UNIVERSITY OF GHANA

ESTABLISHING THE KOJINA GOLD DEPOSIT AS A VIABLE MINING VENTURE FOR SAMEVA LTD, GHANA

BY

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THIS DISSERTATION IS SUBMITTED TO THE UNIVERSITY OF GHANA, LEGON IN PARTIAL FULFILMENT OF THE REQUIREMENTS FOR THE AWARD OF MSc MINERAL EXPLORATION DEGREE.

OCTOBER, 2014

DECLARATION

This dissertation is an outcome of a research undertaken by Edem Kwami Mensah Abba towards the award of MASTER OF SCIENCE IN MINERAL EXPLORATION in the Department of Earth Science, University of Ghana.

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This dissertation has not been submitted in whole or in part to this university or elsewhere for a degree.

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ABSTRACT

Assessment of viability of an ore deposit depends on the accurate estimation of the quantity and quality of the commodity, which is a function of the sample data quality. SAMEVA Limited, a fully owned Ghanaian company, wants to acquire Kojina Gold deposit as an investment opportunity in the gold industry of Ghana. The decision to invest in the mining industry comes with huge capital requirement and the associated risks are very high. The major objectives with regards to evaluating the viability of the deposit are; to define appropriate mineralized zone based on available drill core data, use an appropriate estimation method to create a model that accurately predicts grade and tonnes and finally use a suitable financial model to establish the viability or otherwise of the Kojina project. An Ordinary krigged geostatistical model was used to estimate the quantity and quality of gold within the deposit based primarily on diamond drill holes. The Net Present Value (discounted cash flow) method of financial analysis was used to arrive at the investment decision on the Kojina deposit. Capital and Operating Costs estimates used were based on benchmarking of adjacent mine operations within the catchment area of Kojina. Using appropriately planned drilling and representative sampling as well as predominant use of diamond core gives more accurate information with no contamination and has structural orientation which allows for better geologic and structural mapping therefore helping define the mineralization zones more accurately. The mineralization zone defined based on reverse circulation holes gave 37,413,998 tonnes at 1.47g/t whereas that based on the diamond holes gave 26,674,888 tonnes at 1.57g/t. Again, based on the more accurate deductions from diamond holes, the Net-Present value of the project was estimated at USD 187,796,325 with a payback period of three years.

DEDICATION

This project work is dedicated to all grand-children and great grand-children of the late Togbe Abba of Have-Etoe.

Education has no end!

ACKNOWLEDGEMENT

My utmost thanks to God almighty for giving me the strength and energy to go through this master's programme successfully. My appreciation goes to the Technical services manager, Mr. Trevor Silverton, as well as Management of Chirano Gold Mines Ltd for providing funding for all aspects of the programme.

To my supervisors; Prof. Johnson Manu, Mr. Richard Afenu and all other lecturers of the Department of Earth Sciences, University of Ghana, Legon for their tutorship and guidance throughout our studies at the University, I am very grateful. To my course mates; Fred, Dela, Augustine, Paul and Mike; I say "boys abre".

Finally, to my beautiful women; Esime and Enya for not fussing over daddy's constant absence and Selorm for the encouragement, 'dze ko' moments, and appreciation of the times. Love you gals!

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CHAPTER ONE

INTRODUCTION

1.1 Background

A number of studies have revealed that unreliable reserve estimates has led to several mine failures. In the 1980s, a study on 35 Australian gold mines found that 68% failed because they could not deliver the planned head grade (Tatman, 2001). Another study in 2003 of 41 underground mines showed that 60% of ore reserve estimates fell outside the expected range, with some very seriously in error (Tatman, 2001). It was revealed in those studies that the errors resulted from unreliable sampling data, insufficient samples, poor choice of estimation technique, unrealistic cost estimates, or a combination of these.

Notable examples of projects, which have had reserve problems, are Hayden Hill, California, USA. In 1993, Amax Gold reported a US$64 million write-down of the carrying value of Hayden hill. Schwabb et al (1994) assert that the grade/tonnage shortfall resulted from an absence of adequate drilling, obscured by variogram analysis that on close inspection could not be supported by geology.

Also, Grouse Creek mine in Idaho, USA was permitted in 1992 with an expected annual production of 3.1 t/a of gold, later changing to 2.2 t/a (Darling, 2013). Three (3) years later, the owners announced a write-down of the entire US95 million investment because it encountered significant shortfalls in both grade and tonnage of the ore being mined.

The accuracy of resource estimates is key to the economic evaluation of natural resources. Accurate resource estimation establishes a basis for successful mining operations. The overestimation or underestimation of mineral reserves can have severe consequences for future mine planning. Underestimation could result in an otherwise profitable operation being written off while overestimation on the other hand may lead to the construction of a mine where no profitable orebody exists. More commonly, it results in the life-of-mine (LOM) being shorter than anticipated. Incorrect mineral resource calculation and unrealistic cost and revenue estimates are major risks for project start-ups in the mining industry worldwide (Burmeister, 1988).

Guarnera (1997) sums up this geological risk in mining projects as “No single feature has caused so many mining projects to fail as have reserves not being what were originally estimated”.

SAMEVA Limited is a fully owned Ghanaian company looking for investment opportunities in the mining industry of Ghana. SAMEVA’s previous venture into the industry had led to huge losses. This time around they are taking into account some detailed pragmatic steps both geological and economical for an accurate assessment.

The Kojina deposit is SAMEVA’s new interest. The deposit is adjacent to the producing mine of Kinross’ Chirano gold project and several other exploration properties.

Reliability of information provided by professionals for use in decision-making will boost confidences for further investment by the Ghanaian private sector.

1.2 Problem Statement

SAMEVA Ghana Limited made huge investments into the now defunct Sewum gold mines. During production, it was discovered that the actual grades being mined fell way below the predicted tonnes and grades. The resource work done at Sewum had been based on sparse and unreliable data coming from different regimes of sampling techniques and company standards.

SAMEVA wants to enter into mining once more, the new venture is the Kojina gold deposit owned by DrillCorp Exploration Limited, on the Sefwi- Bibiani gold belt. Based on available detailed data on the property, SAMEVA has therefore contracted FlyGroup Consult (a mineral resource evaluation group) to create a reliable model of the deposit to be used for assessment of viability before a decision on the acquisition is made.

1.3 Objectives

The objectives are to:

- verify drilling and sampling data to ensure sample integrity and representativity.
- use an appropriate estimation method to create a model that accurately predicts grade and tonnes.
- use a suitable financial model to establish the viability or otherwise of the Kojina project.

1.4 Justification

The use of diamond holes, primarily, in resource estimation ensures the absence of downhole contamination and gives in-situ information on geological, mineralization and structural orientation especially for a hydrothermal deposit, which is structurally controlled. Structurally controlled ore bodies tend to have sharp ore waste boundaries, which must be properly modeled to avoid smearing of grades in unmineralized rock. Ordinary kriging has the inherent ability to handle the clustering, which occurs in mineral exploration drilling. The issue of unrealistic cost estimates and revenue forecasts are addressed by benchmarking existing operations. The approach used in this research hopes to address the problems identified by Bullock (2011) as the reasons for failed mine start-ups; recovered grade and tonnes being lower than predicted, capital and operating costs being higher than expected and finally, the sales revenue being lower than projected.

This evaluation will therefore not only project the economic viability of the Kojina deposit but will also serve as guideline in considering future projects that may be signed.

1.5 Location

The Kojina deposit, which is approximately 29 km^2 in size, is located in south-western Ghana with many historical and current open pit and underground gold mines located along strike to the northeast of the Project (Kesse, 1985). It is about 180 km southwest of Kumasi. The nearest operating mine is Chirano Gold Mines limited, which is some 35km away from Kojina deposit. From the township of Sefwi-Bekwai, the site is accessible by 1st class road onwards to Sefwi-Akontonbra and a 20-kilometer dust road beyond Ashiem. The Project is centered on 5°59’ north latitude and 2°27’ west longitude (Figure 1).

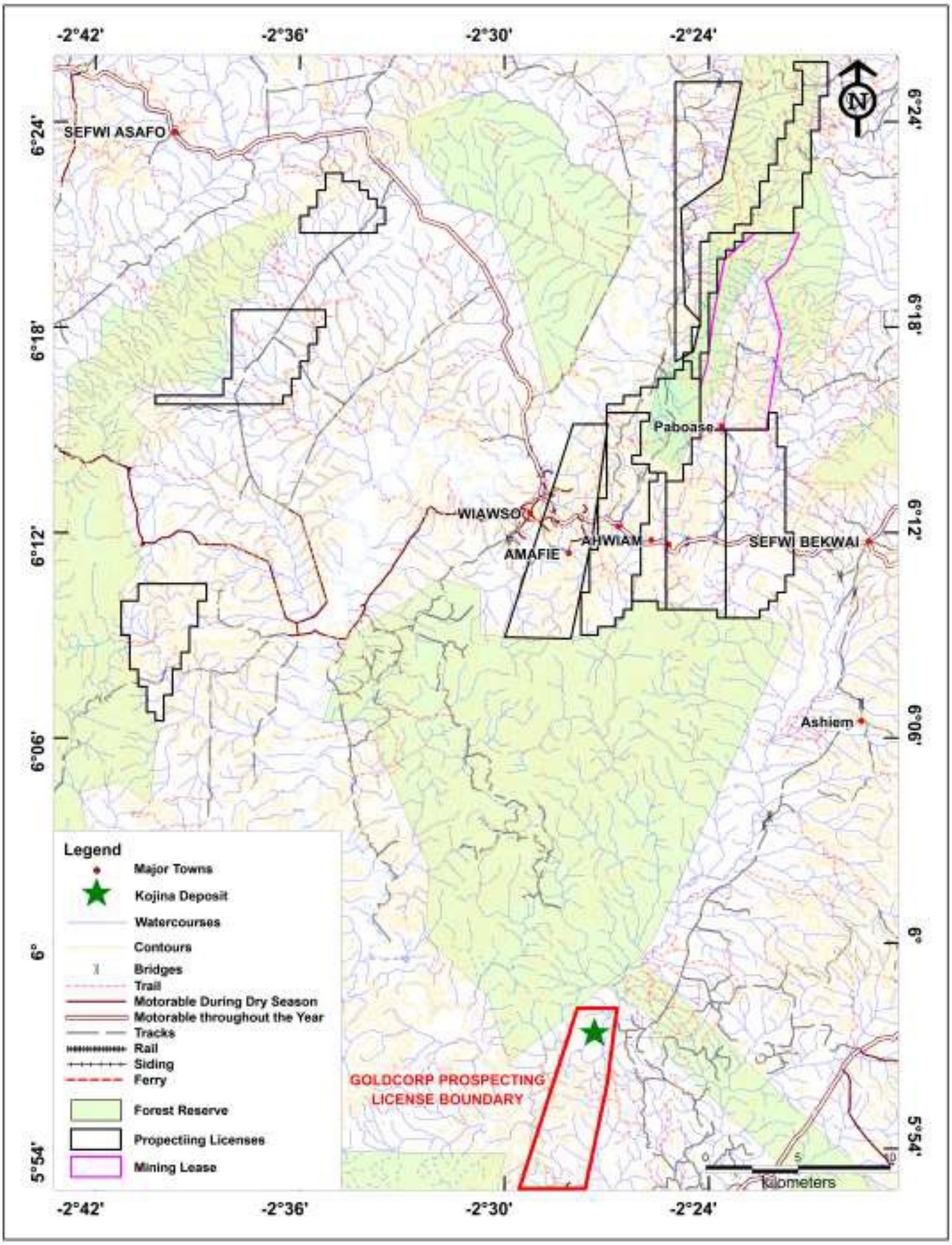

Figure 1: Location map showing the Kojina deposit (red box)

1.6 Physiography and Occupation

The physiographic region of the place is of the forest-dissected plateau (Dickson and Benneh, 1988). The heavy rainfall in the region and the consequent forest vegetation which prevents sheet erosion explains the strongly dissected nature of the plateau. The Ankobra River with its several tributaries drain through the broad flat valley of the concession. Covered with steep hills, which rise to an average of 240 meters ASL, rugged and gently undulating topographies, the concession is characterized by rocks of the Upper Birimian as well as Tarkwaian rocks. (Dickson and Benneh, 1988).

The major occupational activities within the area are commercial farming and small scale mining operations. Besides cocoa and oil palm farming done on a commercial scale, subsistence farming to produce cassava, plantain, yam, pepper and citrus is also done in the area. In addition to the above produce some non-traditional cash crops which have gained popularity in the municipality include rice and pineapple.

CHAPTER TWO

LITERATURE REVIEW

This section entails all available geological information with respect to previous data that have been compiled considering the regional geology, local geology and also on the Kojina deposit as well. This spans from lithological characteristics, regional extents, structures, mineralization, alteration, geochronology and proposed geodynamic or geological settings. The latter part of the literature considers a detailed example of similar viability in Ghana, specifically in the Northern part of the country but of the same regional geological scope.

2.1 Regional Geology

Early Proterozoic Birimian greenstone assemblages (2100 Ma) constitute the major geologic units in Ghana. This occupies about twenty percent of the total area of Ghana, and form part of the West African craton that outcrops extensively in other parts of the subcontinent, including Ivory Coast, Burkina Faso and Mali (Hammond and Tabata, 1997). The greenstone units in Ghana are characterized by five evenly spaced volcanic belts trending NE-SW interspersed with sedimentary basin (Figure 2).

In Ghana, the volcanic belt consists of metamorphosed volcanic rocks of tholeitic to cal-alkaline composition, where the metasedimentary basins contain metamorphosed volcaniclastics, wackes and argillitic sedimentary rocks (Kesse, 1985). Available field evidence suggests that the volcanic and sedimentary rocks are lateral equivalents (Leube et al., 1990). Within the Birimian Supergroup, northwest striking mafic metavolcanics belts are separated from the intervening metasedimentary basins by major faults, these faults probably early syn-Birimian sedimentary basin down-faulting (Hirdes and Leube, 1989). The Tarkwaian system is dominated by coarse clastic sedimentary rocks. Age dating suggests that the Birimian and the Tarkwaian sediments broadly overlap in the period 2140 to 2100 Ma (Pigois et al, 2003).

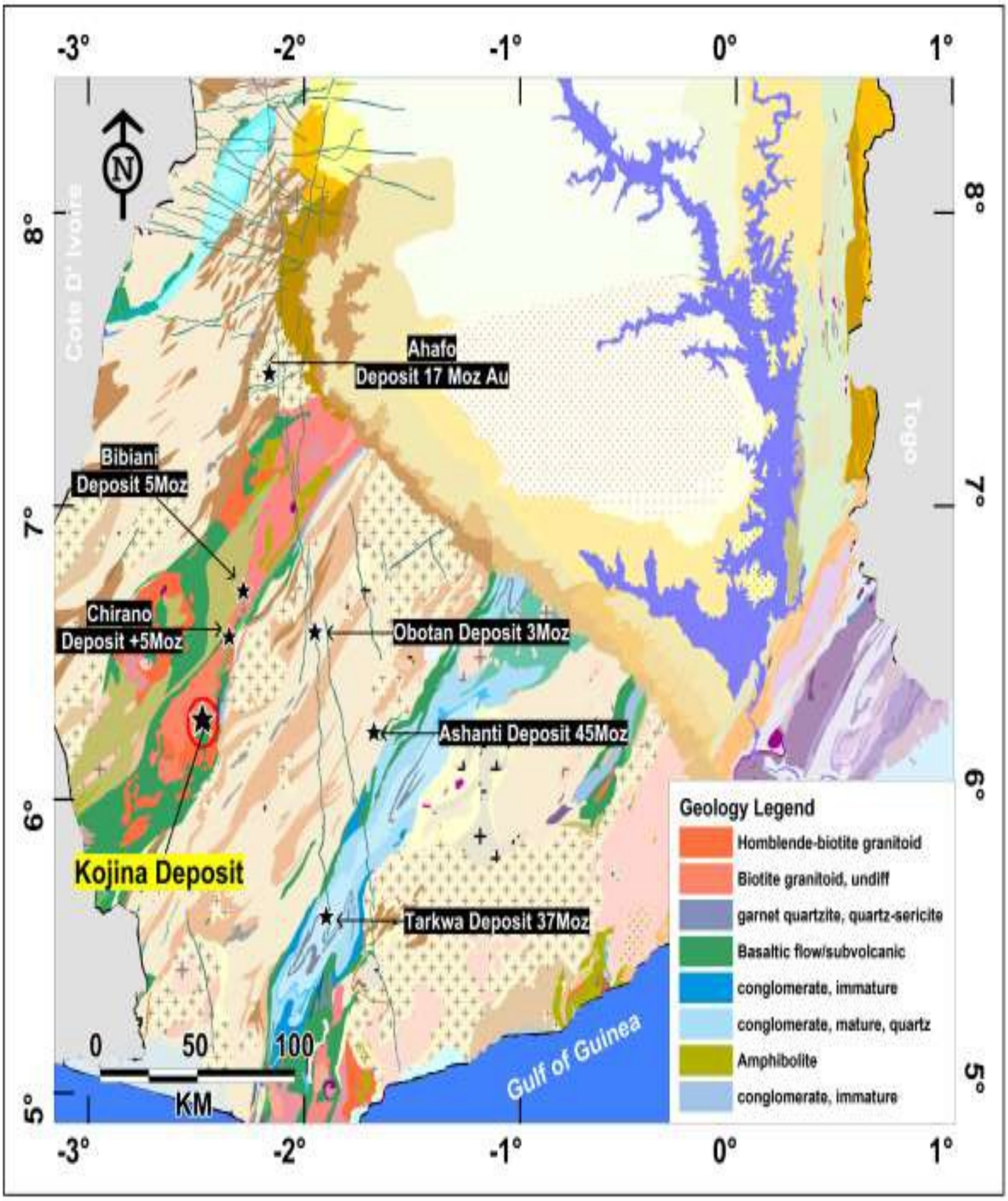

Figure 2: General geology of the Sefwi gold region

2.2 Local Geology

The Kojina gold deposit is within the Proterozoic gold district of southwest Ghana. The concession is along the central western margin of the Sefwi volcanic belt and it covers about 20 km of strike-length along the very prospective, fault bounded contact between the Sefwi volcanic belt and the adjacent Kumasi basin metasediments and intermediate granitoid batholiths to the east. The western part of the concession is dominated by thick sequences of Birimian mafic metavolcanics that form much of the Bibiani range whereas the lower lying areas to the east are dominated by Birimian sediments.

The Sefwi volcanic belt is one of several similar belts in the region, separated by the sedimentary basin. The belt and the basin architect comprise rocks of Birimian age, with the belts being dominated by mafic volcanics and the basin typified by deep fine-grained marine sediments (Stuart, 2007).

The Birimian meta-volcanic rocks in the Chirano district consist of fine-grained basalts, fine to medium-grained dolerites and medium to coarse-grained gabbro with some minor tuffacous sedimentary rocks. The meta-volcanic rocks show minor metamorphism and deformation. The unaltered basalt has about 60% and 40% of pyroxene and plagioclase as its mineral composition respectively. Altered basalts comprise pervasive chlorite-carbonate alteration. Dolerite rocks closer to the mineralised lode horizon however have been altered by greenschist facies assemblage of chlorite, carbonate and albite. There are significant felsic units which are inter-fingering in the mafic. The felsic material is mainly tonalites. Towards the lode horizon the units become more foliated and mylonised (Stuart, 2007).

The felsic units at Chirano are mainly granodiorite to tonalite in composition. They are medium to coarse- grained and equiangular to porphyritic in texture. It has variable colour partly due to the alteration assemblages. The colour however ranges from light pink to grey. The coarser and more equigranular granodiorite-tonalite appear to be more prevalent as large, massive bodies in the hanging-wall nearer the shear zone, whereas the sub-porphyritic variety outcrops in the foot- wall portion of the deposits. The Chirano Shear zone has been intruded by smaller felsic units. Granitoids occur within both the Birimian volcanic rocks and the sedimentary units (Figure 3).

Structures, Mineralization and Alteration

The DrillCorp shear is the major structure which controls the mineralization of the DrillCorp deposit and it is a splay of the Bibiani shear zone. The Bibiani shear is the fault between the Tarkwaian and the Birimian sedimentary rocks and the DrillCorp shear also forms the contact between the Birimian metavolcanics and the Tarkwaian sediments.

Gold mineralization is characteristic of classic Pre-Cambrian Greenstone-hosted quartz vein style gold deposits. This style of gold mineralization is the most important type of gold occurrence in West Africa and is commonly referred to as the" Ashanti-type" in this region of West Africa. It is associated with intense hydrothermal alteration, predominantly manifest as an ankerite-albite-pyrite-muscovite assemblage. Generally, the tenor of mineralization correlates with intensity of alteration, veining and brecciation.

Primary sulphide minerals are present in concentrations of 1% to 2% but can attain concentrations up to 5% (Stuart, 2007). There is a close correlation between gold and the presence of disseminated pyrite and albite-ankerite alteration. The mineralized horizon is characterized by a high degree of alteration, foliation, veining and brecciation.

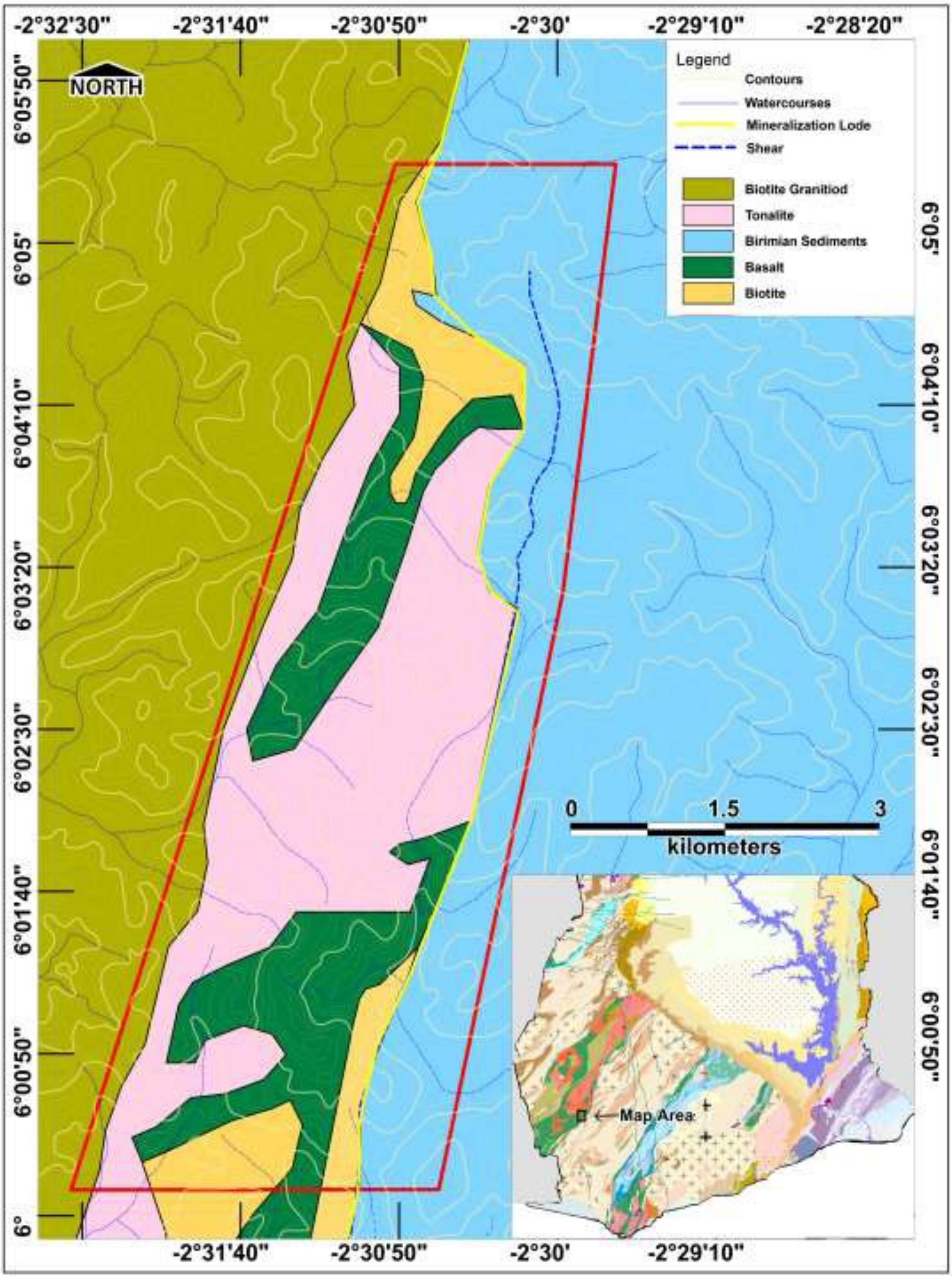

Figure 3: Geology of the Kojina Gold Deposit, showing mineralization trend

The mineralization normally occurs within 50 meter west of the DrillCorp Shear Zone. The common alteration types are sericite, pyrite, carbonate and chlorite alteration.

Exploration at Kojina has been very extensive, employing methods of geophysics, geological mapping, prospecting, remote sensing, geochemistry, etc. to identify areas of interest that warrant further investigation. Kojina has lots of geophysical information especially induced polarization, and aerial photography which were used to delineate areas for soil sampling, trenching and augur drilling. The GIS and remote sensing were done by Terra Resources Limited led by principal consultant Barry Bourne. (DrillCorp technical report no. 13_003)

2.3 Similar viability Study in Ghana.

Awotwe (2003) conducted a pre- feasibility study of the Dokrupe Gold Project (DGP) near Bole in Northern Region of Ghana. The project which is owned by the Northern Goldfield Limited (NGL) was looking for funds to exploit near surface auriferous deposits at Dokrupe and Baju near Bole. To date, NGL's exploration programme has established indicated and inferred reserves totaling 3 million tonnes at an average grade of 3.1 g/t. The nature of the deposits and metallurgical characteristics of the ore bodies indicate that the deposits could be exploited using open-pit mining method at a yearly production rate of 635,280 t over the next 5 years. The run-of-mine ore would be processed by the Carbon-in- leach (CIL) method to achieve a recovery of 90%. Semafo Ghana Limited (SGL) has expressed interest to enter into a joint venture agreement with NGL and by this agreement, NGL would source funds to complete the exploration programme and develop the Dokrupe Gold Project provided the project would be economically viable once the indicated and inferred reserves became proven. Based on funding agreement, there was the need for Dokrupe Gold Project to establish its economic viability or otherwise on the assumption that the indicated and inferred reserves are proven.

With details on the location of the deposits, the exploration data, the proposed mining system and the processing method, the total capital cost and yearly operating cost were estimated using detailed cost estimation method as well as base cost data from mines operating in similar conditions in Ghana. Using an estimated yearly revenues of a base gold price of $9.65/g ($300/oz) and with the use of Ghana Investment Law pertaining to mining, computerized cash flow model — ECASHFLOW was used. This programme has been used to analyse the economic viability of the project using 100% equity and 100% loan (at an assumed Minimum Rate of Return of 12%) as a base case scenario. The Net Present Value (NPV), Internal Rate of Return (IRR) and Discounted Pay Back Period (DPBP) were used as economic indicators to assess the viability of the project. Detailed financial analyses were conducted into the costs of capital, cost of equity and cost of loan, from which optimum capital structure was determined. Based on this, the weighted average cost of capital was determined and used as the Minimum Rate of Return (MRR) to assess the economic viability of the project.

In addition, sensitivity analyses were conducted to verify the effect that changes in any of the economic parameters like capital cost, operating cost and revenue would have on the project's viability. Furthermore, risk analysis was carried out using the Monte Carlo simulation technique, which takes into account the simultaneous random variation of economic parameters on the project's viability. As a result, risk was quantified and the probability of achieving specific returns on the investment also measured. The project was found to be economically viable and financially sound since the NPV was $2 157 862.95 and the Internal Rate of Return (IRR) of (36.46%) was greater than the weighted average cost of capital (19.04%) which is considered as the minimum rate of return. The risk analysis conducted revealed a risk of 35% associated with the project.

CHAPTER THREE

METHODOLOGY

All the methods adopted are systematically outlined below, and these include validation of all available data on drilling and sampling, statistical treatment of the data, appropriate data transformation and the incorporation of reliable modeling tools for the resource evaluation, ore modeling and financial analysis. Details of each approach in relation to the method is dully considered and explained.

3.1 Drilling and Survey

Advanced exploration systems reverse circulation (RC) and diamond drilling (DD) were used to define the resource of the Kojina gold deposit. Boart longyear and Geodril Ghana Ltd were the contractors used for all the drilling at Kojina. Drillhole information was captured electronically at DrillCorp using mini handheld devices called motion PDA's. Previously, Dell's iPAQ using Fusion's Century Technology was used for the data logging (DrillCorp database report, (2013) to ensure the following;

- Standardized procedures and data integrity
- Secured and centralized data
- Increased confidence in interpretations

Diameter of core used for this project was HQ, a 63.5 mm diameter core, extracted using a triple-tube wire-line system capable of extracting core under the worst of ground conditions. The Kojina area is drilled on a fairly regular grid of 50 along strike by 25 across. Most of the holes were drilled at an angle between 60 and 70 degrees towards grid east, typically intersecting the mineralisation zones at 25 meter to 50 meter intervals on section.

Numerous permanent and semi-permanent survey control stations have been established by the company's licensed surveyors. Surface surveying affecting the current resource estimate was carried out using Sokkia total station electronic distance measuring (EDM) instruments.

Downhole surveys were done historically by FlexIT tool, a single-shot surveying instrument but more recently by the contractor called Wellforce, UK Ltd, which did it using the gyroscope surveying tool, a multi-shot able to measure all parameters at any number of depths in a single run into a hole. An azimuth and dip reading is taken every 30m, and at the end of hole.

Grid System

The project area has a local grid system; Kojina grid system. This grid was chosen so that each localised mineralization would have its strike aligned orthogonally to a local north-south trend. The rotation between these two grids is approximately 45°, the seed points for transforming these to UTM coordinates are;

Point 1		
568,247.65E	69,9529.75N	WGS/84
15,086.59E	38,559.73N	Kojina Grid
Point 2		
567,065.39E	69,4749.19N	WGS/84
16,338.48E	33,797.21N	Kojina Grid

3.2 Sampling, Sample preparation and analysis

For RC drilling, sample mixing is prevented by blowing the hole clear at the end of each sampled interval. RC samples are split with the Jones' riffle splitter (Figure 4) to get 2-3kg for laboratory analysis. Diamond core is cut into two halves. One halve is stored for future study and logging, and the other halve was generally sampled at 1m intervals, unless geological conditions dictated otherwise, in which case sample intervals could be shorter. Generally, only mineralized zones were sent to the lab but a few non-mineralised samples were sent as sterilization samples.

Quality Assurance and Quality Control (QAQC)

At Kojina, the types of QAQC checks used are:

- *Standards* (STD)

These are samples of known (usually certified) grade that are submitted to monitor the accuracy of a laboratory, i.e. the ability of the laboratory to get the correct or known result. The standards were bought from Rocklabs in Australia. The permitted standard deviation is +_10%. (Rocklabs Limited, 2013)

- *Blanks* (BLK)

Blank (barren samples) inserted routinely into known ore zones to check on possible contamination from the laboratory.

- *Field Duplicate (DUP)*

Field Duplicates are collected at the drill rig (for RC chips). Diamond cores however are split at the core shed (Figure 4) for duplicate samples. Duplicates were used to estimate sampling and laboratory precision.

ALS Chemex, Kumasi, a subsidiary of ALS Global Ltd, did all the assaying for DrillCorp. Results are sent to DrillCorp in electronic format as an encrypted certificate with accompanying CSV file for easy loading into DrillCorp's Fusion database. Assay files received from the lab are never altered. Any inconsistencies are rather referred back to the lab for rectification. All assaying has been by 50 gram Fire Assay with AAS finish. Umpire checks are sent to Genalysis Lab in Australia from a random selection of pulps and rejects.

Figure 4: Mechanized core cutting process by DrillCorp

3.3 Approaches to Project Valuation

There are three generally accepted approaches to mine project valuation (Manu, 2014). These are the cost approach, market approach and earnings approach.

The cost approach

The fundamental concept is that a purchaser would not be justified in paying more for a property than it would cost him to acquire land and construct improvements having comparable utility, assuming no undue delay. The cost approach is merely applicable in valuing mining properties and is the least reliable method of valuation (Manu, 2014). The correlation between construction costs and value of the property is very imperfect since the very nature of mineral exploration and mining dictates that the discovery value of an ore deposit is generally greater than the cost incurred in that discovery.

The market or comparable sales approach

The market approach is viewed by most appraisers and courts as the best since it reflects the balance of supply and demand. The market approach is rarely used for mining property transactions because assumptions are hardly present and making it very difficult to ascertain the actual true value of the sale. The Analyst studies the market for similar assets in attempting to determine the market value of the item in question. Practical problems of this approach when applied to mining transactions is that there is little comparative data available due to limited sales of mining properties. Each mineral deposit is unique in quality, size, geological location, degree of development, and many other parameters, and market data are of modest value at best.

The income or earnings approach

The Income Approach is used widely in valuing mineral properties (Manu, 2014). The value of the asset or investment- type property is estimated by calculating future annual net earnings from the producing mine or asset and then discounting this earning's stream to the present time using an appropriate interest rate. The assumption is that the purchaser would not be justified in paying more to acquire income producing than the present value of the income stream from the property.

If comparable sales data are unavailable or one is estimating the value of a commodity in-situ, it is possible to arrive at a value estimate by combining the selling price of the commodity produced with the associated costs of producing the commodity from the property in question. By properly incorporating this data into a discounted cash flow analysis, it is possible to arrive at an estimate of property value even in the absence of actual production (Manu, 2014).

The Discounted Cash flow method of Analysis

This approach falls under the income (earnings) approach. The method is used to estimate the attractiveness of an investment opportunity. Discounted cash flow (DCF) analysis uses future free cash flow projections and discounts them (most often using the weighted average cost of capital) to arrive at a present value, which is used to evaluate the potential for investment. If the value arrived at through DCF analysis is higher than the current cost of the investment, the opportunity may be a good one.

3.4 Resource Estimation

The resource estimation process was undertaken using standard industry software Supervisor 8.1 for variography and kriging neighbourhood analysis; sGEMS for declustering; and Surpac version 6.3 for all other related work.

The estimation process was laid out in 5 phases;

Phase 1 - Data Preparation

Phase 2 - Investigation of statistical and spatial patterns

Phase 3 - Building the model

Phase 4 - Model Validation

Phase 5 - Resource reporting.

3.4.1 Phase 1 - Data preparation

Data preparation included data verification, de-clustering and data compositing, basic statistics and treatment of outliers (extreme values), as well as creating 3-dimensional lithological, structural and mineralization models.

Data Verification

The entire drillhole database of the Kojina area was received in CSV format as an export from the Fusion database. The information received was validated against original field sheets, survey contractor's Gyro reports and assay lab certificates by comparing side by side in an excel spreadsheet (Table 1).

Table 1: Validating database against field logs

		ORIGINAL FIELD LOG SHEET				FUSION DATABASE				
Prospect	Hole ID	Northing	Easting	RL		Northing	Easting	RL	Date	Comment
KOJINA	KOJRC1980D	88633.4	49644.03	2305.8		88633.4	49644.03	2305.8		OK
KOJINA	KOJRC1982D	88633.7	49642.24	2305.75		88633.7	49642.24	2305.75		OK
KOJINA	KOJRC1984D	88634.82	49642.61	2305.8		88634.82	49642.61	2305.8	16/04/12	OK
KOJINA	KOJRC1985D	88189.48	49135.84	2389.79		88189.48	49135.84	2389.8	10/4/2012	rounded to 1 d.p
KOJINA	KOJRC1986D	88458.84	49465.4	2299.05		88458.84	49465.4	2299.05		OK
KOJINA	KOJRC1987D	88471.69	49487.63	2291.66		88471.69	49487.63	2291.66	16/04/12	OK
KOJINA	KOJRC1988D	88540.28	49525.37	2290.25		88540.28	49525.37	2290.25	16/04/12	OK
KOJINA	KOJRC1989D	88418.39	49452.89	2310		88418.39	49452.89	2310	27/04/11	OK
KOJINA	KOJRC1990D	88549.29	49452.22	2300.99		88549.29	49452.22	2300.99	5/6/2012	OK

The holes chosen for verification represented 10% of the entire database randomly selected. The process consisted of checks on: Collar Coordinates (easting, northing, elevation, hole length); Survey (azimuth, dip, depth); Lithology (rock code, interval); and Assay (Au values, sample number, sample interval). Appendix G contains results of the checks. In addition, actual field verification of collars (Figure 5) was also done on some random holes.

Figure 5: Verifying accuracy of Collar location

Accuracy and precision of Lab results

Plots of original samples versus duplicates, original samples versus repeats etc were generated using excel charts. The failure criteria used for certified reference materials (CRMs) was per manufacturer's guideline summarized as follows;

- Any standard value falling outside of ±3SD is a failure (inaccurate),
- Any two or more consecutive values falling between ±2SD and ±3SD on the same side of the mean are failures (Rocklabs Limited, 2013).

Data compositing and De-clustering

To ensure that the samples for estimation represent equal volume (support), compositing was done. The composite length was determined from statistical analysis as the sample interval with the highest rank (Figure 6). A 3m length composite was also done to be used later for validation.

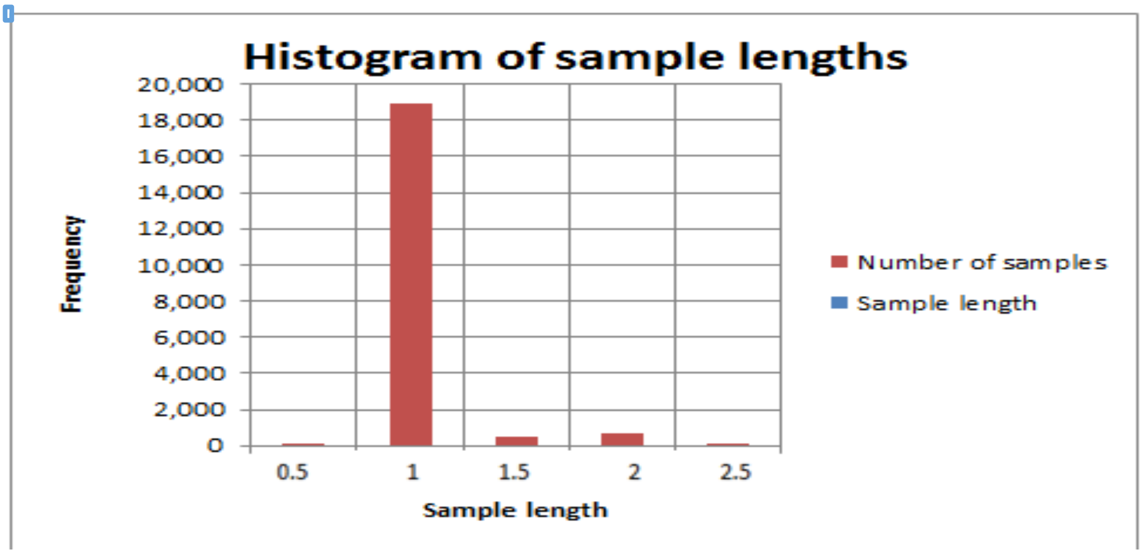

Figure 6: Determining composite length by histogram

In order to reduce the influence of the crowded samples on the estimation process, declustering was done using GSLIB (geostatistical software library, developed by Stanford university, USA) cell de-clustering which assigns weights to samples.

For a given cell, the weight of each sample is calculated as follows,

$$w'_j = \frac{\frac{1}{n_i}}{number \quad of \quad cells \quad with \quad data} \cdot n$$

where n_i is the number of samples in the cell in which sample j is located and n is the total number cells with samples.

The cell size used was determined from a graph of grade versus sample cell sizes (Figure 7) at the point on the graph where the data first flattens out.

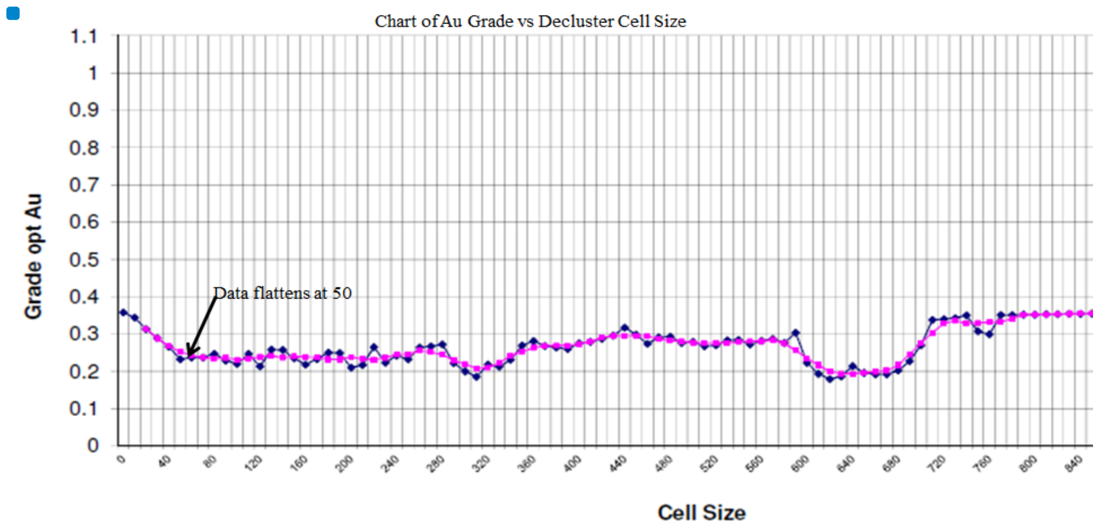

Figure 7: Determining cell de-clustering size

Basic statistical analysis was performed on the composite string to determine the statistical properties of the data being used for the geostatistical estimation. The statistics include the mean, median, variance, standard deviation, coefficient of variation, skewness and kurtosis as shown in Table 2.

Table 2: Descriptive statistics of raw data

No. of Points	23,087
Mean	0.84
Standard Deviation	1.07
Variance	1.14
Cofficient of Variation (CoV)	1.28
Skewness	5.49
Kurtosis	61.78
Maximum	21.40
75%	1.15
50%	0.54
25%	0.22
Minimum	0

The Range, inter-quartile range, variance, and standard deviation indicate the measure of spread of the data.

A histogram of the raw data was plotted to determine skewness (Figure 8a).

The positively skewed data was then log transformed to achieve normality (Figure 8b), which is a requirement for using ordinary kriging.

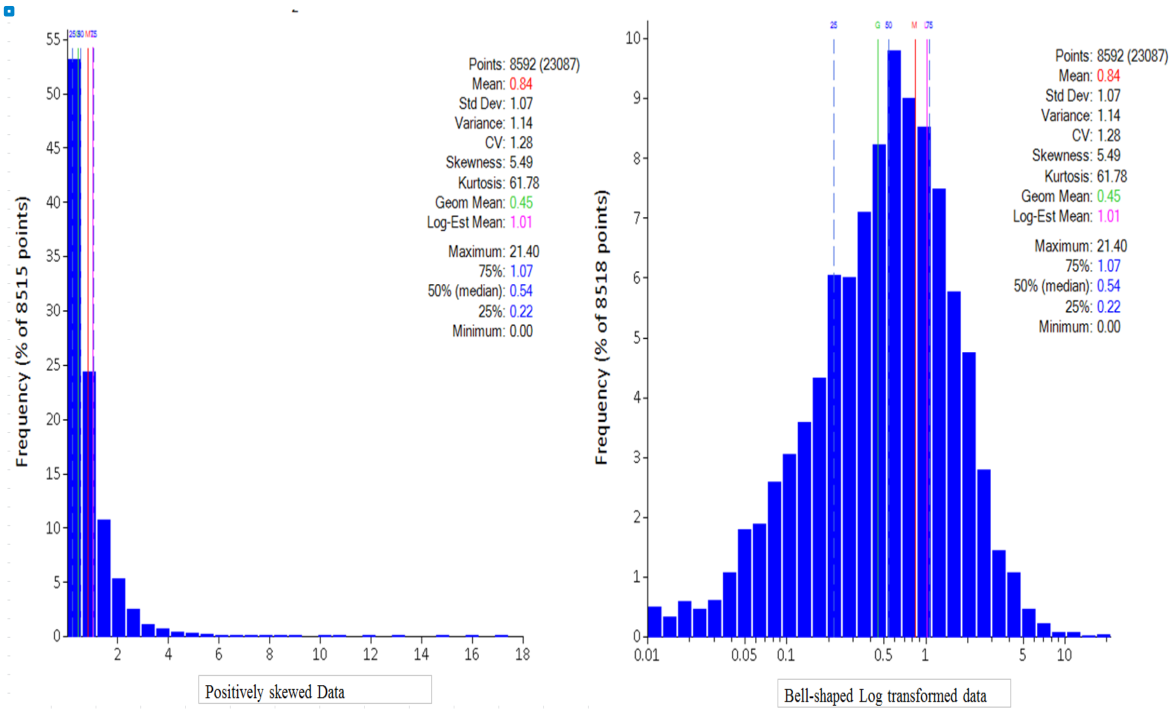

Figure 8: Log transformation of positively skewed data

Treatment of extreme values (outliers)

Although the coefficient of variation (Table 2) was not too high, capping was used to reduce the influence of the few outlier gold values. To arrive at the top-cut value, a log probability plot was used. The first point on the graph which shows a break in trend of the data describes the capping value.

The inflexion point (Figure 9) on the graph was used as top-cut value. A topcut of 10g/t was applied to the data.

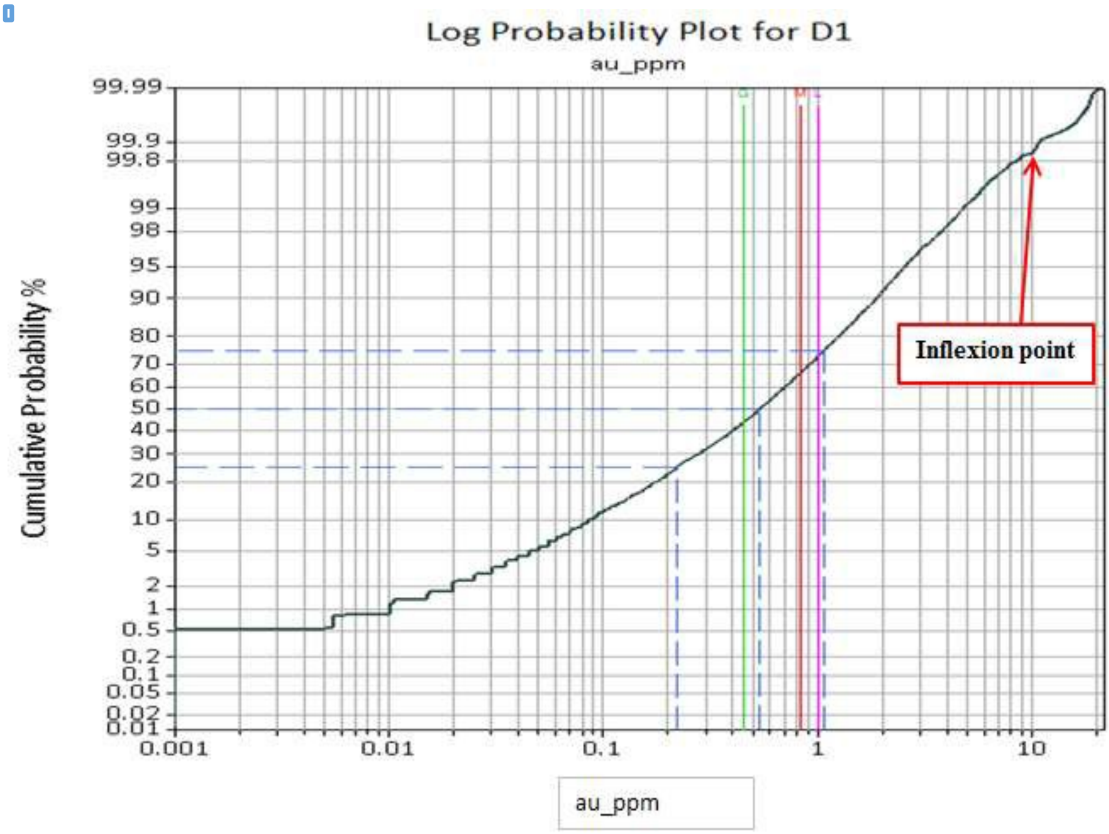

Figure 9: Log probability plot for gold (Au) –Selecting the top-cut value

Wireframes, surfaces and ore solids

All the 3-dimensional (3D) models of lithology, shear zones, alteration and mineralization were created by digitizing sectional interpretations of lithology, alteration, or mineralization in cross-section mode across the deposit. The strings were then triangulated to build 3D solids. The topographic surface was created by extracting all collar values into a string to form the surface. The oxidation surface points were extracted from geotechnical core log data. Points on each drillhole logged as oxide /fresh were extracted as strings. These strings were then formed into DTM's (digital terrain models) using Surpac's "create DMT from string" function.

Two mineralization envelopes were built based on diamond holes only and RC holes only. The gradational contact in the RC holes led to a wider interpretation of mineralization zone (Figure 10).

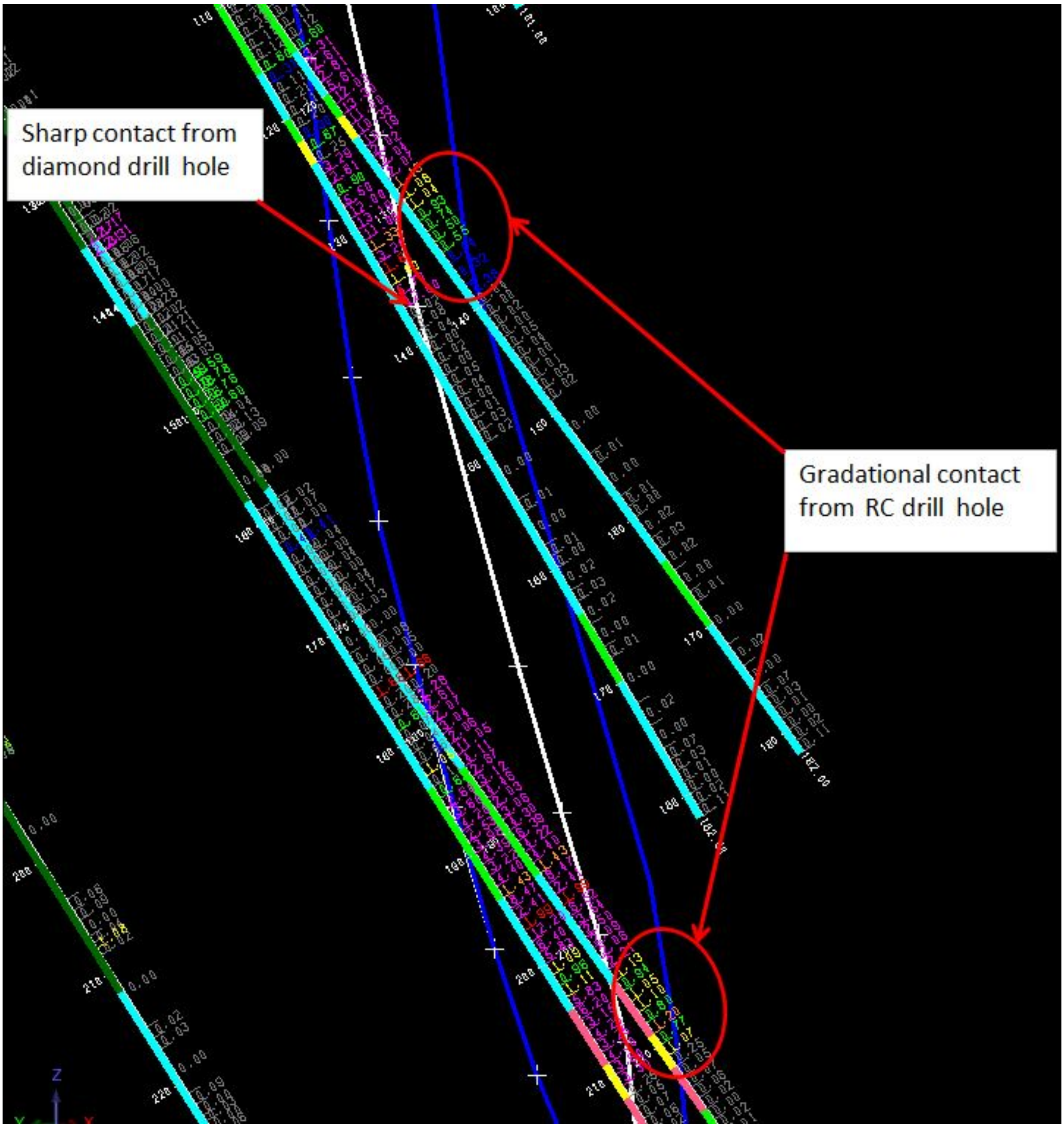

Figure 10: Mineralization zones as interpreted from Diamond holes verses RC holes

3.4.2 Phase 2- Investigation of statistical and spatial patterns

The semi-variogram was used in identifying and modeling the trends in the data. The prepared data was imported into Supervisor software and processed. Two sets of variogram were generated; the downhole variogram and the planar variogram. The downhole variogram was generated first to know the nugget effect to be used in the planar variogram. A line of best fit was put through the sample pairs which equation is used by Surpac as the trend predictor for the kriging process. The major and semi-major anisotropic ratios describe the relationship between the principal directions. The two variogram maps were extracted along the minor axis to generate the anisotropic ratio and the kriging parameters for the estimation process. This minor axis is the down dip or plunge of the orebody. An ellipsoid was created from these ratios and superimposed on the dataset to confirm the trends in the data. All results were back transformed, per industry standard, using the back-transform model below (Figure 11).

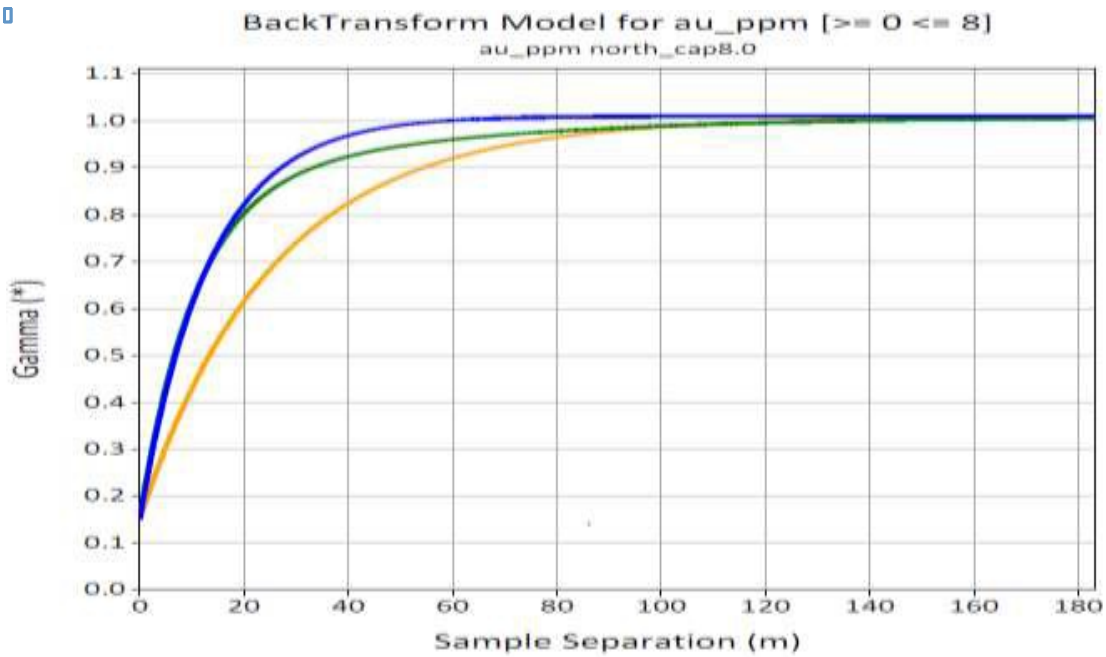

Figure 11: Back transform Model

Kriging neighbourhood analysis

The Supervisor kriging neighbourhood analysis (KNA) utility was used. KNA helps determine the best block size, discretization, minimum and maximum number of composites to use for estimation, and search ellipse size by domain. (Figure 12)

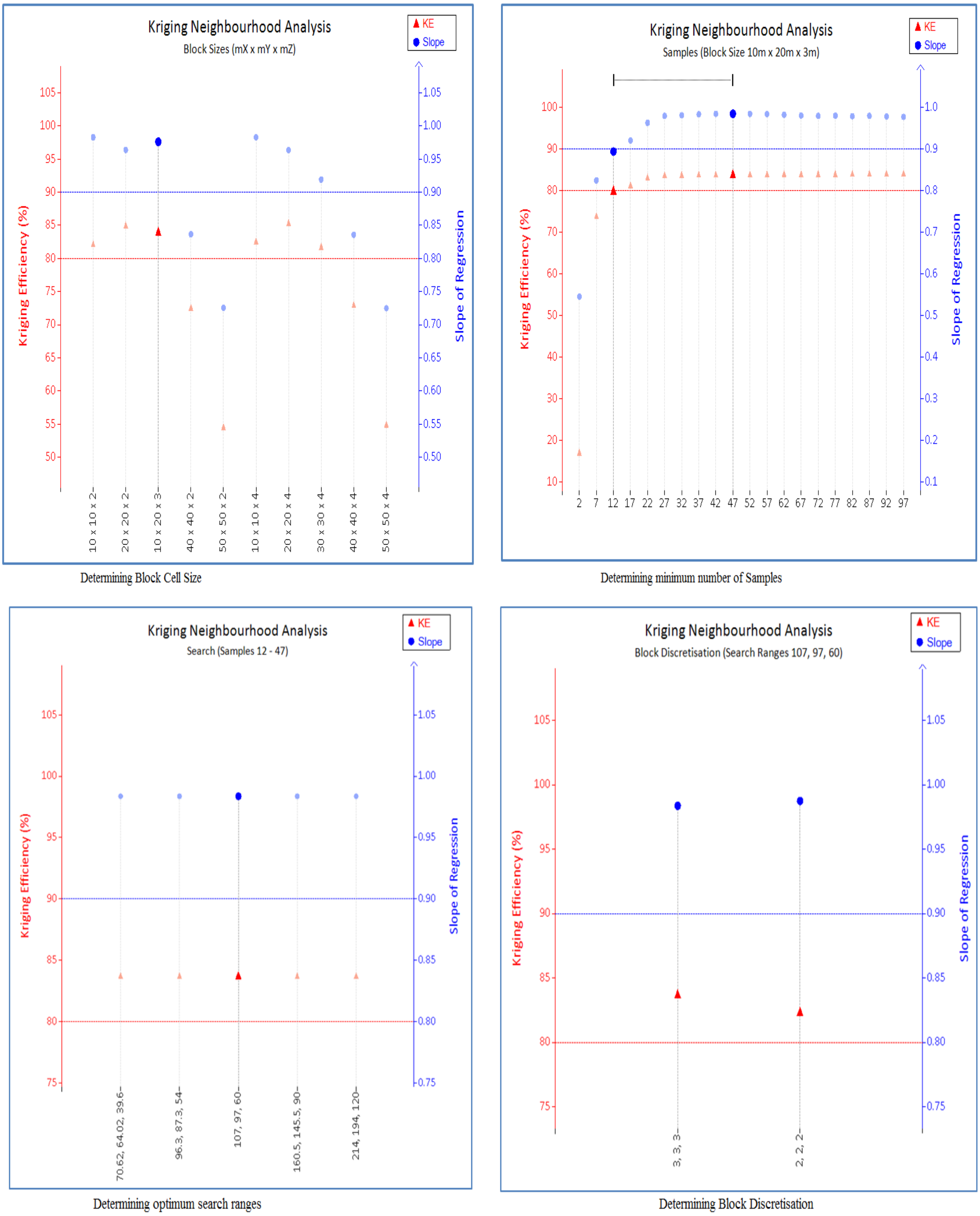

Figure 12: KNA –Determining block size, number of samples, search radius

3.4.3 Phase 3- Building the model

The entire dataset was displayed in 2D grid, both plan view and longitudinal view. The limits of the data guided the choice of the block model extents (Table 3). The selection of block size was driven by the borehole spacing (easting and northing), which is 25m as well as the assumed mining bench height (elevation) of 3m.

Table 3: Block model extents

Block model parameters			
	X	Y	Z
Minimum	48885	88270	1350
Maximum	49470	89220	2500
Parent Block Size	10	20	3
Sub Block Size	5	10	3

The model was created using Surpac version 6.5. Attributes (Table 4) were created to store the block model properties as summarized below.

Table 4: List of attributes used in the block model

Block Model Attributes			
Attribute Name	**Type**	**Background value**	**Description**
au_nn	Real	-99	nearest neighbour gold estimate expressed in g/t
au_ok	Real	-99	ordinary krigged gold value expressed in g/t
specific gravity (SG)	Real	2.75	in-situ dry bulk density expressed in g/cm^3
redox	Float	Fresh	oxidation state of rock expressed as “fresh” or “oxide”
krig_var	Float	-	kriging variance of the estimate
class	Integer	4	mineral resource classification
nsamp	Float	-	the number of data points used to estimate grade;
nhole	Integer	-	the number of drillholes used to source data for estimation;
Pass	Integer	-	the estimation pass that successfully informed the cell

Grade Estimation

Grade estimation was done twice; based on the mineralization as interpreted from the Diamond holes and then as interpreted from the RC holes. The estimation was done using ordinary kriging and nearest neighbour methods. The nearest neighbour method was later used as one of the validation checks. Estimation for gold grade was done with a 3-pass approach in each case, using search ellipses defined by variogram ranges. The first estimation pass used a search radius equal to the full variogram range. The range was doubled for pass two, and tripled for the third estimation pass. Based on the KNA, a minimum of three holes were used for the first and second pass estimations; two drill holes for the third pass. No more than eight samples per single drill hole were used for the estimation. Thus the highest confidence blocks were estimated in pass 1 and the lowest confidence blocks in pass 3.

3.4.4 **Phase 4- Block Model Validation**

Three methods of validation were used.

1. Visual validation

Visual inspections on cross sections of composite values compared to the estimated blocks were made for reasonableness. Comparisons were made using the 1 m composites that were used for the block estimates. Also, 3 m composites were compared to the blocks as they would have a variance closer to that of the blocks. Some of the sections are displayed in Appendix D

2. *Swath plots*

A comparison of the estimated block grades with the composite data along the three principal directions i.e. evaluation of conformance of the block estimates with the composite grades by comparing au_ok (ordinary krigged estimates), au_uncap (uncapped composite data) and the au_cap (capped composite data).

3. Alternative Estimation technique

A Nearest neighbour estimation was done as a check on the ordinary krigged estimate. The two were compared using an excel scatter plot; estimated grade on y-axis, estimated point location in y, x, z directions on the x-axis.

3.4.5 Phase 5 - Resource classification and Reporting

A combination of drillhole spacing and variogram range were used in arriving at the classification. Visual recognition of continuity was obtained by examination of drillhole data in long section showing geology and assays. Polygons were digitized in this view and solids were built to flag blocks as inside or outside of each class designation. The mineral resources were classified as Measured, indicated or inferred, to reflect decreasing levels of confidence based on geological interpretation and distribution of the data (Figure 13). As a 'rule of thumb', measured material is where the nominal spacing between drillholes is less than 25 meters, indicated where the spacing is greater than 25 but less than 40 meters, and Inferred where the spacing is greater than 40 but less than 75 meters. Where there are blocks estimated greater than 75 meters, this would be considered 'Other' mineralization.

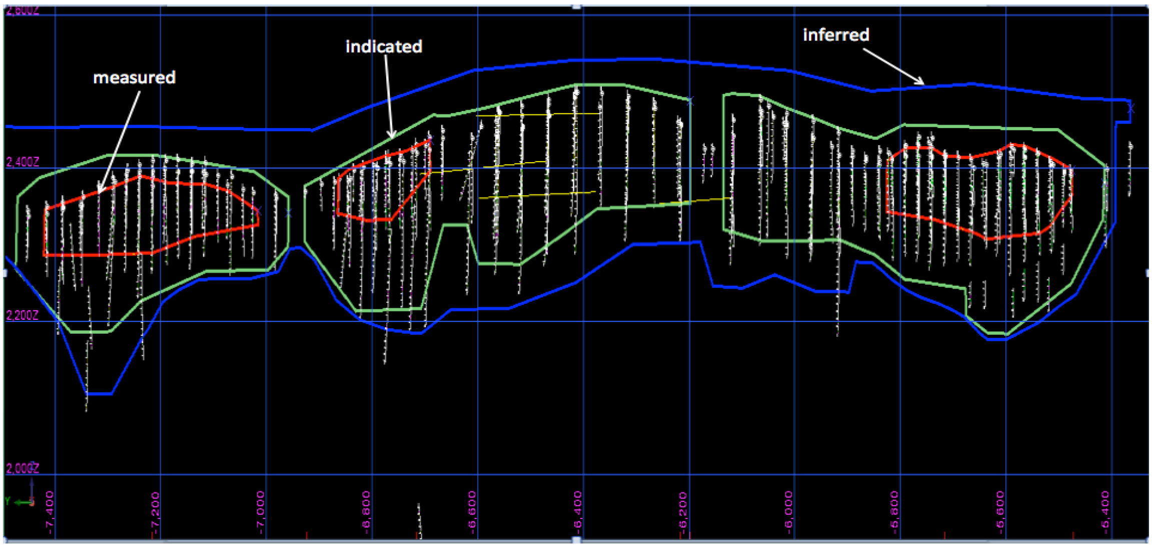

Figure 13: Digitized polygons for classification of mineral Resources

Resource and reserve tonnage calculation

Tonnage for each respective block was obtained by weighting volumes corresponding to the interpreted ore zones and the specific gravity of the block. Specific gravity values were assigned to blocks based on its position above or below the given oxidation surface. Blocks above the surface were assigned an oxide value of 1.56 g/cm^3 whereas blocks below were assigned a fresh rock value of 2.75 g/cm^3.

3.5 Financial Analysis

The economics of the SAMEVA Project were evaluated using a real, after-tax discounted cash flow (DCF) model on a 90% project equity basis, calculated as:

$$DCF = \frac{CF_1}{(1+r)^1} + \frac{CF_2}{(1+r)^2} + \ldots + \frac{CF_n}{(1+r)^n}$$

CF_n = cash flow in year n;
r = discount rate;
n = n^{th} year of investment

Projected production, revenues, operating costs, capital costs and taxes were considered in the financial model (Figure 14) below. The main economic assumptions were a US$1,200/oz gold price and a 10% discount rate.

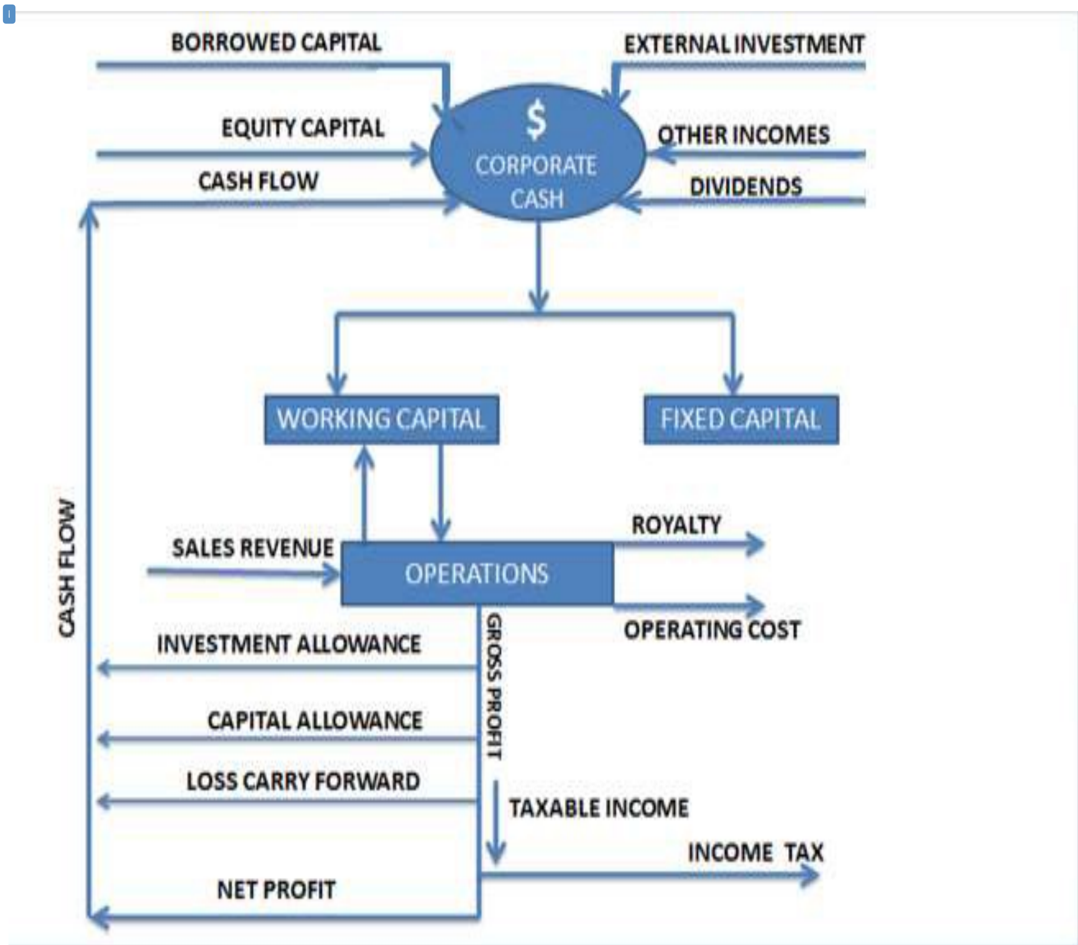

Figure 14: Cash flow model for mineral project evaluation in Ghana (modified after Stermole, 1993)

3.5.1 Cost estimates

Capital cost estimates

Capital costs estimates include pre-production costs, ongoing construction and replacements costs, infrastructure, vehicles, closure and rehabilitation costs.

Operating and processing costs

Estimates on operating and processing costs were based on figures from the adjacent producing mines of Chirano Gold mines and Noble Gold Mines, as well as the national mining costs index. Details of capital expenditure and operating cost estimates are attached as Appendix H and I

3.5.2 Sensitivity Analysis

Analysis of project sensitivities was done using a spider diagram. As shown in Table 5, the analysis was done by adjusting the three most crucial parameters from -10% to +10% from the base case estimates. The tabulated data (Table 5) was used to plot a graph in excel to describe changes in NPV (Figure 15) as the three parameters were modified.

Table 5: Project sensitivities

Sensitivity Area	Percentage Change	Values		Percentage Change	
		NPV (USD M)	IRR(%)	NPV(%)	IRR(%)
Gold Price	-10%	295	45.0	(22)	(18)
	-5%	337	50.0	(11)	(9)
	0%	379	55.0	0	0
	5%	420	59.0	11	7
	10%	462	63.0	22	15
Total Capital Cost	-10%	381	57.0	0.5	4
	-5%	380	56.0	0.3	2
	0%	379	55.0	0.0	0
	5%	378	54.0	(0.3)	(2)
	10%	377	53.0	(0.5)	(4)
Total Operating Cost	-10%	380	57.5	(0.5)	5
	-5%	379	56.5	(0.3)	3
	0%	379	55.0	0.0	0
	5%	378	53.9	0.0	(2)
	10%	377	52.1	0.3	(5)

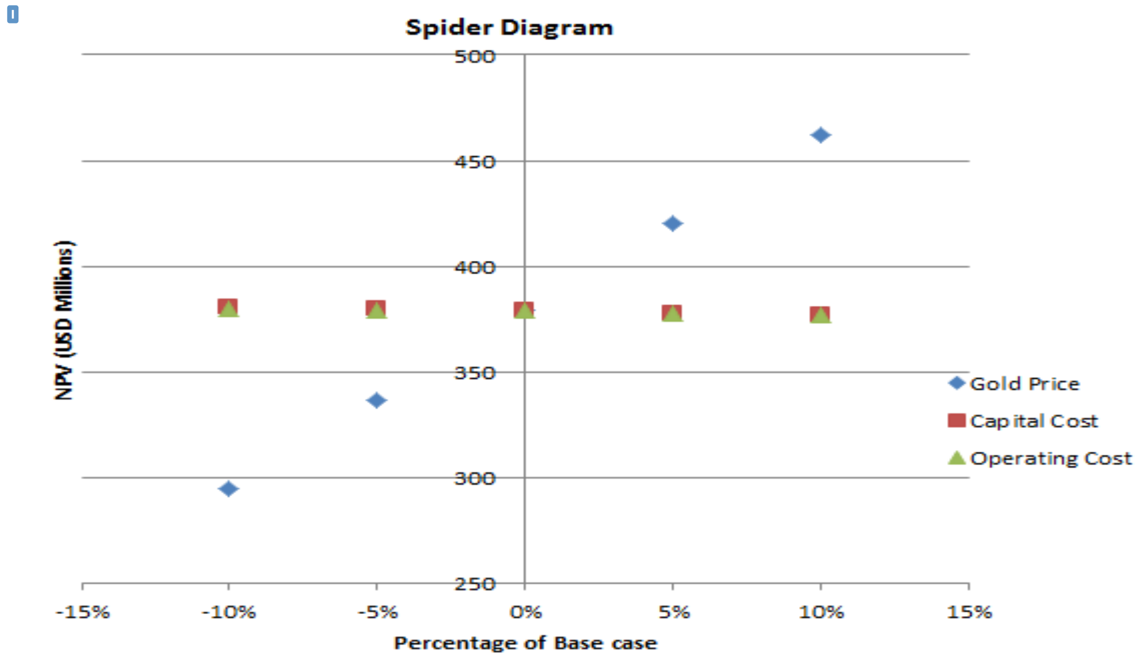

Figure 15: Determining the effect of price/cost changes on NPV

CHAPTER FOUR

RESULTS AND DISCUSSION

4.1 Deduction on Data validation and Integrity

Spatial (geostatistical) analysis of the data – that is, the calculation and modelling of semi-variograms is a precursor to any of the kriging methods. The semi-variogram (Figure 16) is a graph relating the degree of similarity between sample grades to the distance between them along a given orientation.

Structural analysis through the semi-variogram was used to describe the spatial variability of the gold attribute. The calculation of variograms in different directions (Figures 18 to 20) along strike, across strike and down dip gave an important insight into the (an)isotropy of the orebody. The primary and secondary planar variograms were used to determine the direction of maximum continuity along strike and across strike of the orebody respectively.

Fitted to these experimental variograms are a series of mathematical models, which when used in the kriging algorithm created the spatial continuity observed in the dataset (Hadlow, Khosrowshah and Vearucombe, 1993). The grades of samples separated by distances greater than the range of influence are uncorrelated, implying that such samples cannot be used to estimate locations outside this range. The variogram model of data follows the classical definition of a spherical model; a linear behaviour at small separation distance near the origin but flattens at large distances (Figure 16) with the tangent of the origin reaching the sill at about two-thirds of the range (Vann, 2007).

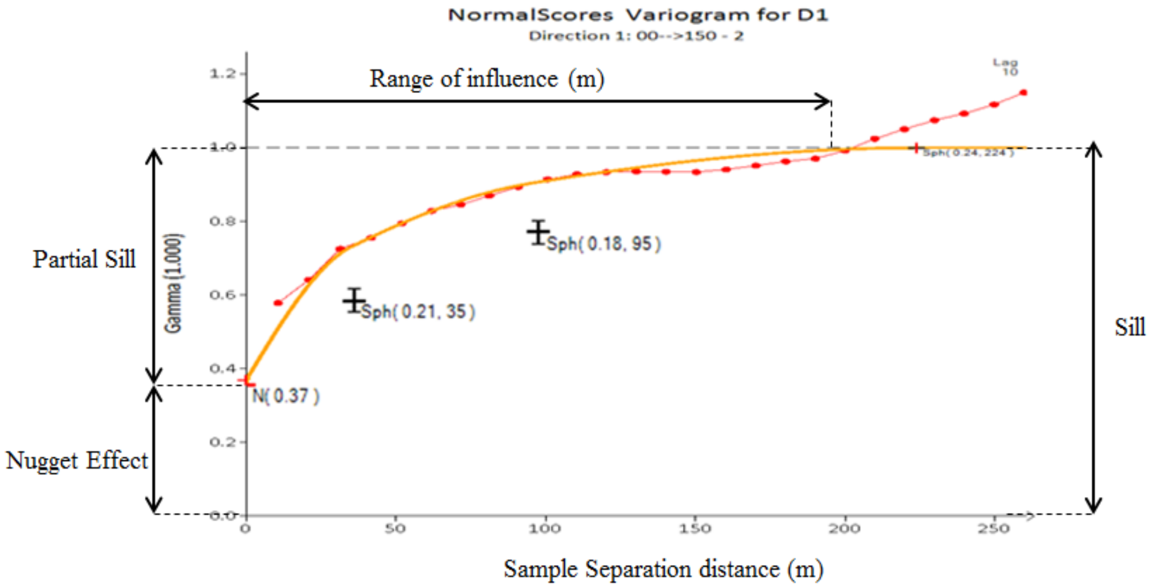

Figure16: The components of a variogram model

Nugget effect describes reproducibility of results when the sampling is repeated at the same location. It incorporates both the natural inherent variability of the deposit and the artificial variability due to human processes; sampling size, sample preparation, analysis, etc. (Snowden, 2001). Variography results produced nugget effect (Figure 17) lower than 25% of the total sill, indicating relatively low variability and good continuity between adjacent composites. The low nugget effect also testifies to good quality assurance and quality control (QAQC) and sampling protocols used since ensuring rigorous QAQC protocols during sample collection, transport, preparation and analysis can reduce the artificial component of the nugget effect (Carrasco, 2010).

Recognition of the level of the nugget effect is vital to resource estimation. The more homogeneous the mineralization, the lower the nugget effect. The low nugget achieved implies finely disseminated mineralization, which will tend to give easily reproducible results but heterogeneous mineralization will be sensitive to the method of sampling and could give variable results from a single location.

The higher the nugget effect, the lower the likelihood of being able to achieve a high degree of selectivity during mining.

This affects classification in that a high nugget effect resource quoted at a low cut-off may have a higher degree of resource confidence than if it were reported at a higher cut-off.

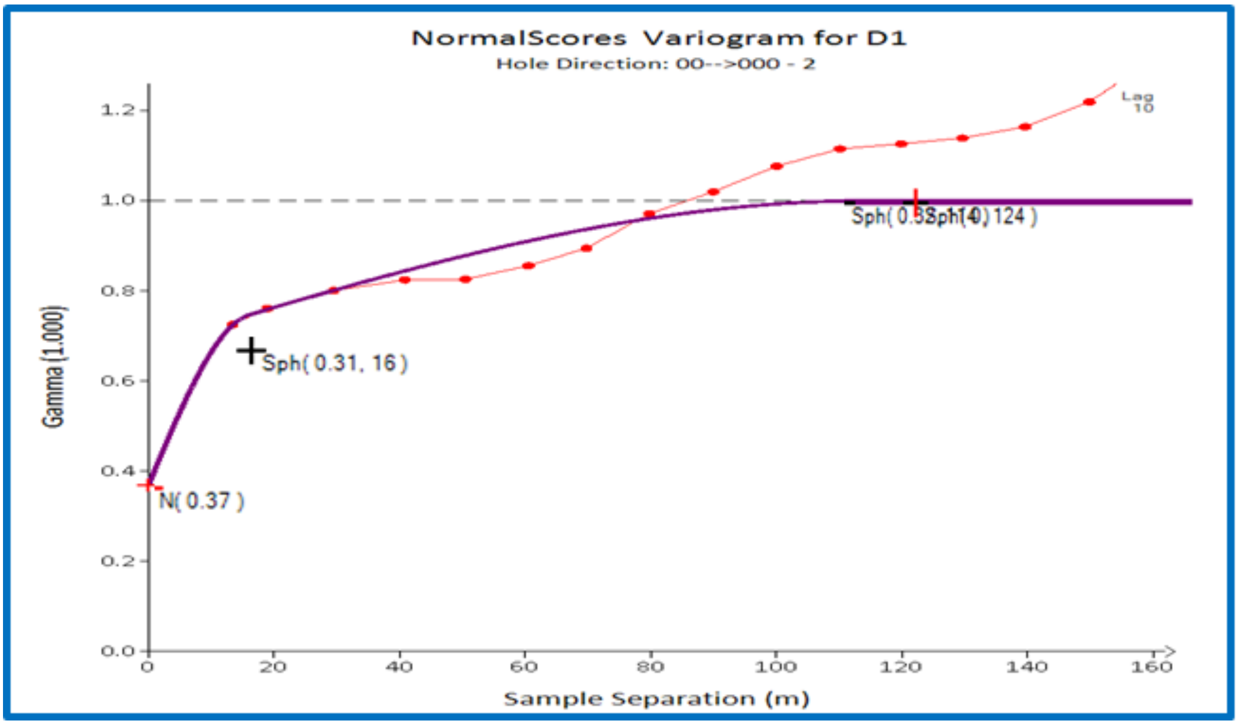

Figure17: Downhole variogram for Au - determining the nugget effect

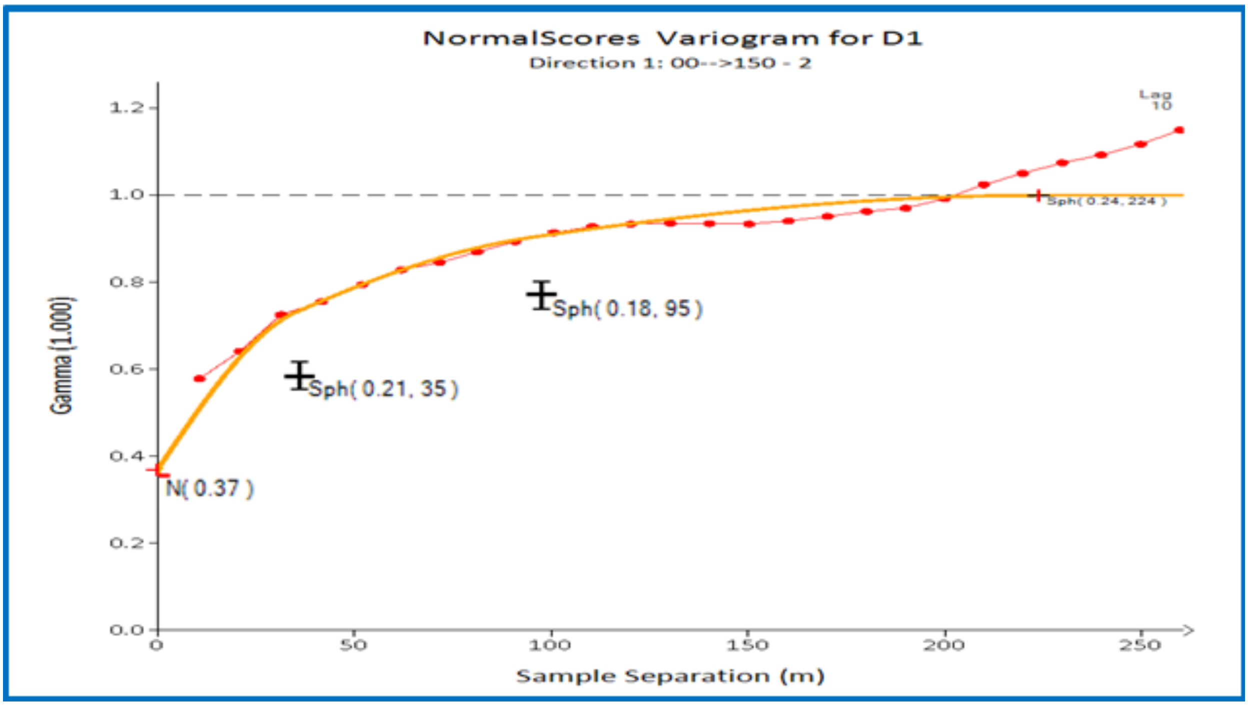

Figure 18: Variogram modelling for direction 1

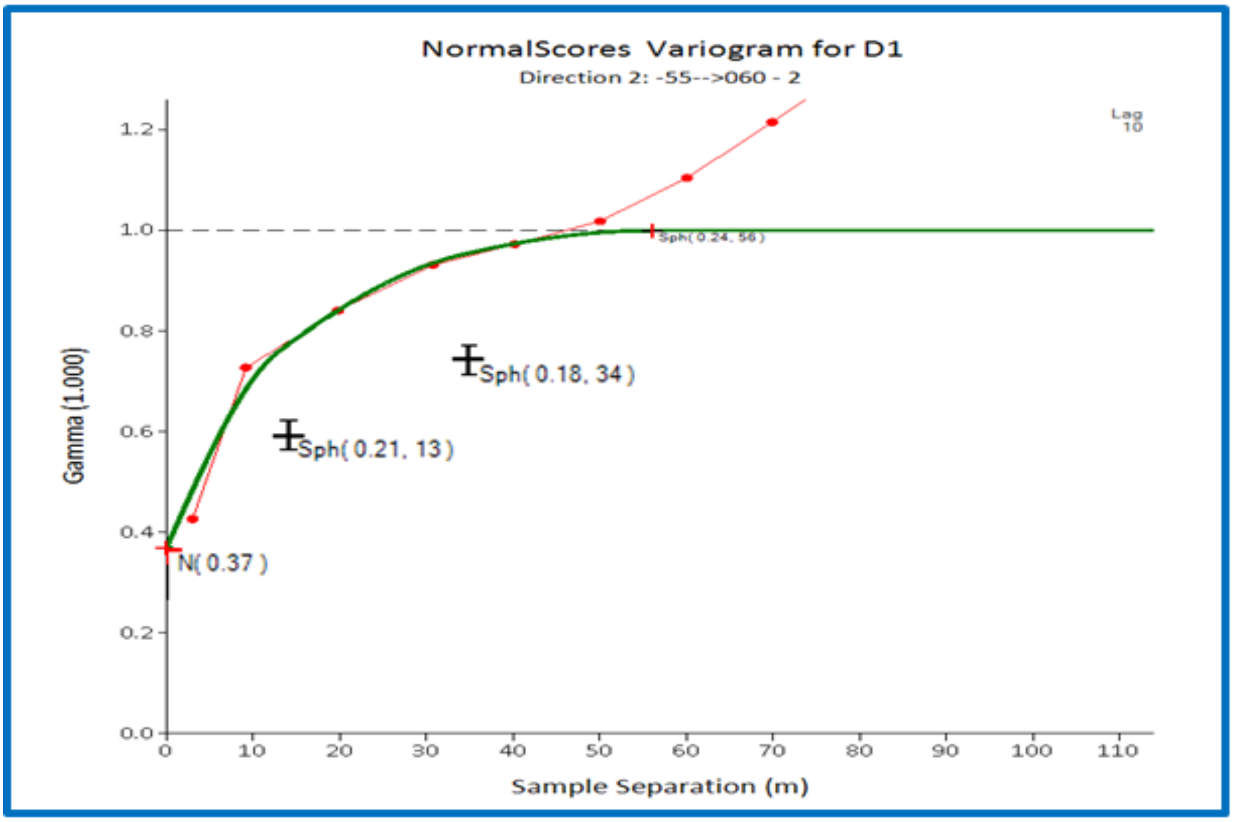

Figure 19: Variogram modelling for direction 2

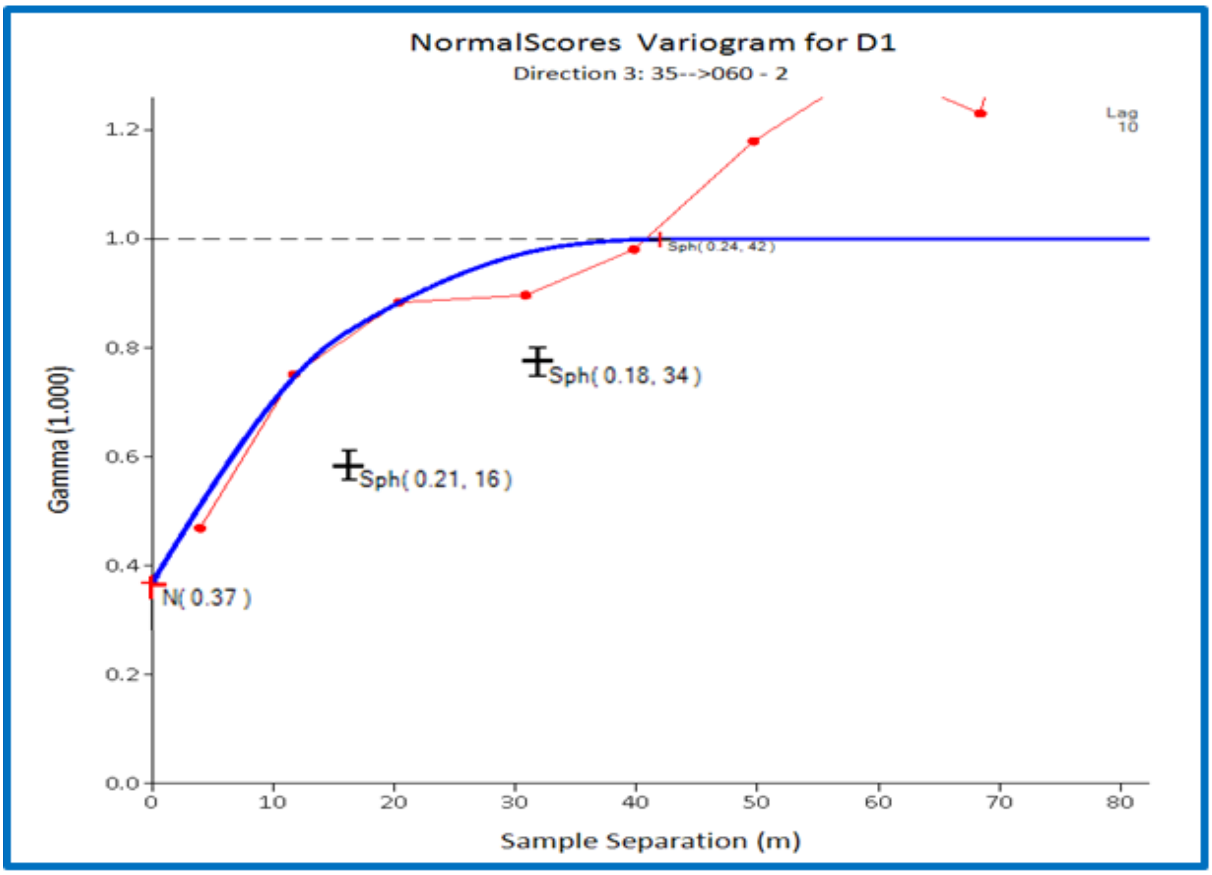

Figure20: Variogram for modelling direction 3

4.2 The Resource Model and Estimation

The main factors that were considered in arriving at the Mineral Resource/ Reserve Estimates (Tables 6 to 8) were the amount of data, distribution and quality of the data, reliability of geological interpretation, assumptions regarding costs and commodity prices. In determining block size to be used, it should ideally bear some relation to the mining equipment planned for use, often referred to as the selectivity. The concept of the selective mining unit is the smallest parcel of ground on which mining decisions, such as the allocation to ore or waste, may be made.

The block dimensions should also be considered in relation to the sampling grid; blocks that are too small will result in over smoothing of the sample data and subsequent very low precision results. Over smoothing results in conditional bias, whereby high-grade blocks are underestimated and low-grade blocks are overestimated (Krige, 1996). Supervisor's KNA tool was used as a guide in arriving at the block size.

Drillholes are spaced at 25 m sections and 25 to 75 m on long sections. A block size of 10 m x 10 m x 6 m was selected in order to accommodate the nature of the mineralization and be amenable for the open pit potential. The block model was sub-celled on a 5 mx 5 m x3m pattern in the YZ plane which allows the parent block to be split in each direction to more accurately fill the volume of the wireframes, thus more accurately estimate the tonnes in the resource.

The resource estimate was done using the ordinary kriging method on a capped and composited borehole dataset. Whatever the estimation approach adopted, all techniques which seek to interpolate grades into blocks (except the nearest neighbour method) depend on the sample search procedure. The methods by which samples are selected for subsequent weighting are critical to the process, and in some cases (Carras, 1998) are of more consequence than the estimation algorithm itself. A good sample search plan should have some or all of the following features; de-clustering of data via octant or quadrant selection, restrictions on the number of samples from one drill hole, minimum and maximum numbers of samples specified for search, preferred search directions (i.e. anisotropic search); and restriction of the influence of high- grade samples.

Validation of the results was conducted through the use of swath plots (Figure 21), visual inspection (Figure 22), and global statistical comparison of the model against a nearest neighbour (NN) model. As is typical for exploration data, the composites were slightly more variable than the block grades (Figure 21). This may imply some smoothing in the resource model estimates. The potential for smearing high-grade samples elsewhere within the deposits was controlled by the kriging process.

As depicted by Figure 23, the ore zone is quite well-defined.

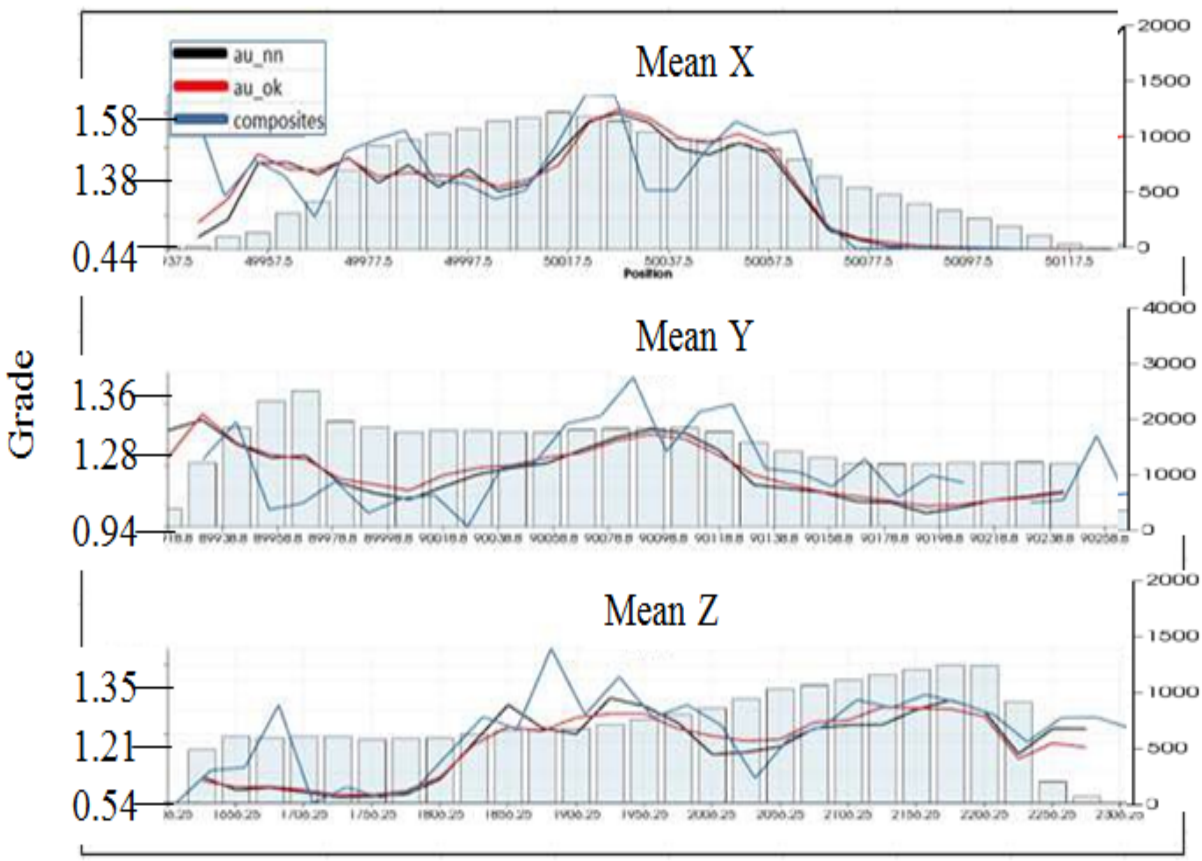

Figure 21: Comparing Ordinary krigged estimates to Nearest neighbour estimates

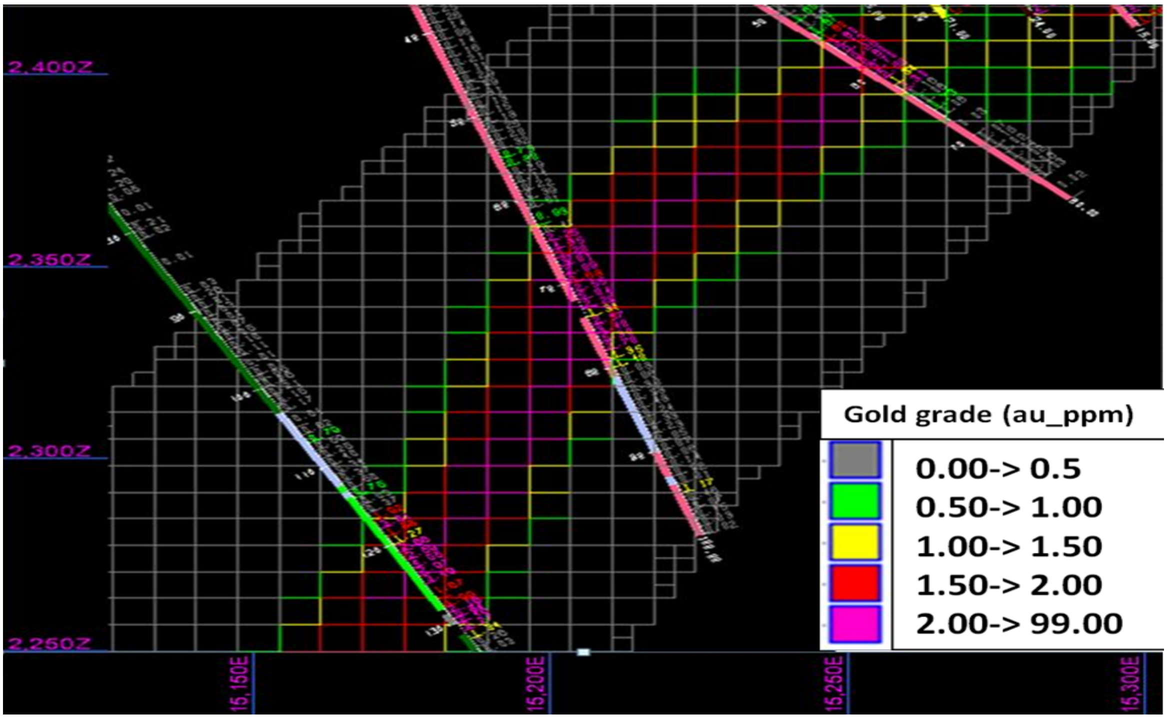

Figure 22: Section showing good comparison between drillholes and estimated grades

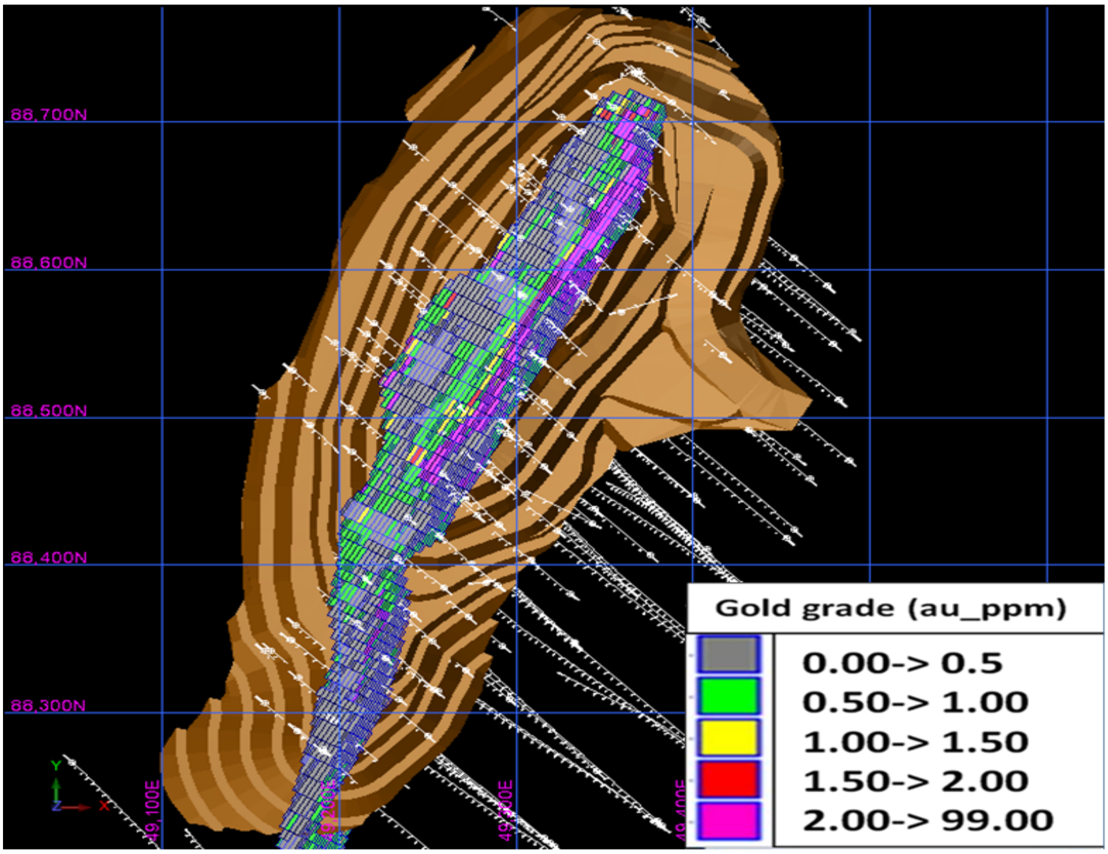

Figure 23: Plan view of model at 0.5g/t cut-of

Table 6: Resource summary indicating Metal content at increasing cut-off

cut-off g/t	Tonnes	Au Grade g/t	contained ounces
0.20	137,676,970	0.78	3,452,603
0.30	112,213,205	0.87	3,138,730
0.40	74,580,205	1.19	2,853,391
0.50	54,885,865	1.47	2,593,992
0.60	46,847,742	1.55	2,334,593
0.70	40,341,111	1.62	2,101,133
0.80	34,396,105	1.71	1,891,020
1.00	26,615,668	1.79	1,531,726
1.10	22,686,688	1.89	1,378,554
1.20	18,289,126	2.11	1,240,698

Table 7: Resource summary by category

Resource Classification	Tonnage	Grade	Ounces
Measured	28,237,758	1.60	1,452,583
Indicated	16,425,907	1.28	675,974
Inferred	10,455,010	1.44	484,036
Total	55,118,674	1.47	2,612,593

Table 8: Reserve summary

Reserve Classification	Tonnage	Grade	Ounces
Proven	26,714,874	1.45	1,241,349
Probable	10,699,124	1.29	443,740
Total	37,413,998	1.40	1,685,089

Criteria for Classifying Mineral Resources as Measured, Indicated or Inferred

- Confidence in geological and grade continuity as depicted in variograms (Figure 24)
- Quantity and distribution of drillhole data (Figure 25)

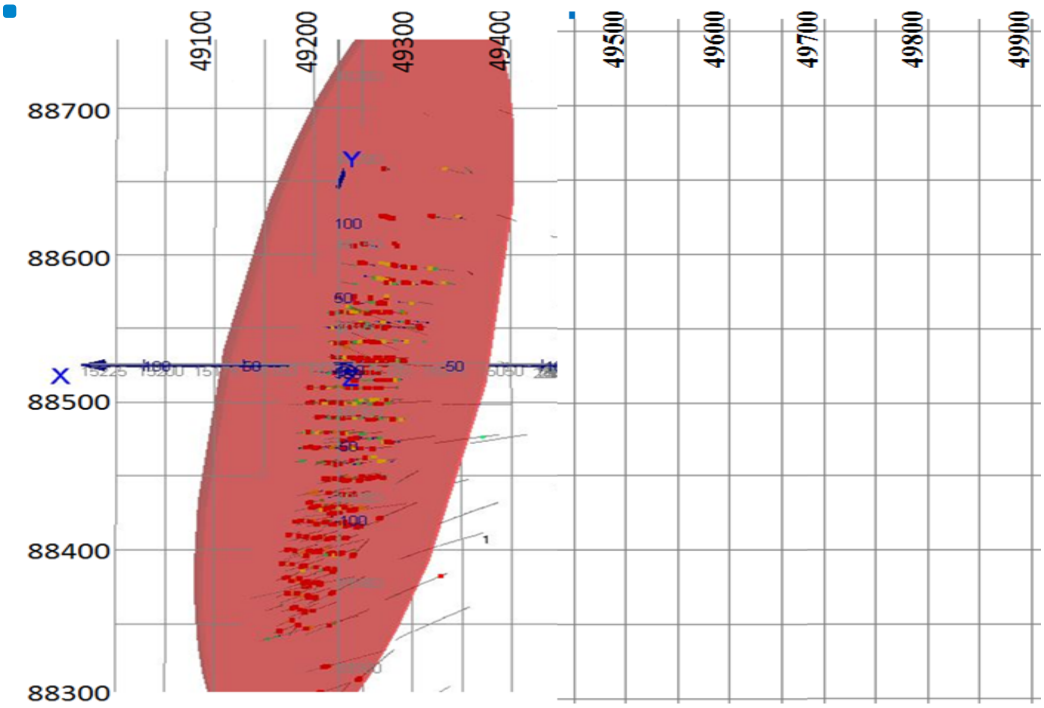

Figure 24: Variogram ellipsoid showing Grade anisotropy in the data

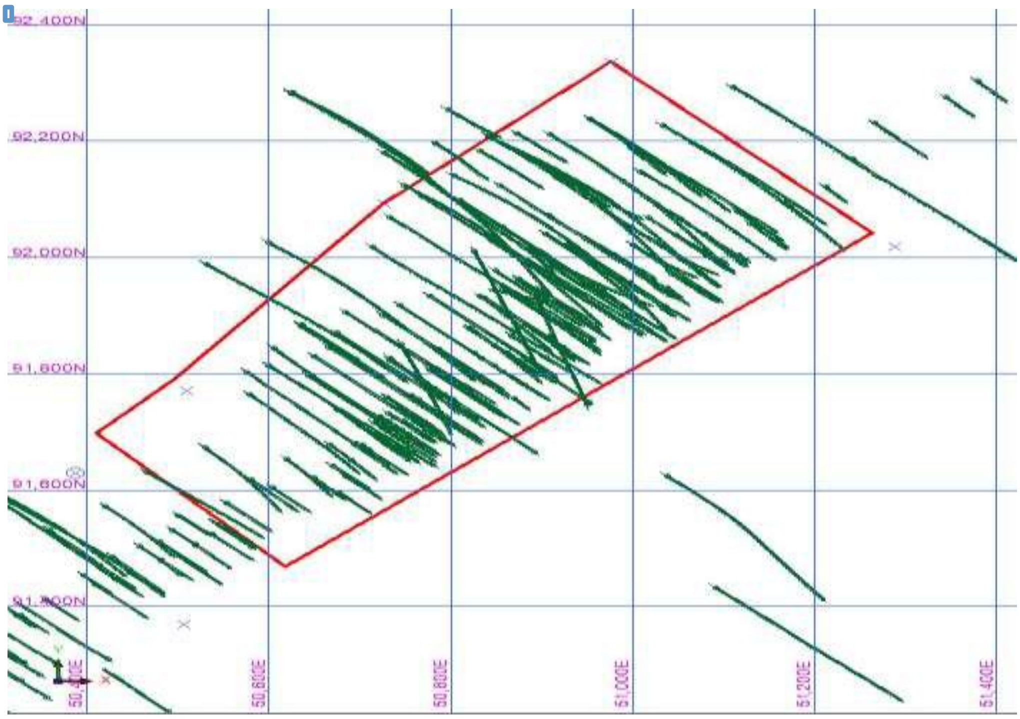
Figure 25: Plan view of drill holes showing good sampling coverage

4.3 Financial analysis

To overcome the causes of unreliable financial estimates, cost parameters and assumptions were based on benchmarking of the adjacent mining operation of Chirano Gold Mines Ltd and Noble Gold ltd. This was done to ensure estimates were close to reality.

Some assumptions upon which the financial analysis was done are that mining will be by conventional open pit bench mining using truck and shovel method, a gold price of US$1,200/oz, a 10% discount rate, metallurgical recovery tests yielding 96% for oxide and 92% for sulphides. The project was evaluated using a real, after-tax discounted cash flow model (Figure 26) on a 90% project equity basis. The annual rates of production were then estimated based on the projected excavator capacity. Based on the recoveries and gold price of USD 1,200, corporate tax of 35%, royalties of 5.6% and the other costs, revenues could then be calculated as shown in Appendix I.

Table 9: Financial analysis results –RC versus DD data

Project Data	Estimated Value (RC Data)	Estimated Value (DD data)
Life of Mine	11 years	9 years
Total Gold Produced	1.6 million ounces	1.2 million ounces
Total Ore Mined	37 million tonnes	26 million tonnes
Total Material Mined	121 million tonnes	109 million tonnes
Open Pit Strip Ratio	2.30	2.60
Initial Project Capital cost	US$ 32 million	US$ 32 million
Total Mining Cost per Ounce Mined	US$204/oz	US$256/oz
Total Mining Cost per Tonne	US$2.83/t	US$2.83/t
Base Case Gold Price	US$1200/oz	US$1200/oz
Before Tax Net Present Value @ 0%	US$ 834 million	US$ 529 million
After Tax Net Present Value @ 0%	US$ 542 million	US$ 343 million
After Tax Net Present Value @ 5%	US$ 379 million	US$ 252 million
After Tax Net Present Value @10%	US$ 270 million	US$ 187 million
After Tax Internal Rate of Return	0.59	0.58
Payback Period (from start –up)	2.12 years	3.1 years

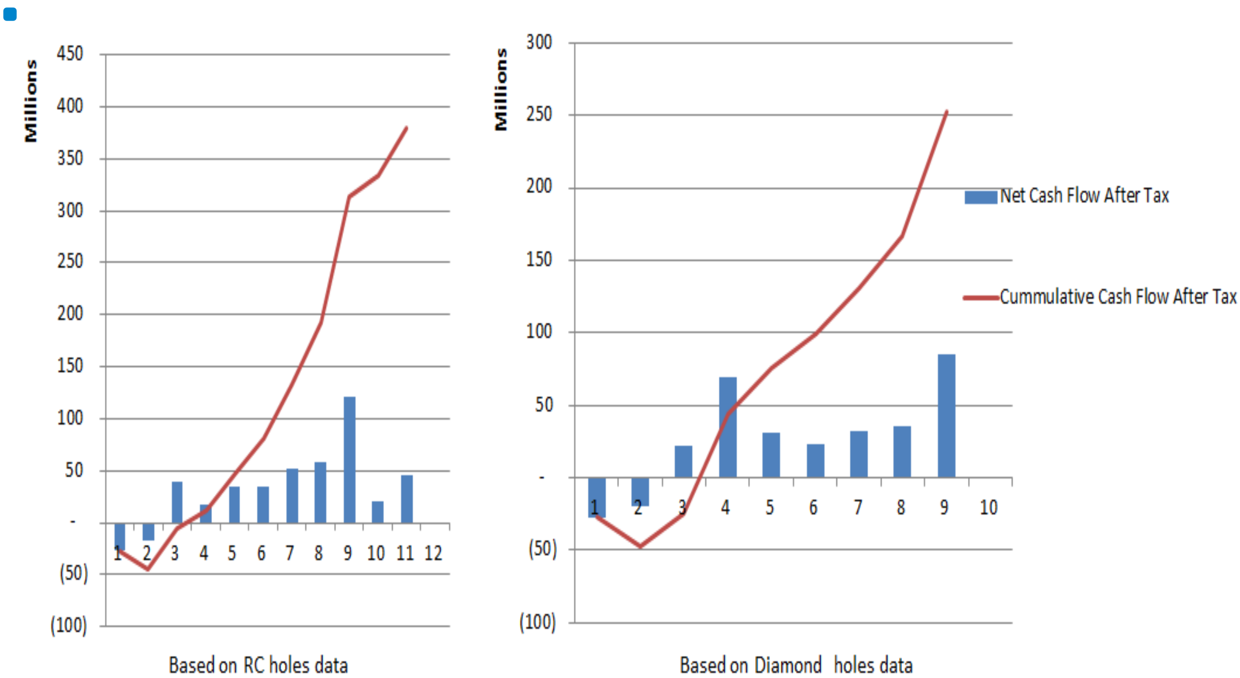

Figure 26: Discounted cash flow model – RC data vrs DD data

Project Sensitivities

Sensitivity analysis is a tool for project failure prediction and a potential risk mitigation tool. A sample spider plot (Figure 27) showing the effects of metal price changes, changes in Capital cost, and operating costs on NPV. The sensitivity analysis indicates that changes in operating and capital costs do not have any appreciable impact on the project because these parameters lie virtually flat on the curve and are not steep as the gold price parameter. Stability of gold price on the market, which unfortunately cannot be controlled by the investor, can impact very negatively on the project.

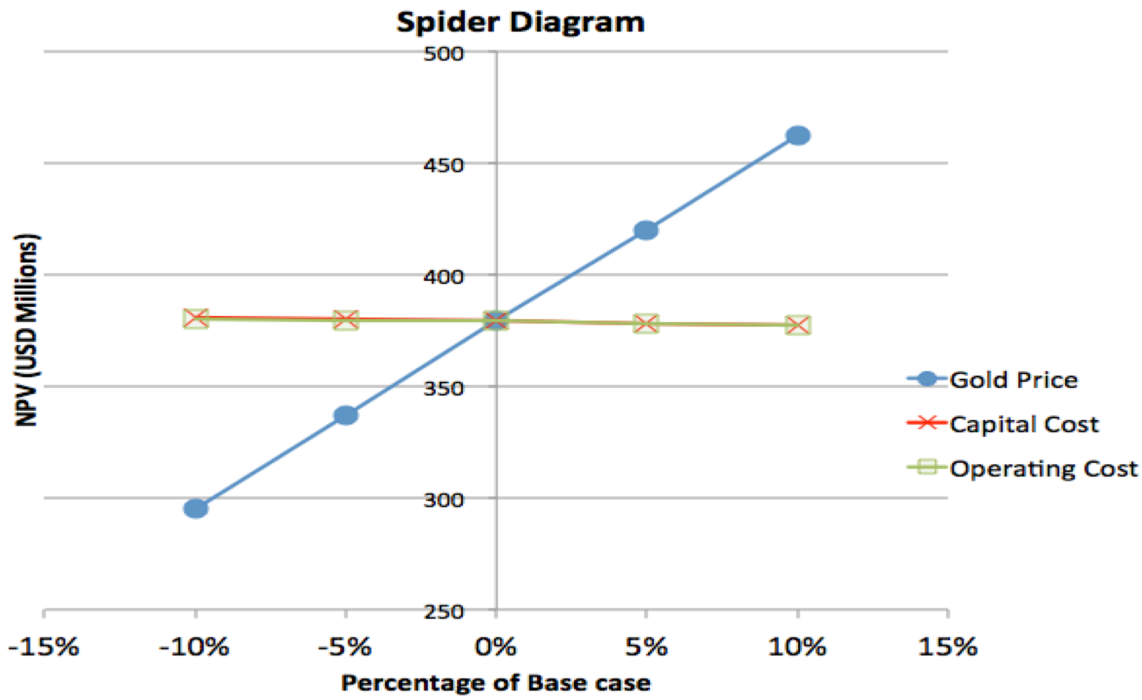

Figure 27: Sensitivity Analysis Graph

The factors that cause poor resource estimation: unreliable data and/or insufficient data; poor geological interpretation; inappropriate estimation methodology; inadequate checks and alternative estimates, inappropriate financial analysis; reliance on unsustainable market and pricing forecast; excessive optimism on production forecasts (English, 1984) have been tackled in this research work. Ground truthing, which refers to actual site visits and checks on data, was a key component of this work.

Table 10: Financial analysis results and sensitivities

Financial metric	Unit	Gold Price (US$/oz)				
		1,100	1,200	1,300	1,350	1,400
NPV at 0% discount rate	US$ (million)	447.22	542.34	637.47	685.03	732.59
NPV at 5% discount rate	US$ (million)	310.18	379.71	449.24	484	518.77
NPV at 10% discount rate	US$ (million)	218.65	270.92	323.19	349.33	375.47
Internal rate of return	%	50%	59%	67%	71%	75%
Payback year	Year	2017	2017	2017	2017	2017

CHAPTER FIVE

CONCLUSION AND RECOMMENDATION

5.1 Conclusion

This work provides insight concerning the mineral resource evaluation of the Kojina gold deposit on the Sefwi-Bibiani gold belt. The major findings regarding the project objectives are stated as following;

i. Using appropriately planned drilling and representative sampling as well as predominant use of diamond core gives more accurate information since the sample is in-situ with no contamination and has structural orientation which allows for better geologic and structural mapping therefore helping define the mineralization zones more accurately.

ii. The mineralization interpretation based on reverse circulation holes gave 37,413,998 tonnes at 1.47g/t whereas that based on the diamond holes gave 26,674,888 tonnes at 1.57g/t.

iii. Based on the more accurate interpretation from diamond holes, the Net-Present value of the project was estimated at USD 187,796,325 with a payback period of three years.

1.2 Recommendations

1. SAMEVA Ltd should go ahead with the acquisition of the Kojina deposit and immediately commence feasibility studies.
2. That diamond drill holes be included when evaluating structurally controlled mineral deposits.

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APPENDICES

Appendix A – Database Tables Summary

Surpac Minex Group	2014				
Kojina Project - Database Tables Summary					
Number of drillholes: 182	Total drilled length:	25,827.28m			
Table Name	**Data Type**	**Records**			
assay	interval	24,322			
collar		182			
geology	interval	3,105			
redox	point	164			
rqd	interval	3,271			
survey		2,037			
	Hole Id	**Northing**	**Easting**	**Elevation**	**Depth**
MIN. NORTHING	CHRC2080	87838.26	49183.26	2322.11	100
MAX. NORTHING	CHRC1567	88946.59	49366.04	2412.48	108
MIN. EASTING	CHRC1817	88266.91	49048.59	2427.95	150
MAX. EASTING	CHRC1868D	88558.10	49489.21	2300.15	186
MIN. ELEVATION	CHDD1823	88471.69	49487.63	2291.66	58
MAX. ELEVATION	CHRC1817	88266.91	49048.59	2427.95	150
MIN. DEPTH	CHCH001	88388.67	49282.48	2339.28	35
MAX. DEPTH	CHRC1238D	88272.00	49475.14	2318.70	611.5

Appendix B – Drillhole listing used for Resource Estimation

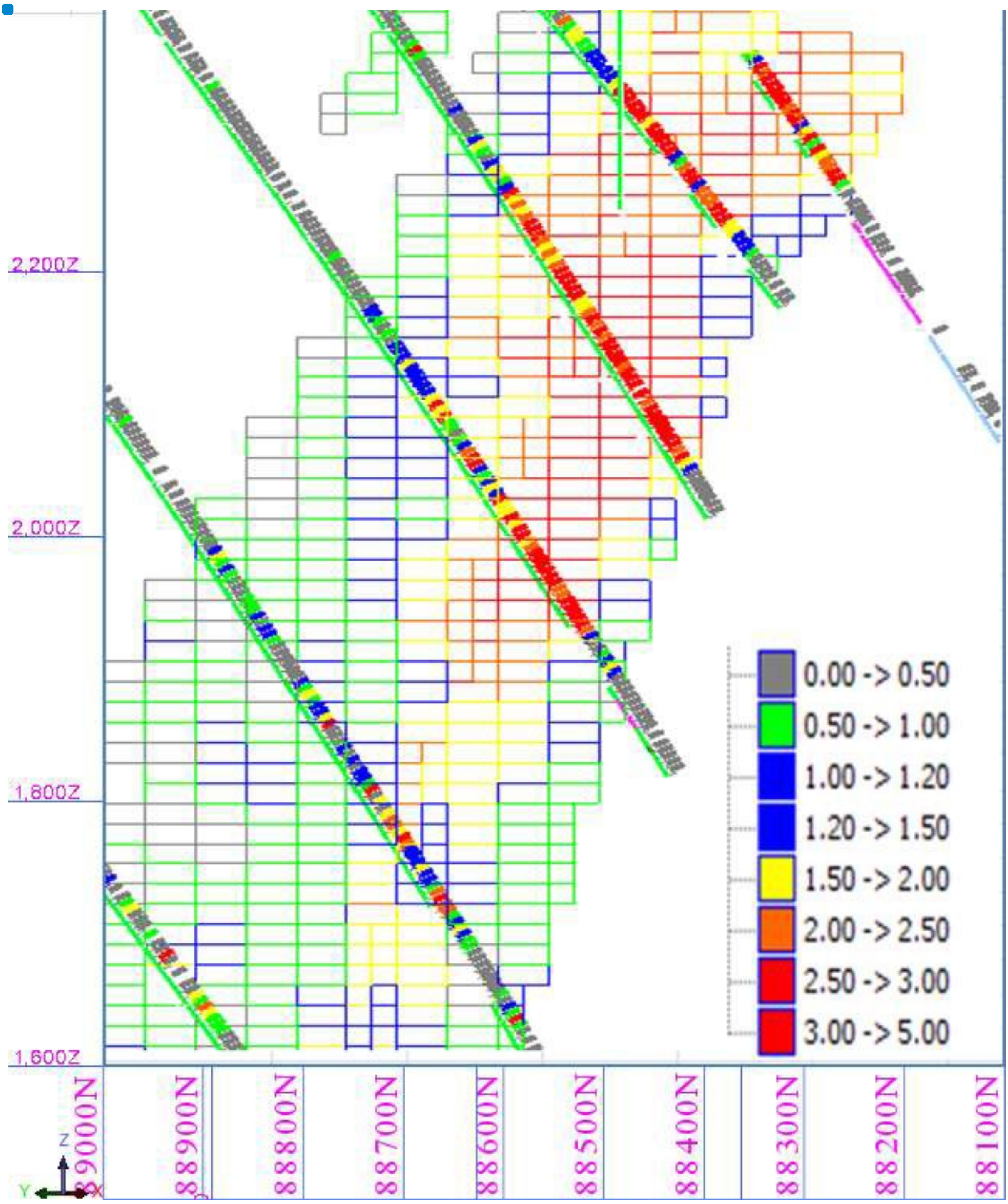
Drillhole listing used for Resource Estimation

Prospect	HoleID	North	East	RL	Azi	Dip	ERC	EOH	Final_C	HoleTyp	Start_RC	Stop_RC	Start_DD	Stop_DD
KOJ	GC_RC690	32203	19204	2411.4	90	-55	120	120	N	RC	7/8/2005	7/9/2005		
KOJ	GC_RC691	32202	19252	2401.5	90	-50	120	120	N	RC	7/9/2005	7/9/2005		
KOJ	GC_RC692	32053	19139	2385.6	90	-50	117	117	N	RC	7/9/2005	7/10/2005		
KOJ	GC_RC693	32099	19141	2383.7	90	-50	93	93	N	RC	7/10/2005	7/10/2005		
KOJ	GC_RC694	32154	19141	2385	90	-55	87	87	N	RC	7/10/2005	7/11/2005		
KOJ	GC_RC695	32203	19204	2411.4	90	-50	100	100	N	RC	7/11/2005	7/11/2005		
KOJ	GC_RC696	32202	19252	2401.5	90	-50	87	87	N	RC	7/11/2005	7/11/2005		
KOJ	GC_RC697	32151	19351	2413.1	90	-50	87	87	N	RC	7/12/2005	7/12/2005		
KOJ	GC_RC699	32053	19240	2427.5	90	-50	93	93	N	RC	7/13/2005	7/13/2005		
KOJ	GC_RC700	32097	19221	2423.1	90	-50	120	120	N	RC	7/14/2005	7/14/2005		
KOJ	GC_RC701	32609	19213	2428	90	-60	120	120	N	RC	7/17/2005	7/17/2005		
KOJ	GC_RC707	32522	19249	2396.3	90	-65	90	90	N	RC	7/25/2005	7/25/2005		
KOJ	GC_RC708	32526	19213	2390.4	90	-55	129	129	N	RC	7/26/2005	7/26/2005		
KOJ	GC_DD1708	39546	14960	2408.2	87.5	-62		302	Y	DD			11/2/2011	11/14/2011
KOJ	GC_DD1711	39478	14975	2421.7	90.8	-54		287	Y	DD			11/14/2011	11/19/2011
KOJ	GC_DD1712	40125	14927	2358.7	90.9	-52		244	N	DD			11/17/2011	11/26/2011
KOJ	GC_DD1713	39575	15014	2401.2	89.9	-55		200	Y	DD			11/22/2011	11/26/2011
KOJ	GC_DD1715	39525	15022	2409.1	89.8	-62		220	Y	DD			11/26/2011	11/30/2011
KOJ	GC_DD1716	40175	14930	2360	89.1	-47		245	Y	DD			11/26/2011	11/30/2011
KOJ	GC_DD1717	39503	14968	2416.9	89	-60		270	Y	DD			11/30/2011	12/3/2011
KOJ	GC_DD1718	40070	14985	2385.7	84.7	-60		218	Y	DD			12/1/2011	12/6/2011
KOJ	GC_DD1719	39600	14987	2397.3	90.1	-58		228	Y	DD			12/6/2011	12/9/2011
KOJ	GC_DD1720	40125	14925	2358.9	88.7	-64		305	Y	DD			12/6/2011	12/12/2011
KOJ	GC_DD1721	39625	14980	2397.2	90.6	-50		203	Y	DD			12/9/2011	12/12/2011
KOJ	GC_DD1722	40070	14989	2385.8	84.6	-47		178	Y	DD			12/12/2011	12/15/2011
KOJ	GC_DD1723	40025	15010	2410.3	90.4	-63		201	Y	DD			12/12/2011	12/16/2011
KOJ	GC_DD1726	40022	14953	2416.4	88.8	-63		137	Y	DD			12/16/2011	1/13/2012
KOJ	GC_DD1727	40000	15018	2410.7	90	-64		162	Y	DD			1/13/2012	1/17/2012
KOJ	GC_DD1728	39925	15054	2407.4	90.1	-63		152	Y	DD			1/13/2012	1/18/2012
KOJ	GC_DD1729	39975	15026	2411.3	90.5	-63		151	Y	DD			1/17/2012	1/19/2012
KOJ	GC_DD1730	39900	15041	2408.5	89.9	-58		140	Y	DD			1/18/2012	1/20/2012
KOJ	GC_DD1731	39953	15034	2411.9	92.6	-57		151	Y	DD			1/20/2012	1/23/2012
KOJ	GC_DD1732	40159	14995	2358.4	95.9	-54		143	Y	DD			1/21/2012	1/23/2012
KOJ	GC_DD1733	40025	15026	2405.1	90	60		151	Y	DD			1/22/2012	1/24/2012
KOJ	GC_DD1735	39850	15054	2389.9	90.2	-60		146	Y	DD			1/25/2012	1/27/2012
KOJ	GC_DD1737	40048	15019	2399.2	88.1	-57		160	Y	DD			1/27/2012	2/1/2012
KOJ	GC_DD1742	40021	14956	2416.2	88.4	-65		292	Y	DD			2/18/2012	2/24/2012
KOJ	GC_DD1744	40075	14723	2388.7	86.2	-63		671	Y	DD			2/28/2012	3/13/2012
KOJ	GC_DD1747	39500	14731	2487	85.4	-63		733	Y	DD			3/13/2012	3/28/2012
KOJ	GC_DD1781	39345	14981	2439.4	90	-60		157	Y	DD			6/11/2012	6/13/2012
KOJ	GC_DD1784	39344	15101	2474.2	89.9	-60		167	Y	DD			6/14/2012	6/16/2012
KOJ	GC_DD1789	39350	15043	2444.4	91	-60		154	Y	DD			6/19/2012	6/22/2012
KOJ	GC_DD1794	39342	15101	2474.2	89.9	-60		78	Y	DD			6/22/2012	6/23/2012
KOJ	GC_DD1796	39400	15003	2429.4	90	-60		224	Y	DD			6/24/2012	6/27/2012
KOJ	GC_DD1803	39300	15083	2470.3	90	-60		132	Y	DD			7/3/2012	7/7/2012
KOJ	GC_DD1808	39251	15040	2470.1	90	-60		144	Y	DD			7/8/2012	7/12/2012
KOJ	GC_RC1008	39475	15080	2425.7	90	-60	117	117	Y	RC	3/15/2007 0:01	3/15/2007		
KOJ	GC_RC1009	39500	15067	2420.7	90	-61	126	126	Y	RC	3/16/2007 11:04	3/16/2007		
KOJ	GC_RC1012	39575	15077	2400.8	90	-63	116	116	Y	RC	3/17/2007 22:00	3/17/2007		
KOJ	GC_RC1013	39625	15079	2386.8	90	-67	120	120	Y	RC	3/16/2007 23:56	3/16/2007		
KOJ	GC_RC1015	39600	15130	2403.3	90	-57	50	50	Y	RC	3/17/2007 10:05	3/17/2007		

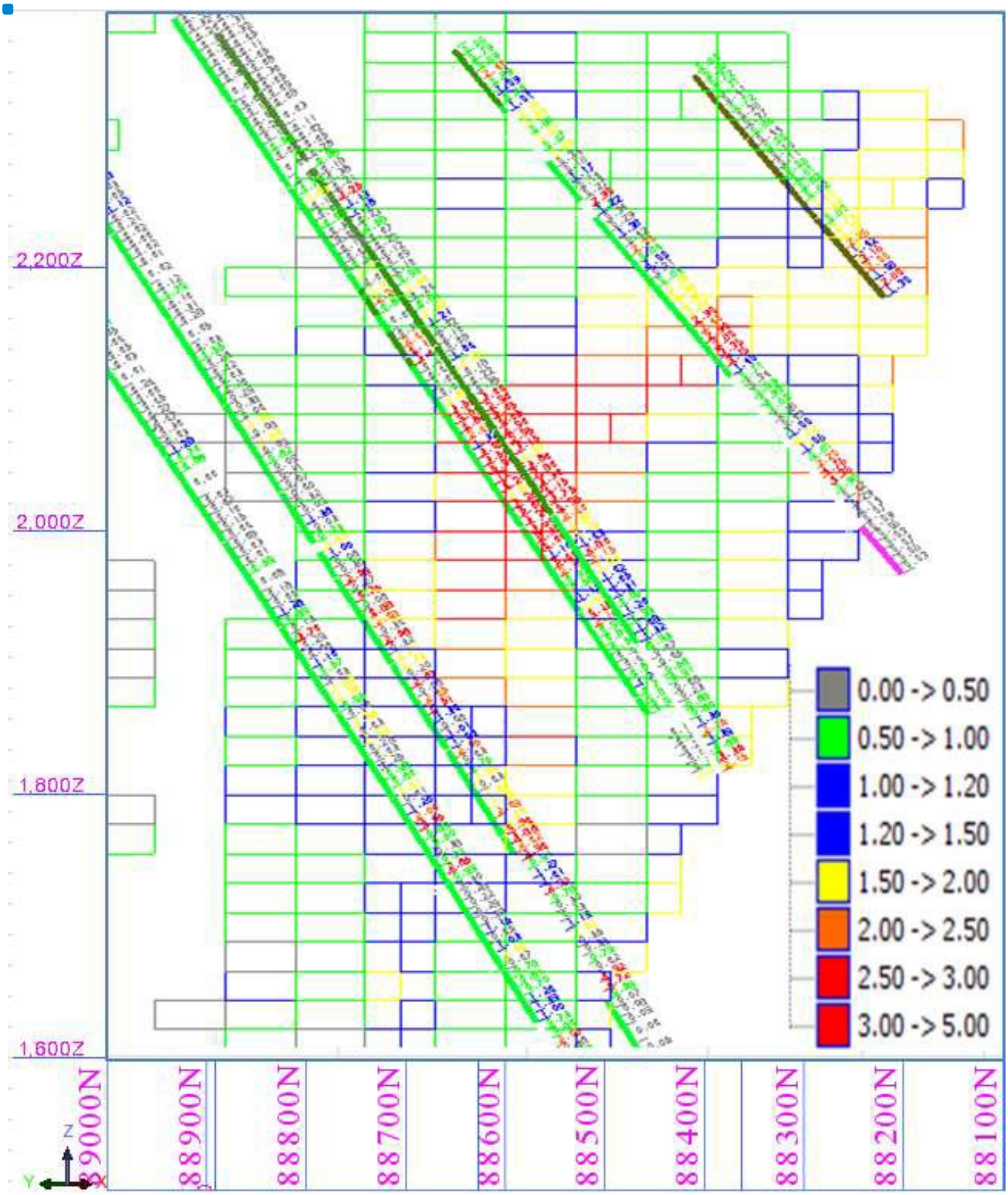
Prospect	HoleID	North	East	RL	Azi	Dip	ERC	EOH	Final_C	HoleTy	Start_RC	Stop_RC	Start_DD	Stop_DD
KOJ	GC_RC1017	39552	15130	2416.6	90	-59	68	68	Y	RC	3/17/2007 8:37	3/17/2007		
KOJ	GC_RC1018	39550	15068	2407.6	90	-60	114	114	Y	RC	3/17/2007 21:15	3/17/2007		
KOJ	GC_RC1019	39925	15098	2397	90	-68	85	85	Y	RC	3/20/2007 8:16	3/20/2007		
KOJ	GC_RC1020	39450	15138	2450	90	-69	59	59	Y	RC	3/20/2007 14:07	3/20/2007		
KOJ	GC_RC1021	39475	15139	2441.6	90	-66	60	60	Y	RC	3/20/2007 8:55	3/20/2007		
KOJ	GC_RC1022	40025	15061	2397	90	-62	114	114	Y	RC	3/20/2007	3/21/2007		
KOJ	GC_RC1023	39500	15132	2432.7	90	-56	44	44	Y	RC	3/21/2007 8:34	3/21/2007		
KOJ	GC_RC1024	40170	15085	2347.1	90	-59	40	40	Y	RC	3/21/2007 12:02	3/21/2007		
KOJ	GC_RC1025	39618	15127	2398.8	90	-73	41	41	Y	RC	3/21/2007 14:46	3/21/2007		
KOJ	GC_RC1026	40235	15069	2338.1	90	-45	58	58	Y	RC	3/22/2007 9:18	3/22/2007		
KOJ	GC_RC1027	39904	15093	2396.3	90	-60	83	83	Y	RC	3/22/2007 11:33	3/22/2007		
KOJ	GC_RC1028	40200	15086	2341.1	90	-45	46	46	Y	RC	3/22/2005 16:43	3/23/2007		
KOJ	GC_RC1030	40050	15062	2393.5	90	-58	95	95	Y	RC	3/22/2007 9:23	3/23/2007		
KOJ	GC_RC1032	40075	15061	2388.7	90	-52	80	80	Y	RC	3/23/2007 15:32	3/24/2007		
KOJ	GC_RC1034	40131	15083	2363.3	90	-57	55	55	Y	RC	3/24/2007 14:01	3/25/2007		
KOJ	GC_RC1035	39646	15123	2386.7	90	-56	57	57	Y	RC	3/25/2007 10:50	3/25/2007		
KOJ	GC_RC1802	39395	15061	2440.1	90	-60	126	126	Y	RC	7/4/2012 1:29	7/5/2012		
KOJ	GC_RC1804	39301	15029	2451.9	90	-60	156	156	Y	RC	7/3/2012 14:52	7/4/2012		
KOJ	GC_RC1805	38834	15021	2482.2	90	-60		192	Y	RC	7/5/2012	7/6/2012		
KOJ	GC_RC1806	38825	15083	2468.5	90	-60	96	96	Y	RC	7/6/2012	7/6/2012		
KOJ	GC_RC1807	38714	15077	2428.8	90	60	150	150	Y	RC	7/6/2012	7/7/2012		
KOJ	GC_RC1902	40000	14559	2404.6	90	-60	186	186	Y	RC	10/12/2013 17:43	10/15/2013		
KOJ	GC_RC1903	40400	14707	2417.7	90	-61	170	170	Y	RC	10/18/2013 13:36	10/20/2013		
KOJ	GC_RC1905	40500	14743	2415.3	90	-60	154	154	Y	RC	10/21/2013 12:15	10/22/2013		
KOJ	GC_RC1906	40694	14607	2439.2	101	-60	114	114	Y	RC	10/24/2013 13:01	10/25/2013		
KOJ	GC_RC1908	39999	14664	2404.6	270	-50	192	192	Y	RC	10/26/2013	11/3/2013		
KOJ	GC_RC1924	39300	15052	2456.6	90	-58	150	150	Y	RC	2/16/2014 13:11	2/17/2014		
KOJ	GC_RC1927	39300	15008	2453.8	90	-67	240	240	Y	RC	2/17/2014 10:09	2/18/2014		
KOJ	GC_RC1928	39300	15134	2482.7	90	-72	125	125	Y	RC	2/19/2014 11:30	2/19/2014		
KOJ	GC_RC1929	39200	15035	2500.8	90	-64	180	180	Y	RC	2/19/2014 14:16	2/20/2014		
KOJ	GC_RC1930	39202	15089	2494.2	90	-67	130	130	Y	RC	2/20/2014	2/20/2014		
KOJ	GC_RC1931	39102	15002	2498	90	-50	200	200	Y	RC	2/20/2014 10:20	2/21/2014		
KOJ	GC_RC1932	39098	15129	2480.3	90	-60	35	35	Y	RC	2/21/2014	2/21/2014		
KOJ	GC_RC1933	38900	14956	2495.1	90	-51	200	218	Y	RC	2/21/2014 15:53	2/22/2014		
KOJ	GC_RC1934	38750	15072	2440.3	90	-75	160	160	Y	RC	2/22/2014	2/22/2014		
KOJ	GC_RC1936	38750	15076	2440.3	90	-62	110	110	Y	RC	2/22/2014 13:33	2/23/2014		
KOJ	GC_RC1938	38700	15102	2418.7	90	-59	100	100	Y	RC	2/25/2014	2/25/2014		
KOJ	GC_RC1939	38805	15115	2460.7	90	-65	50	50	Y	RC	2/26/2014 16:10	2/27/2014		
KOJ	GC_RC1940	39005	15062	2462	90	-54	120	120	Y	RC	2/25/2014 11:01	2/26/2014		
KOJ	GC_RC1942	38800	15053	2460.3	90	-58	128	128	Y	RC	2/26/2014	2/26/2014		
KOJ	GC_RC1943	38900	15100	2442.7	90	-65	50	50	N	RC	2/27/2014 14:13	2/28/2014		
KOJ	GC_RC1950	39250	15142	2485.4	90	-65	80	80	Y	RC	3/4/2014 7:31	3/4/2014 18:00		
KOJ	GC_RC1951	39150	15040	2499.6	90	-62	166	166	Y	RC	3/5/2014	3/5/2014		
KOJ	GC_RC1954	39050	15136	2479	90	-65	50	50	Y	RC	3/5/2014	3/5/2014		
KOJ	GC_RC1955	39050	15092	2486.1	90	-64	100	100	Y	RC	3/5/2014 9:24	3/6/2014		
KOJ	GC_RC1956	39049	15046	2480.3	90	-66	140	140	Y	RC	3/6/2014	3/6/2014		
KOJ	GC_RC1957	39153	15096	2480.8	90	-62	104	104	Y	RC	3/6/2014 6:39	3/6/2014 23:00		
KOJ	GC_RC1958	39345	15132	2476.8	90	-60	50	50	Y	RC	3/7/2014	3/7/2014		
KOJ	GC_RC1959	39348	14983	2438.6	90	-58	200	200	Y	RC	3/7/2014	3/7/2014		
KOJ	GC_RC1961	39202	15092	2494	93	-45	110	110	Y	RC	3/10/2014	3/10/2014		
KOJ	GC_RC1962	39197	15030	2501.1	90	-74	218	218	Y	RC	3/8/2014 9:48	3/9/2014		
KOJ	GC_RC1963	39350	15085	2463.3	90	-68	104	104	Y	RC	3/11/2014	3/11/2014		
KOJ	GC_RC1964	39250	15040	2470.2	90	-45	130	130	Y	RC	3/11/2014	3/11/2014		
KOJ	GC_RC1965	38850	15069	2476.9	90	-64	104	104	Y	RC	3/12/2014	3/12/2014		

Prospect	HoleID	North	East	RL	Azi	Dip	ERC	EOH	Final_C	HoleTyp	Start_RC	Stop_RC	Start_DD	Stop_DD
KOJ	GC_RC310	39765	15110	2362.8	90	-45	160	160	N	RC	10/31/2001	11/1/2001		
KOJ	GC_RC311	39881	14685	2418.6	90	-45	150	150	N	RC	11/1/2001	11/2/2001		
KOJ	GC_RC318	39507	15104	2423.3	90	-70	90	90	N	RC	1/27/2002	1/27/2002		
KOJ	GC_RC319	39450	15106	2440.5	90	-45	70	70	N	RC	1/28/2002	1/28/2002		
KOJ	GC_RC320	39557	15102	2409	90	-70	90	90	N	RC	1/28/2002	1/28/2002		
KOJ	GC_RC321	39604	15107	2396.2	90	-75	90	90	N	RC	1/28/2002	1/28/2002		
KOJ	GC_RC322	39655	15098	2383.3	90	-75	90	90	N	RC	1/28/2002	1/28/2002		
KOJ	GC_RC323	39710	15101	2370.6	90	-45	70	70	N	RC	1/28/2002	1/29/2002		
KOJ	GC_RC324	39857	15106	2378.8	90	-75	70	70	N	RC	1/29/2002	1/29/2002		
KOJ	GC_RC325	39899	15102	2392.5	90	-45	70	70	N	RC	1/29/2002	1/29/2002		
KOJ	GC_RC326	39809	15085	2369	90	-45	70	70	N	RC	1/29/2002	1/29/2002		
KOJ	GC_RC327	39767	15073	2366.3	90	-50	75	75	N	RC	1/29/2002	1/29/2002		
KOJ	GC_RC328	40050	15092	2385.5	90	-50	70	70	N	RC	1/29/2002	1/30/2002		
KOJ	GC_RC329	40000	15090	2390.4	90	-60	70	70	N	RC	1/30/2002	1/30/2002		
KOJ	GC_RC330	39951	15099	2395.9	90	-55	70	70	N	RC	1/30/2002	1/30/2002		
KOJ	GC_RC331	40101	15070	2377.4	90	-45	72	72	N	RC	1/30/2002	1/30/2002		
KOJ	GC_RC332	40137	15055	2352.2	90	-45	80	80	N	RC	1/30/2002	1/30/2002		
KOJ	GC_RC333	40170	15050	2349.7	90	-65	100	100	N	RC	1/30/2002	1/31/2002		
KOJ	GC_RC334	40200	15050	2350.7	90	-50	78	78	N	RC	1/31/2002	1/31/2002		
KOJ	GC_RC335	40237	15027	2345.2	90	-50	84	84	N	RC	1/31/2002	1/31/2002		
KOJ	GC_RC336	40299	15027	2370.3	90	-45	95	95	N	RC	2/1/2002	2/1/2002		
KOJ	GC_RC337	40395	15000	2399.1	90	-45	100	100	N	RC	2/1/2002	2/1/2002		
KOJ	GC_RC338	40437	15032	2379.8	90	-75	90	90	N	RC	2/1/2002	2/1/2002		
KOJ	GC_RC339	40490	15026	2352.8	90	-45	80	80	N	RC	2/1/2002	2/1/2002		
KOJ	GC_RC340	39809	15084	2369	90	-70	90	90	N	RC	2/4/2002	2/4/2002		
KOJ	GC_RC341	39450	15105	2440.2	90	-70	85	85	N	RC	2/4/2002	2/4/2002		
KOJ	GC_RC394	39482	15117	2431.4	90	-45	50	50	N	RC	8/4/2002	8/4/2002		
KOJ	GC_RC395	39483	15114	2431.3	90	-75	70	70	N	RC	8/4/2002	8/4/2002		
KOJ	GC_RC396	39530	15115	2419.6	90	-45	45	45	N	RC	8/4/2002	8/4/2002		
KOJ	GC_RC397	39530	15113	2419.5	90	-75	70	70	N	RC	8/4/2002	8/4/2002		
KOJ	GC_RC398	39531	15075	2412.8	90	-60	100	100	N	RC	8/4/2002	8/4/2002		
KOJ	GC_RC399	39580	15111	2403.7	90	-45	45	45	N	RC	8/4/2002	8/4/2002		
KOJ	GC_RC400	39580	15108	2403.6	90	-75	70	70	N	RC	8/5/2002	8/5/2002		
KOJ	GC_RC401	39629	15102	2384.1	90	-45	50	50	N	RC	8/5/2002	8/5/2002		
KOJ	GC_RC402	39630	15100	2384	90	-70	75	75	N	RC	8/5/2002	8/5/2002		
KOJ	GC_RC403	39680	15100	2380.1	90	-45	45	45	N	RC	9/5/2002	8/6/2002		
KOJ	GC_RC404	39977	15073	2400.4	90	-45	80	80	N	RC	8/6/2002	8/6/2002		
KOJ	GC_RC405	39977	15071	2400.6	90	-70	100	100	N	RC	8/6/2002	8/6/2002		
KOJ	GC_RC406	39952	15096	2396.5	90	-80	80	80	N	RC	8/6/2002	8/6/2002		
KOJ	GC_RC407	40000	15069	2397.7	90	-75	100	100	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC408	40025	15089	2389.8	90	-60	60	60	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC409	39926	15100	2397.3	90	-45	80	80	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC410	39876	15085	2391.7	90	-45	60	60	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC411	39876	15083	2391.8	90	-70	90	90	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC412	40071	15097	2383	90	-60	60	60	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC413	40101	15068	2377.3	90	-75	90	90	N	RC	8/7/2002	8/7/2002		
KOJ	GC_RC414	40201	15046	2350.9	90	-70	93	93	N	RC	8/8/2002	8/8/2002		
KOJ	GC_RC415	40171	15024	2357.2	90	-60	120	120	N	RC	8/8/2002	8/8/2002		
KOJ	GC_RC416	40134	15047	2352.2	90	-75	105	105	N	RC	8/9/2002	8/9/2002		
KOJ	GC_RC417	40408	15009	2391.5	90	-45	70	70	N	RC	8/10/2002	8/10/2002		
KOJ	GC_RC418	39828	15084	2373.3	90	-45	55	55	N	RC	8/10/2002	8/10/2002		
KOJ	GC_RC419	39829	15082	2373.3	90	-70	90	90	N	RC	8/10/2002	8/10/2002		
KOJ	GC_RC420D	39999	15119	2377.7	270	-60	50	49.5	N	DD	8/10/2002	8/11/2002		
KOJ	GC_RC529D	39557	15130	2416.4	360	-90	35	35	N	DD	9/28/2002	9/29/2002		

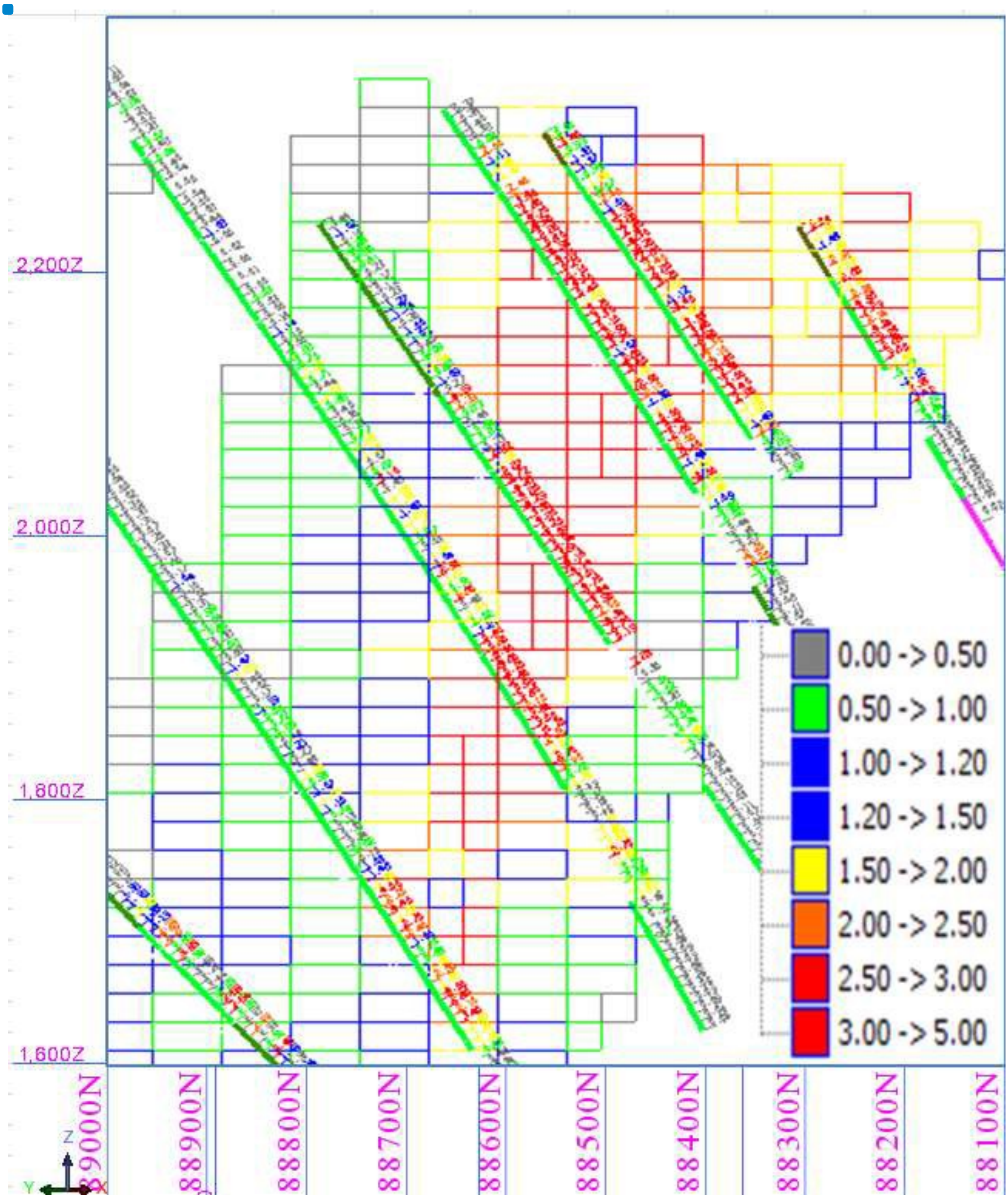
Appendix C – Visual Validation of Model



Visual validation - 88125 Section (looking north)



Visual validation - 88925 Section (looking north)

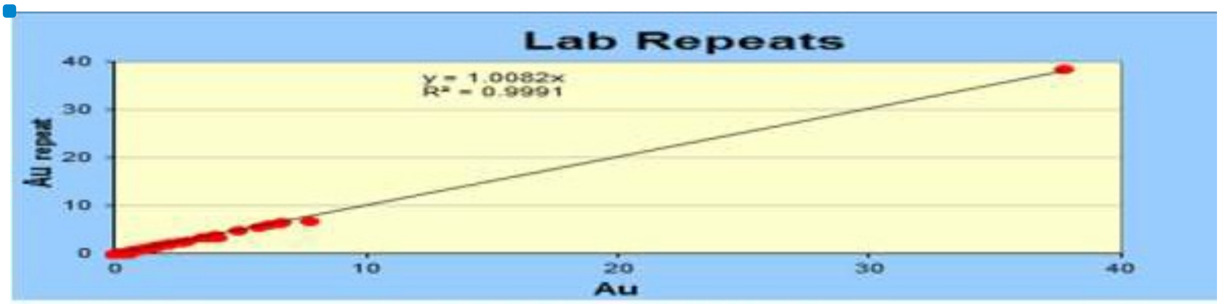


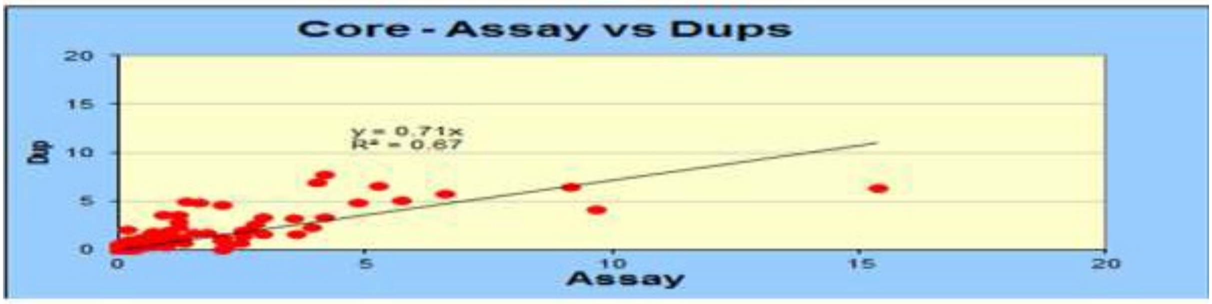
Visual validation - 88275 Section (looking north)

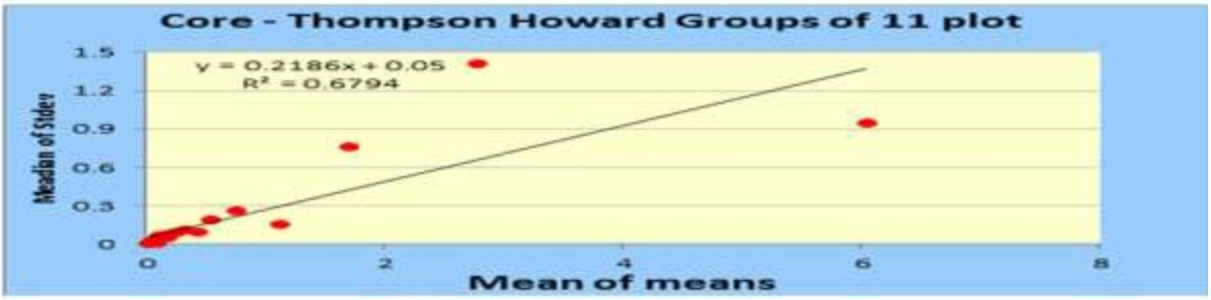
Appendix D – Summary of QAQC samples Performance

Standard	Standards used	Over limits	Bias	comments	Action Item
OxA71	13	pass	No	Pass	No further action
OxC88	65	pass	No	Pass	
OxE86	156	2	No	2 reported above the upper limit	Batches re-assayed
OxF65	130	pass	No	Pass	No further action
OxH82	26	pass	No	Pass	
OxH66	13	pass	No	Pass	
SH41	806	Pass	No	Pass	
Oxi81	13	pass	No	Pass	
OxJ80	26	pass	No	Pass	
OxJ47	13	pass	No	Pass	
SK33	676	Pass	No	Pass	
SN26	286	5	No	5 reported below the lower limit	Batches re-assayed
SN38	52	pass	No	Pass	No further action
SN50	91	Pass	No	Pass	
SP27	52	pass	No	Pass	
SP37	91	pass	No	Pass	

Appendix E – Charts of QAQC samples Performance







Appendix F - Data Verification Report

DATA VERIFICATION

The Kojina database consists of 55 diamond holes (19 are twinned), 68 RC holes and 59 RC with diamond tails.

Summary of Data used

Table Name	Data Type	No. of Records
collar		182
survey	point	2037
assay	interval	24,322
redox	point	164
geology	interval	3105
rqd	interval	3271

A QA/QC analysis was performed on the 15 new holes at Kojina and 20 random ones from previous drilling campaigns. Original field logs, survey information and original lab assay reports were checked for consistency with the master Fusion database.

The sources of error that were taken into consideration were: importation errors, numerical errors (significant figures and rounding), data accuracy and consistency. All 15 new holes, as well as the 20 random ones were verified making 19% of the total database. Drillholes were chosen to represent the database as a whole and to include different hole lengths, spatial variation within the deposit, assay lab alternatives, temporal controls and employee variability.

The QA/QC process determined consistency on a variety of controls such as: Collar Coordinates (easting, northing, elevation, length), Survey (azimuth, dip, depth), Lithology (rock code, interval), and Assay (Au values, sample number, sample interval).

The holes verified are the fifteen (15) new ones GC_DD1921, GC_DD1922, GC_DD1925, GC_RC1873D, GC_RC11944, GC_RC1945D, GC_RC1946D, GC_RC1949D, GC_DD1926, GC_DD1935, GC_DD1936, GC_RC1870D, GC_RC1771D, GC_DD1950, GC_DD1951 and twenty (20) random ones; GC_DD1315, GC_DD1325, GC_DD1511, GC_DD1018, GC_DD1012, GC_DD918, GC_RC98, GC_RC100D, GC_RC135, GC_RC151D, GC_DD1200, GC_DD874, GC_RC816D, GC_RC248D, GC_RC105D, GC_RC454D, GC_DD1202, GC_DD866, GC_RC819D and GC_RC646D.

COLLAR

There were no true errors within the collars. The mismatch noticed was because the holes were new, and the mine surveyor was yet to update the new drillholes with collar coordinates. The surveyor went ahead to update these for our use. Also duplicate collars were from twinned holes.

Inconsistencies found at the end of some holes were explained away by the setup of the database itself.

SURVEY

Discrepancies of azimuth, dip, depth and downhole directional surveys were verified. Contract drillers utilized a Gyro tool to generate downhole survey data. There were no errors identified.

ASSAY

Gold grades, sample number and sample interval were verified from original logs and original assay certificates against the master database. There were 3 sample types; core, regular 1m RC, and composited 3m RC.

Missing sample intervals were genuinely not sampled. Chirano uses the ALS Global (Ghana) lab in Kumasi to obtain gold grades. For umpire checks, Genalysis (Australia) is used. Digital copies are stored on a main server and backed up on portable drives. Hard copies of original assay certificates are stored at the mine.

The data set verified represented 100% of the new holes and 3% of the sample group.

Lithology

Rock type codes represent rock types. The Codes and lithology intervals were not verified due to the use of electronic logging in the field. The PDAs used are designed to do validation, and these are transferred directly into the database.

Other Data Checks

There were no other data checks

Appendix G – Capital Costs Summary

Main Area	Plant Area	Facility	Total (USD)
Mining Costs	Mining-General	Mining Compound and E, Lii ld. roy,-	26,916
	Mining-General Total		26,910
	Other Mining Costs	Other Mining Costs	0
	Other Mining Costs Total		0
Mining Costs Total			26,910
Management Costs	EPCM	EPCM - Home Office	2,057,724
		EPCM - Site Construction	819,658
		EPCM - Pre-Commissioning	82,205
		EPCM - Commissioning	115,264
		EPCM - Expenses	485,406
	EPCM Total		3,560,257
	Specialist Consultants	Tailings Darn Consultant	271,960
		Geotech nica l Consultant	20,000
		Architectural Services	60,000
	Specialist Consultants Total		351,960
	Vendor Representatives	. i en dor Representatives - Ger e.-3	224,400
	Vendor Representatives Total		224,400
Management Costs Total			4,136,617
Owners Costs	Owners Costs-General	Owners Project Mng't Team	531,750
		Construction Insurance's	400,000
		PrefPost Shipment Inspection	578,953
		Sunk Costs	0
	Owners Costs-General Total		1,510,703
	Plant & Ad min Pre-production	Plant Pre-production Labour	0
		Admin Pre-production Expenses	287,000
		Opening Stocks	289,000
		Training	557,000
		Admin Pre-production Labour	970,000
		First Fill Reagents & Consumable!,	786,000
	Plant & Admin Pre-production Total		2,889,000
	Spare Parts	Insurance Spares	634,400
		Consumable Spares	260,000
		Commissioning Spares	52,000
	Spare Parts Total		946,400
	Fees/Taxes/Duties	Duties/Taxes	750,000
	Fees/Taxes/Duties Total		750,000
	Land/Crop Compensation	Crop compensation	697,229
		Resettlement	377,720
	Land/Crop Compensation Total		1,074,949
Owners Costs Total			7,171,052
Working Capital	Working Capital-General	Plant	1,033,333
	Working Capital-General Total		1,033,333
Working Capital Total			1,033,333
Contingency Total			3,646,855
Import Duties Total			90,042
Grand Total			32,016,166

Appendix H – Operating Costs Summary

Cost Centre	Oxide Ore					Primary Ore				
	Tonnage 2,000,000 tpa		Fixed Cost	Variable Cost		Tonnage 2,000,000 tpa		Fixed Cost	Variable Cost	
	US$/Year	US$/t	US$/Year	US$/Year	US$/t	US$/Year	US$/t	US$/Year	US$/Year	US$/t
Process Plant										
Labour	1,004,014	0.502	1,004,014	0	0.000	1,004,014	0.502	1,004,014	0	0.000
Operating Consumables	3,406,939	1.703	915,163	2,491,777	1.246	4,300,355	2.150	1,167,259	3,133,096	1.567
Power	1,544,940	0.772	777,176	767,764	0.384	2,741,883	1.371	1,016,565	1,725,318	0.863
Maintenance and Repairs	1,250,106	0.625	1,125,095	125,011	0.063	1,404,618	0.702	1,264,156	140,462	0.070
Laboratory	161,167	0.081	161,167	0	0.000	161,167	0.081	161,167	0	0.000
ROM Ore Handling	500,000	0.250	0	500,000	0.250	500,000	0.250	0	500,000	0.250
Subtotal	**7,867,165**	**3.93**	**3,982,614**	**3,884,551**	**1.94**	**10,112,037**	**5.06**	**4,613,161**	**5,498,876**	**2.75**
Mining										
Labour	684,034	0.342	684,034	0	0.000	684,034	0.342	684,034	0	0.000
Dewatering Rehabilitation	400,000	0.200	400,000	0	0.000	400,000	0.200	400,000	0	0.000
Day Works	120,000	0.060	120,000	0	0.000	120,000	0.060	120,000	0	0.000
Grade Control	1,150,000	0.575	0	1,150,000	0.575	1,150,000	0.575	0	1,150,000	0.575
Power	50,000	0.025	50,000	0	0.000	50,000	0.025	50,000	0	0.000
Subtotal	**2,404,034**	**1.20**	**1,254,034**	**1,150,000**	**0.58**	**2,404,034**	**1.20**	**1,254,034**	**1,150,000**	**0.58**
General & Administration										
Labour	789,972	0.395	789,972	0	0.000	789,972	0.395	789,972	0	0.000
Owner's Expenses	1,312,700	0.656	1,312,700	0	0.000	1,312,700	0.656	1,312,700	0	0.000
Contracts	1,021,430	0.511	1,021,430	0	0.000	1,021,430	0.511	1,021,430	0	0.000
Subtotal	**3,124,102**	**1.56**	**3,124,102**	**0**	**0.00**	**$3,124,102**	**1.56**	**$3,124,102**	**0**	**0.00**
TOTAL	**$13,395,302**	**$6.70**	**8,360,750**	**5,034,551**	**2.52**	**$15,640,174**	**$7.82**	**8,360,750**	**5,034,551**	**2.52**

Appendix I : Kojina Financial Analysis – Cash Flow

APPENDIX I: Kojina financials Summary

	LOMP Total	year 1	year 2	year 3	year 4	year 5	year 6	year 7	year 8	year 9	
MINING PHYSICALS	1,685,089										
Total Tonnes	109,509,479	9,350,763	8,566,733	10,932,138	7,461,049	11,471,086	15,227,484	15,545,059	20,378,508	10,576,658	
Total Ore Tonnes	30,689,796	-	474,262	3,526,496	1,663,833	4,645,482	5,546,782	5,225,141	5,544,790	4,063,010	
Grade	1.42	-	1.28	1.23	1.34	1.34	1.20	1.58	1.70	1.42	
Mined Ounces	1,400,224	-	19,517	139,906	71,923	200,179	214,446	265,628	303,057	185,567	-
Total Capital Waste Tonnes	17,443,234	9,350,763	8,092,471								
Total Operating Waste Tonnes	61,376,449			7,405,642.11	5,797,216.31	6,825,604.43	9,680,702.33	10,319,917.61	14,833,718.11	6,513,647.85	
Total Waste Tonnes	**78,819,683**	**9,350,763**	**8,092,471**	**7,405,642**	**5,797,216**	**6,825,604**	**9,680,702**	**10,319,918**	**14,833,718**	**6,513,648**	**-**
Strip Ratio	**2.6**	**-**	**17.1**	**2.1**	**3.5**	**1.5**	**1.7**	**2.0**	**2.7**	**1.6**	**-**
PROCESSING PHYSICALS											
CIL Fresh Ore Tonnes	11,519,766	-	-	-	-	388995.6397	909917.0205	1634734.268	2974853.066	5611265.899	
CIL Fresh Ore Grade	1.83	-	-	-	-	1.38	1.49	1.78	1.89	1.91	
CIL Fresh Ounces	679,597	-	-	-	-	17,289	43,560	93,406	180,767	344,576	-
Recovery	2.3%	91.4%	91.4%	91.4%	91.4%	91.4%	91.4%	91.4%	91.4%	91.4%	91.4%
Recovered Ounces	**15,802**	**-**	**-**	**-**	**-**	**15,802**	**39,814**	**85,373**	**165,221**	**314,942**	**-**
CIL Oxide Ore Tonnes	9,935,945	-	417083.8976	2925099.265	1236159.913	1892291.595	1482009.283	1555057.1	428243.8639	-	-
CIL Oxide Ore Grade	1.54	-	1.33	1.37	1.51	1.60	1.73	1.61	1.70	-	-
CIL Oxide Ounces	491,037	-	17,833	129,226	59,818	97,609	82,450	80,709	23,392	-	-
Recovery	59%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Recovered Ounces	**289,262**	**-**	**16,942**	**122,765**	**56,827**	**92,729**	**78,327**	**76,673**	**22,223**	**-**	**-**
Medium Grade Fresh Ore Ton	2,679,399	-	-	-	-	39266.17688	179324.91	485557.3408	888771.18	1086479.112	
Medium Grade Fresh Ore Gra	1.17	-	-	-	-	1.24	1.28	1.20	1.10	1.20	
Medium Fresh Ounces	100,899	-	-	-	-	1,565	7,380	18,679	31,421	41,854	-
Recovery	1%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Recovered Ounces	**1,409**	**-**	**-**	**-**	**-**	**1,409**	**6,642**	**16,811**	**28,279**	**37,669**	**-**
Low Grade Oxide Ore Tonnes	2,539,779	-	57177.846	371815.3692	427673.0172	683693.0694	590891.7168	298540.1694	109987.416	-	
Low Grade Oxide Grade	0.88	-	0.92	0.89	0.88	0.88	0.88	0.88	0.87	-	
Low Grade Oxide Ounces	71,933	-	1,684	10,680	12,105	19,311	16,651	8,429	3,073	-	-
Recovery	56%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%
Recovered Ounces	**40,278**	**-**	**1,549**	**9,825**	**11,137**	**17,766**	**15,319**	**7,755**	**2,827**	**-**	**-**

Low Grade Fresh Ore Tonnes	-										
Low Grade Fresh Grade	-										
Low Grade Fresh Ounces	-	-	-	-	-	-	-	-	-	-	-
Recovery	0%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%
Recovered Ounces	-	-	-	-	-	-	-	-	-	-	-
Total Ore Tonnes	26,674,888	-	474,262	3,296,915	1,663,833	3,004,246	3,162,143	3,973,889	4,401,856	6,697,745	-
Total Ore Grade	1.57	-	1.28	1.32	1.34	1.41	1.48	1.57	1.69	1.79	-
Total Mined Ounces	1,208,773.23	-	15,613.95	111,924.61	161,538.67	128,619.41	130,032.48	160,978.04	190,921.78	309,144.30	-
			19,517	139,906	101,923	135,774	150,041	201,223	238,652	386,430	
	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%
Recovered Ounces	1,112,071.38	-	**14,364.83**	**102,970.64**	**148,615.58**	**118,329.85**	**119,629.88**	**148,099.80**	**175,648.04**	**284,412.76**	-
COST		year 1	year 2	year 3	year 4	year 5	year 6	year 7	year 8	year 9	Jan-00
Unit Mining Cost	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**	**2.83**
Total Capital Cost	**$32,077,216**	**$15,300,000**	**$16,777,216**								**$0**
Total Mining Cost	**$309,584,525**	**$26,434,712**	**$24,218,251**	**$30,905,278**	**$21,092,470**	**$32,428,890**	**$43,048,269**	**$43,946,056**	**$57,610,270**	**$29,900,330**	**$0**
Mining Operating Cost	**$260,272,306**	**$0**	**$1,340,743**	**$30,905,278**	**$21,092,470**	**$32,428,890**	**$43,048,269**	**$43,946,056**	**$57,610,270**	**$29,900,330**	**$0**
Mining Capital Waste Cost	**$49,312,219**	**$26,434,712**	**$22,877,508**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**
G & A	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**
Processing Cost	**$388,919,869**	**$0**	**$6,914,736**	**$48,069,015**	**$24,258,684**	**$43,801,914**	**$46,104,044**	**$57,939,300**	**$64,179,054**	**$97,653,122**	**$0**
Total Cost	**$730,581,610**	**$41,734,712**	**$47,910,203**	**$78,974,293**	**$45,351,154**	**$76,230,803**	**$89,152,313**	**$101,885,356**	**$121,789,324**	**$127,553,453**	**$0**
UNIT COST											
Total Mining Cost per Tonne	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$2.83/t	$0.00/t
Total Mining Cost per Ounce	$256/oz	$0/oz	$1172/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz
Total Mining Cost per Ounce	$278/oz	$0/oz	$1593/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz	$0/oz
Total Cost per Ounce Recover	$657/oz	$0/oz	$3335/oz	$767/oz	$305/oz	$644/oz	$745/oz	$688/oz	$693/oz	$448/oz	$0/oz
REVENUE	1,334,485,651.14										
Revenue	$1,334,485,651	$0	$17,237,797	$123,564,766	$178,338,694	$141,995,824	$143,555,857	$177,719,761	$210,777,645	$341,295,306	$0

Cash Flow Before Tax	**\$603,904,041**	**(\$41,734,712)**	**(\$30,672,406)**	**\$44,590,473**	**\$132,987,540**	**\$65,765,021**	**\$54,403,544**	**\$75,834,405**	**\$88,988,321**	**\$213,741,854**	**\$0**
Forestry and Royalties	\$74,731,196	\$0	\$965,317	\$6,919,627	\$9,986,967	\$7,951,766	\$8,039,128	\$9,952,307	\$11,803,548	\$19,112,537	\$0
Net Cash Flow Before Tax	**\$529,172,844**	**(\$41,734,712)**	**(\$31,637,722)**	**\$37,670,846**	**\$123,000,573**	**\$57,813,255**	**\$46,364,416**	**\$65,882,099**	**\$77,184,773**	**\$194,629,317**	**\$0**
Corporate Tax	\$185,210,496	(\$14,607,149)	(\$11,073,203)	\$13,184,796	\$43,050,201	\$20,234,639	\$16,227,546	\$23,058,735	\$27,014,671	\$68,120,261	\$0
Net Cash Flow After Tax	**\$343,962,349**	**(\$27,127,563)**	**(\$20,564,519)**	**\$24,486,050**	**\$79,950,373**	**\$37,578,616**	**\$30,136,870**	**\$42,823,364**	**\$50,170,103**	**\$126,509,056**	**\$0**
Cummulative Cash Flow After Tax		**(\$27,127,563)**	**(\$47,692,082)**	**(\$23,206,032)**	**\$56,744,340**	**\$94,322,956**	**\$124,459,826**	**\$167,283,190**	**\$217,453,293**	**\$343,962,349**	
Net Cash Flow After Tax (dis	**\$252,326,652**	**(27,127,563)**	**(19,585,257)**	**22,209,569**	**69,064,138**	**30,916,020**	**23,613,027**	**31,955,454**	**35,654,955**	**85,626,309**	**\$0**
Cummulative Cash Flow After Tax (discounted)		**(\$27,127,563)**	**(\$46,712,819)**	**(\$24,503,250)**	**\$44,560,887**	**\$75,476,908**	**\$99,089,934**	**\$131,045,388**	**\$166,700,343**	**\$252,326,652**	
Net Present Value											
NPV @ 0%	**\$ 343,962,349**	(27,127,563)	(20,564,519)	24,486,050	79,950,373	37,578,616	30,136,870	42,823,364	50,170,103	126,509,056	-
NPV @ 5%	**\$ 252,326,652**	(27,127,563)	(19,585,257)	22,209,569	69,064,138	30,916,020	23,613,027	31,955,454	35,654,955	85,626,309	-
NPV @10%	**\$ 187,796,325**	(27,127,563)	(18,695,018)	20,236,405	60,067,898	25,666,700	18,712,625	24,172,673	25,745,195	59,017,408	-