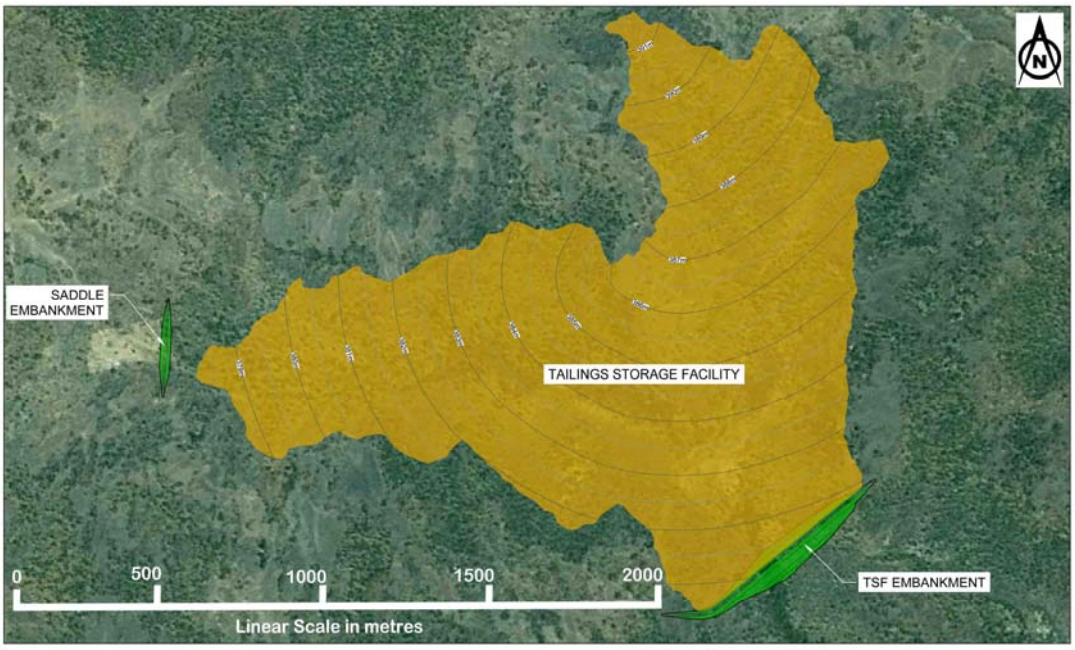
18.9.1 Main TSF

The residue from the gravity circuit will consist of slimes and sand will be pumped to a TSF located 2.3km to the south of the processing plant as shown in Figure 18.4. Figure 18.2 shows the detailed footprint of the TSF while a section through the main wall design is shown in Figure 18.3.

The TSF is designed as a valley impoundment dam with a compacted earth embankment wall. Full details of the design and geotechnical investigation of this wall are provided in Appendix C.

Figure 18.2

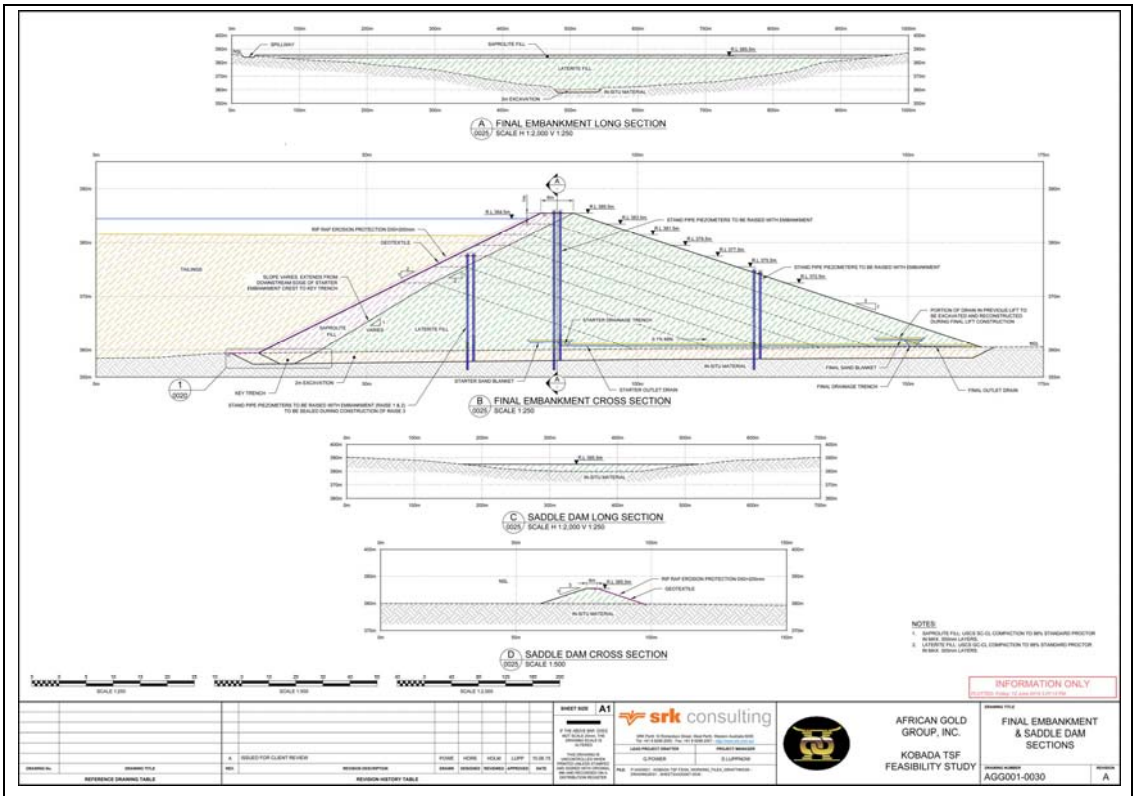
Main TSF layout



Source SRK November 2015

Figure 18.3

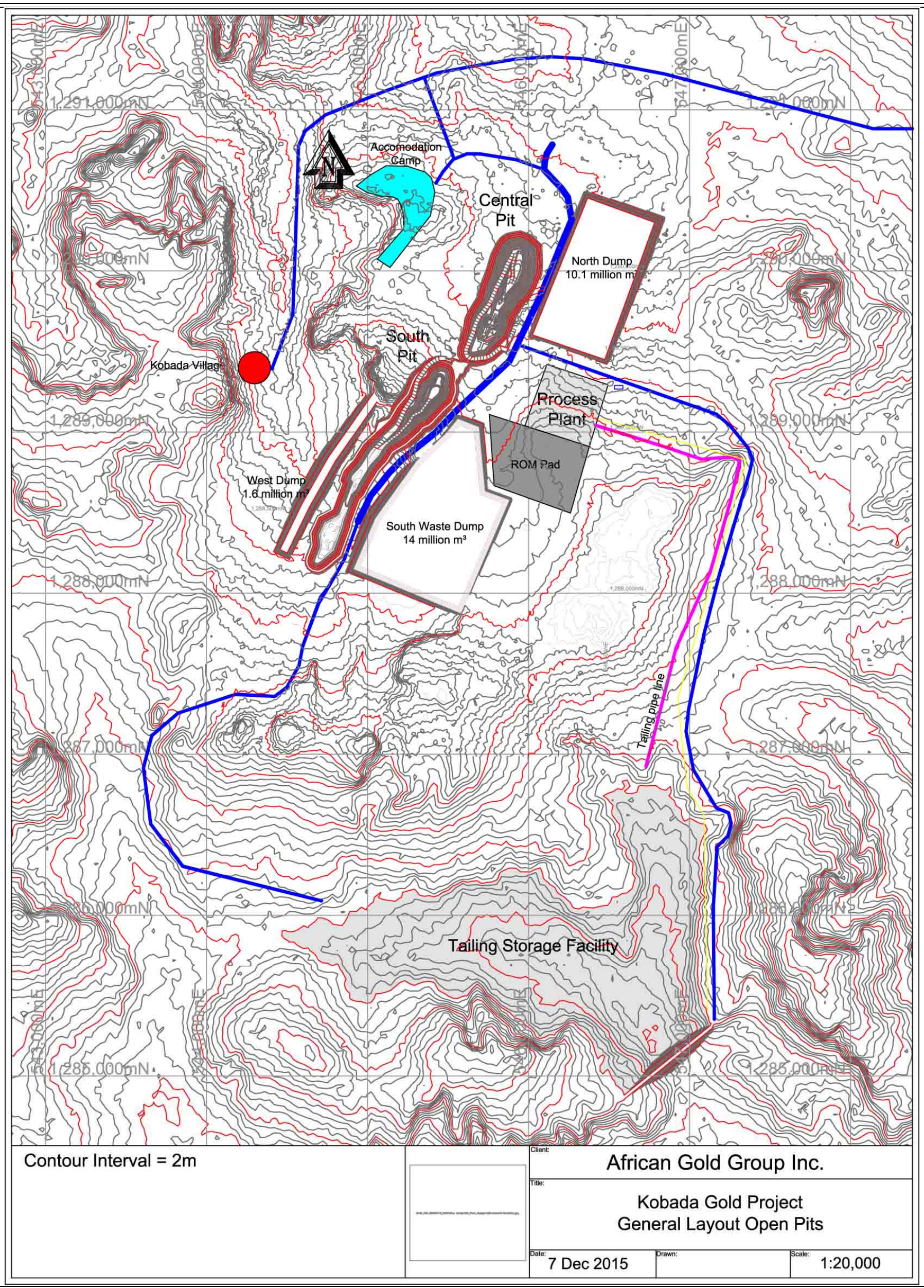
TSF Wall design



Source SRK November 2015

Figure 18.4

General Site Infrastructure Layout



18.9.2 Leach residue TSF

It is proposed to wash and filter the tailings material from the leaching circuit to ensure maximum recovery of gold. The filter cake will be processed through a cyanide detoxification circuit to

Detoxification will reduce weak acid dissociable (WAD) cyanide level in the leach plant tailings to below those recommended under the international Cyanide Code (<50 ppm). The dry tailings will then be disposed within the waste rock storage facilities.

18.9.3 Water Storage

The water balance for the processing plant and tailings storage facility was completed based on figures within SRK Consulting's TSF report (Appendix C). The water balance for processing is included in Appendix D. This calculation confirmed that there is a water shortfall in the dry season of 229 m^3/h, however in the wet season this is made up for by a 477 m^3/h excess. The excess water from the wet season would be stored in the TSF for use in the dry season.

Table 18.1
Reclaim Water Balance

	Process Water Usage (m^3/h)	TSF Reclaim Water (m^3/h)	Shortfall / Excess (m^3/h)
Dry Season	924	695	-229
Wet Season	924	1401	477
Average	**924**		**249**

Analysis of annual rainfall, catchment for the main TSF and process water requirements indicated that there was sufficient water from precipitation and water recycling to sufficient water for processing activities year round.

18.10 Construction Timetable

Project construction and commissioning is expected to be completed over an 18 month period as detailed in Table 18.2.

The critical aspect of this construction schedule is the construction of the bridge across the Fie River prior to the wet season. This will allow access to site regardless of river height. This will allow the balance of construction activity to proceed without delays, including the construction of the accommodation camp and other important site works. It will also ensure that mobilisation of the plant is not affected by high water levels in the Fie River.

Table 18.2

Construction Schedule

	Wet Season			Dry Season								Wet Season			
Item	Jul-16	Aug-16	Sep-16	Oct-16	Nov-16	Dec-16	Jan-17	Feb-17	Mar-17	Apr-17	May-17	Jun-17	Jul-17	Aug-17	Sep-17
Access road and Fie River Bridge															
Accomodation camp construction															
Inert TSF construction															
Process Plant Detailed Engineering															
Process Plant Manufacture															
Process Plant mobilisation															
Process Plant erection and commissioning															
Commencement of mining operations															

Manufacturing of the processing plant will occur at Gekko Systems manufacturing facilities in Ballarat, Australia. The modular nature of this plant means that many individual components will be the size of 40' shipping containers and therefore transportable by conventional truck and shipping. Only the scrubber, vibrating screen and rougher cyclone cluster will require break bulk freight. At site, construction will consist of foundations, water supply and the provision of other services.

Another important consideration is that of initial water supply. The construction of the Inert TSF is also included prior to the wet season. This will allow this facility to be initially utilised as a water storage dam which will supply process to the operation.

19 MARKET STUDIES AND CONTRACTS

No formal market studies have been undertaken.

The PEA envisages that gold produced at the mine would be shipped from site, with commensurate security arrangements, to a specialist refining company. There are a number of other operating gold mines near the Project that have well-established logistics for shipping gold from this part if Mali.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Environmental and Social Impact Assessment

The Environmental and Social Impact Assessment (ESIA) for the Project was commissioned by AGG and undertaken by independent consultants Environment and Social Development Company SARL during 2014. This document was submitted in December 2014 to support the application for the environmental permits. The ESIA is included in Appendix F.

20.2 Environmental Permitting

Environmental permits for the operation were granted on 2nd June 2015.

20.3 Mining License

The Exploitation License for the operations was granted on the 31st July 2015. Details of the mining license are provided in Appendix A.

20.4 Considerations of Social and Community Impacts

20.4.1 Project Setting

The Kobada gold project spans three municipalities, namely the Rural Municipalities of Nouga, Kaniogo, and Séléfougou. These municipalities are located in the Kangaba Circle and in the second administrative region of Mali, the Koulikoro region. According to a 2009 census there was 32,479 people in these municipalities.

There are 2 hamlets on the Kobada exploration license. These are the villages of Foroko and Kobada. In addition to there are 3 hamlets in the area of influence of the project (immediate periphery of the project). These hamlets are Kòdjouni, Nièouléni and Chakabougou.

Important observations relating to the social services and infrastructure in the area are listed below.

- Security: There are no police stations or associated facilities in close proximity to the proposed development;
- Water supply: Limited infrastructure is available to meet the current needs and requirements of the local community, and an increase in the number of people in the Project area is likely add pressure on the existing resources;

- Education and skills: Limited facilities are available to the local community and skills are also limited;
- Medical Facilities: Limited medical facilities are available in the area, the closest government facility is in Selengue.
- Local trade and commerce: Limited facilities are currently available for trade and commerce with the nearest centre being Selengue;
- Access Roads: the main road to Kobada from Selengue is in poor condition, and travel times across the 48km distance can take more than 2.5 hours.
- Recreational facilities and sport: Additional facilities will be required to cope with the number of employees and demographic to be introduced to the area as part of the project development;
- Food security and storage: agricultural development to date has been limited due to the relatively low reliance of the community on locally produced food;
- Financial institutions: Facilities such as banks and micro-lenders are currently not present in the area.

20.4.2 Local Employment Plan

AGG aims as far as practicable to employ Malian citizens, both male and female, to work at Kobada, and maximising the employment those from the local community. This will benefit both Mali, in terms of creating jobs and passing on skills.

20.4.3 Community Development Plan

A process of stakeholder consultation has commenced and will be conducted in parallel with the ESIA.

20.5 Mine Closure

Mine closure and reclamation will be performed in accordance with Malian regulations and guidelines. All buildings and facilities not identified for a post-mining use will be removed from the site during the salvage and site demolition phase. Mine closure costs have been estimated to be US$3.9 million. A detailed mine closure plan is to be developed for the project.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The estimated pre-production capital costs for an annual mine production of 1.6 Mtpa was estimated to be $45.4 million. Sustaining capital expenditures were estimated to be an additional US$36.7 as summarized in Table 21.1.

Table 21.1
Pre-Production and Sustaining Capital Cost Estimate

Item	Pre-production US$ millions	Sustaining US$ millions	Comment
Processing Plant	35.8	2.2	Gekko estimate based on preliminary design
Tailings Storage	3.3	9.6	SRK/AGG estimate based on final design and schedule of rates
Infrastructure	2.9		AGG and contractor estimates
Owners Costs	3.0		Mobilisation
General Contingencies	0.4		Excludes processing plant
Capitalised Waste Mining		21.0	Waste stripping in excess of average strip ratio
Closure Costs		4.0	Rehabilitation and closure plan
Grand Total	45.4	36.7	

21.1.1 Processing Plant

The breakdown of the capital estimate for the processing plant is detailed in Table 21.2. The estimate was prepared in January 2016 and has an estimation accuracy ranging from -5 to +25%.

The estimate is based on preliminary designs, and the cost associated with "approved for construction" design process was included in the estimate.

Table 21.2
Processing Plant Cost Estimate

Main Area	Cost (US$ million)
10 – Scrubber and Crusher	2.96
15 – Cyclone Pre-concentration	1.51
20 – Gravity Concentration	2.15
30 – Concentrate Grinding	0.82
50 – Leaching and Filtration	4.46
55 – Detoxification and Filtration	1.79
60 – Resin Absorption	1.23
70 – Gold Room	1.03
80 – Reagents	1.03
90,100 – Utilities	1.49

110 – Infrastructure	0.12
120 – MCC's, Electrical	3.86
Direct Subtotal	22.44
EP and Project Management	3.08
Shipping	1.61
Earthworks	0.46
Civils	0.43
Installation and Construction Management	5.84
Commissioning	1.98
Indirect Subtotal	13.39
Total Capital Cost*	35.83

Full details of this cost estimate are provided in Appendix D.

21.1.2 Tailings Storage Facility (TSF)

The tails storage facilities were designed by SRK consulting, and all earthwork quantities were taken from this design. Unit rates were sourced by AGG and provided by two earthwork contractors in West Africa.

The TSF study was completed for the mining schedules presented in the PEA published in December 2014. This schedule contemplated the processing of 24 million tonnes of material. The schedule presented in this report processes the Proved and Probable Mineral Reserve of 12.7 million tonnes. Allowance has been made to reduce the sustaining capital requirements from US$12.85 million to US$9.02 million, as detailed in Table 21.3. This is in line with the reduction in the final wall height from 384mRL to 381mRL or reducing the final wall height from 23 metres to 20 metres.

Table 21.3
Main TSF Cost Summary

Main Area	Initial Capital Cost (US$ millions)	Sustaining Capital Cost (US$ millions)
Preliminary and general	0.51	1.49
Earthworks – TSF embankment and basin	2.21	6.83
Embankment underdrainage	0.19	0.28
Additional embankment works	0.13	0.19
Supply and installation of monitoring equipment	0.01	0.02
Water management – diversion channel	-	0.22
Total (US$ millions)	**3.05**	**9.02**

A second TSF was design for the residue from the intense leach reactors. Whilst the leach circuit now includes cyanide destruction technology on the intense leach reactor (ILR) tailings stream. This means that lining of the ILR TSF should not be required, however this cost has

been kept to ensure the highest environmental standards are maintained. Table 21.7 details the capital requirements for the ILR TSF.

Table 21.4
ILR TSF Cost Summary

Main Area	Initial Capital Cost (US$ millions)	Sustaining Capital Cost (US$ millions)
Preliminary and general	0.04	.09
Earthworks – TSF embankment and basin	0.13	.10
Facility Lining	0.07	.35
Total (US$ millions)	**0.24**	**0.54**

Full details of the estimate are provided in Appendix C.

21.1.3 Access Infrastructure

Access infrastructure requirements were provided by AGG personnel and the costs were quoted by Enterprise Générale Traore & Freres SARL (EGTF), a Malian road and general construction contractor. The breakdown in these costs is detailed in Table 21.5.

Table 21.5
ILR TSF Cost Summary

Item	Initial Capital Cost (US$ millions)	Comment
Fie River Bridge	1.01	Permanent concrete pier bridge
Access road – section 1	0.18	Selingue to Selefougou
Access road – section 2	1.06	Selefougou to Fie River
Access road – section 3	0.65	Fie River to Kobada
Total (US$ millions)	**2.90**	

Details of design and cost estimates are provided in Appendix E.

21.1.4 Owners Costs

A number of items are included in the broad heading of owner's costs. Table 21.10 summarises these costs

Table 21.6
Owners Costs

Item	Initial Capital Cost (US$ thousands)	Comment
Accommodation buildings	520	Based on 6 room buildings with twin share accommodation for employees, and single for staff.

Table 21.6 Owners Costs		
Item	Initial Capital Cost (US$ thousands)	Comment
Kitchen, dining, laundry, medical centre	430	Kitchen and dining already constructed, waiting fit out.
Offices	210	Mine offices and security building
Services	220	Power, sewage and water supply
Sub total	**1,380**	**Camp and offices**
Processing - crane	99	20% down on equipment finance
Processing – plant excavator	48	20% down on equipment finance
First fills and spares	116	
Other	60	Security systems and training
Sub total	**323**	**Processing**
Maintenance workshop	100	
Security fencing	400	Fencing of entire mine area
Computer hardware and software	50	
Light vehicles	152	This represents 20% down on equipment lease finance for light vehicles and light trucks
Sub total	**702**	**General and Admin**
Mobilisation of mining contractor	600	
Grand Total	**3,005**	

21.1.5 Mining Capital Cost

The mineralization at Kobada outcrops at surface, and there is no pre-stripping requirement for the mining operation. Ore production occurs immediately from the start of mining. AGG plans to utilize a mining contractor for the entire life of mine operation and US$600,000 has been allowed for contractor mobilization.

During mining of the pits, part of the waste stripping for the ultimate pit limit has been capitalized. The average strip ratio for the life of mine is 3.28 to 1. In any month where the strip ratio exceeds 3.28 to 1, the additional waste strip was capitalized. That is waste mined up to a strip ratio of 3.28 to 1 is considered an operating cost and expensed. Waste in excess of 3.28 to 1 is capitalized and amortized over the remaining gold production.

US$21 million of waste mining was capitalized from November 2018 until July 2021.

21.1.6 Closure Capital Costs

Closure costs have been estimated as $4.0 million. This is based on an estimate of $8,000 per hectare of surface disturbance, plus allowances for pit bunding, building removal and a 10% contingency.

No allowances have been made for salvage value of plant and equipment.

21.2 Operating Cost Estimates

21.2.1 Processing

Processing costs for Kobada were estimated to be $6.63 per tonne of ore processed, as detailed in Table 21.7.

Table 21.7

Summary of Processing Costs

Item	US$/t
Utilities (Power and Water)	2.46
Labour and Administration	0.56
Reagents	0.46
Consumables	0.82
Maintenance	1.16
Laboratory	0.50
Mobile Fleet	0.60
Total	6.55

A full breakdown of the operating cost estimate by Gekkos is provided in Appendix D. Additional allowances were made of the inclusion operating and ownership of the equipment listed in Table 21.8.

Table 21.8

Mobile fleet for processing

Item	Number of units	Operating Cost US$/hr	Ownership Cost US$/month	Total Cost* US$/t ore
Cat 318 excavator	1	55.86	3,500	0.29
LH580 wheel loader	1	73.81	4,200	0.13
CAT TH62 tele handler	1	10.00	1,129	0.02
Grove RT765E-2 Crane	1	20.00	5,478	0.05
Toyota Troop Carrier	1	13.78	833	0.03
Toyota Landcruiser utility	2	13.78	833	0.05
Isuzu NPS300 Dual Cab with Hiab	1	13.78	1,027	0.03
Diesel welders & plant	1	20.40	333	0.01
Total			19,144	0.60

*Total cost is the life of mine average over 12.7Mt of ore processed.

Operating costs inclusive fuel and lubricants, ground engaging tools (where applicable), tyres, maintenance and major component replacement. These operating costs are exclusive of labour.

The ownership cost assumes the purchase price of mobile plant and equipment is spread over the useful life of the machine and assumes that all equipment is financed. Interest fees bank charges and insurances have been allocated under G&A costs, and have been assumed to be 12% of the purchase price of each piece of plant or mobile fleet.

21.2.2 Mining

It was assumed that mining will be undertaken as a comprehensive contract. Total mining costs are summarized in Table 21.9 and average US$2.35 over the life of mine.

Table 21.9
Mining Cost Summary

Item	US$ million	US$/t mined	US$/t ore
Technical services	6.6	0.12	0.52
Grade control drilling	14.2	0.26	1.11
Drill and Blast	0.5	0.01	0.04
Load and haul	63.9	1.17	5.02
Dayworks and subcontracting	43.0	0.79	3.38
Total Mining Costs	128.2	2.35	10.07

NB: This includes US$21 million of waste stripping expense, which was capitalized in the cash flow calculations in Section 22.

Indicative mining costs were obtained from two independent contractors and these were incorporated into a first principles estimate to ensure that the estimate reflected the variability in the mining schedule. The estimate contemplated the detail of the mining schedule, with material destinations and haulage distances (ramp and horizontal) factored into mining productivity estimates.

It was assumed that mining was undertaken on a continuous 24/7 roster, with 12 hour shifts and 3 crews working two week rosters with a break of one week. The rationale for this roster combination was to ensure that the operation can attract suitably qualified personnel from outside the local area whilst training people from local villages. While it was assumed that this roster continues for the life of the mine, a more productive roster options may be implemented once the mine has been established and the development of skills in the local area has been effective.

Technical Services

Technical services costs were estimated to average US$69,000 per month, with salaries and wages accounting for $49,000 per month. Total staff includes a Mining Manager, Chief Geologist, mining engineer. Two geologists, four geology assistants, a chief surveyor and two junior surveyors for a total head count of 12.

Grade Control

Expenditure on grade control drilling and assaying is based on a 10m x 10m drill pattern. Over the life of mine the average monthly expenditure was US$149,000 per month, with grade control costs equivalent to US$1.11 per tonne of ore mined.

Drill and Blast

The only material hard enough to require drill and blasting techniques is the laterite rock type, particularly when iron (Fe) content is very high. This usually only occurs at the top of the laterite horizon. An allowance of US$0.60 per tonne was made for drill and blast activities through this material. Laterite only makes up around 2% of the total material movement, and consequently drill and blast activities only account for US$0.01 per tonne of material mined.

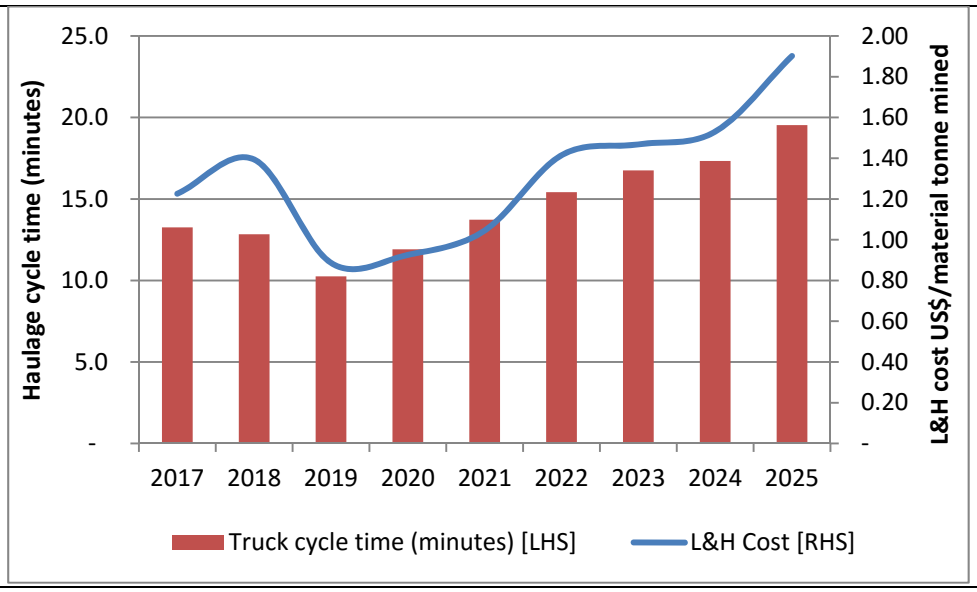
The saprolite ore type is free digging and does not require any blasting.

Load and Haul

Loading and haulage (L&H) accounts for an average of US$1.17 per tonne of material mined. Over the life of the project, L&H costs vary depending on the staged pit being mined, depth and haulage distance to waste dumps and the ROM pad. Figure 21.1 shows the L&H cost per tonne of material mined against the average annual truck cycle time and illustrates a general trend of increasing cycle time (and haulage distance) over time and a corresponding increase in load and haul costs over time.

Figure 21.1

Truck cycle time and load and haul cost

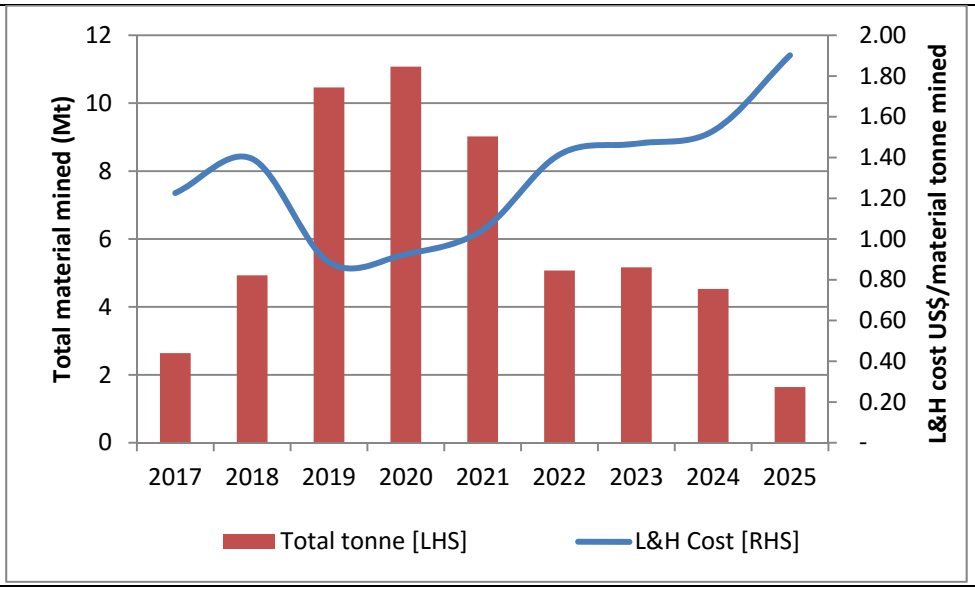


The other important factor to consider is the monthly mining rate and the impact that this has on unit costs. Figure 21.2 shows L&H costs against the annual total material movement. There is a clear correlation between periods of higher movement and lower cost. This is ex-

pected since higher over movement generally relates to improved efficiency, and fixed cost items (like road maintenance) are expensed over a greater tonnage movement. This results in lower unit costs.

Figure 21.2

Total material movement and load and haul cost



Dayworks and Subcontracting

An allowance of US$0.50 per tonne of ore was made for general day works which allows for activities like pit dewatering and ground control. Similarly, and allowance of US$0.60 per tonne of material mined was made for subcontracted services.

21.2.3 General and Administrative

Total G&A costs were estimated to be $3.54 per tonne of ore mined. The breakdown of this cost estimate is detailed in Table 21.10.

Key areas of expenditure include general administration which incorporates the General Manager, accounting functions, bank and finance charges, telecommunications, vehicles and permitting expenses.

Expenditures under community relations included an allowance of 1.5% of gross revenue from gold sales to fund the Company's obligations under the Community Development Plan, which was established in conjunction with the mining license and environmental permits. These amounts are exclusive of $400,000 per annum to fund the operations of a medical clinic that will service the needs of the operation, but also be available to the local village.

G&A charges also include the operating expenditures associated with the accommodation village.

Table 21.10 General and Administrative Costs		
Item	**US$ per annum**	**US$/t ore**
General Administration	1.4	0.90
Community Relations	0.9	0.59
Emergency Response	0.1	0.05
Safety, Health & Environment	0.2	0.11
Medical Clinic	0.4	0.23
Security	0.6	0.38
Warehouse & Logistics	0.2	0.12
Training	0.1	0.08
Accommodation	1.7	1.06
Total	**5.6**	**3.54**

22 ECONOMIC ANALYSIS

22.1 Caution and Forward-Looking Information Statements

The results of the economic analysis discussed in this section represent forward- looking information as defined under Canadian securities law. Actual results may differ materially from those expressed or implied by forward-looking information.

Forward-looking statements include, but are not limited to, statements with respect to the economic and feasibility parameters of the Kobada Project: the cost and timing of the development of the Project; the proposed mine plan and mining method, stripping ratio, processing method and rates and production rates; grades; projected metallurgical recovery rates; infrastructure, capital, operating and sustaining costs; the projected life of mine and other expected attributes of the Kobada Project; the net present value (NPV) and internal rate of return (IRR) and payback period of capital; cash costs and all-in sustaining costs; the success and continuation of exploration activities; estimates of Mineral Resources; the future price of gold; the timing of the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations that may be assumed; requirements for additional capital; environmental risks; political risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

- There being no signification disruptions affecting the future development and operation of the Project;
- The exchange rate assumptions for the foreign currencies in which quotes were provided and U.S. dollar being approximately consistent with the assumptions in the study;
- The availability of certain consumables and services and the prices for diesel, fuel oil, reagents, electricity and other key supplies being approximately consistent with assumptions in the study;
- Labour and materials costs being approximately consistent with assumptions in the study;
- Permitting and arrangements with local peoples being consistent with current expectations;
- The timelines for exploration and development activities on the Project;

- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant government bodies and other relevant stakeholders within the expected timelines;
- Certain tax rates, including the allocation of certain tax attributes to the Project;
- Assumptions on ownership may change when the new operating company has been formed and the convention has been negotiated;
- The availability of financing for AGG's development activities;
- Assumptions made in Mineral Resource estimates, including current mineral title being maintained; geological interpretation, grade, recovery rates, gold price assumption, assumptions used in assessment of reasonable prospects of economic extraction; and general business and economic conditions.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by forward-looking statements. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the feasibility study and economic parameters discussed herein not being realized; decrease of future gold prices; cost of labour, supplies, fuel and equipment rising; actual results of current exploration; adverse changes in Project parameters; discrepancies between actual and estimated production, Mineral Resources and recoveries; exchange rate fluctuations; delays and costs inherent in consulting and accommodating rights of local peoples; title risks; regulatory risks and political or economic developments in Mali; changes to tax rates; risks and uncertainties with respect to obtaining necessary surface rights and permits or delays in obtaining same; risks associated with maintaining and renewing permits and complying with permitting requirements; and other risks involved in the gold exploration and development industry; as well as those risk factors discussed elsewhere in this Report.

22.2 Key Economic Assumptions

The economic analysis for the project has been completed based on the following key assumptions:

- Gold price US$1,200 per ounce;
- 35% corporate tax rate, with accelerated depreciation;
- 3% gold production royalty;
- Project ownership 90% AGG and 10% Malian Government; and
- Exchange rate: USD1 = XAF612.

The base case assumptions used in the financial model are listed in Table 22.1.

Table 22.1 Financial Model Assumptions		
Parameter	**Units**	**Value**
Gold price	US$/troy ounce	$1,200
Discount rate	%	5.0%
Corporate tax rate	%	35.0%
Government royalty	%	3%
Mining costs	Ave $/t mined	$2.35
Processing cost	$/t processed	$6.55
G&A costs	$/t processed	$3.54
Installation capital	US$MM	$45.4
Sustaining Capital	US$MM	$36.7
Salvage value	US$MM	$0 (end of mine life)
Closure costs	US$MM	$3.9 (end of mine life)

22.3 Base Case Financial Model

Table 22.2 summarises the main economic parameters from the cash flow model. The Project free cash flow was estimated to be US$121.5 million and net present value at 5% discount rate was US$86 million. AGG's 90% equity distribution was US$109.2 million that had a $NPV_{5\%}$ of US$77 million. The Governments free carried distribution was US$12.3 million.

Table 22.2 Financial Model Summary			
Item	**Project**	**AGG 90% equity distribution**	**Gov. 10% equity distribution**
Operating Cashflow (US$MM)	$121.5	$109.2	$12.3
NPV @ 5.0% (US$MM)	$86.1	$77.1	$9.0
NPV @ 10.0% (US$MM)	$60.8	$54.1	$6.7
Internal Rate of Return	43%	41%	N/A
Payback Period (years)	2.6		
Maximum negative equity (US$MM)	$43.9		

The payback period for the project was 2.6 years from initial capital expenditures in July 2016, until positive cumulative cash flow in January 2019. This assumes that capital expenditures commence in July 2016 and commissioning is achieved in September 2017 with strong cash flows from this time.

Figure 22.1 illustrates the capital expenditures, cash flow and cumulative cash flow for the operating company over the life of the project.

Figure 22.1

Project Cash Flow and Cumulative Cash Flow

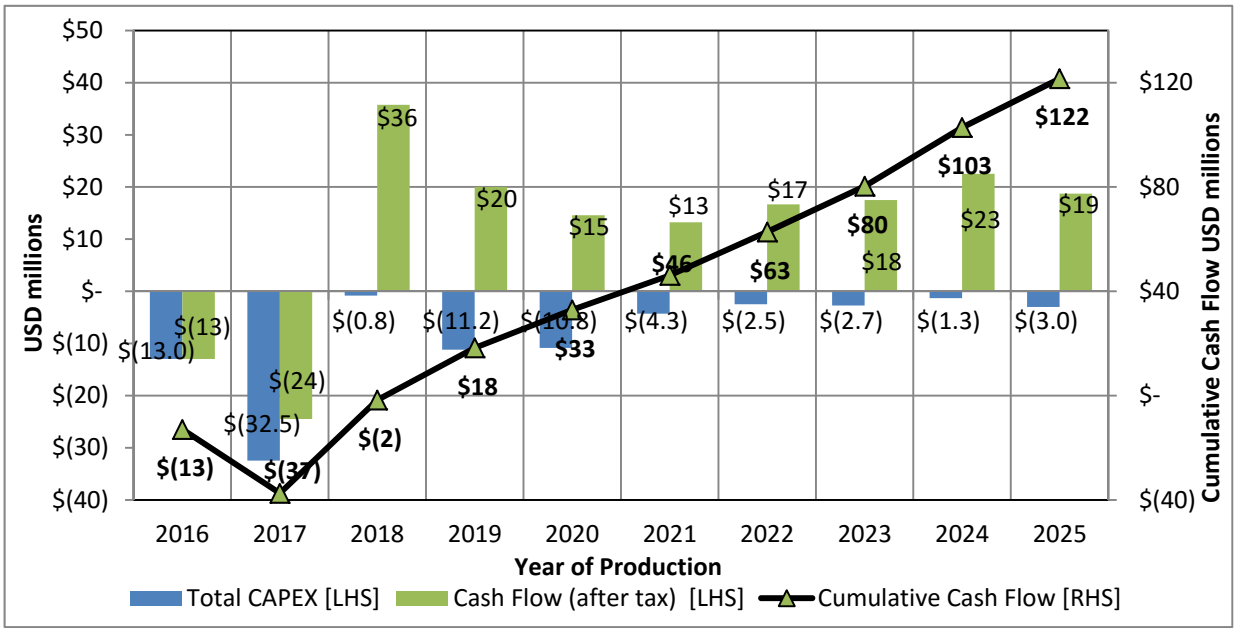


Table 22.3 and Table 22.4 provide details of the annual Income Statement and Cash Flow Calculation over the life of the project.

22.3.1 Royalties

Cash flows include provision for a 3% gross revenue royalty.

22.3.2 Taxation

Cash flows include provisions for Malian corporate taxation. Tax is levied on taxable profit at a rate of 35%, and is subject to accelerated depreciation of capital expenditures.

Amortization includes the US$45.4 million in preproduction capital, US$36.7 million of sustaining capital expenditures (including US$21 million in capitalized waste stripping). Exploration costs totaling US$27 million have also been amortized over the first three years of gold production.

Total taxation was estimated to be $52 million over the life of the project.

Table 22.3

Production Summary and Income Statement

Production		Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Ore Mined	000t	12,735		951	1,807	1,338	1,505	1,842	1,311	1,598	1,564	819
Gold Grade	g/t	1.25		1.49	1.34	1.22	1.18	1.18	1.15	1.19	1.25	1.36
Contained Gold	000oz	511		46	78	52	57	70	49	61	63	36
Waste Mined	000t	41,811		1,689	3,125	9,125	9,572	7,179	3,763	3,570	2,964	824
Total Tonnes Mined	000t	54,547		2,640	4,932	10,463	11,077	9,021	5,073	5,168	4,528	1,643
Total BCM Mined	000 BCM	26,847		1,212	2,379	5,058	5,503	4,508	2,530	2,581	2,260	815
Strip Ratio	W:O	3.28		1.78	1.73	6.82	6.36	3.90	2.87	2.23	1.89	1.01
Tonnes Processed	000t	12,735		518	1,601	1,601	1,605	1,601	1,601	1,601	1,605	1,002
Contained Gold	000oz	511		25	72	64	61	61	60	61	64	43
Recovery	%	82.2%		85%	84%	82%	81%	81%	81%	81%	82%	83%
Gold Recovered	000oz	420.460		22	60	53	50	49	48	50	53	36
Gold Price	US$/oz			1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200

Income Statement		Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Gold revenue	$MM	505	-	25.9	72.3	63.7	59.9	59.2	57.7	59.5	63.3	43.0
Mali Government Royalty	$MM	(15)	-	(0.8)	(2.2)	(1.9)	(1.8)	(1.8)	(1.7)	(1.8)	(1.9)	(1.3)
Net Revenue	**$MM**	**489**	**-**	**25.1**	**70.1**	**61.8**	**58.1**	**57.4**	**56.0**	**57.7**	**61.4**	**41.7**
Mining Costs	$MM	(107)	-	(7.2)	(12.9)	(11.2)	(13.0)	(17.2)	(13.4)	(14.1)	(12.8)	(5.3)
Processing Costs (incl. transport & refini	$MM	(83)	-	(3.5)	(10.4)	(10.5)	(10.5)	(10.4)	(10.5)	(10.4)	(10.4)	(6.9)
Site General & Administrative	$MM	(44)	-	(2.5)	(5.2)	(5.6)	(5.6)	(5.5)	(5.3)	(5.4)	(5.3)	(3.2)
Cash Operating Costs	**$MM**	**(234)**	**-**	**(13.1)**	**(28.5)**	**(27.3)**	**(29.1)**	**(33.1)**	**(29.2)**	**(29.9)**	**(28.5)**	**(15.4)**
Operating Margin - before depreciation	**$MM**	**255**	**-**	**12.0**	**41.6**	**34.6**	**29.0**	**24.3**	**26.8**	**27.8**	**32.8**	**26.3**
Depreciation	$MM	(82)	-	(5.8)	(16.2)	(15.6)	(12.3)	(4.6)	(5.2)	(6.3)	(7.5)	(8.7)
Debt interest and fees	$MM	(2)	-	(0.1)	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)	(0.1)
Income before Tax	**$MM**	**172**	**-**	**6.1**	**25.2**	**18.8**	**16.6**	**19.5**	**21.4**	**21.3**	**25.1**	**17.5**
Mali Tax	$MM	(52)	-	(1.9)	(5.4)	(3.6)	(3.9)	(6.8)	(7.5)	(7.5)	(8.8)	(6.1)
Other Tax	$MM	-	-	-	-	-	-	-	-	-	-	-
After Tax Net Income	**$MM**	**120**	**-**	**4.2**	**19.7**	**15.2**	**12.7**	**12.7**	**13.9**	**13.8**	**16.3**	**11.4**

Table 22.4

Cash Flow Calculation

Cash Flow Calculation		Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Net Income	$MM	120	-	4.2	19.7	15.2	12.7	12.7	13.9	13.8	16.3	11.4
Add back: Depreciation	$MM	82	-	5.8	16.2	15.6	12.3	4.6	5.2	6.3	7.5	8.7
Add back: debt interest and fees	$MM	2	-	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1
Total Adjustments	**$MM**	**84**	**-**	**5.9**	**16.4**	**15.7**	**12.5**	**4.8**	**5.4**	**6.5**	**7.7**	**8.8**
Accounts Recievable	$MM	-	-	(3.4)	0.5	0.2	0.3	(0.1)	0.0	(0.1)	(0.2)	2.7
Accounts Payable	$MM	-	-	1.2	(0.0)	0.1	0.0	0.1	(0.2)	0.0	(0.0)	(1.2)
Total Change in Working Capital	**$MM**	0	**-**	**(2.1)**	**0.4**	**0.2**	**0.3**	**0.0**	**(0.1)**	**(0.1)**	**(0.2)**	**1.5**
Initial Capex	$MM	(45)	(13.0)	(32.5)	-	-	-	-	-	-	-	-
Sustaining Capex	$MM	(16)	-	-	(0.1)	(1.8)	(2.0)	(2.2)	(2.5)	(2.7)	(1.3)	(3.0)
Capitalised waste mining	$MM	(21)	-	-	(0.7)	(9.4)	(8.8)	(2.0)	-	-	-	-
Working capital	$MM	-	-	(2.1)	0.4	0.2	0.3	0.0	(0.1)	(0.1)	(0.2)	1.5
Free Cash Flow	**$MM**	**122**	**(13.0)**	**(24.5)**	**35.7**	**20.0**	**14.6**	**13.2**	**16.7**	**17.5**	**22.5**	**18.7**
Equity Distribution / Funding	$MM	122	(13.0)	(24.5)	35.7	20.0	14.6	13.2	16.7	17.5	22.5	18.7
Government Free Carry	$MM	12	-	-	-	2.0	1.5	1.3	1.7	1.8	2.3	1.9
AGG Net Equity Distribution	**$MM**	**109**	**(13.0)**	**(24.5)**	**35.7**	**18.0**	**13.1**	**11.9**	**15.0**	**15.8**	**20.3**	**16.9**

22.4 Sensitivity Analysis

Examination of the Project's cash flow to the key variable of gold price illustrates the robust nature of the project. Table 22.5 details the undiscounted cash flow, discounted cash flow, internal rate of return and payback period at varying gold prices.

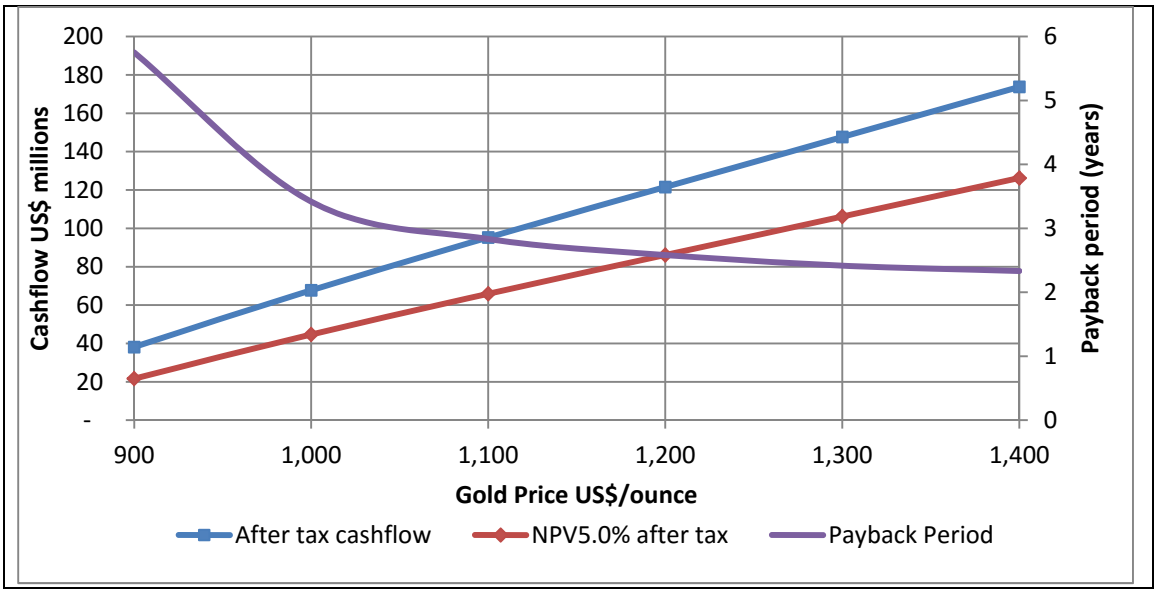
At the base gold price of US$1,200 per ounce, the Kobada Project exhibits an NPV of US$86 million, at a discount rate of 5%. The internal rate of return (IRR) was 43% and payback of capital expenditures achieved in 2.6 years. The NPV and IRR are based on the project's free cash flow which is reported net of all costs and Malian taxes.

Table 22.5
Project Sensitivity to Gold Price

Gold Price	US$/oz	900	1,000	1,100	1,200	1,300	1,400
After tax cashflow	US$MM	38	68	95	122	148	174
$NPV_{5.0\%}$ after tax	US$MM	22	45	66	86	106	126
Internal Rate of Return	%	17%	28%	36%	43%	49%	55%
Payback Period	years	5.8	3.4	2.8	2.6	2.4	2.3

The project is reasonably robust to decreasing gold price, and it is only at a gold price less than US$1,000 per ounce that the IRR falls below 20%. The payback period at US$1,000 per ounce was 3.4 years, which was 10 months longer than at the base price of US$1,200 /oz. At higher gold price there is significant increase in the Project's NPV, although the payback period only improves incrementally. At US$1,400 /oz the $NPV_{5\%}$ was US$126 million, of which US$113 million is attributable to AGG.

Figure 22.2
Project Sensitivity to Gold Price



23 ADJACENT PROPERTIES

This section is not relevant to the Report.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to the Report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resource

- Large laterite plateaus cover most of the area. The underlying saprolite is exposed below the plateau boundaries and is generally of a yellowish ochre colour, whitened in numerous places by intense kaolinization, likely of hydrothermal origin.
- Gold mineralization is associated with narrow, irregular, high-angle quartz veins and with disseminated sulphides in the wall rock and vein selvages. Mineralization occurs as free gold. Sulphides present include arsenopyrite, pyrite and very rare chalcopyrite.
- The Mineral Resource for the Kobada Gold Project has been estimated using Multiple Indicator Kriging ('MIK') for gold. This estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralization, and the style and geometry of mineralization. The estimation was constrained with geological and mineralization interpretations.
- The resource estimation was based on the available exploration drillhole database which was compiled in-house by AGG. The database has been reviewed and validated prior to commencing the resource estimation study.
- To establish appropriate grade continuity, the mineralization model was based upon a nominal 0.3ppm Au indicator mineralization shell estimated using 5m unconstrained down-hole composites. This interpretation is designed to capture the broad mineralization halo that encompasses the geological vein system and is not intended to constrain individual veins or vein clusters. As the grade estimation technique is MIK with change of support technique, this type of mineralization constraint is deemed appropriate.
- The resource estimate for the Kobada deposit has been classified as Measured, Indicated and Inferred Resources based on the confidence levels of the key criteria. Applying these confidence levels, resource classification codes were assigned to the block model using the following criteria:
    - Measured Resources
        - Blocks are predominately estimation pass 1

    - Distance to nearest data of 25m or less (drill section spacing 25m)
    - Minimum of 24 data used (from at least 3 drillholes)
    - Only in areas of optimal drilling orientation (orientation approx. 290°)
- Indicated Resources
    - Blocks are predominately estimation pass 1
    - Distance to nearest data of 25m or less
    - Minimum of 18 data used (from at least 3 drillholes)
    - Drillhole section spacing predominantly 25m.
    - Only in areas of optimal drilling orientation (orientation approx. 290°)
- Inferred Resources
    - Blocks are predominantly estimation pass 2.
    - In areas of sub-optimal drilling orientation (orientation approx. 200°)

- It is considered that AGG company procedures are sufficient to ensure the integrity and validity of the samples taken and that sample preparation and analytical procedures employed are industry standard. It is therefore considered that the data supplied is suitable and adequate for the purposes of resource estimation.

25.2 Mining Methods

- The mining operation is considered simple. The saprolite is free digging, and it is anticipated that only the minimum use of explosives will be required.
- This is a bulk mining method, and whilst the equipment size provides the ability to be selective, generally mining will focus on broad mineralization with grade confirmation from grade control drilling and proper production reconciliation techniques.
- It is planned to use a Malian mining contactor for all mining works. Mining costs were estimated to average US$2.35 over the life of the project. The value of the mining contract is estimated to be around US$107 million over 8 years.
- The Company will provide the management and technical staff to support the mining operation.
- Mining operations will be scheduled continuously year round.

25.3 Recovery Methods

- The main constituent minerals are: SiO_2 (69%), Al_2O_3 (11%) and Fe_2O_3 (12%)
- Gold is generally relatively coarse and fully or partially liberated which allows for high gravity recovery potential.
- The Saprolite gold distribution is much coarser than the particle size distribution, showing potential for preconcentration of the gold value by desliming using cy-

clones. This characteristic is not shown by the Laterite sample tested, as the gold distribution is slightly finer than the particle size distribution.

- Wet scrubbing was found to be superior to dry scrubbing
- Screening following scrubbing (at 1-2mm) will result in the concentration of 50% of the gold values into the oversize fraction, which made up 10 – 30% of the feed mass. This characteristic can be exploited, by crushing and then bypassing this fraction to the gravity concentration circuit
- Hydrocycloning can reject up to 70% of the Saprolite ore feed mass following the scrubbing stage. Gold recoveries to the product stream (cyclone underflow) averaged 94%.
- Gravity concentration was effective in recovering gold from the cyclone underflow, with recoveries improving as the mass yield to concentrate increased. Gravity tails grades of between 0.15 to 0.23 g/t Gold were recorded across all tests on the Saprolite at a mass recovery of 5%. Laterite tail grades were 0.25g/t Au at a 1.18mm crush however they increased to 0.7 g/t Au at 2.0 mm crush size. This reflects incomplete liberation at the coarser crushing size. As a result, the plant design is using a 1.18mm crush size.
- Gold leaching recoveries to solution were high, and leach times can be reduced to <10 hours by grinding down to a P80 of 125 µm.
- The ground leached residue proved to be very amenable to belt filtering and washing.

25.4 Economic Analysis

The Project displays a positive economic outcome at gold prices above US$1,000 per ounce. This outcome is largely a result of the low processing costs associated with the plant design selected, which was estimated at US$6.55 per tonne processed.

The key component in ore processing is the pre-concentration followed by gravity concentration which effectively can achieve up to 95% rejection of the feed mass while recovering more than 82% of the gold. It is this attribute that rejects barren material with relatively low energy inputs. Only the concentrate is ground to less than 180µm and leached, which is the energy intensive part of the ore processing flowsheet. This is the reason the processing cost per tonne is comparatively low compared to traditional CIP/CIL operations.

Mining costs of US$2.35 per tonne mined and G&A costs of US$3.54 per tonne of ore are considered to be of a similar quantum to other oxide projects of similar scale. There is an opportunity to improve the economic outcomes through lowering the waste to ore stripping ratio during the first half of the mine life, although this is contingent on exploration success. It is AGG's stated aim to focus exploration effort on the shallow (less than 25 metres below surface) mineral potential in order to extend the number of years that are maintained with a low strip ratio. Whilst not demonstrable at this time, the strategy is sound.

26 RECOMMENDATIONS

26.1 Geology and Resources

Further drilling is deemed necessary in the areas where the current drill direction is predominantly 200° azimuth as it is not currently possible to define hanging wall or footwall to the mineralization with any degree of precision. The categorisation of a Mineral Resource other than Inferred is currently precluded in these areas and additional drilling in a suitable orientation is necessary to remedy this. A modest amount of RC drilling is considered necessary and approximately 4,000m of RC drilling is proposed.

Insufficient dry bulk density information is still not available for some areas of the deposit. A number of recent drillholes have allowed dry bulk density readings in oxide and laterite, however, additional diamond core drilling is necessary to address this deficiency. Approximately ten HQ diameter diamond drillholes (1,000m) are recommended to be drilled across the strike length of the deposit. Density determination is recommended to be by Archimedes method and the determinations should be undertaken on site to preclude the possibility of damage to friable saprolitic material during transport.

26.2 Minerals Processing

Test work showed that screening following scrubbing (at 1-2mm) will result in the concentration of 50% of the gold values into the oversize fraction, which made up 10 – 30% of the feed mass. This characteristic can be exploited, by crushing and then bypassing this fraction to the gravity concentration circuit.

Additional design work and a variation of the flow sheet are required to take advantage of this characteristic. Gekko recommends that this change is considered in any future process optimisation work.

26.3 Mining

While mining costs have been validated by quotes from Malian mining contactors, there has not been a rigorous tender process. It is recommended the AGG prepare detailed mining tender documents and enter into a competitive tender process.

Table 26.1
Indicative cost of recommended work

Item	Cost US$
Additional drilling – inferred resource upgrade	800,000
Additional drilling – bulk density determination	200,000
Additional plant design for coarse material to bypass of cyclones	20,000
Mining tender process	50,000
Total	**1,070,000**

27 REFERENCES

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Appendix A

“Decret No 2015-0528/PM-RM” – Mining Permit

“Arrete No 2012-2338/MCMI-SG” – Exploration Permit

Appendix B

Kobada Gold Project Preliminary Slope Design Parameters

Appendix C

Kobada Gold Project Tailings Storage Facility
Feasibility Design

Appendix D

Kobada Gold Project Processing Plant Engineering Study

Appendix E

Kobada Gold Project Other Cost Estimates

Appendix F

Kobada Gold Project Environmental and Social Impact Assessments