3D GEOLOGICAL MODELLING AND MINERAL RESOURCE ESTIMATE FOR THE FE2 GOLD DEPOSIT, SADIOLA MINE – MALI

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DECLARATION

I declare that this research report is my own, unaided work. Where use was made of the work of others, it was duly acknowledged. It is being submitted for the Degree of Masters of Science in Mining Engineering at the University of the Witwatersrand, Johannesburg.

This work was conducted on behalf of AngloGold Ashanti Limited.

It has not been submitted before in any form for any degree or examination at any other university.

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ABSTRACT

The FE2 Gold Deposit forms part of Sadiola Mine located in south-western Mali - nearby the border with Senegal - approximately 440km north-west of the capital Bamako, and 70km south of the city of Kayes.

Sadiola Mine is made up of 7 open pits (the Main Pit, FN3, FE2, FE3, FE4, Tambali and Sekokoto). Gold (Au) mineralisation is spatially associated with a complex alteration pattern, pointing to a mesothermal origin for the Au mineralisation.

The Main Pit deposit contains an Oxide portion and a deeper Sulphide zone comprised of unweathered material below the pit. In 2010, mining of the Oxide portion was concluded. Currently, Sadiola does not have the plant capability to treat Sulphides due to its hardness and most of the Oxide Mineral Reserve in the concession has been depleted. The FE2 deposit is expected to provide Oxide Ore for 7 months based on the current mine plan. The Oxide mining on the Sadiola concession has an expected life of 3 years.

Sadiola's future is thus tied to the fate of the Sadiola Sulphides Project (SSP), targeted at exploiting the Sulphide zone Ore. In the absence the SSP materialising to date, focus has shifted to the FE2 deposit to scavenge any remaining Oxide Ore, to prolong mine life.

The previous Mineral Resource model was generated in June 2014. The model was based on grade control drilling information. The current Mineral Resource Estimate (MRE), presented in this research report, was prompted by an Advanced Grade Control (AGC) drilling campaign that took place during October 2014 to identify additional Oxide Ore Mineral Resource (Indicated, Inferred and Blue Sky Potential).

The AGC drillholes (12.5m (X) by 12.5m (Y) drill spacing) have been drilled mostly as infill drilling and all holes had accompanying assay data.

The Ore and Graphite mesh modelling was conducted using the grade interpolation technique in Leapfrog® mining software. The Hardness, Redox, Laterite and Classification wireframes were created in Datamine® Studio 3 software. A lower geological cut-off of 0.32g/t Au was applied to the mineralised domains. Three domains were estimated: EZONE 1 (Laterite and Saprolite Ore); EZONE 2 (Hard Ore i.e. Sulphides) and EZONE 3 (Waste).

All estimation into the Mineral Resource model was done in Datamine® Studio 3. Ordinary Kriging (OK) was used to estimate the Au grades; Inverse Power of Distance (IPD) to estimate "hardness probabilities" for isolated hard/blastable material above the hard/soft contact; and Indicator Kriging (IK) used to estimate the distribution of the Graphitic alteration.

The Au estimation process was optimised using Quantitative Kriging Neighbourhood Analysis (QKNA). The estimates were validated visually, statistically and using swath analyses.

Uniform Conditioning (UC) was used to estimate the recoverable Mineral Resource in EZONES 1 and 2 for the reporting of the distribution of grades above various economic cut-offs. The Selective Mining Unit (SMU) size assumed for the FE2 UC process was 10m (X) x 10m (Y) x 3.33m (Z) and was based on the selectivity achievable with the current mining equipment.

Given the panel size of 25m (X) x 25m (Y) x 10m (Z), there were about 18 SMUs in each panel. A tonnage adjustment factor was applied and was based on a volume representing half the SMU size. It was expressed as a percentage of the panel size (2.7%). Any proportions smaller than this percentage were removed as they would not be practically recoverable (these volumes would be too small to mine with the selected equipment).

The Mineral Resource was classified in accordance with the South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC) and the Australian Joint Ore Reserves Committee (JORC) guidelines. A drill spacing of 25m (X) by 25m (Y) was considered sufficient to classify the Mineral Resource as Indicated, and 50m (X) by 50m (Y) as Inferred.

Areas covered by larger drill spacing were considered to be Blue Sky Potential (not an official Mineral Resource Category, but used for internal purposes by AngloGold Ashanti Limited (AGA) to estimate possible mineralisation potential). No Measured Mineral Resource was defined. The classification criteria are based on studies completed for other, similar Sadiola deposits (such as FE3 and FE4).

The 2014 Mineral Resource model was compared with the updated Mineral Resource model (2015) within a common volume i.e. within the Business Plan (BP) 2015 $1,600 Mineral Resource shell and the $1,200 Mineral Reserve design (below the topography as no mining has taken place at FE2) to quantify if the Oxide Ore potential had increased as a result of the model update (**Table 1**).

The detailed Reconciliation study showed that the new estimate identified an additional 7,191 ounces of Indicated Mineral Resource – of which, 1,893 ounces was previously classified as Inferred Mineral Resource but was upgraded to the Indicated Mineral Resource category as a result of the new Mineral Resource model.

The reason for the increase is due to the new drilling results which resulted in the extension of some of the mineralised zones and showed better continuity for others.

Table 1:*Model reconciliation by broader material types: 2014 vs. 2015 MW cut-off grades*

	2014 Resource Model			2015 Resource Model			2014 minus 2012
	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Ounce diff.
Indicated							
Soft Oxides	1,460,037	1.75	82,351	1,431,892	1.93	88,822	6,471
Hard Oxides	4,126	1.57	208	12,503	2.31	928	720
Transitional	9,188	1.86	550	28,434	2.05	1,872	1,872
Soft Sulphides	-	-	-	564	2.07	38	-512
Hard Sulphides	-	-	-	1,299	2.06	86	86
Total Indicated	**1,473,350**	**1.75**	**83,109**	**1,474,693**	**1.94**	**91,746**	**8,637**
Inferred							
Soft Oxides	53,988	1.67	2,894	18,220	1.71	1,001	-1,893
Hard Oxides	-	-	-	-	-	-	-
Transitional	-	-	-	-	-	-	-
Soft Sulphides	-	-	-	-	-	-	-
Hard Sulphides	-	-	-	-	-	-	-
Total Inferred	**53,988**	**1.67**	**2,894**	**18,220**	**1.71**	**1,1001**	**-1,893**

A checklist of assessment and reporting criteria based on the JORC code showed that no major risks to the model exist.

However, some key recommendations were made and include:

- Testing domaining and variography at various geological cut-offs
- Performing an updated Classification study to confirm the suitability of the Classification criteria used
- Soft Oxide density probe measurements reported in 2015 were significantly higher than in 2014. Further work needs to be done to confirm the validity of the density results before updating the 2015 density values
- Testing estimation software used in the estimation process against similar software in the industry to single out the one that provides the most accurate results
- Further work should be carried out to assess the effect of top cuts and top caps on the resulting Mineral Resource models
- Further work is required on boundary analysis going forward as in reality the Laterite and Saprolite are very different, despite the results of the statistics suggesting that they are similar.

- The latest LIDAR survey had not been provided at the time of Ore wireframe modelling. A new survey needs to be carried out to ensure that drillholes collar positions used in the modelling were correct
- Further work is required to understand what method is best to model the extent of the graphitic alteration and how to optimise the method

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LIST OF UNIT SYMBOLS

Arsenic.. .... As
Beryllium.. .... Be
Billion Years .... Ga
Cobalt.. .... Co
Copper.. .... Cu
Easting Direction.. X
Elevation Direction .... Z
Gold .... Au
Gold Grade.. .... g/t
Grams.. .... g
Meters .... m
Million Ounces .... Moz
Million Tonnes .... Mt
Million Years .... Ma
Northing Direction.. .... Y
Ounces .... oz
Thousand Meters .... Km
Thousand Ounces .... koz
Tonnes.. .... t
US Dollar .... $

LIST OF ABBREVIATIONS

AAC.. Anglo American Corporation
AGA ....AngloGold Ashanti Limited
AGC.. ....Advanced Grade Control
BHID.. Borehole Identification (name)
BP.. .... Business Plan
COV ....Coefficient of Variation
CRM.. .... Certified Reference Material
CV.. .... Covariance
DD.. .... Diamond Drilling
DGM.. .... Discrete Gaussian Model
E...east
EX.. .... Exploration Drillholes
EZONE.. .... Estimation Zone (Domain)
G.... ....Grade above Cut-Off
GC...Grade Control Drillholes
GPS.. .... Global Positioning System
IEF.. ....Information Effect
IK...Indicator Kriging
IPD.. .... Inverse Power of Distance
JORC.. .... Joint Ore Reserves Committee
KKI.. ....Kedougou-Kenieba Inlier
LIDAR.. Light Detection and Ranging
N..... .... north
P.. Proportion above Cut-Off
PDRM.. Programme de developpement des Resources
QAQC.. Quality Assurance Quality Control
QQ.. .... Quantile Quantile
RAB..Rotary Air Blast
RC.. ....Reverse Circulation
S..south
SAMREC........... ......South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves
SD... .... Standard Deviation
SEMOS.. ....Societed'Exploration des Mines d'Or de Sadiola
SGS.. ....Société Générale de Surveillance
SMU.. .... Selective Mining Unit
SSP.. .... Sadiola Sulphides Project

UC.. .... Uniform Conditioning
W.... .... west
WGM.. .... Watts, Griffis and McOut Consulting

LIST OF APPENDICES

1. INTRODUCTION

An understanding of the geology of a deposit is fundamental to the Mineral Resource Estimation process. Mineral Resource estimates are constrained by the 3D geological model (geometry, grade distribution, structural nature, complexity, etc.), of the deposit and hence any geological uncertainty arising thereof.

The quality of the Mineral Resource estimate is further impacted on by the choice and applicability of estimation techniques applied in the estimation process:

"If geostatistics are to give improved reserve estimates, two conditions must be satisfied: geologists must be aware of the methods that are available to them to control the quality of the geostatistical study and geostatisticians must appreciate those areas in which geological input is required if credible results are to be obtained" - (Rendu,1984, p. 166)

The estimation process methodology used is deposit specific because every deposit is unique. The estimation process depends on the Geological model which is itself dynamic since every new hole drilled necessitates the need to update the model based on the new information. It is for this reason that a 3D Geological model was generated for the FE2 Mineral Resource Estimate.

This work was carried out on behalf of AngloGold Ashanti Limited. This research report, is written as part of the requirements for obtaining a Masters in Science in Mining Engineering at the University of the Witwatersrand.

The FE2 Gold Deposit forms part of Sadiola Mine, located in south-western Mali -nearby the border with Senegal -approximately 440km north-west of the capital Bamako, and 70km south of the city of Kayes (**Figure 1**). The Mine has an expected life of 3 years.

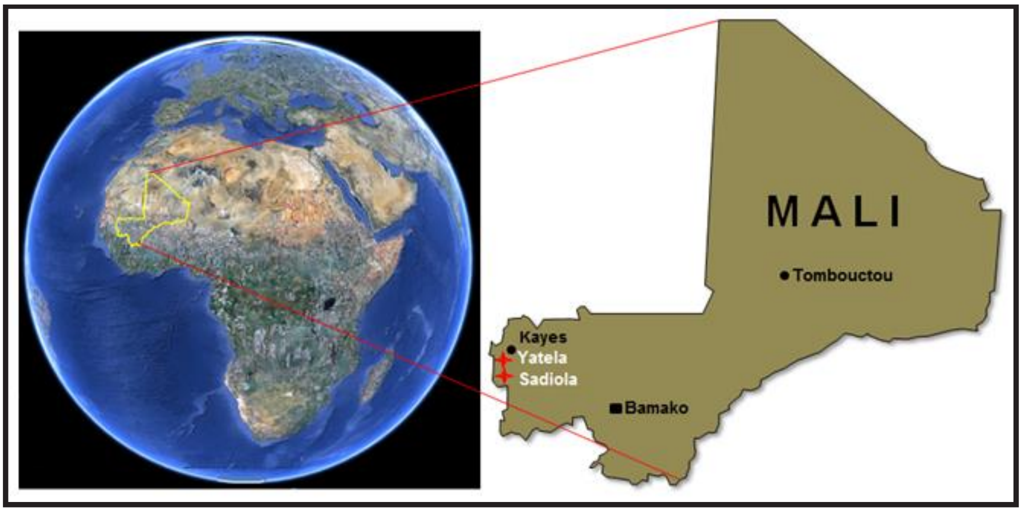


Figure 1: *Sadiola Mine locality map* (SEMOS, 2012, pp 19)

Sadiola Gold Mine is operated by the Societe d'Exploration des Mines d'Or de Sadiola S.A. (SEMOS). The project is a joint venture operation between AGA (41%), IAMGOLD (41%), and the State of Mali (18%) (mining-technology.com, 2015); however it is managed by AGA.

The Sadiola mining permit covers an area of 302 km^2. **Figure 2** shows the concession area along with the location of Yatela Gold Mine - situated 20 km north-west of Sadiola, also a joint venture operation between AGA (40%), IAMGOLD (40%), and the State of Mali (20%), also managed by AGA.

Mining at Sadiola commenced in 1996 and at Yatela in 2000. The two mines combined have produced more than 8.4 million ounces of gold using open pit mining.

Sadiola Mine is made up of 7 open pits (the Main Pit, FN3, FE2, FE3, FE4, Tambali and Sekokoto) (**Figure 2**).

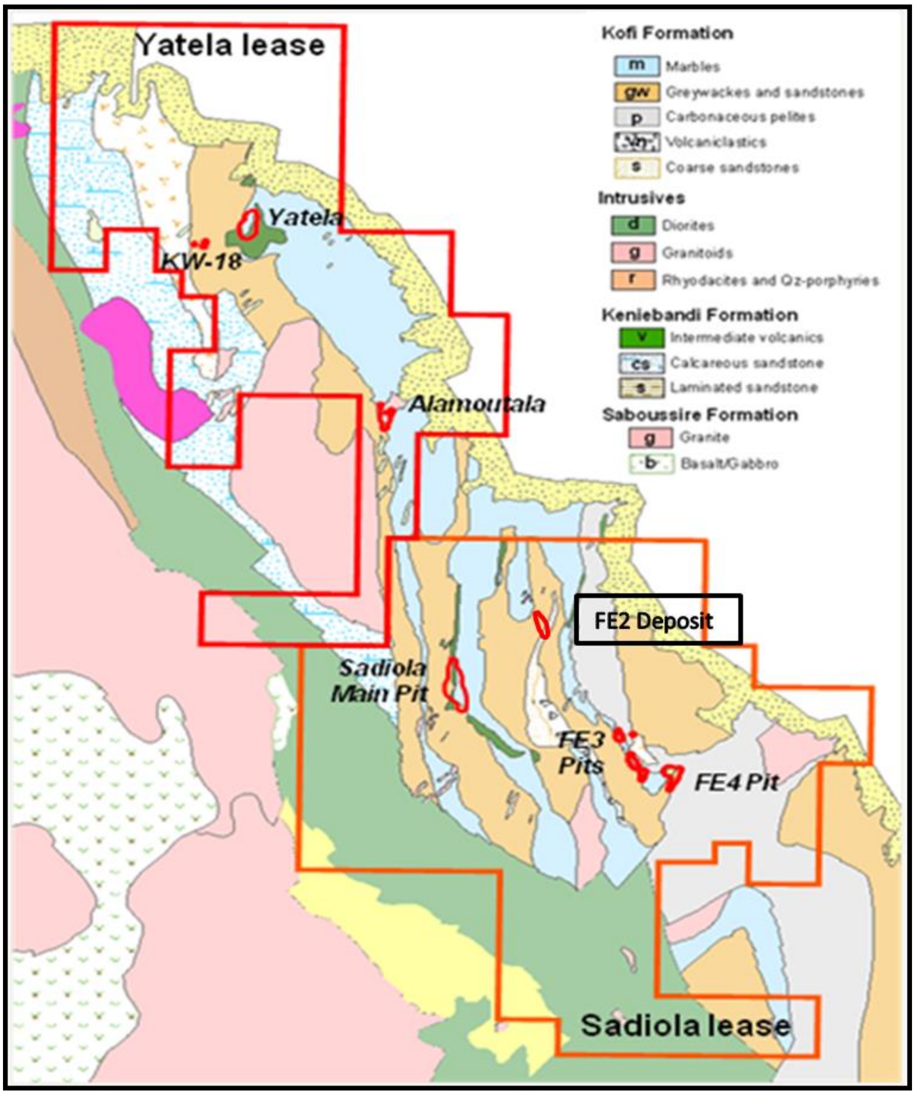


Figure 2: *A lithological map of the Sadiola mining district showing the main mine lease boundaries. Tambali and Sekokoto not shown* (SEMOS, 2012, pp 20).

The Main Pit deposit contains an Oxide portion (Oxides) and a deeper Sulphide zone (Sulphides) comprised of unweathered material below the pit. In 2010, mining of the Oxide portion was concluded. Currently, Sadiola does not have the plant capability to treat Sulphides due to its indentation hardness and most of the Oxide Mineral Reserve has been depleted.

The satellite Ore bodies - FE3 and FE4 - south-east of the Main Pit have since contributed some gold through minor production activities.

This contribution has however declined due to operational challenges, such as declining grades in the FE3 and FE4 pits; Ore losses in the eastern wall of the FE4 pit and extended mill shut-downs and increased operational costs.

In an attempt to salvage the life of mine, plans to expand the Main Pit to access the Sulphides and erect a new plant capable of treating the hard material were drawn up and the Sadiola Sulphides Project (SSP) borne, but to date has been unsuccessful.

Sadiola's future is tied to the fate of the SSP and in the absence of it materialising, focus has shifted to foraging for the final remnants of Oxides in the concession.

The FE2 deposit is one such area that possesses Oxide potential, necessitating the need for an updated Mineral Resource model of the deposit to prolong mine life.

The Sadiola exploration strategy is to build a comprehensive understanding of the remaining Oxide potential in the short term and to extend the Sulphide potential in the longer term.

Oxide exploration on the Sadiola concession has reached maturity and exploration work that was previously focussed primarily on follow up drilling at various prospective targets and identifying new targets has since ceased due to declining gold prices.

A reliable estimate of the FE2 Mineral Resource is therefore critical to the livelihood of Sadiola Mine.

1.1. BACKGROUND INFORMATION

Considerable work has been covered at Sadiola Mine, and is well documented in company reports referred to by SEMOS, 2012 and referenced throughout this dissertation.

The exploration potential at Sadiola was originally based on widespread evidence of artisanal gold workings and small scale mining by locals in the area. Written records of the workings date back 250 years with some believing that this could date back even 1000 years ago due to the extent of the old mine workings.

As part of an aid programme financed by the European Development Fund, a German company named Klöckner Industries, conducted a regional geochemical survey (The Mali Quest 1 Project) for the Malian government. During this time (October 1987 to August 1989), 48,000 samples were collected for geochemical analysis. The samples were sourced near the villages of Sadiola and Dinnguilou and contained high gold, arsenic and antimony anomalies.

In January 1990, the Government of Mali granted exploration rights to Klöckner Industries to conduct a large scale gold exploration programme in the Sadiola area which identified the presence of significant Oxide gold.

In 1991, Watts, Griffis and McOuat (WGM) reviewed the work of Klöckner Industries and prepared a preliminary economic assessment of Sadiola on behalf of IAMGOLD. The preliminary feasibility study spurred on a large exploration drilling programme (from 1991 to 1992) to delineate and confirm the Sadiola Mineral Resource. In December 1992, WGM estimated a probable Mineral Reserve of 22.3 million tonnes of Oxide mineralisation with an average gold grade of 3.3 g/t.

In October 1992, a joint-venture agreement with Anglo American (“AAC”) was signed for the construction and management of any mine developed at Sadiola. A feasibility study on the Sadiola Gold Deposit dated December 1993 and prepared by AAC was presented to the Government of Mali. In August 1994 the Government of Mali issued an exploitation permit (the “Sadiola Mining Permit”).

SEMOS was incorporated on 14 December 1994 as the joint venture company to hold the Sadiola Mining Permit, to exploit the Sadiola gold deposit and to carry out exploration activities within the Sadiola area.

1.2. LITERATURE REVIEW

Whilst extensive literature was reviewed to ensure that the results and findings for this work were correct, the AGA Mineral Resource guidelines were prioritised based on past tried and tested results.

The Mineral Resource estimation process is iterative, requiring a good understanding of common practice and relevant literature. For guidance on the estimation process followed for the research, a series of relevant but not prescriptive papers were reviewed in a monogram on Good Practice in Resource and Reserve Estimation published by the Australian Institute of Mining and Metallurgy.

Key authors referred to in the monogram include Amos, Q G, 2001; Appleyard, G R, 2001; Duke, J H and Hanna, P J, 2001; Guibal, D, 2001; Stephenson, P R and Vann, J, 2001; Stoker, P T and Gilfillan, J F, 2001; whose case studies and guidance set the premise for the work as they demonstrated superior knowledge in each area of the estimation process.

The estimation process is founded on a good understanding of the underlying geology (regional and local) as well as the stratigraphy and mineralisation style. This information in the form of reports, maps and reviews were sourced from the SEMOS site office and corroborated with academic articles such as those by Diene et al., 2012 and Masurel et al., n.d. and Masurel et al., 2012. In addition, detailed work focused on geological descriptions and interpretations has been carried out over the years in collaboration with the University of the Witwatersrand. Notable authors include Professor Kim Hein and Dr Greg Cameron for AngloGold Ashanti. This information assisted in the geological modelling.

The data used for the modelling and analysis were sourced directly from site. All drillhole and surface sampling data for FE2 is stored in a Microsoft SQL database using AGA customised Century Systems Fusion software since 2002. The database is derived from several sources with quality controls in place to prevent errors being introduced to the database.

According to Stephenson and Vann (2001) and Gilfillan and Stoker (2001) rigorous system and quality checks are to be performed on the database to verify that the sampling, assay and survey data informing the estimate is free of errors and that the database is representative, accurate, and precise. Therefore, the collar, survey, alteration, assay, lithology, hardness, and Redox data - provided in .csv format - was checked to ensure that the data used for estimation (historical grade control drillholes and newly drilled advanced grade control data) were error free. Where new information became available, the historical grade control holes were used to guide the mineralised wireframes and updates.

Light detection and Ranging (LIDAR) surface topography strings were defined from a LIDAR survey carried out in 2013. The data was sourced from the on-mine survey department and was confirmed to be the latest available. LIDAR refers to the remote sensing technique that utilises light (pulsed radar) to measure distances to the earth to generate a 3D model of the earth's surface. These strings were validated and used in Datamine® to create a wireframe of the topographic surface using the Digital Terrain Model (DTM) function.

The FE2 mineralisation is controlled by a combination of lithology, structure, weathering and alteration. Wireframes were generated for the topography, weathering surfaces, hardness boundaries, extent of the graphitic alteration and gold mineralisation.

Stephensen and Vann (2001), emphasise that the interpretations derived from drillhole sample data represents often less than 0.001% of the geological body and any errors at this level can dramatically bias grade and tonnage results. The geological meshes serve as the basis for the geological model which in turn is the foundation of the resource estimate therefore care must be taken to ensure that no error is propagated or introduced in the process.

A key step to be taken is the approach of re-interpreting the mineralized envelopes created using Leapfrog® software's grade interpolation technique. This step prevents allowing the software to make assumptions on the geology without understanding.

Carras (2001) highlights the importance of the user's knowledge:

"Most resource modelling procedures are attempts at modelling from sparse data, based on assumptions made by the geologist and mining engineer, and assumptions inherent in the mathematical modelling algorithms used. Assumptions made by the geologist and mining engineer are often well stated, understood and often questioned. Assumptions inherent in the modelling algorithms are very rarely understood or stated and seldom questioned".

Operational mines such as Sadiola, have a good geological understanding that is supplemented by a large amount of historical data. This knowledge of the Ore body behaviour guides the modelling and has been fine-tuned over the years using reconciliation results which were also assessed.

The wireframe interpretations are used in Datamine® Studio 3 to code the drillhole samples according to mineralisation, lithology, weathering and structure. Samples within the mineralised envelope are deemed as "Ore" and those outside, as "Waste".

As a result, the Domaining exercise was less of an arbitrary process since the new wireframe was guided by the 2014 grade boundaries which were constructed with a sound understanding of the grade continuity and geological controls on the grade distributions.

Domains are defined as zones which are geologically and statistically homogenous (supported by variography and statistical analysis) (Duke and Hanna, 2001). Glacken and Snowden, 2001, define domains as areas or volumes within which the characteristics of the mineralization are more similar than outside the domain.

The domains for FE2 were defined using grades in an iterative process of selecting mineralised intersections in each borehole using Leapfrog® Mining Software.

Domains should conform to the geology. In this case, the geological units are the same as the mineralisation domains therefore the grade modelling is constrained entirely by the geological modelling and the resource model is a reflection of the geology (Glacken and Snowden, 2001). Glacken and Snowden (2001) draw attention to Domaining as a process that assists in reducing the problem of preferential data clustering and its bias on statistical analysis and variography that arise because of the natural tendency to drill or take more samples in higher grade areas, causing data to be misrepresented.

Glacken and Snowden (2001), also state that summary statistics should be presented to detect if any trends are evident in the data. Plotted data distributions were used for this to depict any trends evident within the domains to assist in selecting interpolation techniques; defining subsets within the data and highlighting outliers and extreme grade values and essentially establish if a relationship between variables exists.

This is followed by Exploratory Data Analysis which ensures that the domains are well understood and quantified. Blackwell and Sinclair

(2001, pp. 181-191) explain the relationship between domains and variography. The authors describe domains as being the basis for variography which in turn, is the first step in geostatistical analyses and thus fundamental to the success of the estimation process.

The practical considerations for the estimation methods used in the Mineral Resource Estimate were further guided by the work of Blackwell and Sinclair (Chapter 10, pp 215-241 2002) who illustrate how Kriging is an optimal block or point estimation technique.

Kriging weights are allocated using a least squares procedure that minimises the estimation variance therefore making the sample weights unbiased. Correct semi-variogram models that capture the grade continuity are a requirement for Kriging to work.

Experimental variograms are estimates of the ‘underlying’ variogram and some irregularity is generally expected according to Guibal (2001). Supervisor® (v8) geostatistical software will be used to calculate and model the variograms and evaluate the directions of continuity.

Kriging, however, is a ‘minimum variance estimator’ only if the search neighbourhood is properly defined. Bertoli et al., (2003), explain that a Quantitative Kriging Neighbourhood Analysis (QKNA) should be carried out to determine what optimum combination of search neighbourhood and block size results in conditional un-biasedness during Kriging, as defined by the user.

The true block grades are never known but the relationship between the true block and estimated values are inferred based on the assumption that the variogram models are representative of the domains (stationarity) and that a linear regression can define the relationship between true and estimated grades at the specified support – blocks in this case.

During Quality Kriging Neighbourhood Analysis (QKNA), the neighbourhood was optimized to ensure the best regression statistics in order to reduce or eliminate conditional bias. The process involves smoothing because the data set is exhaustive (information effect) and the variance of the estimated block values will be lower than that for the true block values. QKNA assists in deciding how much smoothing is needed for conditional unbiasedness.

The choice of estimation method applied depends on the appropriateness of the method to the deposit's geology and the available data.

Indicator Kriging (IK) was used to estimate the extent of the Graphitic alteration. The concept of IK is discussed by Blackwell and Sinclair (2001, pp 252).

For this research, the graphite codes in the drill logs were used. IK is good when dealing with categorical data. The main motivation for using IK is the fact that it is non-parametric. All the samples that contained graphite alteration were given a value equal to 1 and the remaining samples a value equal to 0 i.e. the data undergoes a non-linear transformation to indicator values (0 or 1) based on the presence or absence of graphite alteration. Values that are greater than a particular alteration intensity received these indicator values. The results of IK provides probabilities for the condition i.e. presence or absence of graphite alteration.

Ordinary Kriging is then applied to the indicator transformed values to provide a value between 0 and 1 for each point estimate. The resulting estimates were plotted and did not capture the known extent of the graphite alteration well. A Leapfrog® interpolant for graphitic alteration was also created and used instead, since the results were more realistic than that of the IK.

"Hardness probability" estimation was run to identify isolated hard/blastable material using the Inverse Power of Distance (IPD) method which is used for its robustness and ease of use (Babak and Deutsch, 2008). This technique is the accepted technique used by AGA, due to the reliability of results of past estimates.

Exponents ranging from 1 to 5 were investigated, but the power of 2 proved to be optimal. Past trial-and-error exercises have also proven that the power of 2 is optimal.

Estimates of the "hardness probabilities" are assigned weights based on how close they are to actual values. This technique was used because no prior information is required for the interpolation , unlike OK where a variogram is known and the assumption of stationarity applies (Babak and Deutsch, 2008.). It also assists when little data is available and a quick visualisation of the variable is needed.

The FE2 deposit lends itself well to the use of Ordinary Kriging (OK) as an estimation technique for gold mineralisation. OK is used on composites whose local mean is unknown (Blackwell and Sinclair, 2001, pp. 231).

The geometry of the mineralisation domains are represented as 3D arrays of blocks in the model. Hence, each domain is kriged block by block based on the requirements defined by the user in the QKNA exercise such as search distance optimisation and selecting the minimum and maximum number of samples

Kriging provides the best estimate since it provides the smallest standard error; narrowest confidence interval and most confidence (lowest risk).

The block model grades were estimated using OK in Datamine® Studio 3. Datamine® Studio 3 was also used for data manipulation, earlier statistics, block modelling, validation and reporting.

Boundary analyses were undertaken using the Bloy® Geostats kit to determine whether or not the grade variations across the mineralisation-waste boundaries are “hard” or “soft”.“Hard” boundaries are defined by abrupt changes in grade whilst “soft” boundaries allude to more gradual changes.

Unusually high grade samples (also called extreme values) result in overestimation of a resource. Histograms, log probability plots and mean and variance plots are analysed to determine which grade cap is the most appropriate per domain.

Top cuts (99th percentile or above) are generally applied to remove the extreme grade values from the resource database whilst including the high-grade assays below the top cut that are recognized as a real feature of the assay distribution (Pocock, 2001, citing Enterprise Metals, 1990).

The impact of applying top cuts was evaluated by Pocock (2001). The results showed that the application of a cut-off leads to changing the inherent characteristics of the data. Attempts to apply even very high cut-offs still reduces the variation in mean gold grade as well as the percent relative standard deviation (RSD) between datasets.

In addition, Pocock (2001) found that wide spaced drilling produced strongly biased datasets and showed sensitivity to outliers and a disproportionate contribution of a few samples to the average grade of an estimate (especially in smaller data sets).The suitability of applying cuts or capping to the data was assessed but further work is required in this area in future.

The mining method and mine selectivity to be used is important when considering recoverable resources. When the selective mining units are small in comparison to the data spacing - over smoothed estimates result from using linear estimation techniques like OK (Deraisme and Roth, n.d.).

Therefore, it is important to be able to calculate the distribution of the block grades from the distribution of sample grades also referred to as a change of support (Deraisme and Roth, n.d.). This is achieved via Uniform Conditioning.

"Uniform Conditioning (UC) provides a method for creating a resource model that is representative of the variability of the deposit for a defined selective mining unit (SMU), which if used for mine planning and reserve calculation can increase the confidence in the resulting reports and mine plans" (dataminesoftware.com, 2015).

In mining, resources are estimated into larger mine planning blocks called panels but are mined as selective mining units (SMU). SMUs are defined as the smallest volume of material on which the decision between Ore and Waste is based. The estimation of recoverable resources therefore depends on the volume on which the Ore/Waste decision is made i.e. the support effect (Neufeld et al, 2005). The UC process to be followed is based on the work by Neufeld et al, 2005.

The aim of UC is to estimate the tonnage and the metal content of blocks inside a panel conditionally to the sole panel grade, which is estimated assuming local stationarity (e.g. Ordinary Kriging) (geovariances.com, 2015).

Since the grade estimation process is complex, it is essential to test if the resultant estimates are a good representation of the input sample data. AngloGold Ashanti uses a validation checklist to ensure that the quality of the data represented by the model is error free. Some error checks include generating statistics and grade plots to ensure that the composited input data is free from negative grades and absent values.

The block model estimates are validated as follows:

- Visually comparing the model estimates against input grades
- Comparison of the global and input means
- Sectional plots of number or composites, model grades and composite grades
- Grade-tonnage curves
- No negative grades occur
- All blocks have and estimate and a density for tonnage weighting in statistics
- No overlapping of blocks occurs

Classification of tonnage and grade estimates is done according to differing degrees of geosientic confidence.

In general, to move a Mineral Resource from infered to measured, the level of confidence should increase. One way, to increase the confidence is to use estimates based on optimised drill hole spacing. Drillhole spacing exercises are carried out to see at which spacing the confidence is the highest.

AGA's bases its classification on the Mineral Resource and Reserve Committee's guidelines. The metal content above the Ore cut-off is measured to an accuracy of 90% confidence at 15% error over a period of 3 months for Measured and for 1 year for Indicated.

The 15% error 90% confidence method is adapted from Anglo American and the idea is to estimate the average grade above cut-off with less than 15% relative error and 90% confidence.

The classification criteria are based on studies completed for other, similar Sadiola deposits (such as FE3 and FE4) - an updated classification study to confirm its suitability is recommended.

The mineral resource has been classified in accordance to the South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC) and Joint Ore Reserves Committee (JORC) Code.

The JORC code checklist of assessment criteria to identify any inherent risks to the resource estimate is also included in this report.

Model reconciliations quantify the differences between the new and previous model since the same methodologies were applied. These differences are determined by comparing a common volume between the two models i.e. the old versus the new model.

The reconciliation is also based on in-house standards.

1.3. RESEARCH QUESTION

The previous Mineral Resource model for the FE2 gold deposit was generated in June 2014. The model was based on Advanced Grade Control (AGC) drilling information available at that time.

The current Geological Model and Mineral Resource Estimate was prompted by an Advanced Grade Control drilling campaign that took place during October 2014 in an attempt to identify additional Oxide Ore.

The question is therefore:

"Does the FE2 deposit contain additional Oxide Ore Mineral Resource (Indicated, Inferred and Blue Sky Potential) and if so, how much?"

1.4. STATEMENT OF OBJECTIVES OF THE DISSERTATION

The main objectives of this research were to, based on the new AGC drilling information, assess the Oxide Ore Mineral Resource potential of the FE2 gold deposit including:

- Produce a 3D Geological model of the deposit
- Analyse the assay data using classical Statistics and Geostatistical techniques
- Estimate the Oxide potential of the Mineral Resource
- Generate a Mineral Resource Model for the FE2 deposit
- Generate a Uniform Conditioning model for the FE2 resource
- Test the reliability of the estimates
- Reconcile the new model with the previous model and against the sample data

1.5. RESEARCH METHODOLOGY AND THESIS STRUCTURE

The dissertation follows a quantitative methodology supported by industry research and the company guidelines stipulated by AngloGold Ashanti. The thesis is structured to outline the entire estimation process in 11 Chapters with the final chapters left for recommendations for future work and Risk Analyses.

The methods employed in this study included:

- Compilation and validation of drill hole data provided by site (bias testing; boundary analyses; reviewing data collection, lithological logging, sample preparation and analysis, quality assurance and quality control, bulk density and appraisal of database integrity)
- Statistical and Geostatistical analysis of the data and an evaluation of the results
- 3D geological modelling of wireframes for the gold mineralisation and graphite alteration and Digital Terrain Models for the REDOX, Hardness and Laterite zones

- Exploratory Data Analysis to test and develop estimation zones (stationarity, compositing, domaining and cut-off grade determination)
- Variography (down-hole and directional variograms, determine nugget and variogram ranges; assessment of variogram parameters)
- Quantitative Kriging Neighbourhood Analysis (analysis of slope of regression, krige weights, minimum and maximum number of samples, block size determination, discretisation and sear range optimisation)
- Block Modelling (boundary analysis, search strategies, grade capping and cutting study, block model fields)
- Mineral Resource Estimation (Estimate Au, graphite and hardness probability values into the resource model using appropriate estimation techniques)
- Post process of Krige results with Uniform Conditioning in Isatis® Geostatistical Software
- Model Validation
- Assess the risk associated with the model and classify the mining panels in accordance to the guidelines stipulated by AngloGold Ashanti Limited
- Detailed reconciliation study including Classification of the Mineral Resource
- Comparing the new geological model and Mineral Resource estimate with previous geological models and Mineral Resource estimates for FE2

2. GEOLOGY

This chapter outlines the geological interpretations used for the modelling; the type of data used in the modelling (including the different drilling campaigns contained in the database used for the estimate) and is a summary of the process followed to create the geological model and estimation domain selection.

Simplified geological profiles for the FE2 deposit are available on the geology archive compiled by IAMGOLD that can be accessed on their website (iamgold.com, 2015). For more detailed explanations, the reader is directed to the papers by Diena et al., (2012) , Masurel, et al., n.d. and Masurel et al., 2012.

2.1. REGIONAL GEOLOGY

The Sadiola deposit is located on the West African craton in the Malian portion of a Paleoproterozoic inlier known as the Kedougou-Kenieba window (KKI). It is bound by the Kenema-Man Shield in the northeast; the Pan-African Mauritanide Hercynian Belt in the west; and by un-deformed Neoproterozoic and Palaeozoic sedimentary formations of the Taoudeni in the south (**Figure 3**) (Diena et al., 2012).

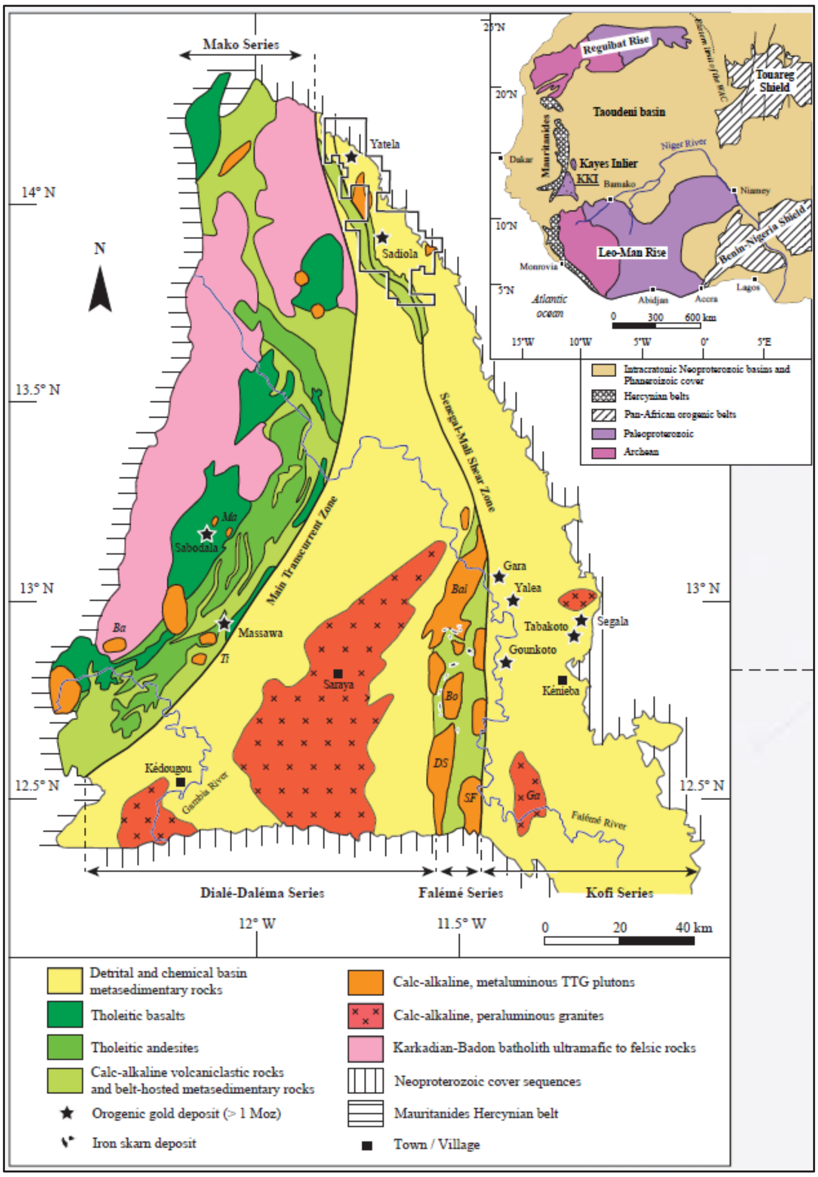


Figure 3: *Map of Regional Geology showing Kedougou-Kenieba Inlier and surrounding Pan-African Belts* (Masurel, et al., n.d)

The Birimian components of the KKI have been interpreted as a collage of at least two N-S trending terranes. To the west, an older (+/- 2.2 Ga) greenstone belt volcano-sedimentary succession intruded by major Calc-alkaline batholiths belongs to the Saboussire Formation. It is separated from the dominantly sedimentary Kofi Formation by the major north to northeast trending Senegalo-Malian Shear Zone (Aida et al., 2012 and Masurel, et al., n.d).

A more detailed view of the KKI and the shear zones is presented in **Figure 4** together with several additional significant gold bearing deposits that are hosted within the splays of the Senegalo-Malian Shear Zone (Masurel et al., n.d.).

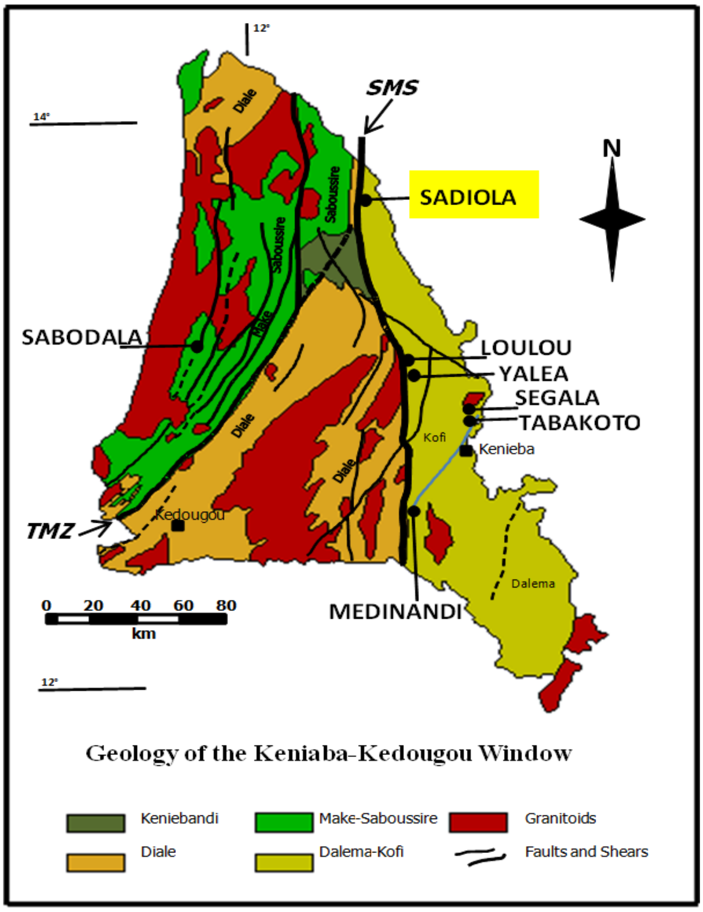


Figure 4: *Geology of the Kedougou-Kenieba Inlier showing the regional Main Transcurrent Shear Zone (MTZ and the Senegalo-Malian Shear Zone (SMS)* (iamgold.com, 2015)

The Kofi Formation is significantly younger and has been intruded by Calc-alkaline batholiths dated at 2.0 – 2.05 Ga. Metamorphic grade includes Greenschist facies, with Amphibolite grades developed locally near major intrusions (Aida et al., 2012 and Masurel, et al., n.d).

The Sadiola deposit is located in the north central section of the window and is hosted by sediments of the Kofi Formation, which have been intruded by numerous felsic intrusives. The sediments consist of fine-grained Greywacke - believed to be distal turbidites; and impure carbonates with minor tuffs and acid volcanics.

The mineralisation has a strong structural control and is spatially associated with a complex weathering and alteration pattern as depicted in **Figure 5**, possibly associated with a mesothermal origin - typical for gold emplacement in West African Birimian rocks (Aida et al., 2012 and Masurel, et al., n.d).

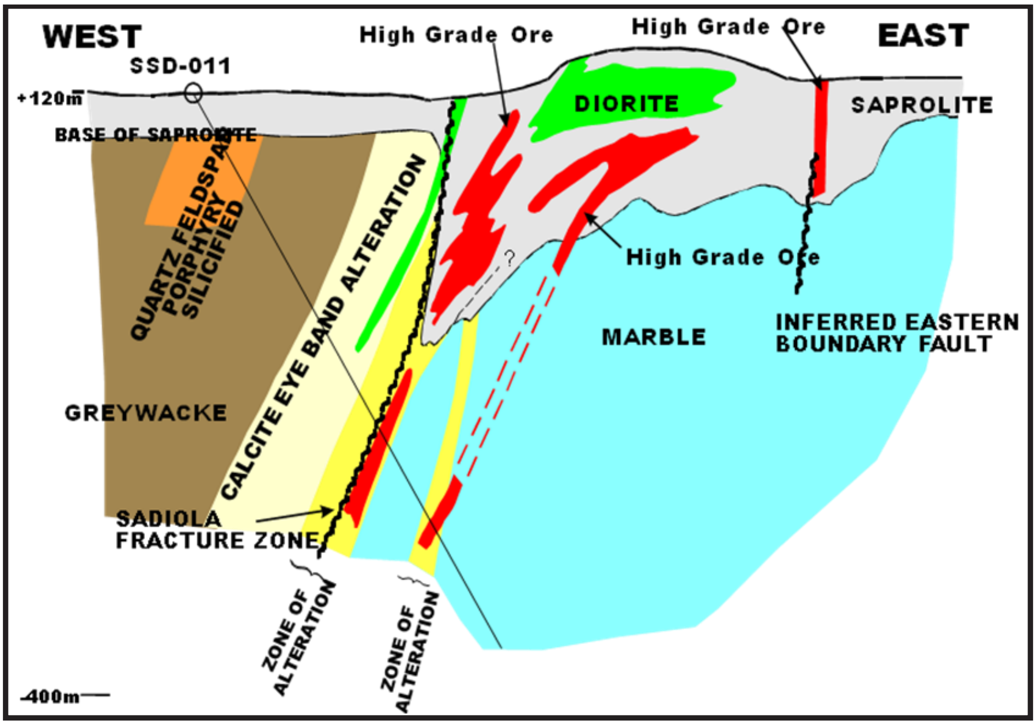


Figure 5: *Geology of the Sadiola Hill Gold Mine Type Cross Section* (SEMOS, 2012, pp 25)

2.2. LOCAL GEOLOGY

The FE2 deposit is a Gold (Au), Arsenic (As), Copper (Cu) ± Cobalt (Co) and Beryllium (Be) anomaly, located approximately 6 km north-east of Sadiola along the contact between marble and metapelites. It occurs on the western limb of a syncline structure identified in the FE3 and FE4 pits. The lithologies are folded and dip gently to the east (25-50 degrees) and comprise of graphitic metapelites overlying impure carbonates.

The FE2 deposit is cross cut by a NNE-SSW structure, intruded by a dolerite dyke (**Figure 6**).

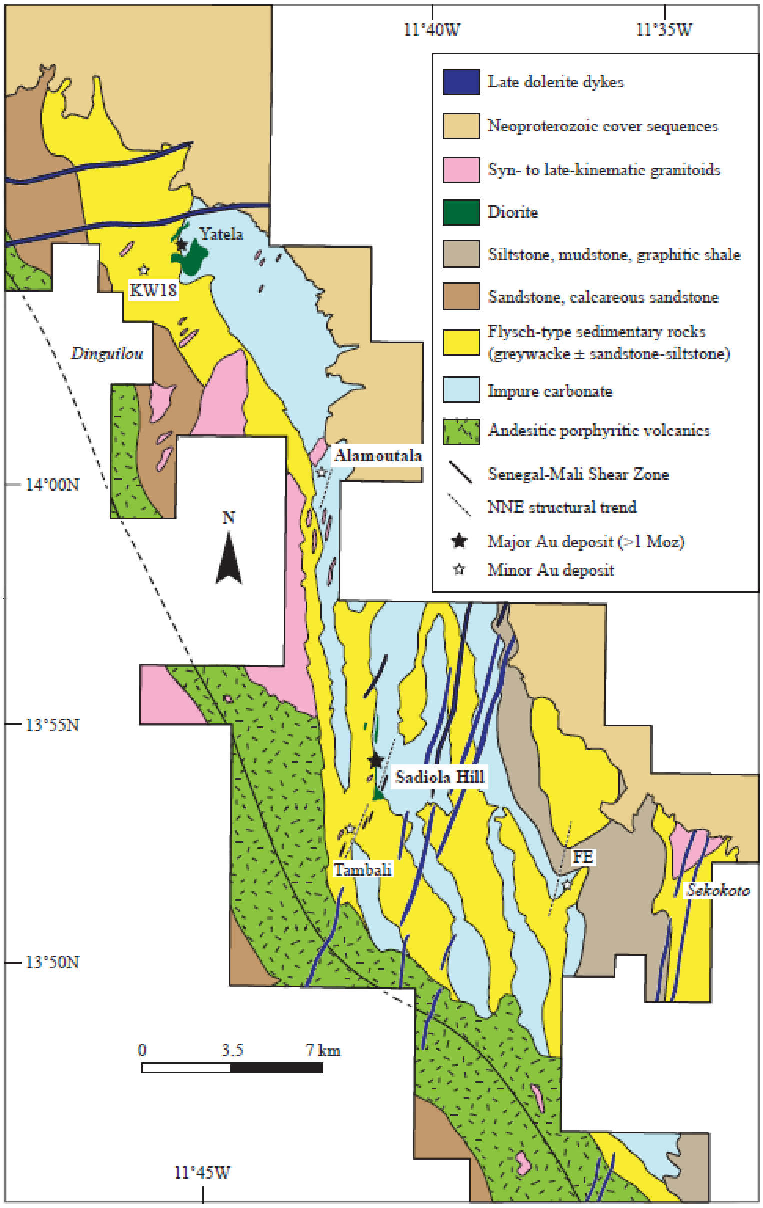


Figure 6: *Geological Map showing the site crosscut by a NNE-SSW structure which was later intruded by dolerite dykes* (Masurel, Thebaud, Miller, & Ulrich)

2.3. MINERALISATION

Pervasive gold mineralisation ranging in grade from 2 g/t to 20 g/t occurs along the SFZ over a strike length of approximately 2,500 metres and remains open to the north and south. The location and geometry of high grade mineralisation appears to be controlled by the confluence of the SFZ with the 020º striking splays, resulting in steeply to vertically plunging zones within the plan of the SFZ.

Gold mineralisation occurs in all of the four major rock types (Marble, Greywacke, Diorite and Quartz-Feldspar porphyry), and is spatially associated with a complex alteration pattern (Aida et al., 2012 and Masurel, et al., n.d).

Alteration assemblages identified to date include Calc-silicate, Potassic, Chlorite–Calcite and Carbonates, and point to a mesothermal origin for gold mineralisation. A summary of the alteration codes included in the model are presented in **Table 2:**

Table 2: *Summary of Alteration types identified at Sadiola Mine*

Alteration Code	Description	Alteration Code	Description
ALB	Albitization	LIM	Limonitic
BIOT	Biotitization	LIMJ	Limonite joint
CHL	Chloritization	KLN	Kaolinitic
CLC	Calcite	SERI	Sericitite
DOLC	Dolomitization	SIO	Silification
EPI	Epidotization	SMECT	Smectite
FSP	Feldspatization	TOUR	Tourmaline
GRAPH	Graphitic	KSIL	Calc-silicate (actinolite-tremolite)
HEM	Hematitic	CEB	Calcite eyes bands
HEMJ	Hematite joint	DEB	Dolomite eyes bands

Gold is associated with both arsenic and antimony dominated Sulphide assemblages including Arsenopyrite, Pyrrhotite, Pyrite, Stibnite and Gudmundite. Primary gold is extremely fine grained, dominantly less than 15 microns, with rare grains approaching 50 microns.

The Sadiola Deposit has been intensely weathered to depths of up to 220 metres.

The dolerite dyke is post mineralisation since it cross-cuts the mineralisation and displaces it. It is also generally barren. Later EW structures have also been identified.

2.4. DATA COLLECTION

The newly drilled AGC holes used for the 2015 model update are saved under the field YEAR = 2015. The detail of the holes pre-dating these are summarised below and saved under the field YEAR=2014 which denotes that they were used in the 2014 model. The description of the drill campaigns provided below is a summary of the account detailed by SEMOS (2012).

2.4.1. OLD DRILL HOLES (USED FOR 2012/2014 MODEL UPDATE)

During 1993, the Marble/Metapelite contact was tested by drilling (SEMOS regional drillholes FE-001 to 003). The drilling showed that the mineralisation dipped shallowly to the east (25-50 degrees). Drillhole FE-003 gave a grade intersection of 2.99 g/t Au over a drilled width of 8.9 meters.

During the early part of 1998, a small reconnaissance was conducted to the south of the FE2 prospect. A total of 9 holes, amounting to 450 meters, were drilled along two fence lines (800m apart) at dip of -60 degrees towards the west and to a depth of 50 meters. The holes were collared to target the nature of contact zone delineated by the IP survey that identified the Au, As and Cu anomaly. One encouraging intersection of 1.0 g/t Au over a 12 meter width at a depth of 38 meters was obtained from AFE2-022 drillhole.

A Reverse Circulation (RC) reconnaissance drilling programme comprising 29 drillholes (1,506 m) was completed during December 1998. The aim of the programme was to verify the main geochemistry anomaly which is situated in the vicinity of the old workings. Drilling was done at 50 m drillhole intervals along three fence lines 200 m apart. Low order mineralisation (0.5 g/t - 1.5 g/t Au) was established over narrow widths. The best intersections were returned from drillholes AFE2-043 and AFE2-031 which contained 3.36 g/t over 6 m and 2.08 g/t Au over 20 m widths, respectively.

In June 1999, as part of a phase I reconnaissance follow-up drilling programme, 958 m of RC drilling (16 holes) were completed. The presence of the diorite dykes within a marble host rock was considered an influencing factor in the localisation of the mineralisation in the area. The best intersection was returned by drillhole AFE2-058 (1.15 g/t Au over a drilled width of 8 m from 16 m to 24 m depth).

In order to address the geological uncertainty, the Programme de developpement des Ressources (PDRM) was tasked to excavate five trenches over an anomalous area in 2000. The first trench, 140 m long, straddled the AFE2-043 intersection and some old workings to the west of this drillhole. The second trench, 100 m long, was excavated immediately south of the FE-001 to FE-003 drilling line (drillholes not found in the current database). The third trench, 140 m long, was positioned across a well-defined N-S trending conductivity contact delineated by the SPECTREM airborne Electro-Magnetic survey.

Trench TR1 intersected three mineralised zones: 33 m width at an average grade of 4.5 g/t Au to the east, 19 m width at 1.8 g/t Au and 4 m width at 8.2 g/t Au to the west. The last intersection coincided with the area of old workings.

A 44 m wide barren area, consisting mainly of graphitic metapelite, separates the mineralised zone. Trench TR2 also intersected two mineralised zones: 32 m width at 2.7 g/t and 3 m width at 2.6 g/t Au. Trench TR3 returned 10 m width of 1.6 g/t Au.

After the encouraging results obtained from the trenching, a short air core programme was drilled to rapidly delineate the mineralised zones between trenches TR1 and TR2 and the possible extension to the north and south. Sixty-three vertical Reverse Air Blast (RAB) holes were drilled on 50 m x 20 m grid spacing. Holes were drilled to a depth of 40 m. This programme was completed during May and June 2000. The assays confirmed the grade obtained by trenching and the extension of the mineralisation to the north and south.

From March to September 2000, an additional programme of a total of 162 RAB drillholes amounting to 5,799 m and 7 RC holes of 340 m, were drilled on 100 m x 25 m grid spacing between trench TR2 and TR3. This programme was aimed at delineating a 30 m – 50 m wide potential mineralisation zone over the strike length of 600 m. Its objective was reached and the strike length of the mineralisation increased from 400 m to 900 m open-ended towards the north.

During May to July 2001, a drilling campaign of 97 inclined RC holes, amounting to 8,892 m, was laid out over the area of interest, reducing the drilling grid resolution to 50 m x 50 m. In addition 4 more trenches (TR4-TR7) were dug. Trench 5 and 6 were dug to the south of FE2. Trench TR4 returned 3 m at 2.20 g/t Au and trench TR7, 4 m at 1.82 g/t Au.

In April to June 2002, 75 RC holes, amounting to 8,670 m, increased the drilling grid resolution to 50 m x 25 m over a strike length of approximately 1.1 km.

During February to May 2005, an additional drilling campaign of 182 RC holes amounting to 16,322 m was carried out for increasing the drilling grid resolution to 25 m x 25 m.

Since 1995, Boart Longyear has completed most of the drilling. RC drilling is undertaken using 115 mm dual tube drill rods fitted with a tungsten carbide drag bit. Hard material is drilled with the use of a face sampling reverse-circulation hammer.

All drill collar positions were surveyed by the SEMOS mine survey team using a differential Global Positioning System (GPS). Holes were also surveyed down-the-hole by making use of Sperry-Sun downhole camera survey equipment.

2.4.2. NEW DRILL HOLES (INCLUDED IN THE CURRENT MODEL)

The last model update was done in June 2014. The current update was prompted by an Advanced Grade Control drilling campaign that took place during October 2014.

The exploration drill data (752 holes at a 25m x 25m grid spacing) used for the previous estimate was combined with the new data (415 holes at 12.5m x 12.5m grid spacing) to inform the new resource estimate.

333 RAB holes (used for previous update) were excluded from the database due to poor sample quality. RAB drilling causes sample chips to be blown out along the side of the drillhole resulting in potential contamination and grade smearing.

2.5. BIAS TESTS

Due to the paucity or holes available for the estimate, it was considered best to combine the 2014 grade control (GC) and exploration (EX) database with the 2015 AGC database (both validated) into one for the estimation. Since these comprised of holes of different support, bias tests were completed to check for discrepancies between GC and EX grades in an area representative of being sampled by both GC and EX holes.

Bias tests included using Quantile-Quantile (QQ) Plots and histograms to compare if the EX holes and GC holes had the same underlying sample grade distributions i.e. equally represented the bias test area.

QQ plots are used to plot the quantiles (fraction of data below a given value) of one data set against the quantiles of another, approximated by a straight line. QQ plots and histograms are good tools to visualise the distribution of the data.

A total of 81 EX holes and 141 GC holes were located inside the bias area. The drill hole spacing does not influence the validity of the bias tests.

The selected bias test area is shown in **Figure 7** and includes the southern end of the modelled area which showed good representativeness of both grade control and exploration drillholes. Only samples occurring within the Ore wireframe were used in the bias testing.

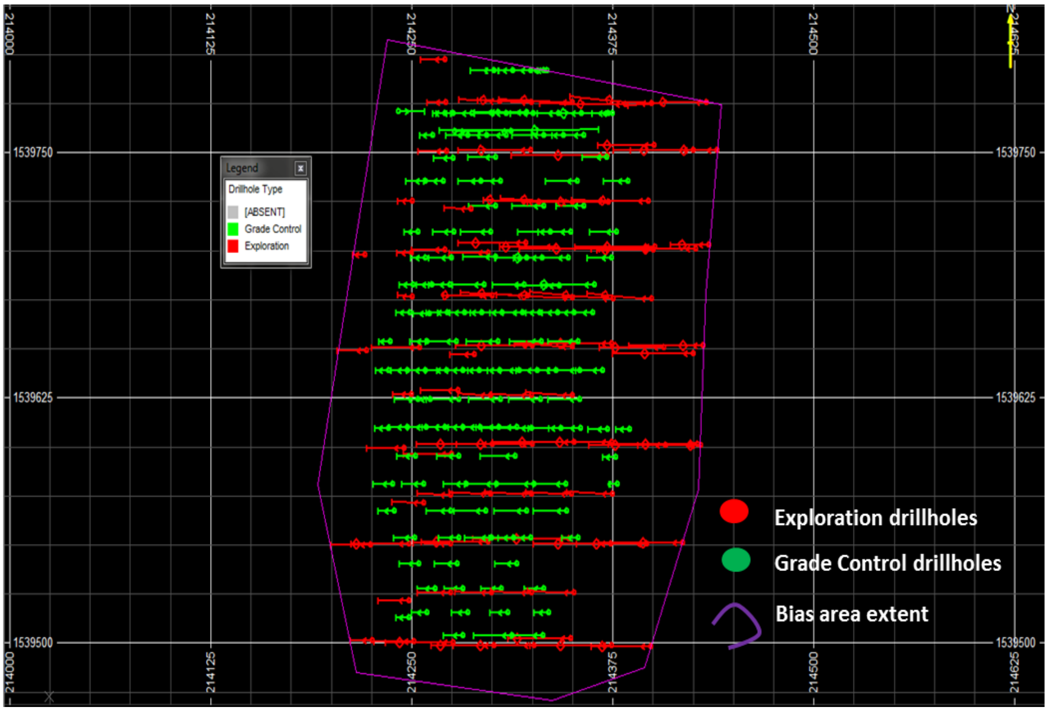


Figure 7: *Bias test samples – new versus old exploration holes*

The previous model estimate included 32 RAB holes. These holes, as stated earlier were removed from the database because the quality of the RAB sample is generally not considered adequate for resource estimation (a significant amount of smearing and downhole contamination can occur).

This may have compromised the integrity of the data used for the estimate previously.

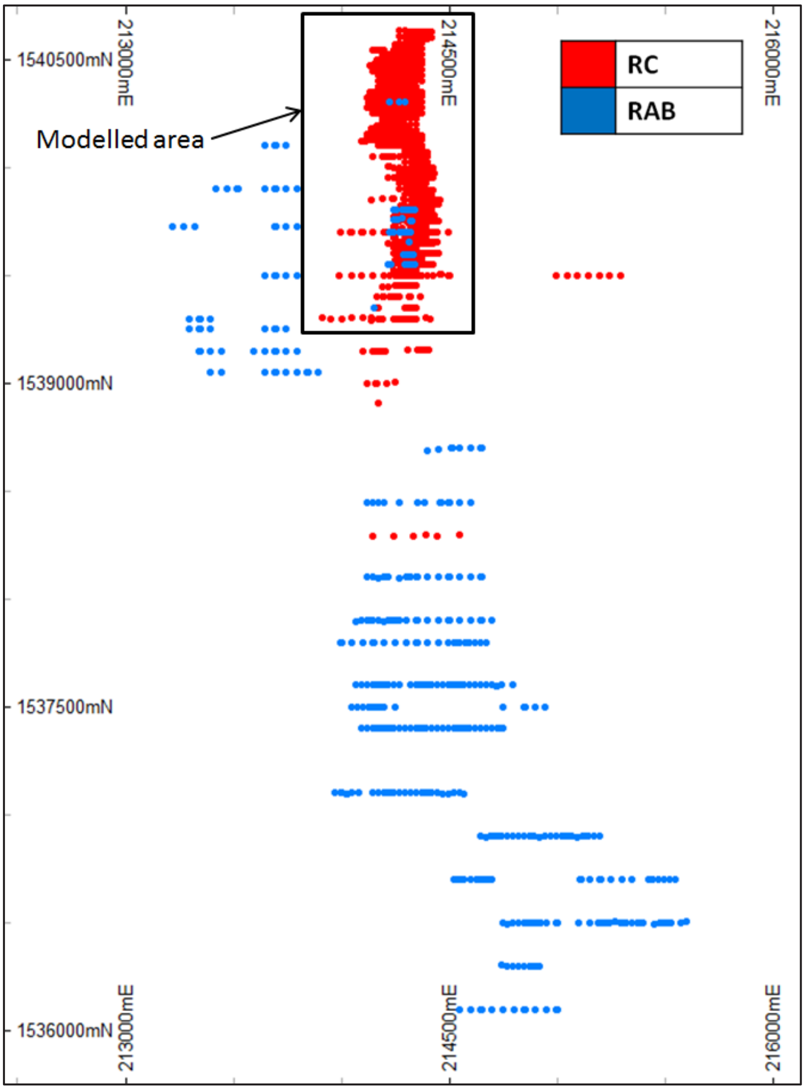
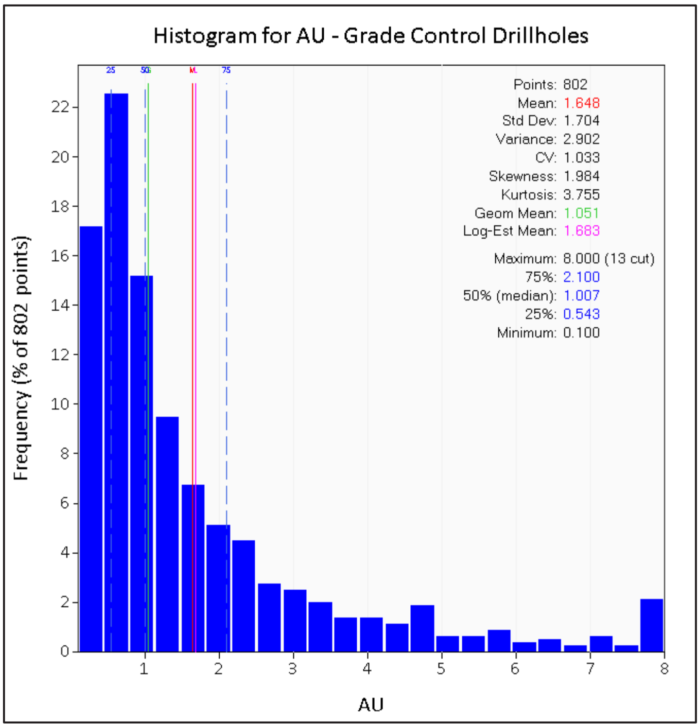


Figure 8: *Plan view showing the location of the RAB and RC drillholes used in the 2014 resource update*.

The histograms and summary statistics of the two bias test datasets (EX and GC) are shown below – both datasets were grade capped prior to the analysis (to 0.1 g/t at the bottom end and 8 g/t at the top end of the distributions) which corresponded to the capping applied previously.



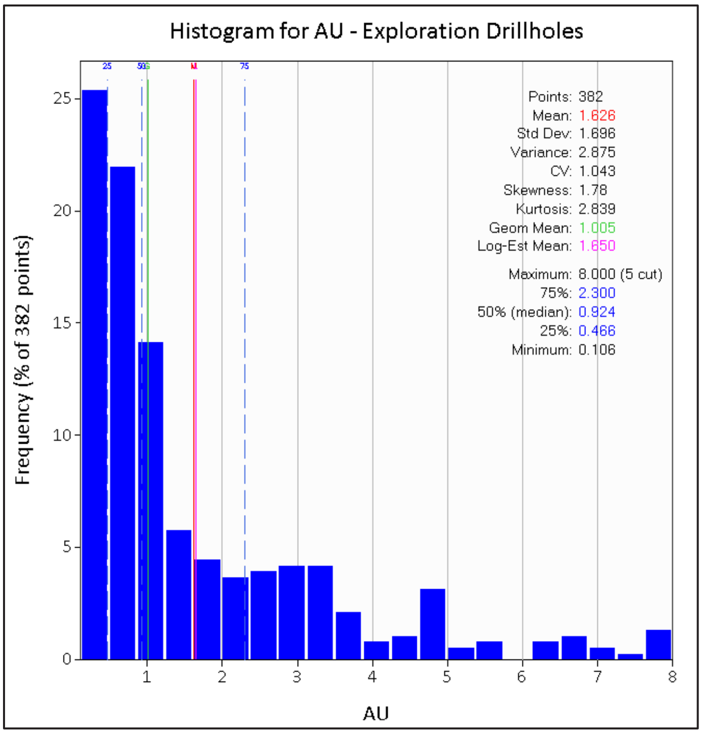


Figure 9: *EX and RC histograms*

The percentage differences between the mean, median and 25th and 75th percentiles are shown below.

Table 3: *Comparative statistics – GC vs. EX*

Statistic	GC	EX	Percentage difference
25th percentile	0.54	0.47	12%
Median	1.01	0.92	18%
75th percentile	2.10	2.30	10%
Mean	1.65	1.63	1%

The GC grades, when compared with the EX grades, showed slight over-reporting at the lower end of the distribution, but under-reporting at the upper end. The mean values differed by only 1% but the medians by 18% with the GC grades again over reporting in the middle end of the spectrum. This observation is also seen in the supporting QQ plot shown in **Figure 10.**

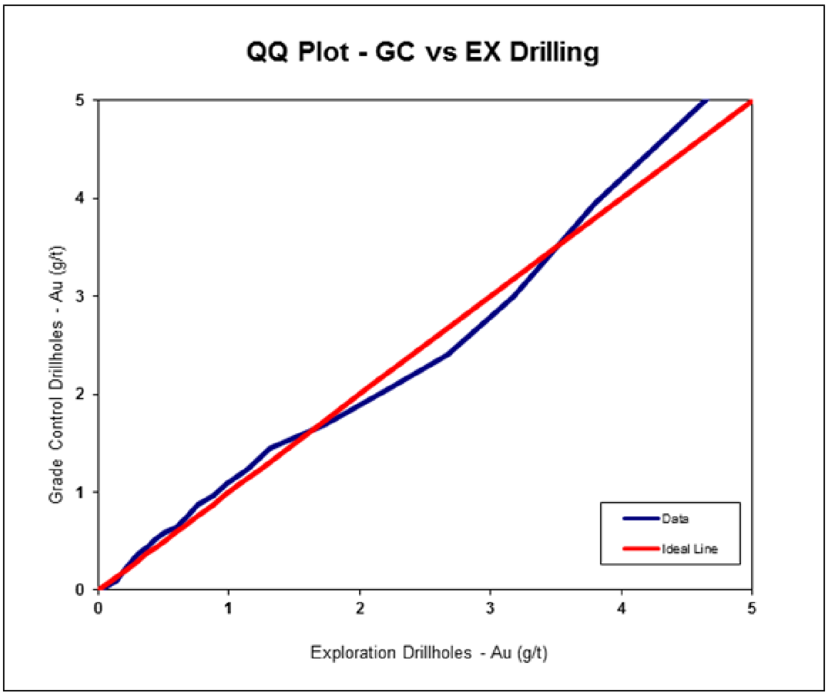


Figure 10: *QQ plot: GC vs. EX*

The reported mean and median values of the EX and GC holes within the bias test area compared well. A good correlation in grades at 1%

difference between the means from the different drilling campaigns was noted, thus reducing risk to the global estimates.

The minor differences identified may be related to the paucity of exploration drill data available.

2.6. LITHOLOGICAL LOGGING

All logging is undertaken using the SEMOS standardised system which was designed to facilitate computer capture and manipulation of logging codes without loss of geological detail.

The system is important to ensure consistency between the various geologists who are responsible for collecting core logging data.

Holes that were drilled before the implementation of the standardised system have also been converted into the new database format. Whilst care is taken to ensure that the logging codes provide description of the rock units, alteration type, alteration pervasiveness, mineralisation, weathering style and weathering intensity; the drillholes in the database provided by site contained lithological logs but did not contain any accompanying description of the lithological codes.

The logging code descriptions were therefore sourced from the 2012 SEMOS Resource Report and is presented in **Table 4**.

Table 4: *Summary of lithological codes used in the database*

LITHOLOGY	DESCRIPTION	LITHOLOGY	DESCRIPTION
ANDS	Andesite	MCGL	Matrix Supported Conglomerate
BRCC	Breccia	MGWK	Meta-Greywacke
CAMB	Calcitic Marble	MISS	Miscellaneous
CGL	Conglomerate	MIX	Mixed Zone
CLAY	Clay	MPEL	Meta-Pelite
CSIL	Calc Silicate	MSLT	Meta-Siltstone
CSST	Karst Blocky Coarse Grained Sandstone	MSST	Meta-Sandstone
DACP	Dacitic Porphyry	MYL	Mylonite

DCMB	Decarbonated Marble	OVB	Overburden
DIOR	Diorite	PCGL	Pebble Supported Conglomerate
DISS	Dissolution	PIC	Pisolitic Clay
DOL	Dolomite	PIG	Pisolitic Gravel
DYKE	Undifferentiated Intrusive	QFP	Quartz-Feldspar Porphyry
FSST	Karst Fine Grained Sandstone	QZTE	Quartzite
GRDR	Granodiorite	REJ	Debris of Previous Mining
GRDT	Granite	RHLT	Rhyolite
HBR	Hydrothermal Breccia	SAP	Saprolite
LAMP	Lamprophyre	SOST	Sourokoto Sandstone
LAT	Laterite	SST	Sandstone
LOSS	Lost Sample	TUFS	Tuffs
LOST	Lost Core	USS	Unconsolidated Sand
MC	Mottled Clay	VOID	Void - No Recuperation

2.7. SAMPLE PREPARATION AND ANALYSIS

RC chip samples were collected over 2 m intervals by employees of SEMOS and was split using a 2-tiered stacked riffle splitter (Jones riffler). Samples were crushed on site using a conventional jaw crusher before submission of an approximate 2-3 kg of sample to Société Générale de Surveillance (SGS) Kayes.

At the laboratory, the samples were dried (typically for 8 hours), then passed through a jaw crusher which reduced the maximum size to <6 mm. A riffle splitter was used to reduce the sample size to 500 g which was then pulverized for a minimum of 3 minutes in a Labtech LM2 chrome steel pulveriser. Depending on the lab and material type, 30 or 50 grams of material were extracted for analysis.

The gold analyses were carried out by traditional Fire Assay followed by Atomic Absorption Spectroscopy (SEMOS, 2012).

2.8. QUALITY ASSURANCE AND QUALITY CONTROL (QAQC)

During the QAQC process assay data is checked and assessed in terms of its reliability, accuracy and precision. The QAQC supports the data validation process.

The full QAQC report provided by SEMOS for the data that informed the estimate is presented in **Appendix A**.

The FE2 site is covered by a total of 1,074 holes drilled over the period 1998-2015 (**Table 5**). Only drillholes that were drilled in the years 2013-2015 are included in the QAQC report. Of these, 416 AGC holes comprise the new drill data. The QAQC results were reviewed and concluded to be of adequate quality for use in the resource estimate.

Table 5: *The count of drillhole types drilled at FE2 to date*

Project	DD	RC	RAB
Exploration	3	487	168
Advanced grade control	0	416	0
Total	**3**	**903**	**168**

*DD=Diamond Drillhole, RC=Reverse Circulation, RAB=Rotary Air Blast

The two primary quality control measures fundamental to assay programmes is to check standards and check duplicates (Roden and Smith, 2001). The routine insertion of QC materials is incorporated into the FE2 sample streams and regular audits and job observations are performed to monitor quality.

Checking standards is a measure of assay accuracy whereas checking duplicates is a measure of precision for the assaying process.

The QC material comprised Certified Reference Materials (CRMs), blanks; field and pulp duplicates and pulp reject repeats from previous sample submissions (**Table 6**).

QC programmes were run in addition to the normal QC insertions and monitoring undertaken by the assay laboratory. The CRMs are commercially certified standards prepared and supplied by Rocklabs Limited for a variety of gold grade ranges.

Table 6: *Summary of the QAQC material in batches of samples received*

Samples and Quality control Material submitted – FE2			
CRM or SRM	**Number submitted**	**% samples submitted**	**Comment**
Standards	178	<1	< QAQC Rev 1.05 guideline of 2014 recommended level of 2%
Pulp blanks	390	1	Within QAQC Rev 1.05 guideline of 2014 recommended level of 1%
Coarse blanks	247	<1	< QAQC Rev 1.05 guideline of 2014 recommended level of 1%
Field duplicates	367	1	Within QAQC Rev 1.05 guideline of 2014 recommended level of 1%
Check assays	NA	NA	included in the 2013 annual check assay programme
Project samples excluding RAB samples	28024	% estimates excludes RAB samples and based on GC recommended levels.	
Project samples	**29989**		

The performance of the certified reference materials (CRMs) was very good. In all, a total of 178 standards from 3 different grade ranges (low, medium and high) were inserted in to the batches of samples submitted for assay.

If a CRM fails the QAQC process, for example, if the CRM results do not fall within ±2 standard deviations of the expected value, the QAQC procedure is reviewed. If other QC checks fail, the work needs to be repeated.

The duplicates (field duplicates and pulp repeats) and blanks (coarse and pulverised) also passed the QAQC process. The assay data was therefore accepted to be used in the modelling and evaluation of FE2 since they were both precise (repeatable) and accurate (unbiased).

2.9. BULK DENSITY

Although Mineral Resource and Mineral Reserve estimates are reported in terms of grade and tonnage, they are produced in terms of three parameters: grade, volume and density (Lipton, 2001).

The bulk density or tonnage factor is a very sensitive value. Small changes have significant implications on how tons and contained metal is estimated (Stephenson and Vann, 2001).

A good background on Bulk Density is presented by Lipton, 2001, in his paper titled *Measurement of Bulk Density in Resource Estimation* which elaborates on the importance of the dry bulk density when converting wireframe volumes to tonnage estimates. When the bulk density fluctuates markedly between samples, it is advised to incorporate the density into the Mineral Resource estimate to prevent grade biases.

A ROCKTYPE field (representing material types) was added to the model and bulk densities were assigned based on these ROCKTYPE values in accordance to the previous model update. The density measurements are generally performed in air and water (for exploration samples) or using a downhole density probe (for GC samples).

Previously, no density measurements were taken for the FE2 deposit. For modelling purposes, the FE3 deposit densities were applied to the FE2 deposit since these two deposits are considered to be similar in terms of rock types, alteration and mineralisation style.

Density measurements for 2015 were carried out for the FE2 pit from the advanced grade control holes (using downhole density probe).

These densities are compared to those used for the 2014 model and are presented in **Table 7**. The densities were mostly comparable except for the Soft Oxide density probe measurements reported in 2015 which were significantly higher than in 2014. Further work needs to be done to confirm the validity of the density results before updating the 2015 density values.

Density data was stored in the ***DENSITY*** field of the model.

Table 7: *Densities used for modelling*

ROCKTYPE	DENSITY - FE2 2014 (t/m3)	DENSITY – FE2 2015 (t/m3)
1 – Laterite and Clay	2.22	2.11
2 – Saprolite (Soft Oxide)	1.97	1.97
3 – Silicified Oxide	2.16	2.05
4 – Soft Sulphide	2.46	2.46
5 – Hard Sulphide	2.70	2.70
6 – Blast Oxide	2.16	2.05
7 – Blast Sulphide	2.70	2.70
8 – Transitional material	Soft (hard=0) 2.21 Hard (hard=1) 2.46	Soft (hard=0) 2.21 Hard (hard=1) 2.46

2.10.DATABASE INTEGRITY

The importance of the resource database is well outlined by Gilfillan and Stoker, 2001, pp. 31-36 who define a valid database as being “the foundation of satisfactory Mineral Resource and Ore Reserve Estimation” (pp. 31).

The database contains the raw observations and measurements that inform the resource estimates i.e. poor quality data will produce inaccurate and unreliable estimates or garbage in=garbage out.

Targeted gold production is often missed or exceeded due to anticipated plant throughputs being lower or higher than predicted.

Several factors could account for these grade discrepancies sourced at different scales. At the simplest level inadequate sampling, drilling and unreliable assays could introduce grade biases into the database. These errors become additive since geological interpretations and estimation techniques are based on the database (Minnitt, 2014). Other biases include:

- Information Effect: Only partial information is available in deciding whether the block is Ore or waste
- Support Effect: The size and shape of the data and how the variance changes according to size
- Regression effect: Over estimation of high grades and under estimation of low grades

Failure to collect reliable data, and preserve its integrity, results in poor estimation. All drillhole and surface sampling data for FE2 has been stored in a Microsoft SQL database managed by Century Systems Fusion software since 2002.

The database is derived from several sources with quality controls in place to prevent errors being introduced to the database:

- Field logs are captured into Excel templates and signed off before loading into Fusion. Key sections include: Collar, Survey, Meta Data, Sample information, Sample QA/QC insertions and geological coding.
- Survey collar positions are updated in the database and plotting positions verified.
- Geologists check on down-hole surveys and drilling methods and that all other related columns in the database such as drillhole depths, drillhole widths, drill rigs and the reason for drilling are correctly populated.
- Quality Control samples and standards are captured. Sample numbers are checked.
- Lithological information is checked

- Assay results are received in electronic format from the laboratories and loaded directly into the database. A random check of 10% of the data is done by the project geologists to confirm the validation. The database is backed up as part of the mine's IT protocol and a copy stored off site.

Despite the degrees of checking that go on in the data capturing process, some human errors may be introduced and before any resource modelling can commence, the database is validated to identify these errors.

2.11. DATABASE VALIDATION

Rigorous system and quality checks were performed on the database to verify that the sampling, assay and survey data informing the estimate were free of errors and that the database was representative, accurate, and precise (Stephenson and Vann. 2001 and Gilfillan and Stoker, 2001).

The database validation eliminates the influence of potential sampling errors and biases (Information, Support and Regression effects) on the resource estimate.

The database comprised of 416 newly drilled advanced grade control holes (12.5m x 12.5m grid spacing) and 752 exploration holes (25m x 25m grid spacing) that were used for the previous estimate.

The total drillhole database considered for the update comprised of 1167 combined GC and EX drillholes.

The location of the new holes drilling campaign is shown in **Figure 11** below.

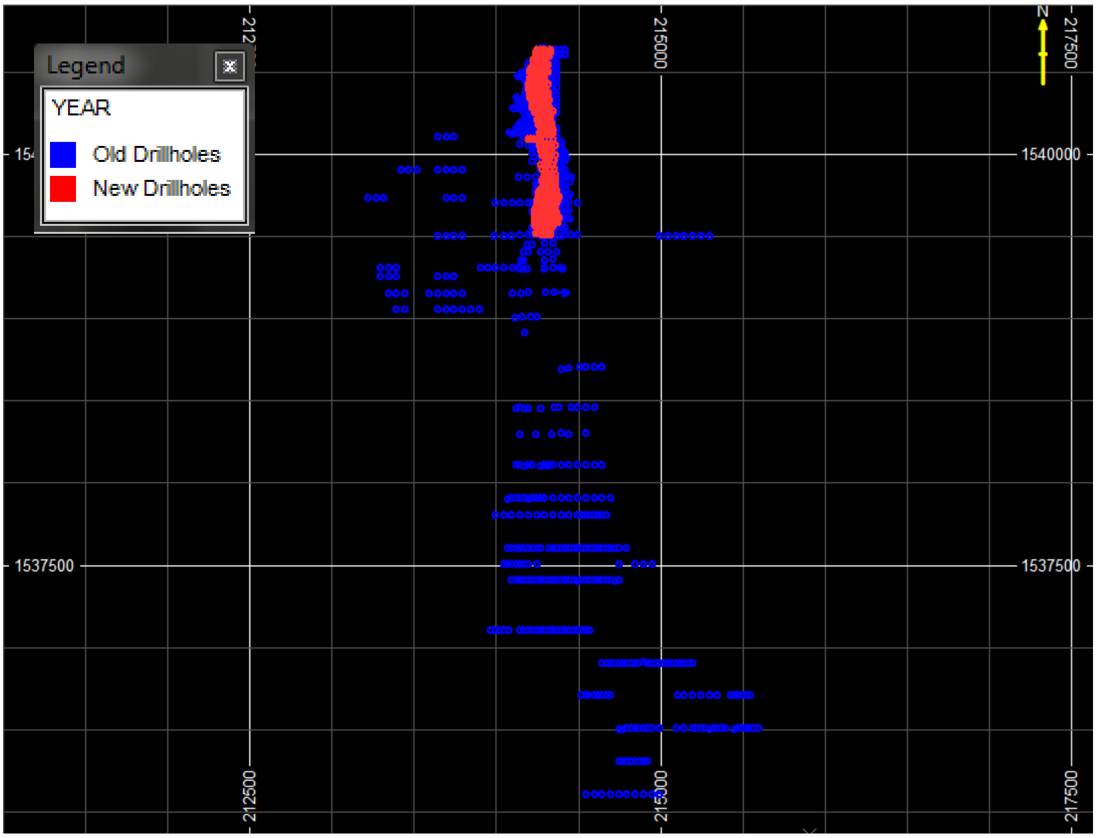


Figure 11: *Location of new and old holes*

The data used for the previous update was obtained from the 2014 Mineral Resource estimate handover file in Datamine® drillhole format . The Mine Geologist at Sadiola provided the new drillhole data for FE2 in .csv format as collar, survey, assay and lithological logs.

The drillhole data sets were combined and then validated. The drillholes were checked for:

- Zero or missing collars
- Errors in collar positions
- Duplicated collars and coordinates
- Interval errors (missing intervals, overlapping intervals, negative length intervals)
- Zero grade values
- Long sample lengths

- Visual inspection (holes terminating mid mineralisation-domain, collars not sitting in correct position, sampling gaps in the reef, drillhole deviations)

The RAB holes were excluded due to poor sampling quality. Other errors detected included duplicated surveys and invalid assay inputs (**Table 8**):

Table 8: *Summary of Errors detected in the Resource Database*

BHID	YEAR	FROM	TO	ERROR	CORRECTION MADE
FV000602	2014	0.0	2.0	Invalid Assay value	(-) was removed and value changed to 0.1g/t Au in accordance to the assay value for all samples within that BHID
FV000741	2014	14	16	Invalid Assay value	(0) was removed and value changed to 1.71g/t Au in accordance to the assay value for all samples within that BHID
FV000446	2014	n/a	n/a	Max Depth = 0	Collar and Survey files available but no accompanying sample data nor "from" or "to" information. The hole was excluded from the final drillhole file to be used for estimation

In addition to these errors, zero assays were found and queried with site personnel. The zero values were the result of voids in lithology and lost sample and therefore set to absent in the database.

Visual checks showed that a few randomly distributed collar positions did not honour the topographic surface (**Figure 12**).

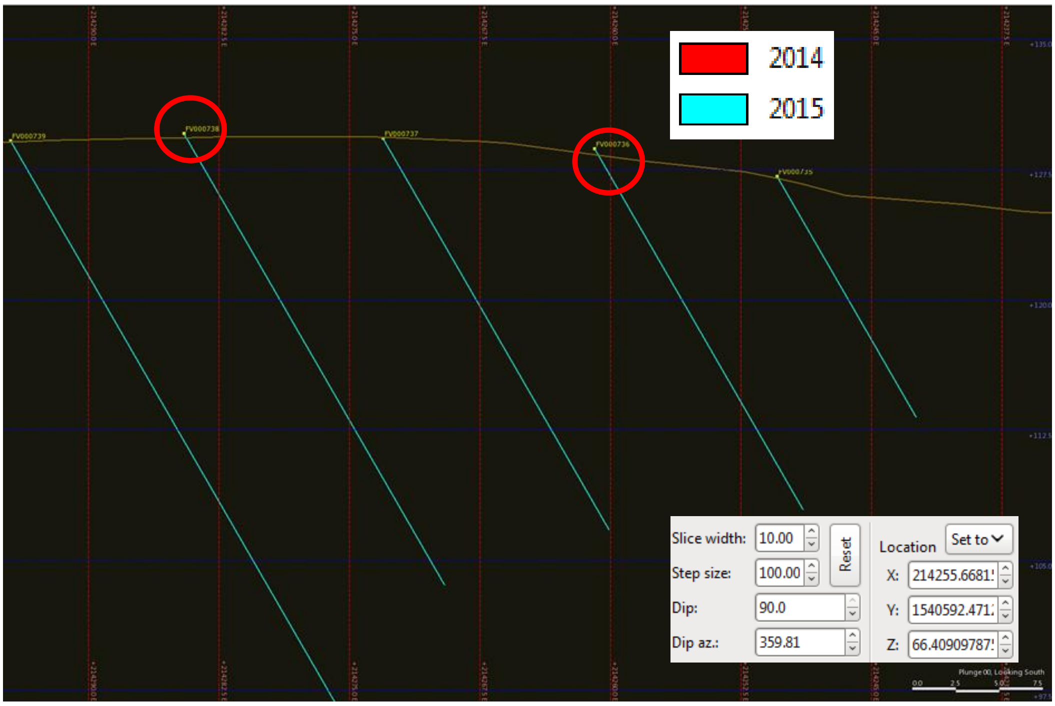


Figure 12: *Leapfrog® Mining Section View showing poor fit between collars and the topography*

To quantify the discrepancies, a new "collar" file of was created whereby the collar points were projected onto the topographic surface. The differences between the original and new ZCOLLARs were calculated and are presented in **Table 9**.

Table 9: *Erroneous collar positions*

	PROJECTED COLLARS			ORIGINAL COLLARS			
BHID	X	Y	Z	X	Y	Z	DIFF.
AFE2-005	214197.2	1539299	139.1822	214197.2	1539299	141.86	2.7
AFE2-031	214297.8	1539699	131.2187	214297.8	1539699	133.33	2.1
AFE2-169	214198.7	1540543	130.1103	214198.7	1540543	134.61	4.5
AFE2-170	214250.9	1540544	131.0857	214250.9	1540544	135.5	4.4
AFE2-243	214427.9	1539601	131.6705	214427.9	1539601	133.94	2.3
FV000001	214248.5	1539513	136.0864	214248.5	1539513	129.5	6.6
FV000110	214251.4	1539682	133	214251.4	1539682	125.5	7.5
FV000756	214256.5	1540628	125.2223	214256.5	1540628	127.18	2.0
IFE2-027	214300.5	1539677	131.4247	214300.5	1539677	134	2.6

*Negative differences signify collars below the topography and positive differences collars above the topography

Leapfrog® mining software (a registered trademark of ARANZ Geo Limited) was used to make the adjustments to the collar depth values so that any collars not lying on the topography were superimposed onto it.

A new collar table was produced with depth values that reflected the topography.

Ideally, superposition of the collars onto the topography cannot be used for modelling, because the source of the error i.e. the topography or collar is unknown. Generally small differences in the topography are expected (1-2 m) due to vegetation and these holes were included in the database.

The holes that showed the greatest discrepancies (FV000001; FV000110; AFE2-169; AFE2-170 and AFE2-243) were then checked against the latest LIDAR surface. The collars still did not honour the topography and were excluded from the estimation.

At the time of Ore modelling however, the latest LIDAR had not been provided and these holes were used to guide the Ore wireframe.

Figures 13-16 show that these holes had little impact on biasing Ore zone definition.

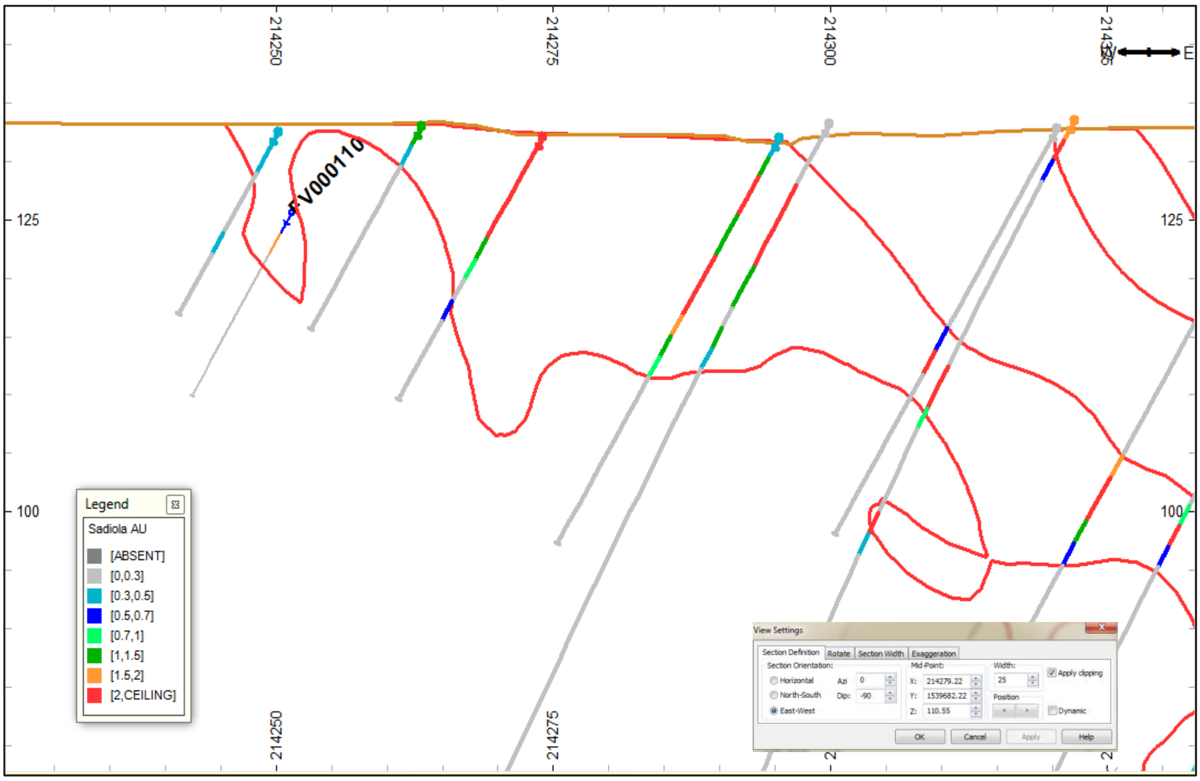


Figure 13: *Section view (looking north) showing drillhole FV000110 in relation to surrounding drill data and the Ore domain*

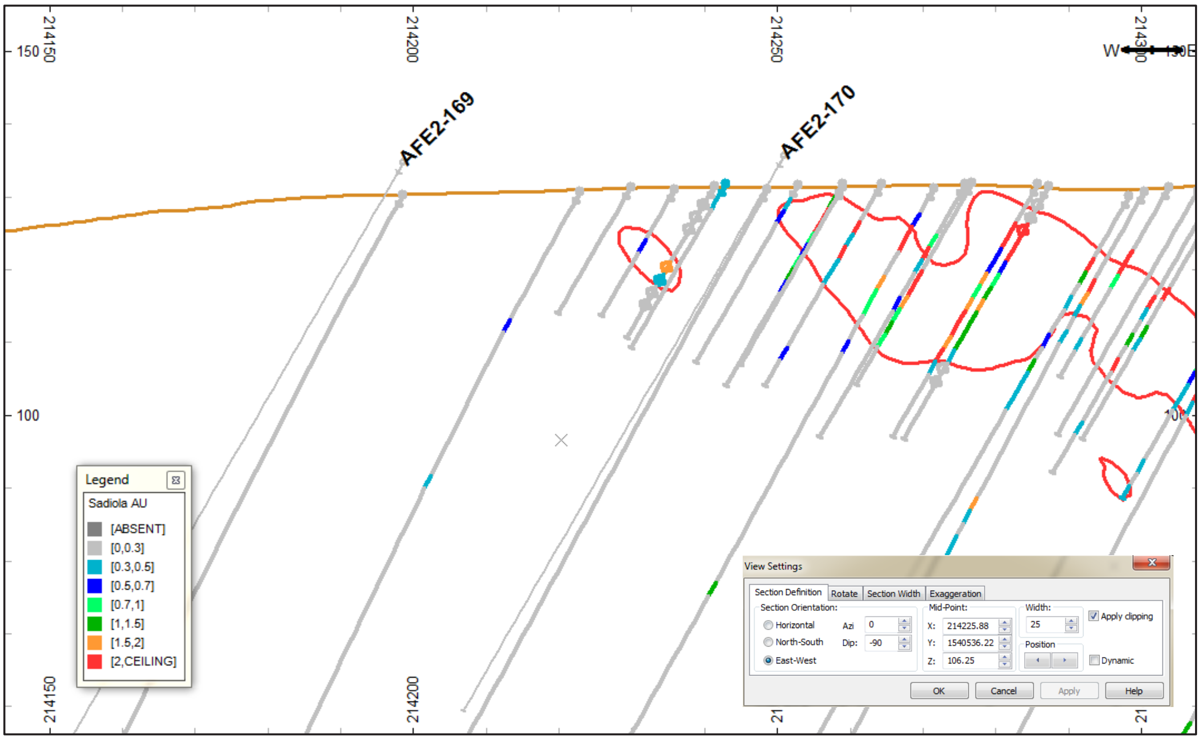


Figure 14: *Section view (looking north) showing drillholes AFE2-169 and AFE2-170 in relation to surrounding drill data and the Ore domain*

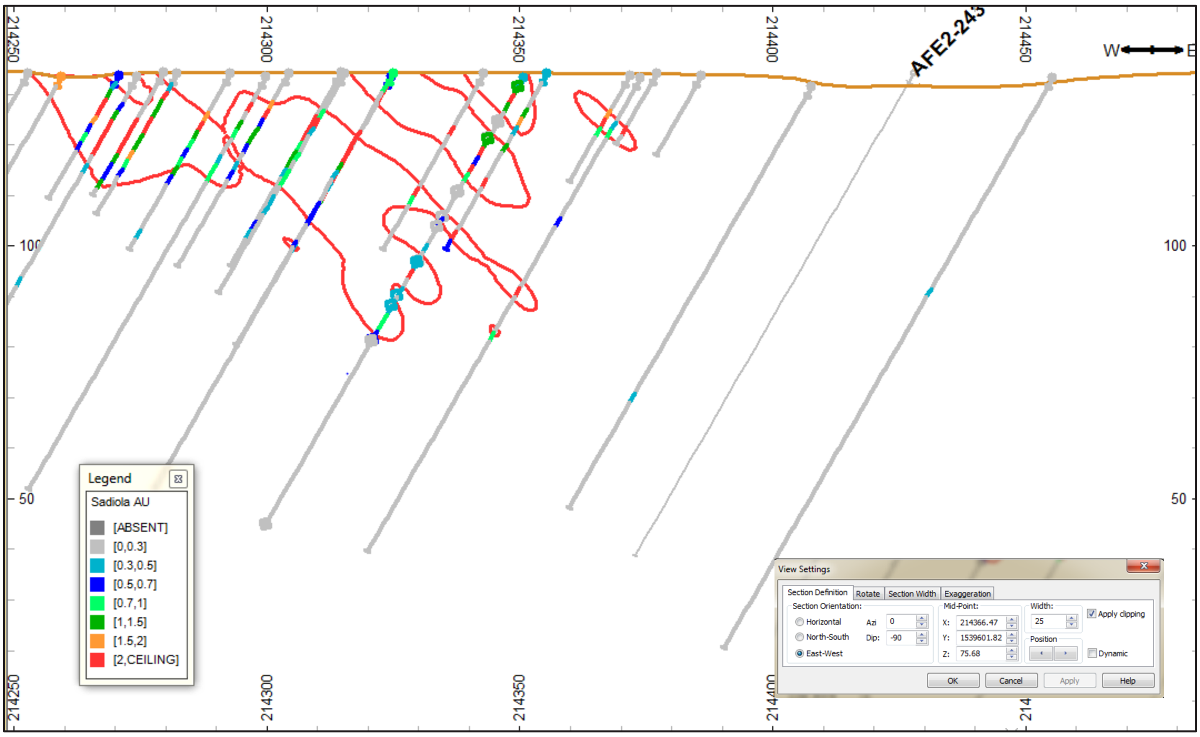


Figure 15: *Section view (looking north) showing drillholes AFE2-243 in relation to surrounding drill data and the Ore domain*

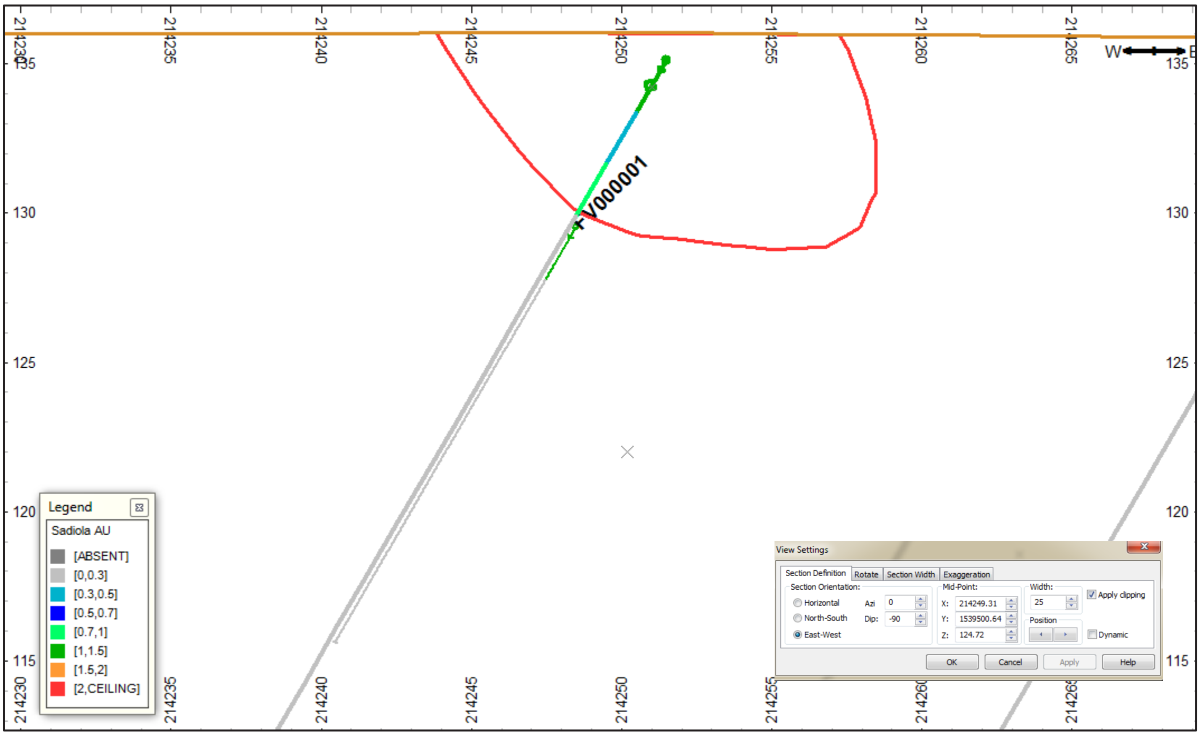


Figure 16: *Section view (looking north) showing drillholes FV000001 in relation to surrounding drill data and the Ore domain*

2.12. GEOLOGICAL MODELLING

The FE2 mineralisation is controlled by a combination of lithology, structure, weathering and alteration. Wireframes were generated for the topography, weathering surfaces, hardness boundaries, extent of the graphitic alteration and mineralisation.

2.12.1. HARDNESS AND REDOX ZONES

Material types (also called rock types) were assigned to the model based on the updated weathering and hardness surfaces (**Table 10**). The updated modelled surfaces fit well with the previous interpretations.

Table 10: *Summary of modelled surfaces*

Surface	Description
Topography	Topographic surface, top contact of the Laterite and Clay material
Laterite	Base of Laterite and Clay material
Redox 2	Contact between oxidized and transitional material
Redox 3	Contact between transitional and fresh material
Hardness	Contact between hard and soft material

The surfaces were updated in Leapfrog® using the drillhole lithology, hardness and Redox codes in the drillhole database. The topographic surface formed the top contact of the Laterite and Clay material, whilst the bottom contact was modelled using the Laterite and Clay lithology codes. In this update, the Clay material was reported together with Laterite material as required by Sadiola Mine.

The HARDNESS codes were used to model the hard/soft boundary and REDOX codes to construct the Redox boundary. Holes drilled prior to 2012 contained a HARDNESS code defined by ‘H’ Values as shown in **Table 11.**

Table 11: *Summary of definition of the hardness codes (HARDNESS) used in the modelling of the hard/soft boundary in 2012*

Hardness	Description
H1	Soft Material
H2	Soft Material
H3	Soft Material
H4	Hard Material
H5	Soft Material
H6	Soft Material
H7	Soft Material
H8	90% Soft Material
H9	Hard Material

Post 2012, the 'H' HARDNESS values were made obsolete and replaced by 'D' HARDNESS values (**Table 12**). The D1-2 codes represent soft material and D3-4 hard material.

Table 12: *Summary of definition of the hardness codes (HARDNESS) used in the modelling of the hard/soft boundary for current update*

Hardness	Description	Blasting
D1	Weak Soft Rock	No blasting required
D2	Mixture, but predominantly weak/soft	Mixture of various hardness, but which is probably mineable without blasting
D3	Mixture, but predominantly strong/hard	Mixture of various hardness, but which would probably not be mineable without blasting
D4	Strong Hard Rock	Blasting required

Table 13: *Summary of drillhole REDOX codes*

REDOX	Description
1	Oxidised Material
2	Transition Material
3	Fresh Material

Table 14: *Rock Type assignment for surfaces*

Rock Type	Description
Laterite and Clay	Material above the Laterite/Clay surface.
Soft Oxide	Material below the Laterite and Clay surface; above the Redox 2 surface (Oxide-trans contact) and above the hard/soft contact.
Hard Oxide	Material below the Laterite and Clay surface; above the Redox 2 surface (Oxide-transitional contact), but below the hard/soft contact.
Transitional	Material between the Redox 2 (Oxide-transitional contact) and the Redox 3 (transitional-fresh contact) surfaces.
Soft Sulphide	Material below the Redox 3 surface (transitional-fresh contact), but above the hard/soft contact.
Hard Sulphide	Material below the Redox 3 surface (transitional-fresh contact); but below the hard/soft contact.

"Hardness probability" estimation using the Inverse Power of Distance (IPD) method was run to account for isolated hard/blastable material located above the hard/soft contact. The estimated probable hardness values were stored in a field called HVAL in the model.

Blasting probability values were assigned to the samples, according to the assigned hardness value. Hard material (below hard/soft contact) was assigned blasting probabilities of 1(100% blast probability) and 0.9; softer material a blasting probability of 0.5, and free dig (soft) material was assigned a blasting probability of zero.

A search volume of 30 m x 30 m x 30 m, with a minimum of 4 and a maximum of 200 samples, was used for the estimation of the blastability probabilities. Where the estimated HVAL exceeded a value of 0.8 in soft Oxides and Sulphides, the rock type was changed to Blast Oxide and Blast Sulphide respectively.

The final material types generated from the surfaces and the hardness probabilities were stored in the ROCKTYPE field of the block model (**Table 15**).

Table 15: *Summary of Material Types*

Rock Type	Description
1	Laterite & Clay
2	Soft Oxide
3	Hard Oxide
4	Soft Sulphide
5	Hard Sulphide
6	Blast Oxide

2.12.2. ALTERATION

A graphite wireframe (**Figure 17**) was created using the interpolation technique in Leapfrog® mining and flagged in the model with a field called ALT.

The modelling was based on the logged alteration codes in the drillhole file. All samples falling inside the wireframe were given a value of ALT=1 and all samples lying outside the wireframe a value of ALT=0.

Graphite alteration can reduce metallurgical recoveries.

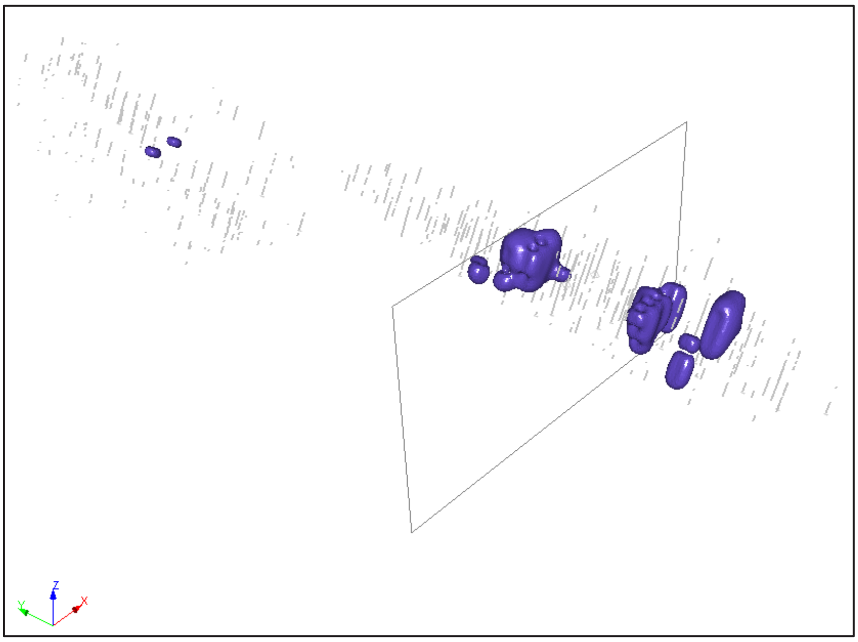


Figure 17: *Extent of Graphite alteration at FE2*

Indicator Kriging (IK) was also used to estimate graphite and was flagged in the model with a field called DOM_GPH. The estimated graphite was visually compared with the field ALT and did not capture the extent of the graphite alteration as well as the graphite wireframe.

The main motivation for using IK is the fact that it is non-parametric. All the samples that contained graphite alteration were given a value equal to 1 and the remaining samples a value equal to 0 i.e. the data undergoes a non-linear transformation to indicator values (0 or 1) based on the presence or absence of graphite alteration. Values that are greater than a particular alteration intensity received these indicator values. The results of IK provides probabilities for the condition i.e. presence or absence of graphite alteration.

2.12.3. MINERALISED ENVELOPES AND MINERALISATION

The drillholes were composited to 2 m intervals (as sampling was at either 1 or 2 m intervals). The composited drill data was used to generate the updated grade envelopes in Leapfrog® software in an attempt to separate out higher grade areas in order to estimate them separately.

In Leapfrog®, the data was transformed to Gaussian space before interpolation; a spheroidal variogram was used and a 0.32 g/t geological threshold grade was selected for the interpolation.

The Leapfrog® User Blog was accessed for guidance on how to optimise interpolant settings. The blog can be accessed at http://blog.leapfrog3d.com/2013/07/26/interpolant-functions-in-leapfrog-geo/.

Interpolant functions indicate how function values are expected to vary as the distance between data points increases. The smaller the distance between points, the more similar the values are expected to be - hence the function values are small. At larger distances between points, the values are expected to vary more hence the function values are larger. This relationship implies that the interpolant function is equivalent to the variogram used in geostatistical modelling.

Leapfrog® uses two base functions to create interpolants i.e. the linear interpolant function and the spheroidal interpolant function.

The linear interpolant function is multi-scale and preferred for general purpose models where the lithology contains only localised areas of high resolution data. In the case of data with distinct finite ranges of influence (such as grade data), the linear interpolant unrealistically extrapolates out from the data instead of falling to zero beyond the range of influence. It is therefore in the case of grade data used only as a quick visual guide for identifying trends in the data.

The spheroidal interpolant function was therefore selected to create the grade envelope based on the finite range spherical variogram used in geostatistical modelling. The spheroidal interpolant function approximates the spherical variogram while forming a smooth interpolant.

The process of choosing a suitable cut-off grade was iterative and aimed at identifying the direction of strongest continuity in the mineralization. Although the manner in which the Ore envelopes were constrained leaned towards being more conservative, a balance between over and under constricting the wireframes was taken to avoid misrepresenting the data.

When the Ore envelopes are unconstrained, large quantities of boundary waste is included. On the other hand, over constraining the Ore envelope may mislead one to think that strong continuity in high-grade mineralisation exists causing an over-estimation of grade and recoverable metal, which has huge consequences for selective mining

The meshes serve as the basis for the geological model which in turn is the foundation of the resource estimate. The interpretations as mentioned earlier, are derived from drillhole sample data which represents often less than 0.001% of the geological body (Stephenson and Vann, 2001). Any errors at this level can dramatically bias grade and tonnage results

Some manual refinement of the envelopes was required to fine-tune the result. This manual adjustment involved using “dummy” high or low grade points to either extend mineralisation where connectivity was less than desired or restrict mineralisation where it was more than desired (or there were unreasonable extensions beyond data support - often termed Leapfrog® “blow outs”).

Low-Grade control points with a default grade of 0.1g/t were inserted to regulate the shape of the interpolant in areas where “blow outs” occurred; irregular unreal shapes were created and where the drillholes ended in the mineralization.

The approach of re-interpreting the mineralised envelopes created using Leapfrog® software’s grade interpolation technique is necessary to avoid allowing the software to make assumptions on the geology without understanding it:

"Most resource modelling procedures are attempts at modelling from sparse data, based on assumptions made by the geologist and mining engineer, and assumptions inherent in the mathematical modelling algorithms used. Assumptions made by the geologist and mining engineer are often well stated, understood and often questioned. Assumptions inherent in the modelling algorithms are very rarely understood or stated and seldom questioned" – Carras, 2001.

The resultant mineralisation interpretation is shown in the image below – it strikes approximately N-S (for 1.1 km) and dips at 50 degrees towards the east (**Figure 18**).

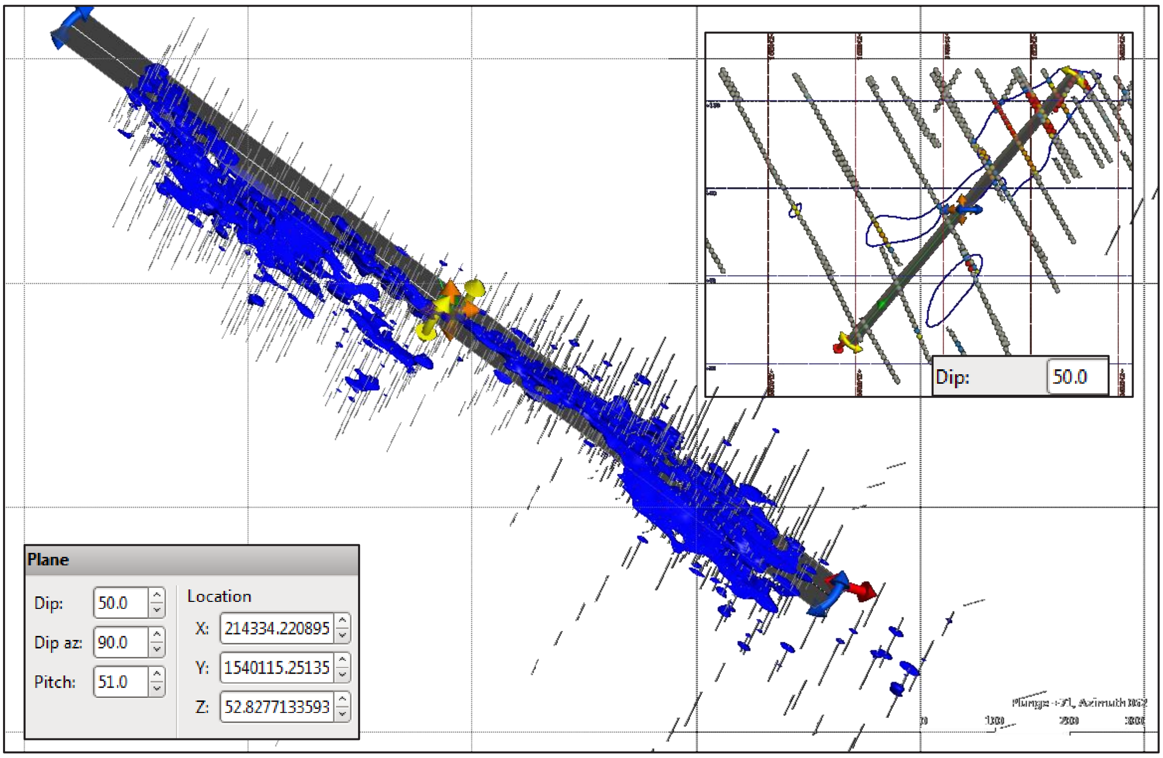


Figure 18: *Isometric and sectional views showing the general trends of the mineralisation at FE2 (shown in blue)*

Near its centre, there is a break in the mineralisation where a dolerite dyke has intruded along a fault which has separated and displaced the mineralisation (**Figure 19**).

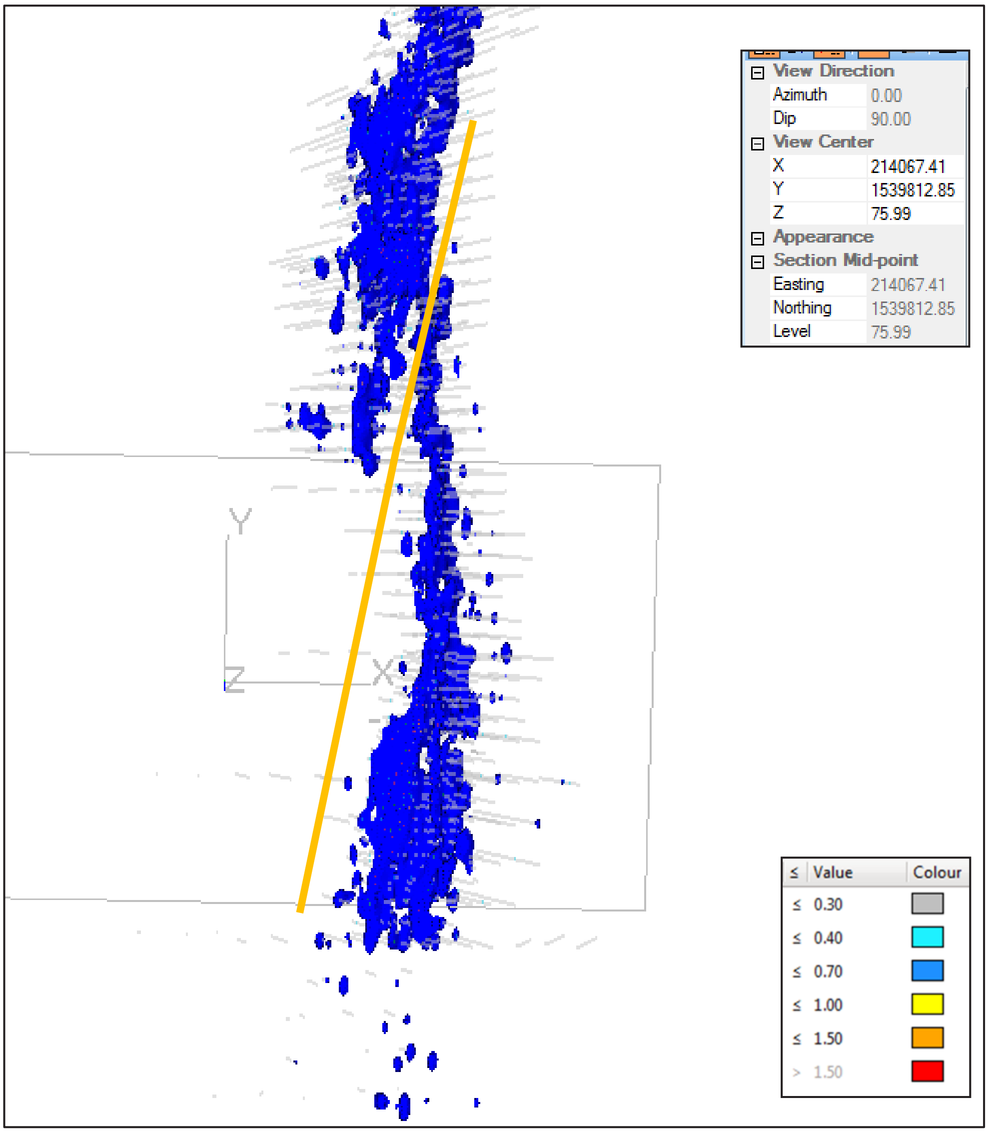


Figure 19: *Isometric view showing the mineralisation envelope (blue) and Dolerite dyke (yellow) with the drillholes coded on Au Grades*

The Leapfrog® parameters used to generate the mineralisation envelope is summarised below (Table 16) and were based on the previously used model parameter and were guided by the previous mesh interpretations.

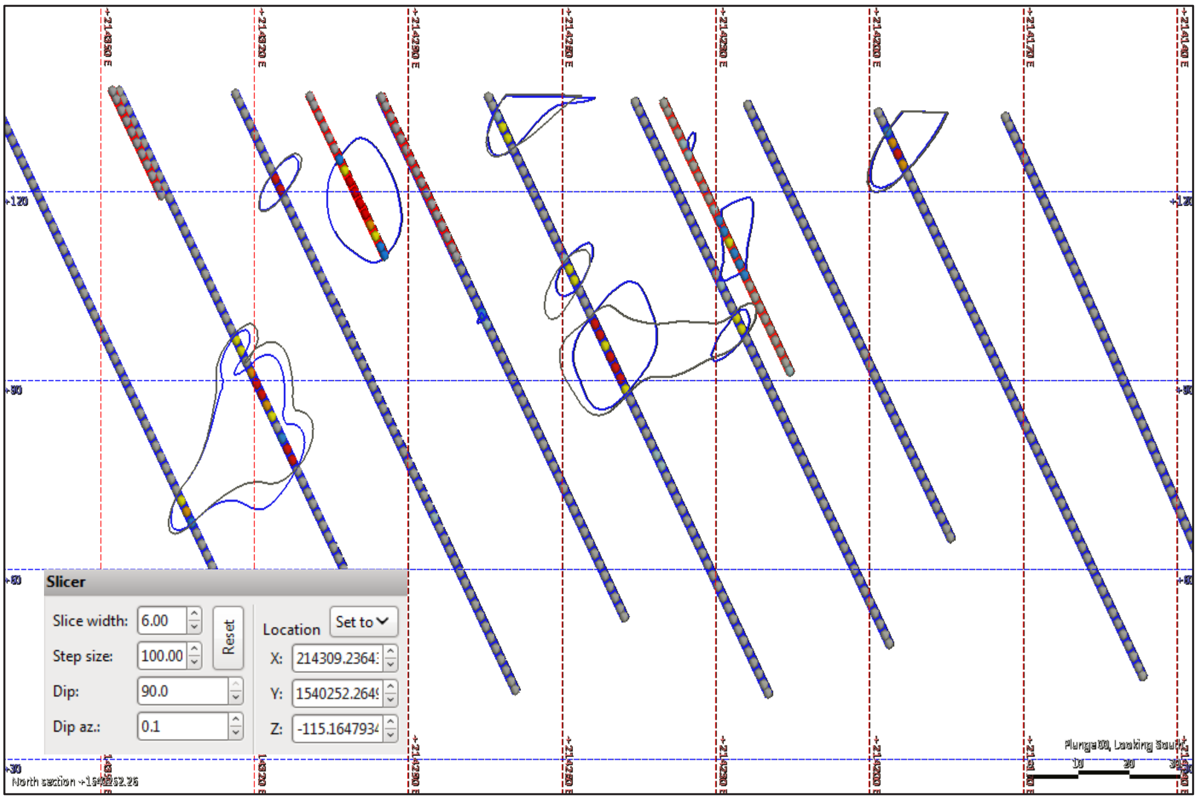
Table 16: *Leapfrog® Interpolation and Surface parameters*

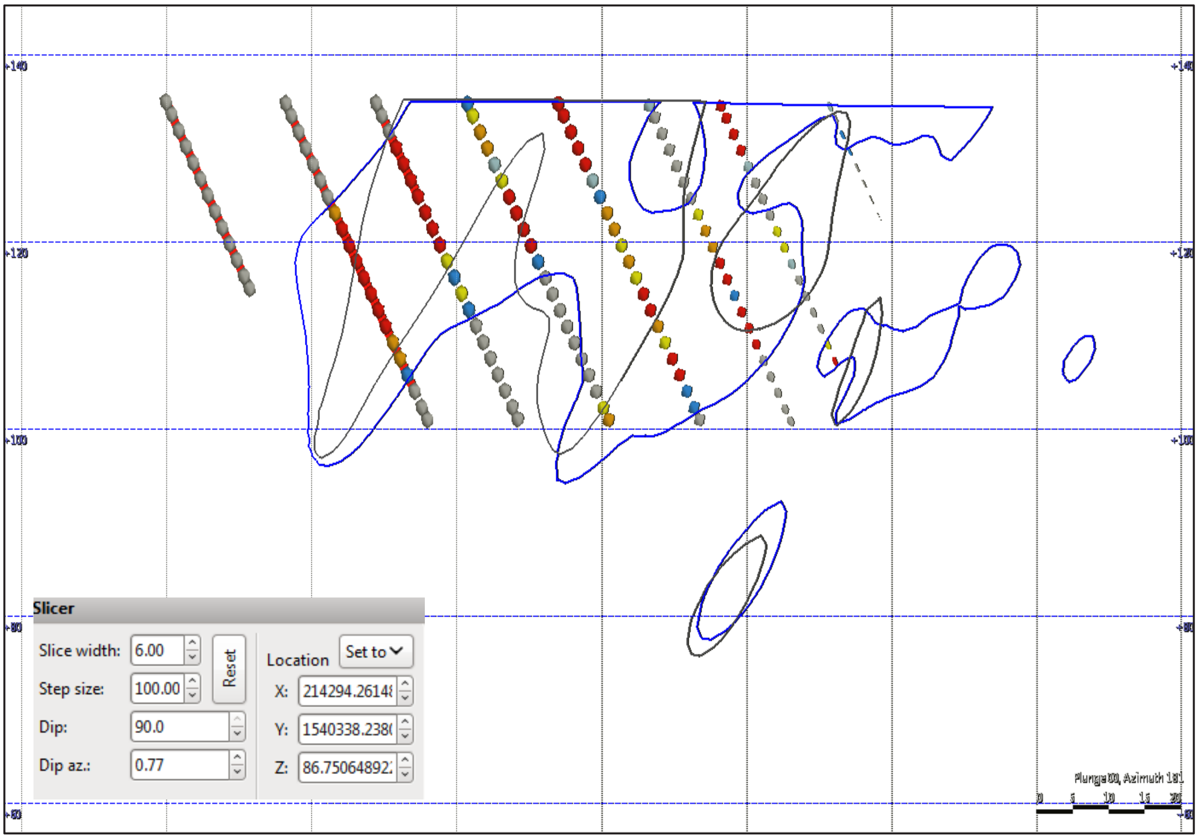
Data Transformation	Gaussian (lower bound 0.0025 g/t and upper bound 10 g/t)
Variogram type	Spheroidal
Sill	0.6
Variogram range	40
Alpha	3
Variogram Nugget	0.0
Drift	Constant (Ordinary Kriging)
Accuracy	0.01
Global or structural trend	Global
Trend Directions	Dip: 51; Dip Azimuth: 90; Pitch: 0
Ellipsoid ratios	Maximum: 3, Intermediate, 2 and Minimum: 1
Threshold	0.32 g/t
Resolution	2m

The previous and updated mineralisation interpretations were carefully assessed and compared well with each other in terms of the geological controls governing the mineralisation.

Adjustments were made to reduce internal waste and better fit the mineralisation trend. These are represented by areas where the old wireframes were narrowed or steepened to support the new drill data.

Figure 20 shows sections where the new interpretations differed significantly from the old interpretations (the new Ore wireframe is shown in blue and the previous one in grey; the drillholes are coloured by grade).


Slicer
Slice width: 6.00
Step size: 100.00
Dip: 90.0
Dip az.: 0.1
Reset
Location
Set to
X: 214309.2364
Y: 1540252.264
Z: -115.1647934
North section +1540252.26
Plunge 00, Looking South


Slicer
Slice width: 6.00
Step size: 100.00
Dip: 90.0
Dip az.: 0.77
Reset
Location
Set to
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Z: 86.75064892
Plunge 00, Azimuth 181

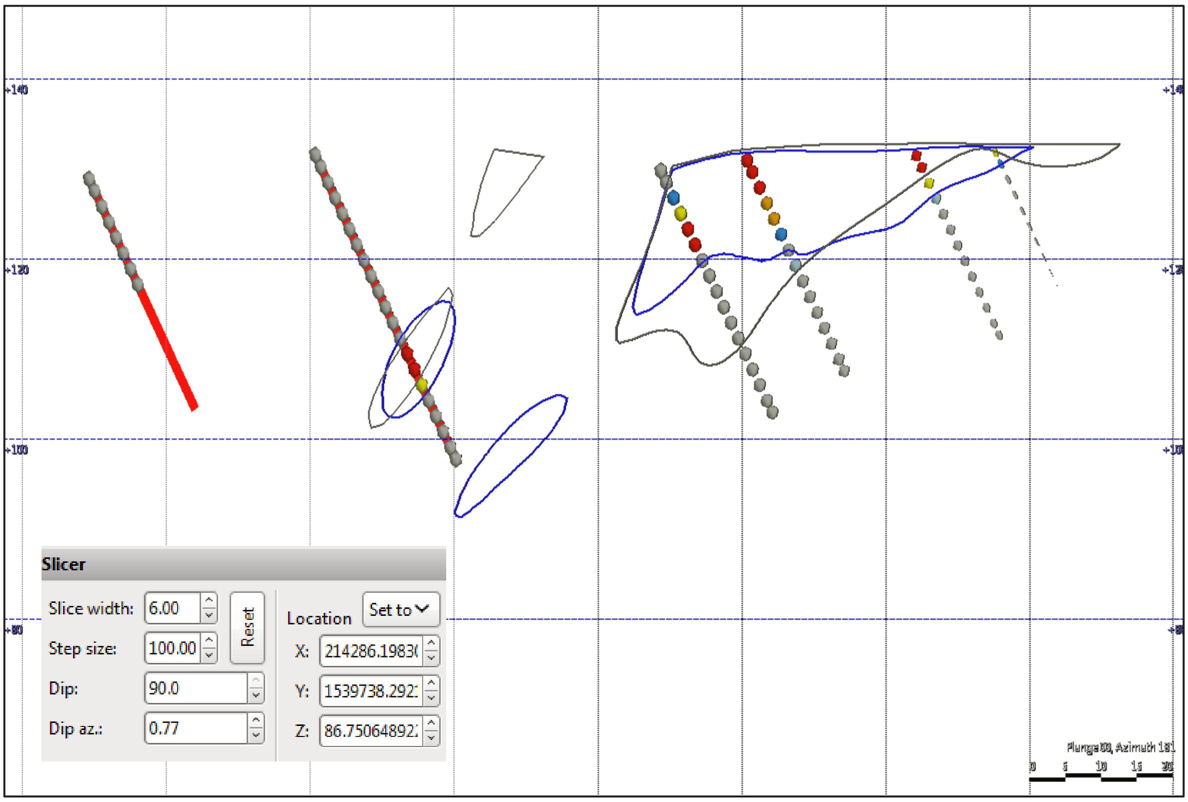

Figure 20: *W-E sections comparing the new (blue) with previous (grey) mineralisation interpretations*

3. EXPLORATORY DATA ANALYSIS

The wireframe interpretations were imported into Datamine® Studio 3 software and used to code the drillhole samples according to mineralisation, lithology, weathering and structure.

Samples within the mineralised envelope were deemed "Ore" and those outside, "waste". Exploratory data analysis of the coded drill sample data is presented in the sections to follow.

3.1. DOMAINING

Domains are defined as zones which are geologically and statistically homogenous (supported by variography and statistical analysis) (Duke and Hanna, 2001). The domains for FE2 were defined using grades in an iterative process of selecting mineralised intersections in each borehole using Leapfrog® Mining Software (**Chapter 4**).

Since grade boundaries are not always clearly distinguishable, boundary analyses were undertaken (**Chapter 8**). An Analysis of Data within the selected domains was also carried out (**Chapter 5 and 7**) to determine grade estimation strategies.

Mineralised domains were identified on the basis of logged samples and grade continuity, and were guided by the previously modelled mineralised wireframes. The spatial limits of these domains are controlled by the 3D mesh (Schofield, 2011) and are examined in boundary analysis section later in the report.

Glacken and Snowden (2001), define domains as areas or volumes within which the characteristics of the mineralization are more similar than outside the domain.

Operational mines have a good geological understanding that is supplemented by a large amount of historical data. This knowledge of the Ore body behaviour guides the modelling and has been fine-tuned over the years using reconciliation results.

As a result, the Domaining exercise was less of an arbitrary process since the new wireframe was guided by the 2014 grade boundaries which were constructed with a sound understanding of the grade continuity and geological controls on the grade distributions.

The domains were constructed to:

- Follow the correct direction of grade continuity
- Prevent smearing of grades across the domain boundary by constraining the width of the envelopes.
- Ensure that the spatial continuity of the high grade data was not exaggerated by ensuring that the domains were not too tight.

Domains should conform to the geology. In this case, the geological units are the same as the mineralisation domains therefore the grade modelling is constrained entirely by the gelogical modelling and the resource model is a reflection of the geology (Glacken and Snowden, 2001).

Domaining assists in reducing the problem of preferential data clustering and its bias on statistical analysis and variography that arises because of the natural tendency to drill or take more samples in higher grade areas causing data to be misrepresented (Glacken and Snowden, 2001).

The domains used for the update are unchanged from the previous model. Each domain represents a volume within which the characteristics of the mineralization are more similar than outside the domain.

The six domains (called KZONES) identified are shown in

Table 17.

Table 17: *KZONES used in the model*

KZONE	Description
1	Laterite Ore
2	Saprolite Ore
3	Hard Ore
4	Laterite Waste
5	Saprolite Waste
6	Hard Waste

The various “Ore” (or mineralised) domains (Laterite, Saprolite and Hard) were compared to determine whether they should be estimated together or separately.

The statistics and distributions of the Hard and Saprolite Ore are compared in **Figure 21** and were found to be fairly similar, but the Saprolite mean Ore grade was higher at 1.65 g/t than the hard Ore mean grade of 1.29 g/t.

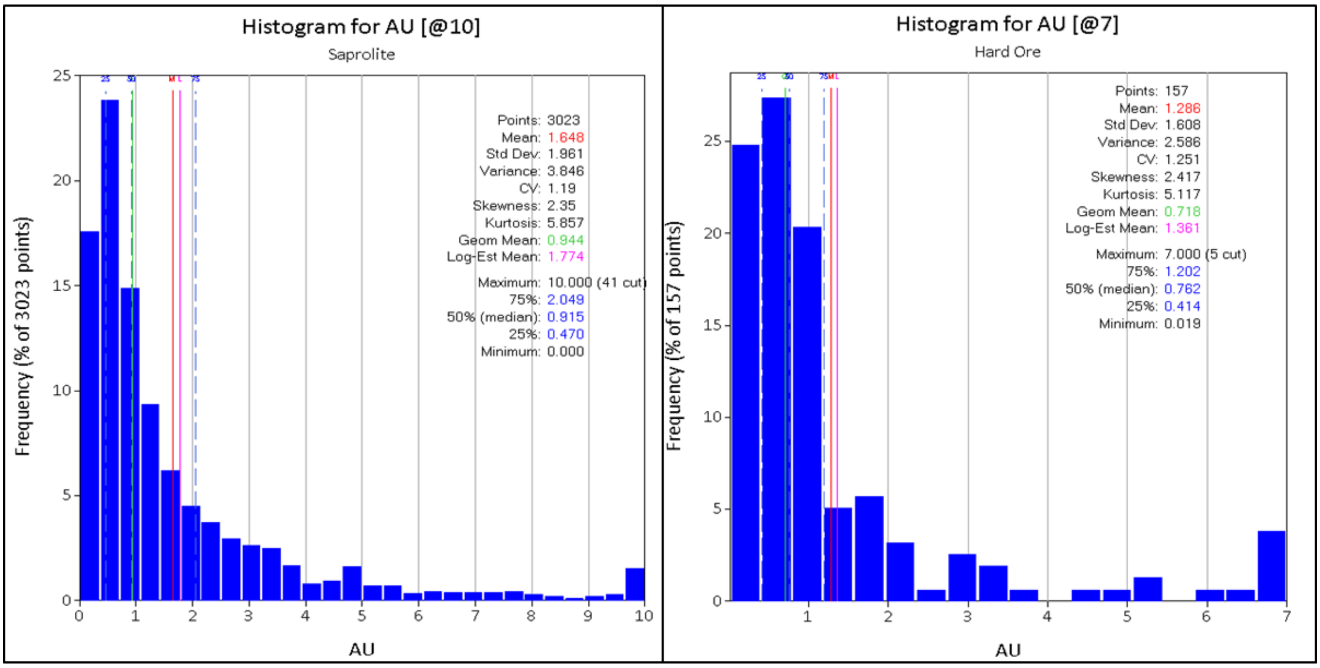

Figure 21: *Histograms for gold grades for Saprolite and Hard Ore*

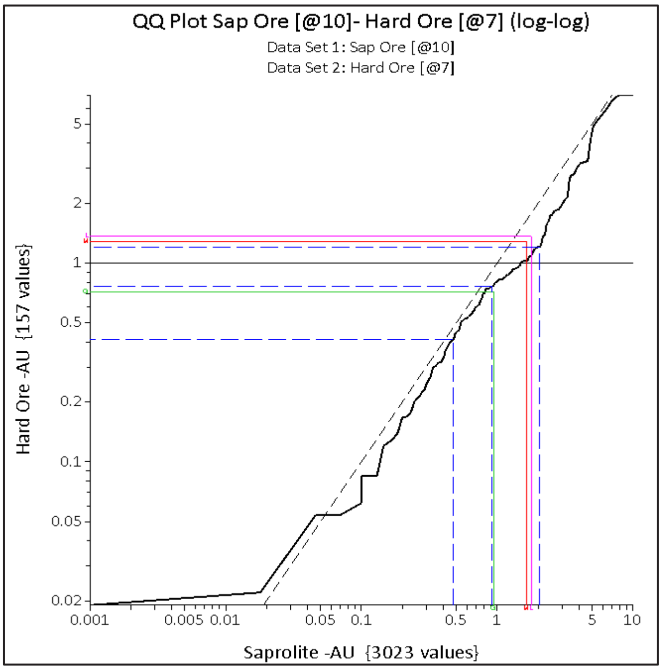

Figure 22: *QQ plot for Saprolite vs. Hard Ore*

In addition to the above, the grade variation along the hard-soft contact was investigated (**Figure 23)**.

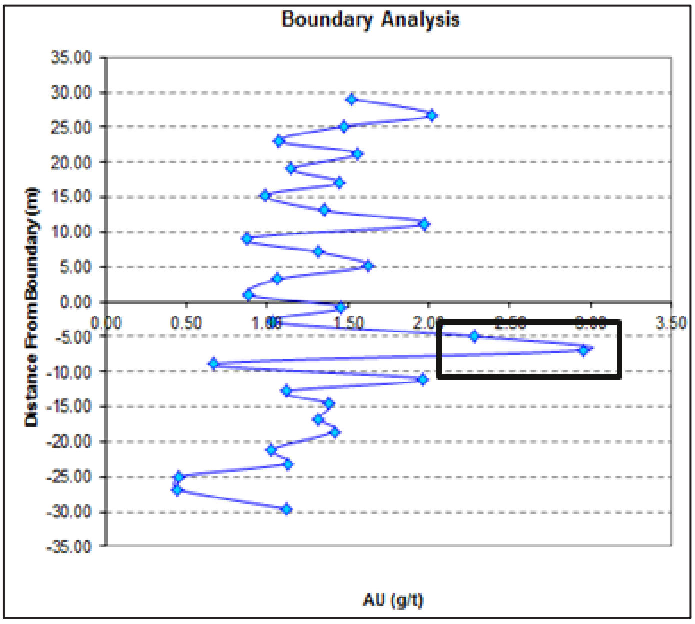

Figure 23: *Boundary Analysis across hard (below)-soft (above) contact*

The result showed slight grade enrichment immediately below the hard-soft contact and that the grade tended to decrease gradually with depth.

As a result, it was decided that the Saprolite and Hard Ore zones be estimated separately, applying a hard boundary between them for estimation.

Comparison of the Laterite and Saprolite mineralisation grades (**Figure 24**) showed comparable means of 1.59 g/t for Laterite and 1.65 g/t for Saprolite.

The QQ-plot also showed that, even though the Laterite dataset comprised of very few samples, the two datasets had similar grade distributions (except at the lower end of the distribution below about 0.5 g/t). The Laterite and Saprolite zones were combined during estimation.

Further work is required on boundary analysis going forward as in reality the Laterite and Saprolite are very different, despite the results of the statistics suggesting that they are similar. It needs to be investigated whether the Laterite was transported or not and if the mineralised zones match. These zones can therefore be combined for variography but not for estimation in future.

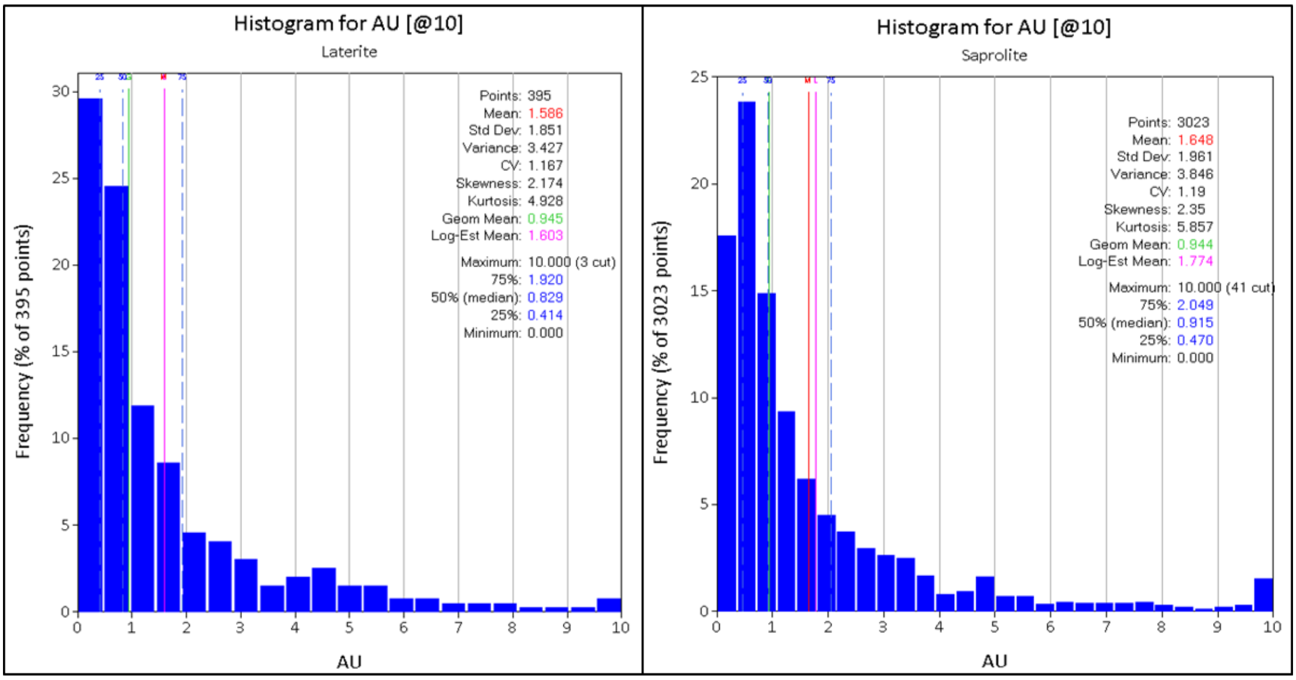

Figure 24: *Histograms for gold grades for Saprolite and Laterite Ore*

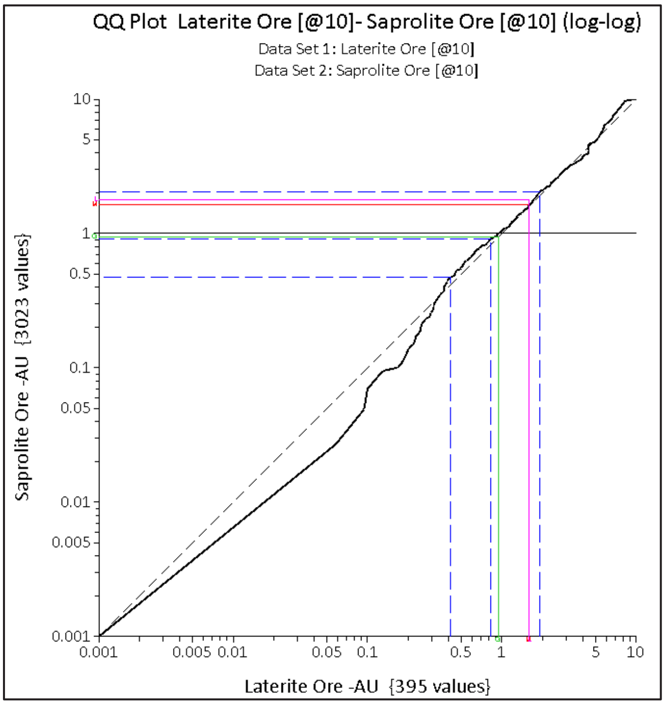

Figure 25: *QQ plot for Saprolite vs. Laterite Ore*

The estimation domain codes were stored in the EZONE field of the drillhole and model files (**Table 18**).

Any block falling inside the Ore envelope was assigned an EZONE value of 1 for Laterite and Saprolite Ore and a value of 2 for Hard Ore. Those blocks that lie outside the Ore envelope were assigned an EZONE value 3.

Table 18: *Estimation Domains used in the model*

EZONE	Description
1	Laterite and Saprolite mineralisation
2	Hard mineralisation
3	All waste (areas outside of mineralisation envelope)

3.2. GEOLOGICAL CUT-OFF GRADE

Cut-off grade theory for Mineral Resource estimates is based on economic viability.

The choice is subjective during exploration phase when there is insufficient data to adequately assess economics. In this case, the decision is generally based on the practical experience to meet the requirements stipulated by the JORC code that the estimates have 'reasonable prospects for eventual economic extraction' (Stephenson and Vann, 2001).

For geological modelling purposes, grade envelopes were created in Leapfrog® software in an attempt to separate out higher grade areas in an iterative process. This threshold grade used for modelling purposes, is referred to as a geological cut-off grade and is not calculated like an economic cut-off which is used by Accountants. A 0.32 g/t Au threshold grade was selected - several grade envelopes were modelled at various geological cut-offs, of which the 0.32g/t Au threshold showed the best grade continuity.

3.3. STATIONARITY

Geostatistics relies on the assumption of stationarity. Based on this concept, data can be combined into domains. A swath plot is a graphical display of the grade distribution from a series of bands/swaths that are generated in several directions through the deposit.

Swath plots assist in identifying if data is correctly domained. Swath plots were generated using the Bloy® Geostats Kit (GSK) to identify any trends in the Au grade data (**Figure 26**).

The GSK software was developed by Bloy® Resource Evaluation and is a customised geostatistics application used by AngloGold Ashanti.

The plots were generated along strike (Northing) at 25m intervals and along elevation at 10m intervals based on the block model cell dimensions. In the soft (Laterite and Saprolite) Ore zone (EZONE 1), the grade gradually decreased with depth and most of the other plots showed more abrupt changes in poorer informed areas. None of the domains were further separated into sub-domains based on the results of the trend analysis

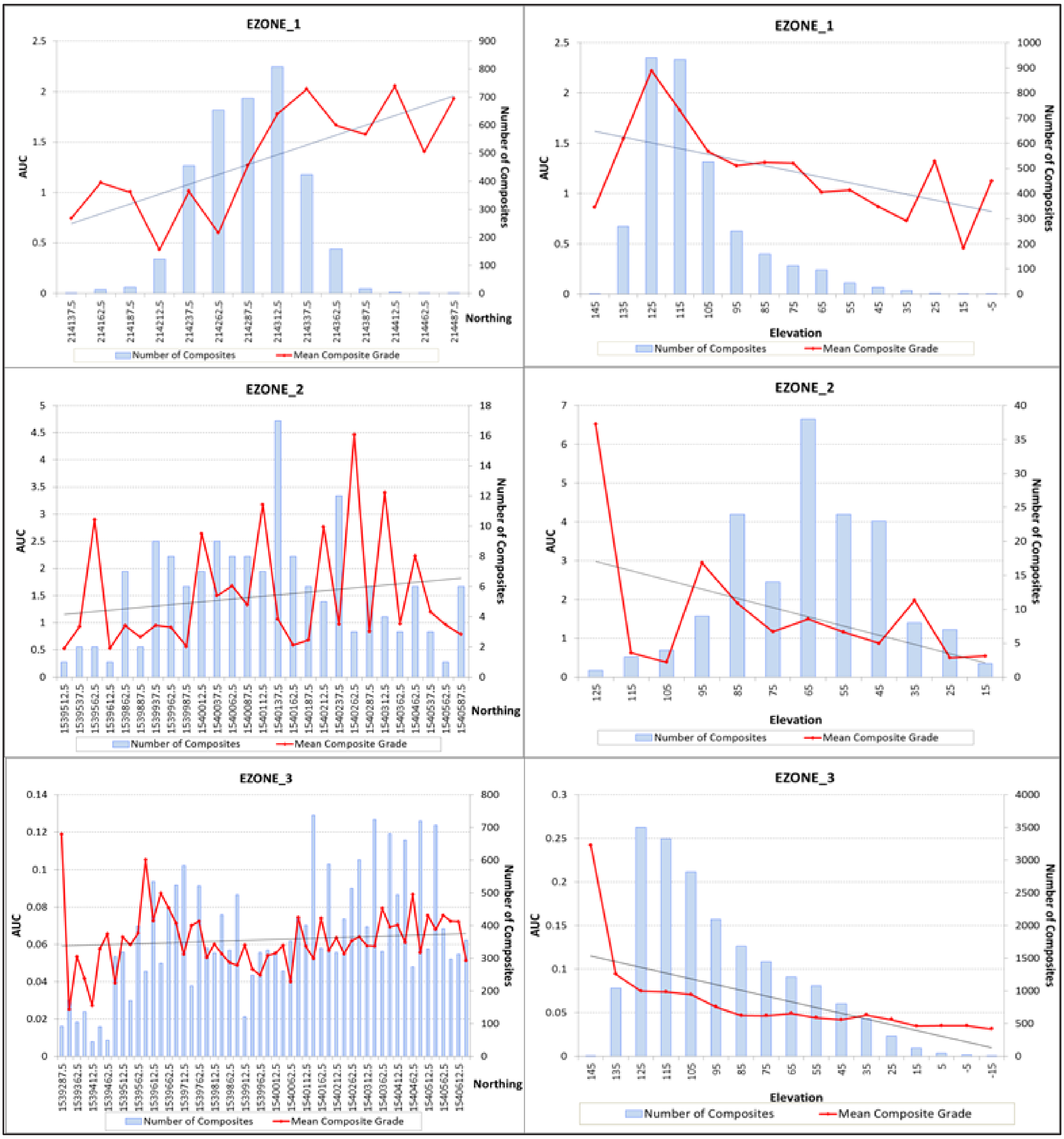

Figure 26: *Swath plots generated using the Bloy® Geostats Kit to identify any trends in Au grades in the data*

3.4. COMPOSITE STATISTICS

Statistical analysis takes place after the data has been composited and declustered.

The summary statistics for the input data (**Table 19** and **Table 20)** and accompanying log histograms (**Figure 27**) are presented below.

Table 19: *Summary statistics for composited and uncomposited data*

	Uncomposited		Composited	
	Length	AU	Length	AU
Total Number of Records	219071	219071	23873	23873
Number of Samples	219071	217728	23873	23650
Number of Missing Values	0	1343	0	223
Number of Values> Trace	219071	217728	23873	23650
Minimum	0.01	0.003	1.00	0.00
Maximum	4.5	71.22	3.00	54.86
Range	4.49	71.22	2.00	54.86
Total	47737.71	134514.4	47706.83	7445.76
Mean	0.22	0.62	2.00	0.31
Variance	0.30	3.62	0.01	1.43
Standard Deviation	0.55	1.90	0.08	1.19
Standard Error	0.001	0.004	0.00	0.01
Skewness	2.94	13.86	-0.32	13.49
Kurtosis	6.69	388.28	40.27	343.27
Geometric Mean	0.07	0.18	2.00	0.04

The differences in the uncomposited and composited statistics were the result of the small grade control sample lengths due to density probe measurements being taken at small intervals. These small lengths destroy the original sample assay length intervals during the Datamine® desurvey process when run with other interval tables. The results of the compositing was however considered satisfactory and was later validated (results presented later in chapter).

Table 20: *Summary statistics – gold grade by estimation domain*

EZONE	Number of samples	Min	Max	Mean	CV
1	3418	0.00	54.86	1.74	1.53
2	157	0.02	16.50	1.45	1.61
3	20293	0.00	1.66	0.06	1.41

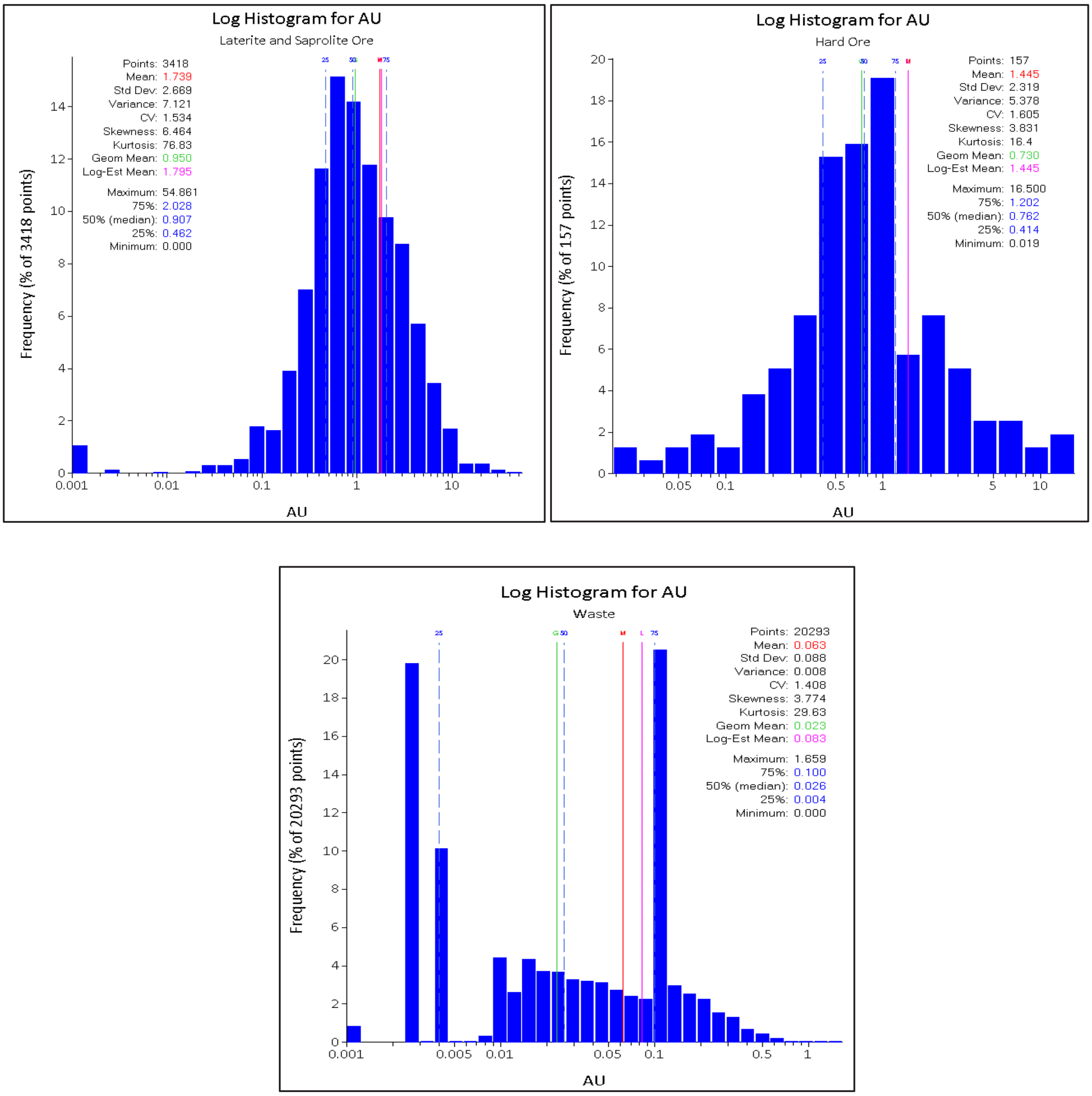

Figure 27: *Log Histograms for estimation zones*

The summary statistics are used to detect if any trends are evident in the data; plot data distributions; depiction of any trends evident within the domains; assist in selecting interpolation technique; define subsets within data; highlight outliers and establish if a relationship between variables exists (Glacken and Snowden, 2001) .

The coefficients of variation for the domains are very high (1.53 and 1.61) which is typical for gold grades which are naturally erratic and have a high nugget.

The samples follow a positively skewed, mesokurtic distribution with a long tail of high grades (Average > Median > Mode).

Top capping of these outliers was applied for estimation to minimise the effects of smearing high grades – This is described later in the report.

The results of the composite validation is presented below. There were no significant differences in total sample lengths and metal accumulations before and after compositing the data.

The results are based on the actual assay table and therefore honoured the original assay sample lengths.

Table 21: *Summary of results for Composite Validation*

	Raw Data		**Composited Data**		**Difference** (Raw minus Comp)	
KZONE	Length (m)	Accumulation m.g/t	Length (m)	Accumulation m.g/t	Length (%)	Accumulation (%)
1	720.66	1192.1691	714.16	1190.2804	1%	0%
2	6029.5	10530.1203	6024.47	10527.3363	0%	0%
3	335.77	464.4251	335.77	464.4251	0%	0%
4	3131.65	287.9299	3126.05	286.8903	0%	0%
5	32314.28	2227.4059	32307.5	2225.8684	0%	0%
6	4759.14	191.4497	4757.97	191.1268	0%	0%

4. VARIOGRAPHY

It is crucial to understand and quantify domains during Exploratory Data Analysis (**Chapter 4**) since Domaining is the basis for variography.

Variography in turn, is the first step in geostatistical analyses and thus fundamental to the success of the estimation process. This chapter is informed by Blackwell and Sinclair, 2001, pp. 181-191).

Variograms define the continuity of mineralization i.e. the spatial variability relationship between samples:

Variogram	**=**	**<u>sum of (sample value - sample value at vector *h* away)</u> ^{2}**
		2(number of sample pairs collected for vector *h*)

The semi-variogram graph (**Figure 28**) shows how the degree of similarity between sample grades relates to the distance between them along any given orientation.

The value at which the graph levels off equals the population variance whilst the distance at which it levels off is called the range of influence.

The grades of samples separated by distances greater than the range of influence are uncorrelated with respect to both their spatial separation and orientation i.e. they are random.

The nugget effect describes the inherent variability plus sampling variability at zero separation distance.

These parameters together allow for the anisotropy of the mineralization to be modelled mathematically.

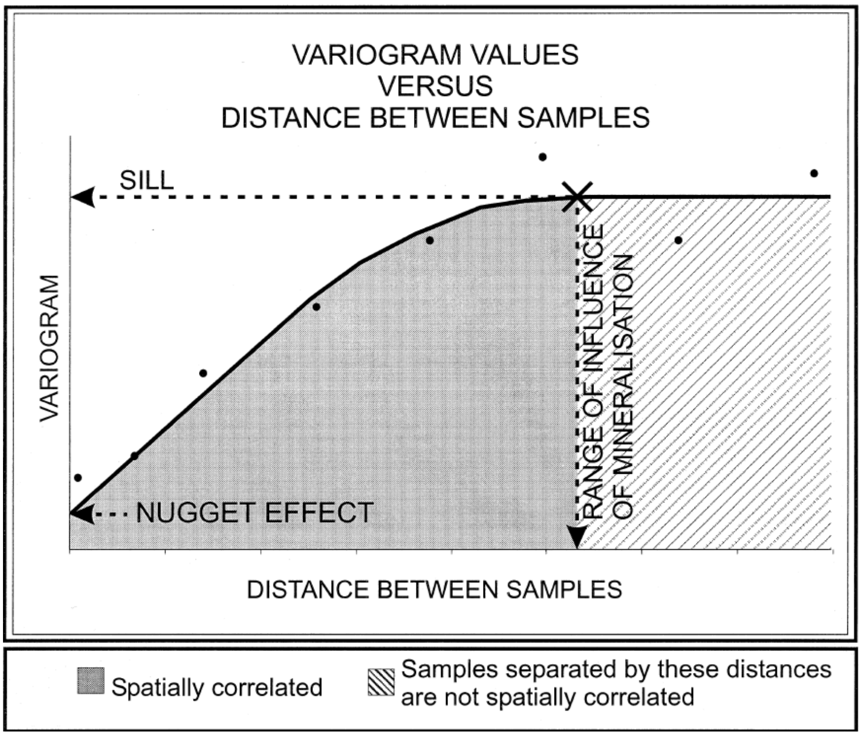

Figure 28: *Experimental semi-variogram and spherical model* (Snowden, 2001, pp 646)

Each dataset in a population represents one of an infinite possibility of subset sample grades for the domain populations (one realization of the random function) (Blackwell and Sinclair, 2001, pp. 183-184).

Since variograms measure the similarity of data (grade continuity), they are representative of the structural character of the underlying random function of the domain grades.

The assumption is that any measurement of data occurring within a domain is representative of that entire domain (stationarity). For gold deposits, where grade dissimilarities are a real part of the data, stationarity is independent of scale and only dependent on the location (X, Y and Z coordinate) of the sample pairs inside the field of the regionalized variable.

An insufficient amount of data was available to obtain a robust variogram model for the Hard Ore domain. The Soft Ore domain parameters (EZONE 1 - Laterite and Saprolite) was therefore used for the Hard Ore (EZONE 2) estimation.

Top and bottom cuts were applied for variogram calculations to improve the resulting variogram structures (see **Table 22**).

Table 22: *Top cuts applied for variography*

EZONE	Top Cut (g/t)	Bottom Cut (g/t)	Before Cutting		After Cutting	
			Mean (g/t)	CV	Mean (g/t)	CV
1	16	0.110	1.74	1.53	1.67	1.16
3	2	0.011	0.06	1.41	0.09	1.02

The statistics and shapes of histograms for Au grades for each domain before and after top cuts were applied, is shown in **Figure 29**.

The histograms are more regular after grades are cut at the portion of the histogram that 'breaks down'.

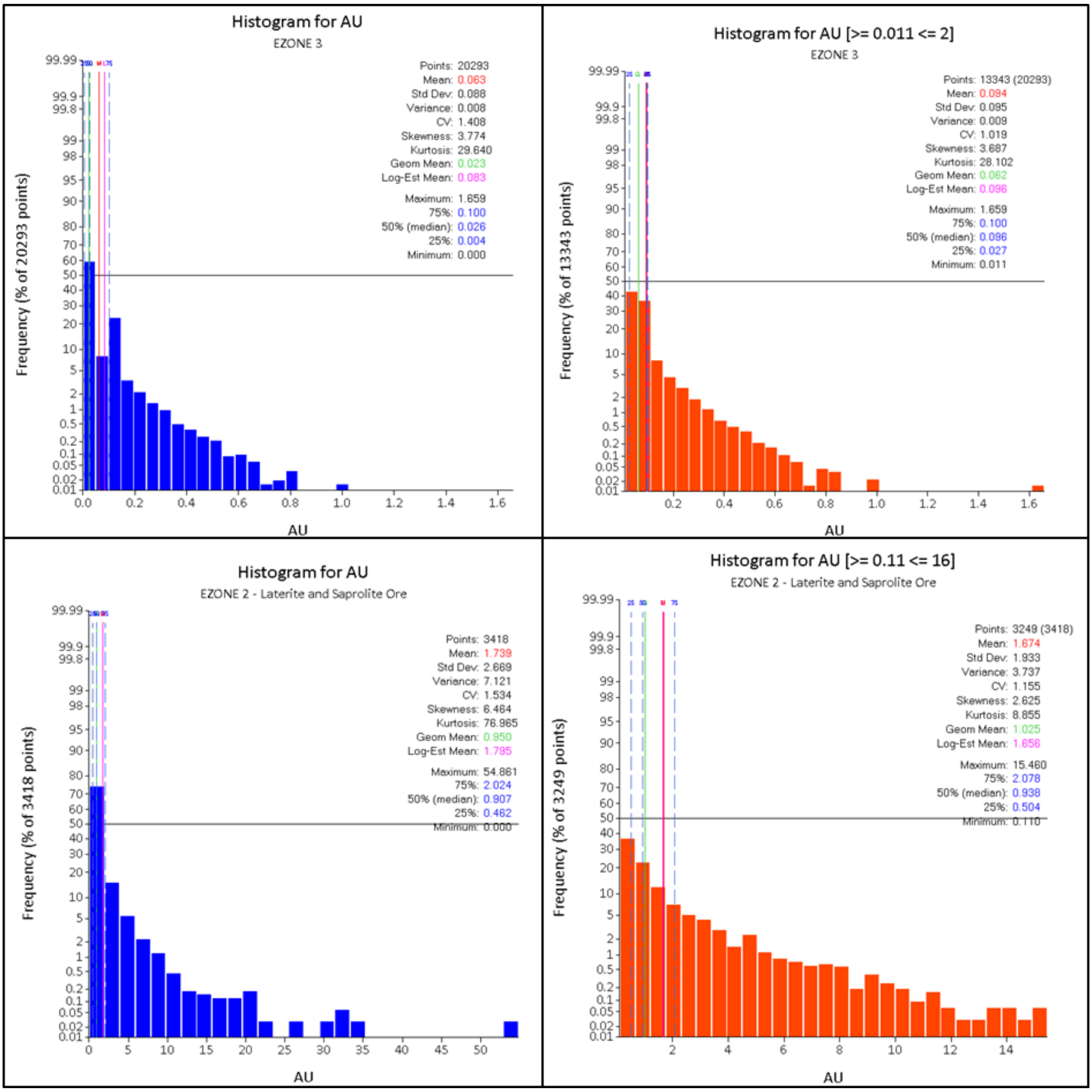

Figure 29: *Histograms for EZONES 1 and 2 (cut and uncut)*

Supervisor® (v8) geostatistical software was used to calculate and model the variograms. Supervisor® was the preferred software.

The choice of software used is dependent on the comfort that the user has with a particular product as well as the user's experience. Supervisor® is user friendly (different directions modelled together) and allows for the interactive viewing of continuity fans and direction definition; interactive lag adjustment to test model suitability and 3D visuals to assess spatial continuity.

The directions of continuity were evaluated by making use of variogram contours (**Figure 30**) on the horizontal, across-strike and dip planes to determine continuity along strike, down dip and across plunge.

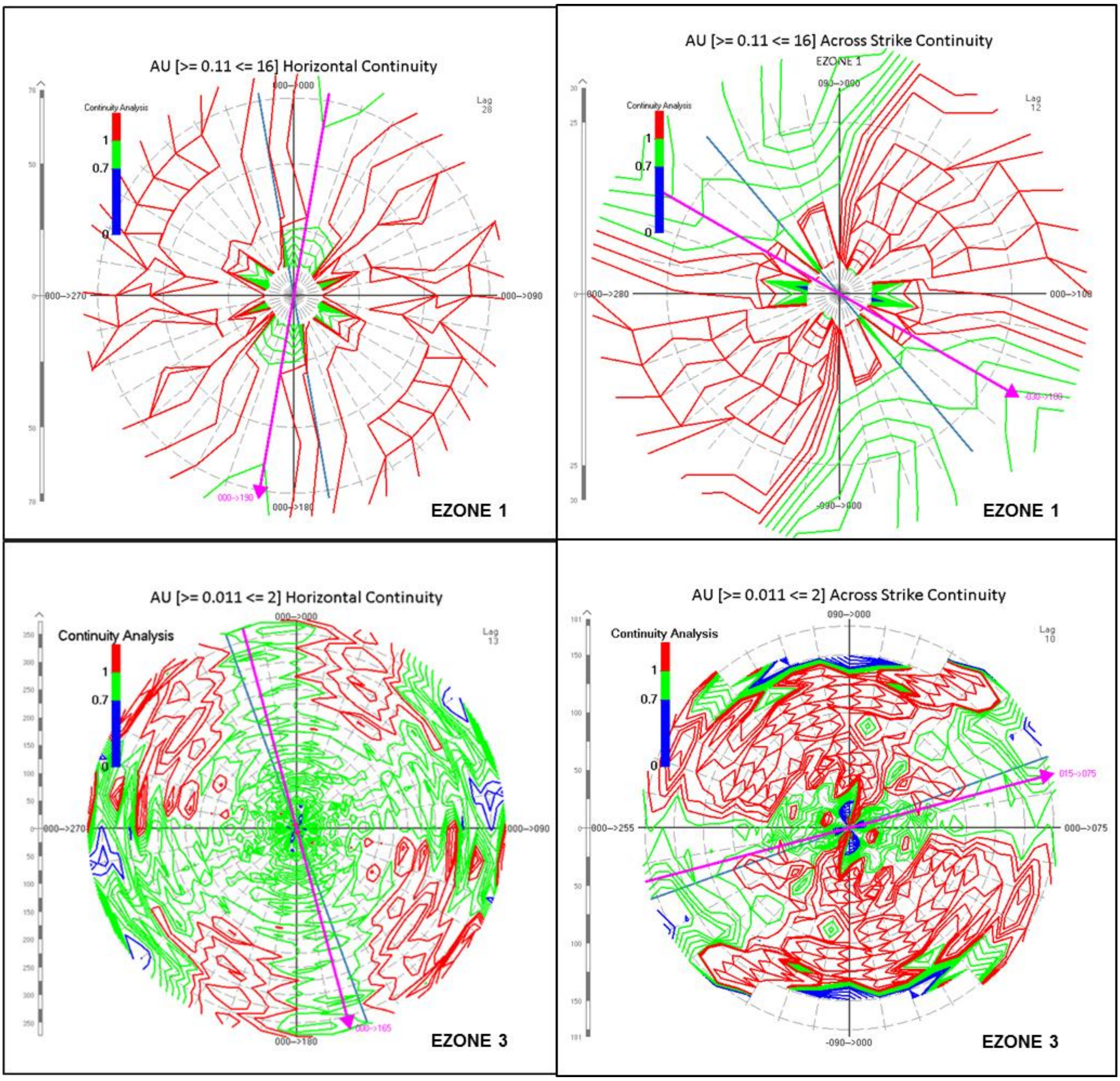

Figure 30: *Var-map continuity fans for EZONES 1 and 3 generated in Supervisor®*

The red arrows on the Var-map continuity fans indicate the direction of strongest continuity as represented by the data. The continuity is represented by the contours on the plot which are tight together.

The resultant calculated experimental variograms were modelled using spherical models with one or two structures. Experimental variograms are estimates of the 'underlying' variogram and some irregularity is generally expected (Guibal, 2001).

The variograms were standardised (rescaled to the original population variance). The nugget effect was determined by extrapolating the first two points of the downhole variogram to the Y-axis to ensure that the shortest ranges were captured.

The main variogram parameters that were tested and optimized are described in **Figure 31**and **Table 23.** Each parameter was modified independently to maximise the number of samples found.

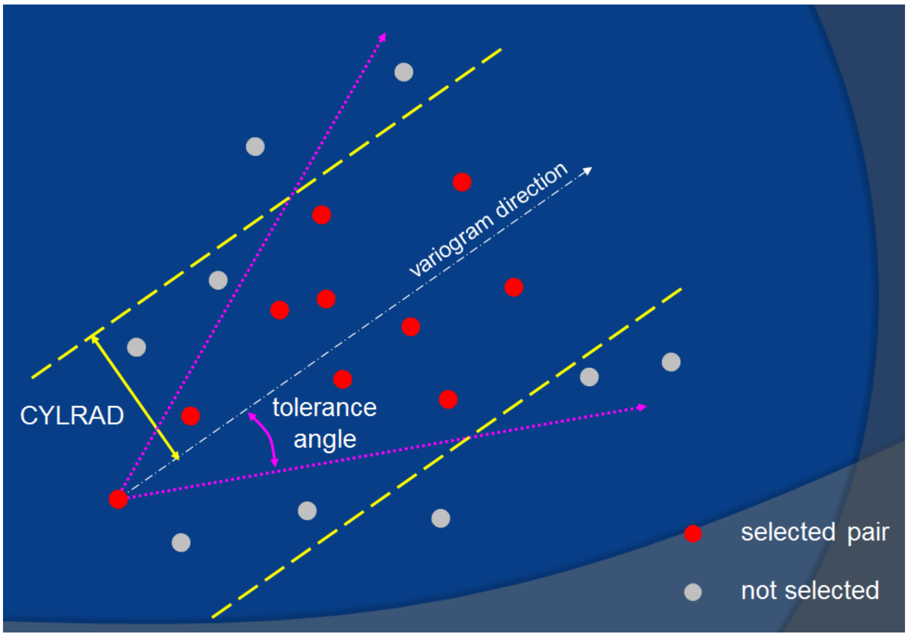

Figure 31: *Main variogram parameters* (Crisp, 2012)

Table 23: *Main variogram parameters*

EZONE	1 and 2				3			
Parameter	**Downhole**	**Along Strike**	**Across Strike**	**Down Dip**	**Downhole**	**Along Strike**	**Across Strike**	**Down Dip**
Direction		**1**	**3**	**2**		**1**	**3**	**2**
Lag distance (m)	2	17	2	21	2	43	19	18
Bandwidth (m)	-	-	-	-	-	-	40	-
Angular tolerance	30	12	30	35	10	10	30	10

4.1. DOWNHOLE VARIOGRAM MODELS

Downhole variograms are defined by the shortest distance (spacing) between samples and is therefore used to quantify the proportion of natural random variability in the data. The first two points of the downhole variogram are extrapolated to the Y-axis since these points have the least variance. The nuggets were applied to all directional variograms belonging to a particular domain since the nugget effect is constant in all directions.

Downhole variograms used to determine the nugget effect are shown in **Figure 32**.

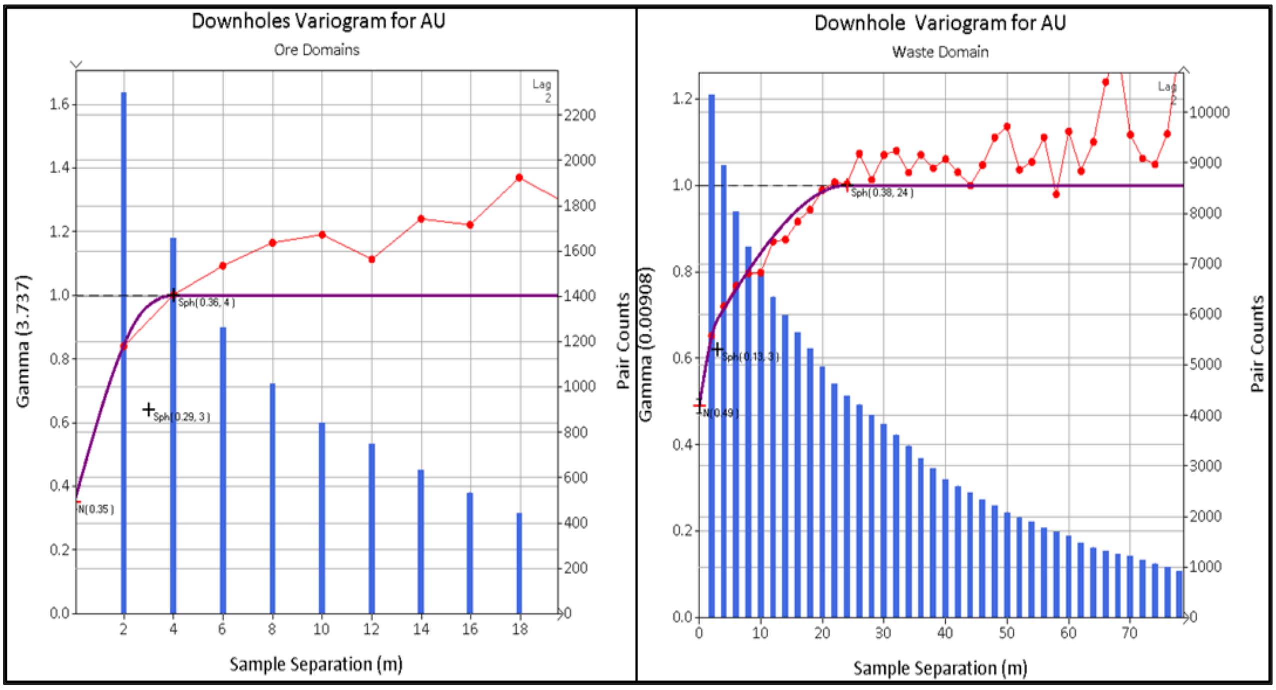

Figure 32: *Downhole Variograms for Ore and Waste Domains. The bar charts represent the number of pairs of samples.*

4.2. DIRECTIONAL VARIOGRAM MODELS

Directional variograms were calculated to identify any changes in grade behaviour i.e. anisotropy in the different directions. The Ore and Waste (not estimated, but contained grade therefore variograms constructed for completeness of work) domains produced anisotropic nested variogram models (**Figure 33**). For both domains the resultant variograms had the same sill in all directions but different ranges, referred to as Geometric anisotropy.

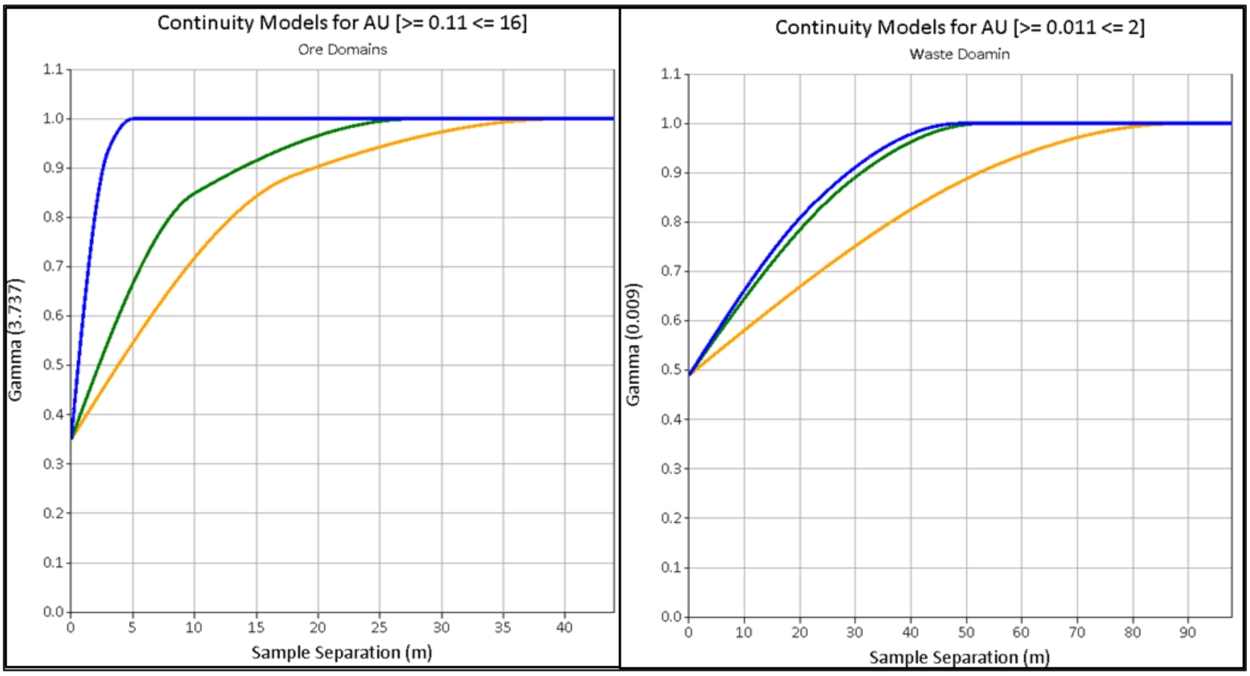

Figure 33: *Geometric anisotropy*

Various lags ranging from 17-43m were used along and across strike and 2m for downhole (composite length) were used. The angular tolerance applied also varied between 12 – 30 degrees for the along and across strike variograms, and between 10-35 degrees for down dip.

The variogram parameters for the estimation zones are described in **Table 24.**

Table 24: Variography parameters

EZONE	C0	C1	C2	Direction	Range X	Range Y	Range Z
1 and 2 (Ore Domains)	0.35	0.34	0.31	00 → 190	18	40	-
				-30 → 100	10	28	-
				60 → 100	3	5	-
3 (Waste Domain)	0.49	0.06	0.45	05 → 146	57	89	-
				14 → 054	29	53	-

				75 → 255	23	49	-

*Standardised to a sill of 1

The shortest direction variograms (along the mineralisation thickness) were typically the poorest (least data pairs along this direction for variogram calculation) and the downhole direction variograms the best (the most data pairs along this direction).

The Datamine® ZXZ angle rotations for across strike (120 degrees) for the variograms were as follows:

Table 25: *Rotation angle conversion*

EZONE	Direction	Angles in Supervisor® (XYZ)	Angles in Datamine® (ZXZ)
1 and 2	1	00 → 190	100
	2	-30 → 100	30
	3	60 → 100	0
3	1	05 → 146	-105
	2	14 → 054	15
	3	75 → 255	160

To test the variogram orientations in Datamine®, a 3D ellipsoid was constructed for the Ore domains.

Supervisor® converts angles from XYZ format to Datamine® ZXZ format. The converted angles were used to construct the ellipsoid.

The ellipsoid shows how the continuity of mineralization varies with direction using the lengths and direction of its three orthogonal axes (**Figure 34**) and was modified to better represent the orientation of the mineralisation **(Figure 35**) since the converted angles may not always honour the orientation of the mineralisation perfectly.

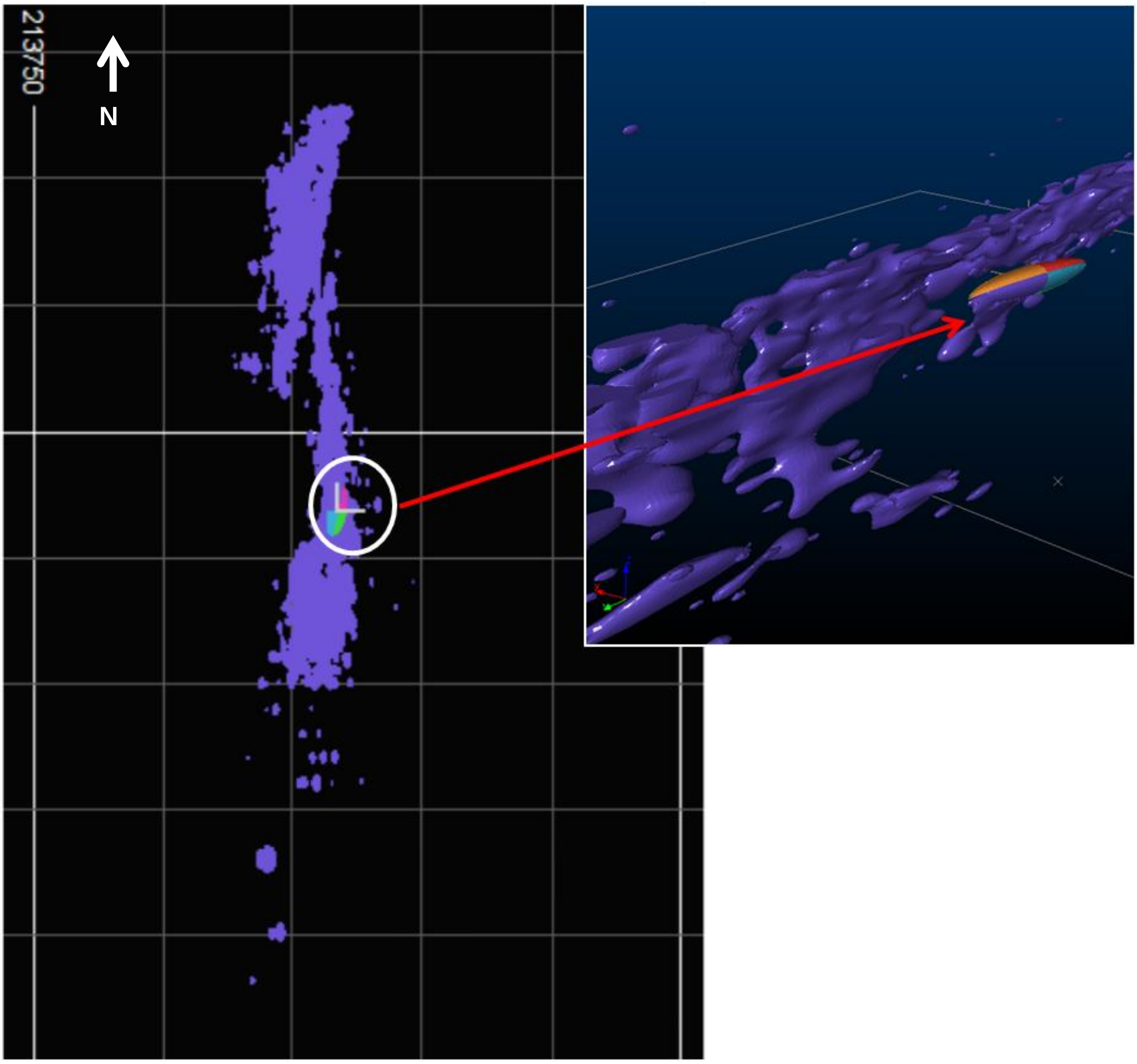

Figure 34: *Isometric views Ore domain ellipsoid against Ore wireframe*

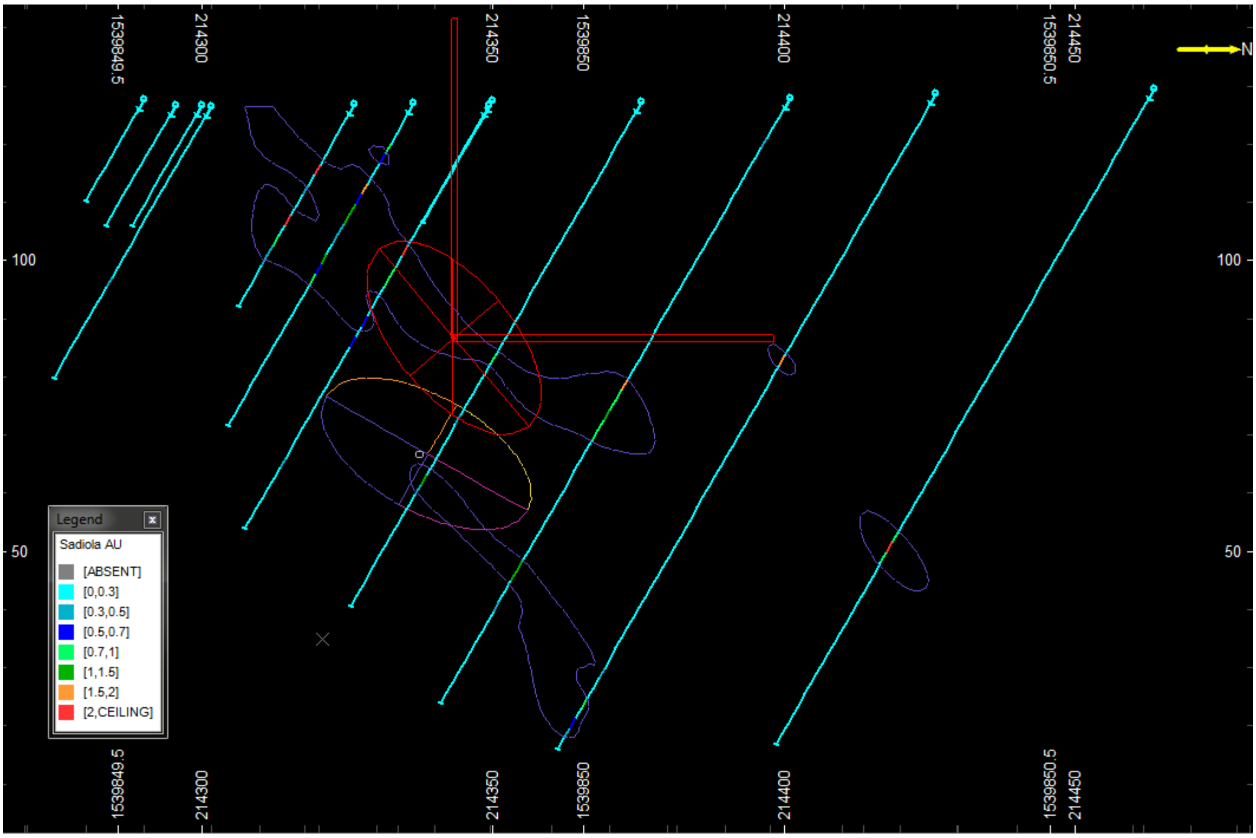

Figure 35: *Orthogonal view of Ore wireframe and orientation of Ore domain ellipsoid based on variogram angles and adjusted Ore domain ellipsoid (in red)*

4.3. DISCUSSION OF RESULTS

The modelled variograms for the Ore (**Figure 36**) and waste (**Figure 37**) domains follow.

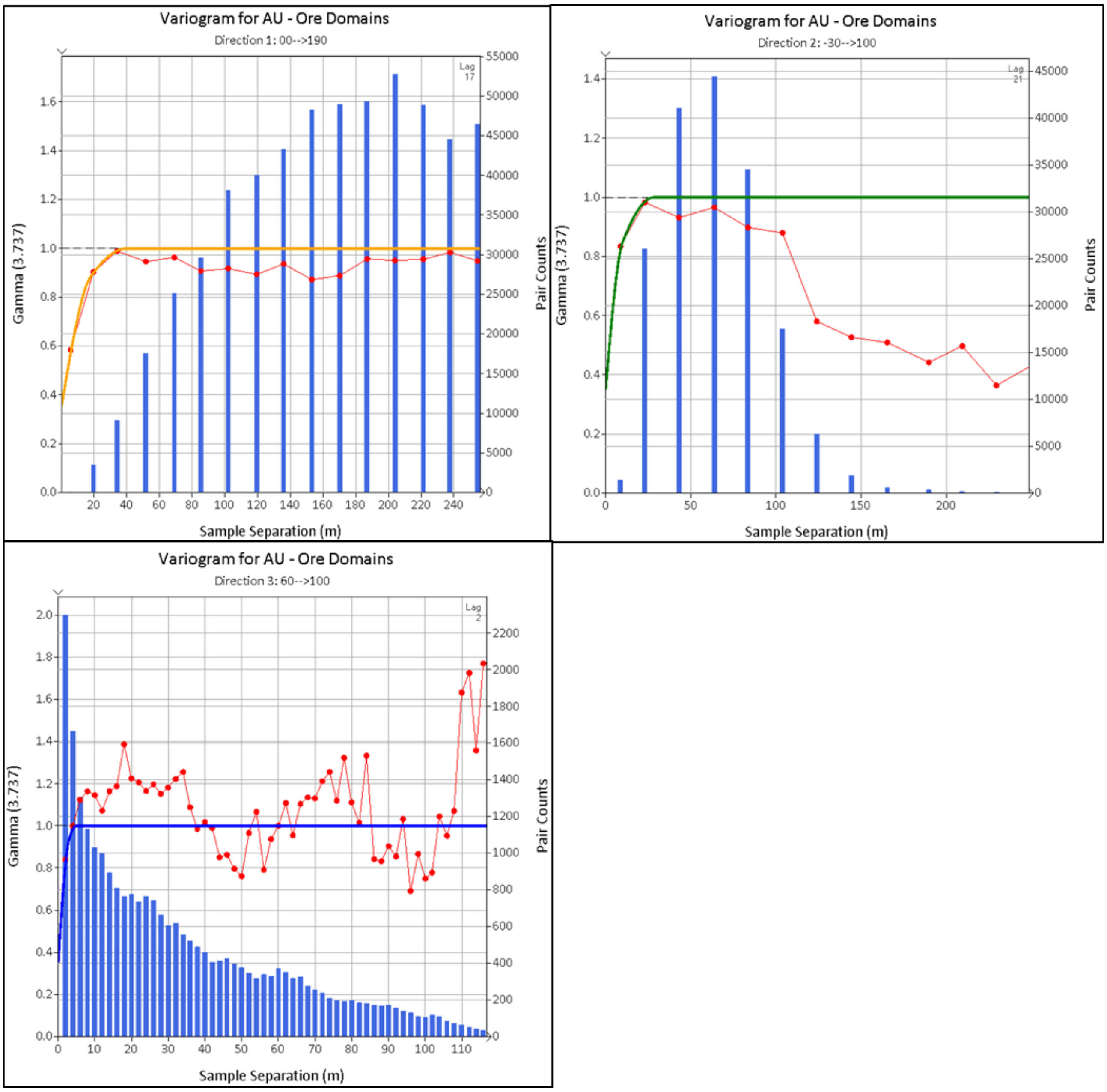

Figure 36: *Directional variograms and fitted variogram models for Ore domain. The bar charts represent the number of pairs of samples.*

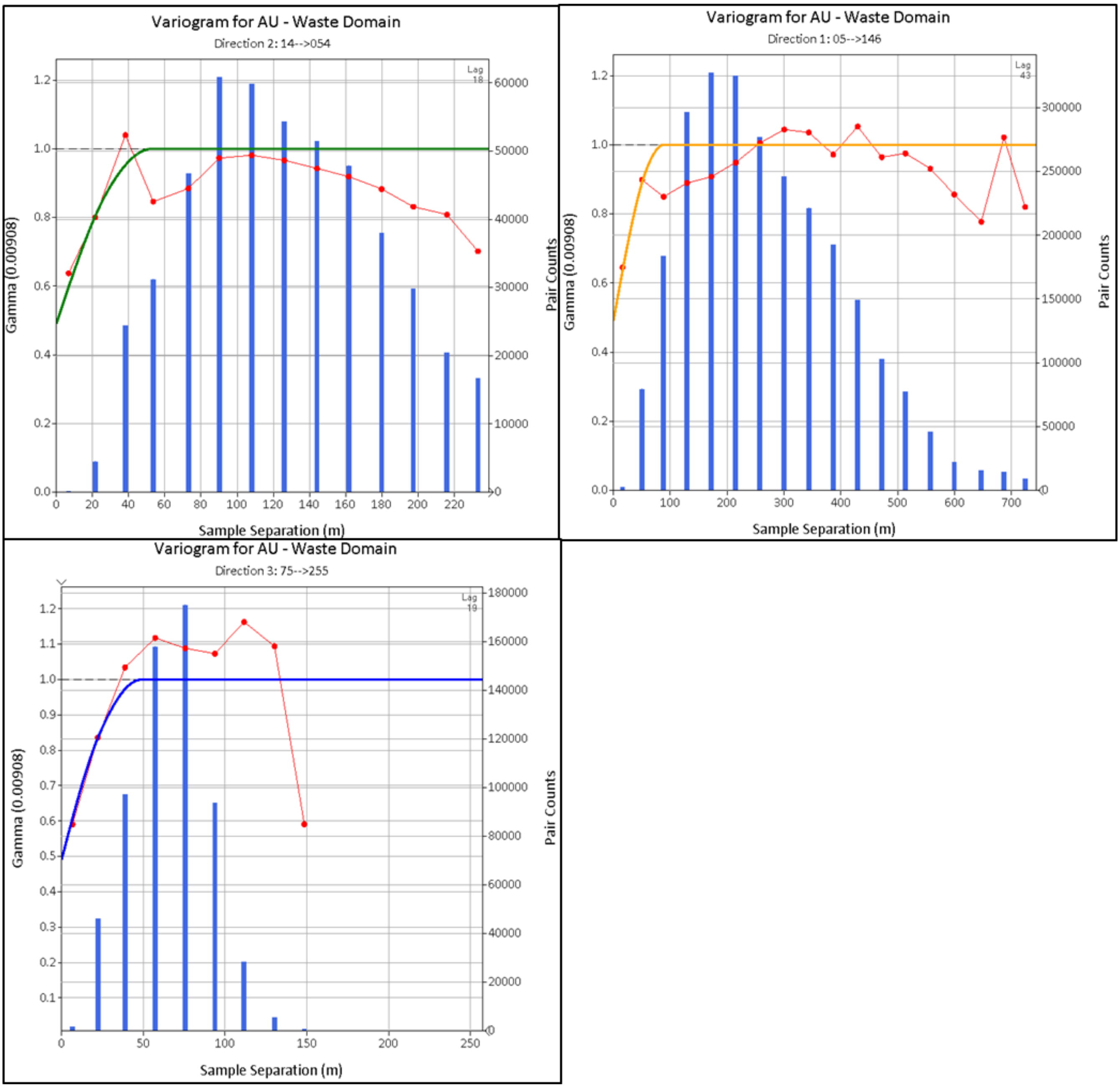

Figure 37: *Directional variograms and fitted variogram models for waste domain. The bar charts represent the number of pairs of samples.*

4.3.1. LAG DISTANCES AND NUMBER OF SAMPLING PAIRS

The first point on the experimental variograms generally have a low number of sampling pairs which may often not considered to be representative of the data. In the case of the waste domain, where it was difficult fitting a model through the first point, the number of pairs at this point was compared to the size of the population and found to represent approximately 200 pairs out of 13,343 samples i.e. 3% of the population. The model fit, though not perfect was therefore deemed adequate.

The distal lags generally show a decreasing number of pairs. In cases where the opposite occurred (at lags beyond the maximum sample separation distance) the variograms become erratic.

4.3.2. SMOOTHNESS OF THE EXPERIMENTAL VARIOGRAM

The saw toothed pattern in the experimental variograms may be an indication of poor lag selection or the result of excluding extreme values (16g/t and 2g/t grade cut offs were applied for Ore and waste domains respectively). Cut offs are applied to those grades occurring in the portion of the histogram of grades where "break-down" is observed. Other methods include cutting off data that falls within the 98th percentile or where erratic changes are observed on a cumulative size frequency distribution curve.

Top caps (setting all grades above 16g/t to 16g/t) were not applied due to the variability of samples (with same grade) reducing to zero, resulting in a reduction of the total population variance and inaccurate nugget values.

It was still possible to identify structures for EZONES 1 and 3, and fit models to the experimental variograms. Irregularities could also have resulted from internal waste being included in the modelling of the Ore envelopes.

4.3.3. THE SHAPE NEAR THE ORIGIN

The shape of the variograms near the origin (fit through the first two variogram points) show a linear character and were therefore fit with a spherical model.

4.3.4. NUGGET EFFECT

The proportion of the nugget effects for the domains is presented in **Table 24** expressed relative to a sill normalised to 1. As a percentage, the nugget values were 35% and 49% for Ore and waste domains respectively.

The Composite lengths were verified to ensure that the nugget effects were representative of the data variability. The density of data and closeness of drill spacing also ensured that the nugget picked up any small scale features.

4.3.5. SILLS AND RANGES

The variogram ranges were assessed visually to identify where the sills stabilise. Where it was not easily identifiable, the stationarity plots were reassessed to check for evidence of trends in the data. The fit of the variogram models were good with regular shapes at short lags, and therefore accepted.

4.3.6. DRIFT

At lags greater than the maximum drillhole spacing (25m) the variograms generally tend to became more unstable. To test if any trends were being picked up scatter plots were generated (**Figure 38**):

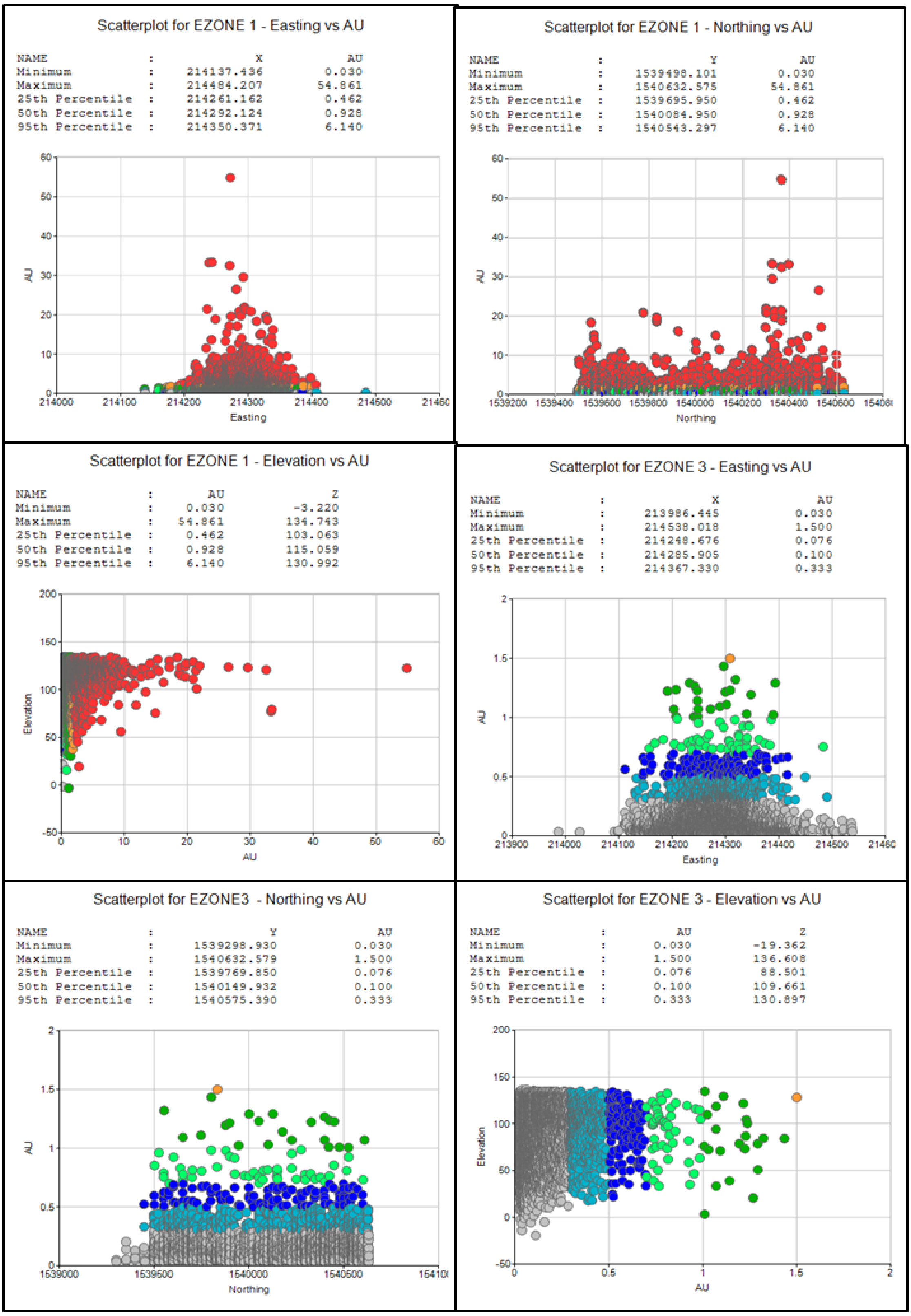

Figure 38: *Scatter plots for Gold grades for EZONES 1 and 2*

In the Soft (Laterite and Saprolite) Ore zone (EZONE 1), the grade gradually decreased with depth. There are no trends visible along Easting for both EZONES. Along Northing, EZONE 1 shows a slight increase in grades.

During the validation process, it is assessed whether or not such trends can be explained. It is recommended that more work be covered in this area to deal with such trends in the data.

4.3.7. HOLE EFFECT

The zigzag pattern or ‘bumps’ in the variograms may also suggest ‘holes’ in the covariance and are generally due to artefacts of sampling. In this case, where data was scarce, the ‘holes’ may have resulted from the lack of pairs. Geologically, the Ore wireframe could contain stringers of waste (internal waste) and this occurrence may also cause ‘holes’.

The Domaining applied to the data was grade-based (0.32g/t cut off). In gold deposits, low grade cut offs give poorly structured variograms due to the mixing of external waste and Ore as well as internal waste. At higher cut-offs clearer structures can be observed. The Domaining could therefore be applied at different cut-offs and then variography applied (Guibal, 2001).

5. QUANTITATIVE KRIGING NEIGHBOURHOOD ANALYSIS

Kriging is a 'minimum variance estimator' only if the search neighbourhood is properly defined.

The objective of Quantitative Kriging Neighbourhood Analysis (QKNA) is to determine what optimum combination of search neighbourhood and block size results in conditional un-biasedness during Kriging, as defined by the user (Bertoli et al., 2003).

This chapter is based on the work by Bertoli et al. (2003).

5.1. CONDITIONAL BIAS

Conditional unbiasedness means that "blocks estimated to have a certain grade Z^*_v will, on average, have that grade". Kriging provides the linear regression solution to the grade interpolation problem of conditional bias.

In practice, there is no perfect correlation between true and estimated grades. The regression line will either be flatter or steeper than the slope of Y (true grade) =X (estimated grade) and conditional bias will be present i.e. the estimated grades will on average either be overestimated (estimates higher than mean grade) or underestimated (estimates lower than the mean grade).

Figure 39 describes this relationship.

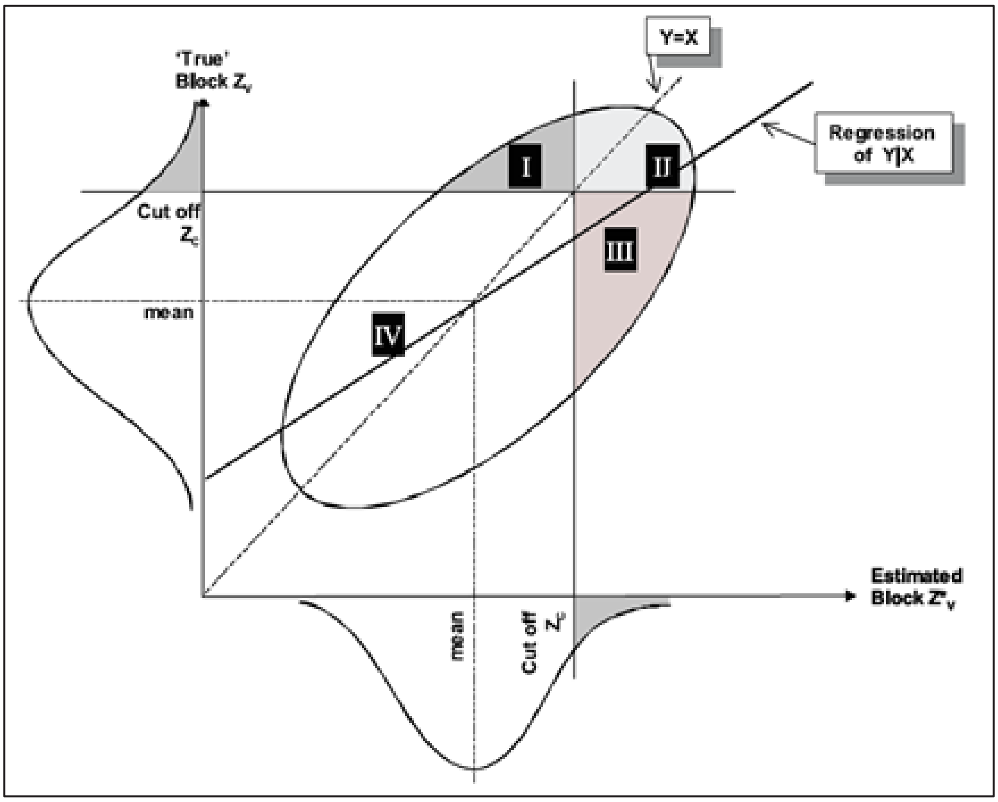

Figure 39: *Conditional Bias due to the information effects using linear regression. Quadrants II and IV are correctly classified as Ore or waste but quadrants I and II incorrectly classified as Ore or waste* (Bertoli et al., 2003, pp. 2)

The true block grades are never known but the relationship between the true block and estimated values are inferred based on the assumption that the variogram models are representative of the domains (stationarity) and that a linear regression can define the relationship between true and estimated grades at the specified support – blocks in this case.

During QKNA, the neighbourhood is optimized to ensure the best regression statistics in order to reduce or eliminate conditional bias.

The process involves smoothing because the data set is exhaustive (information effect) and the variance of the estimated block values will be lower than that for the true block values. QKNA assists in deciding how much smoothing is needed for conditional unbiasedness.

To prevent any conditional biases that are the consequence of using search neighbourhoods which are either too restrictive or unconstrained, the Kriging neighbourhoods are assessed through quantitative tests to check if a particular search neighbourhood is suitable for Kriging. This analysis includes:

- Calculating the slope of regression of the 'true' block grade on the 'estimated' grade
- Assessing the weights of means for a simple Kriging
- Assessing the Kriging weights i.e. proportion of positive versus negative weights
- Testing the Kriging variance

The results of QKNA inform the selection of model block sizes, maximum and minimum sample numbers and the dimensions of the search neighbourhood to be defined.

During the QKNA exercise, the neighbourhood was optimized to ensure the best regression statistics in order to reduce or eliminate conditional bias.

The Kriging efficiency was compared against the block variance. If the Kriging variance is low compared to the block variance, the degree of smoothing is minimised and the grade tonnage relationship is best reflected.

The current QKNA was done using the Bloy® Geostats kit (BGK), which involved optimisation based on a simulated drilling grid which approximates the current drill spacing (12.5m x 12.5m grade control holes) of the data informing the estimate (**Figure 40** and **Figure 41**).

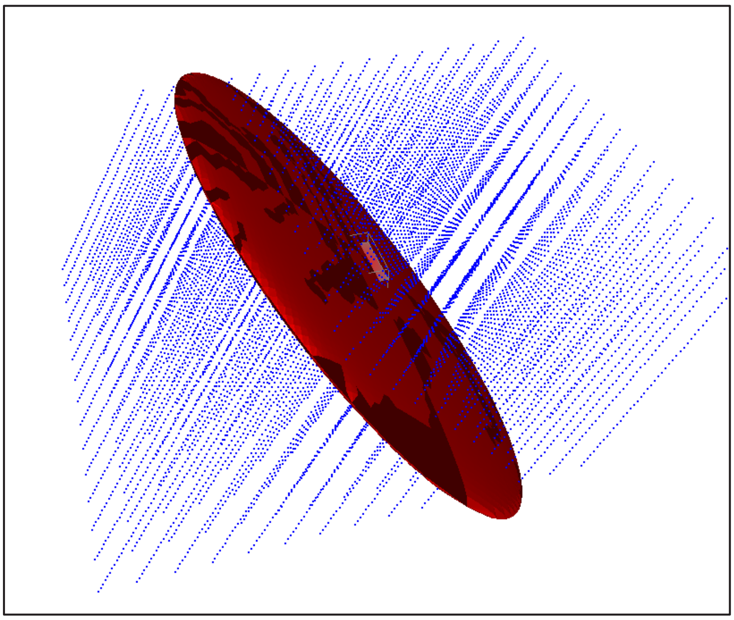

Figure 40: *Simulated drilling grid that approximates the closest drill spacing (12.5m x 12.5m grade control holes)*

The BGK interface is shown in the figure below:

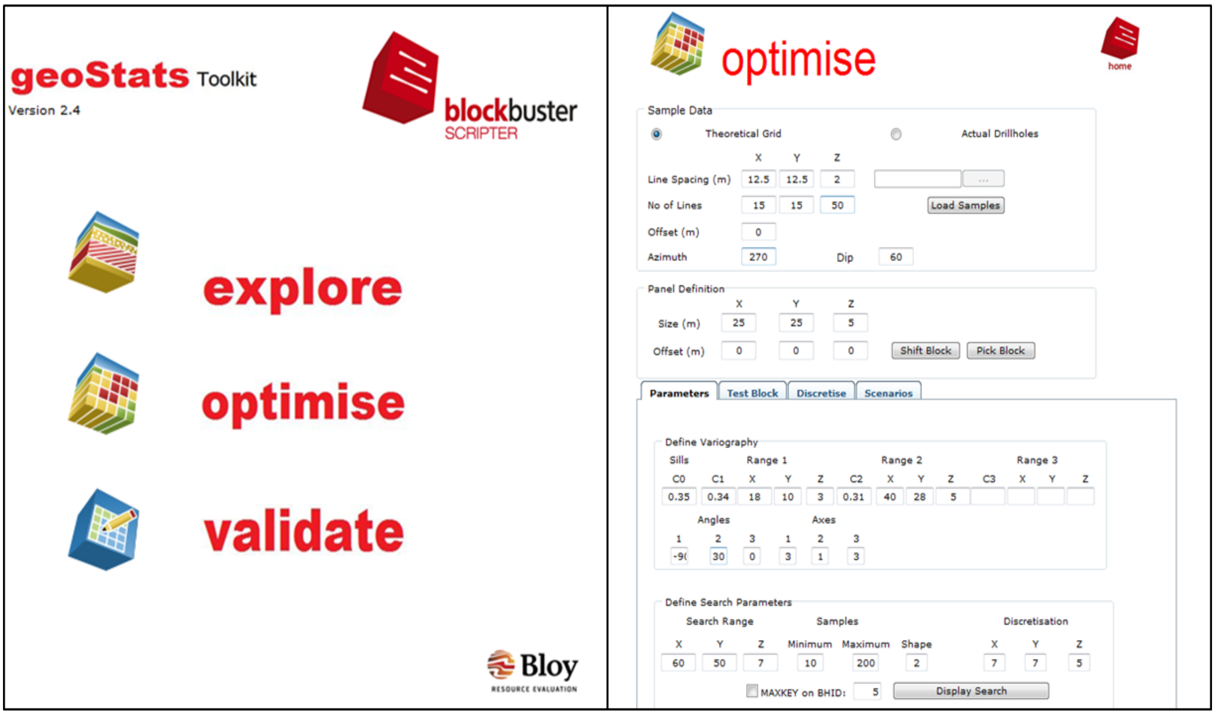

Figure 41: *Script interface*

5.2. SLOPE OF REGRESSION

The BGK outputs the slope of regression as a result. The slope is defined in terms of the covariance and variance of the estimated blocks:

Equation 1:

$$a = \frac{Cov(Z_v, Z_v^*)}{Var(Z_v^*)}$$

where:

a is the slope of the regression

Z_v is the true block grade

Z_v^* is the estimated block grade

A slope of regression equal to one is the requirement for conditional unbiasedness.

The slope can be rewritten in terms of the correlation coefficient p:

Equation 2:

$$a = \rho \frac{\sigma_{Z_V}}{\sigma_{Z_V^*}}$$

where:

a is the slope of the linear regression

ρ is the linear (Pearson) correlation coefficient

σ_{Z_V} is the standard deviation of true block grades

$\sigma_{Z_V^*}$ is the standard deviation of estimated block grades

Correlation may be less than 1 due to smoothing effects of Kriging i.e. estimated grade variability less than true block variability.

5.3. WEIGHTS

For ordinary Kriging, the sum of the weights must add up to 1. Generally the allocated weights are positive. At the margins of optimized searches however, the weights are very small or slightly negative.

The BGK outputs the weights as a percentage of positive and negative weights. The “screen effect” can result in negative weights. It is an in house standard used by AGA that negative weights should not exceed

2% of the total weight. The literature however, accepts anything less than about 5% of the total weight.

5.4. MINIMUM AND MAXIMUM NUMBER OF SAMPLES

To optimise the number of composites, the close spaced theoretical sample spacing of 25m x25m x7m was used together with a discretisation of 5m x5m x 5m.

Search ellipses were oriented according to the approximate orientation of the mineralisation with search distances set to those determined for the search distance optimisation. The maximum numbers of samples were varied and the results recorded to generate the plot below.

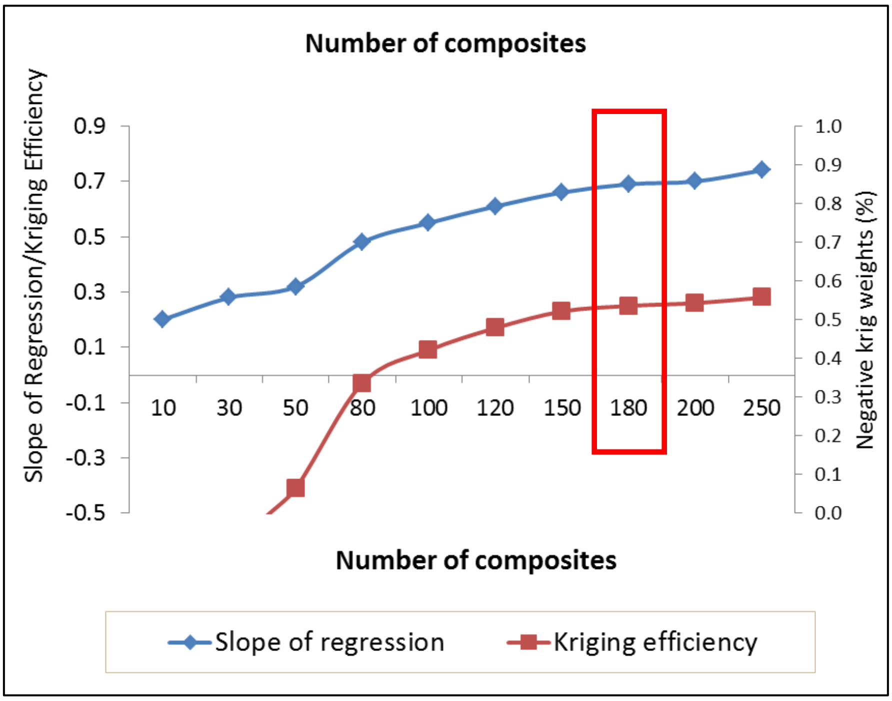

Figure 42: *Number of composites optimisation*

The exercise was repeated for the minimum number of samples. Based on the results, a minimum number of 10 and maximum number of 180 were considered optimal for the Ore zones.

These were chosen at the points at which the slopes of regression and Kriging efficiency graphs flattened out and before a high amount of negative weights were encountered (1-2% was considered acceptable based on AGA guidelines). A test for the waste zone showed that a minimum number of 10 and maximum number of 80 samples are optimal.

5.5. BLOCK SIZE

The block size to be estimated must be comparative to the spacing of the available drilling information informing the estimate to avoid what is known as the 'small block linear estimate problem' described by Stephenson and Vann, 2001, which states that "as block size decreases relative to drill spacing the precision of the estimates decreases, often sharply".

The resultant grade-tonnage curves tend to be distorted and conditionally biased since any cut-off grade greater than zero will equate to incorrect grade and tonnage estimates and thus flawed mine planning (Stephenson and Vann, 2001).

For the block size optimisation (based on the Ore variogram applied for both Hard and Soft Ore estimation) a theoretical sample spacing of 25m x 25m x 7m was assumed together with a discretisation of 5m x 5m x 5m and a minimum of 4 and maximum of 100 samples. Search ellipses were oriented according to the approximate orientation of the mineralisation with search distances set to approximate the variogram ranges.

For the theoretical grid used in the KNA, it was ensured that the block was well informed by samples and that the blocks were not smaller than half the drilling grid dimensions to prevent degradation of the slope of regression and mean at smaller block sizes. The block sizes were then varied and tested (**Figure 43**).

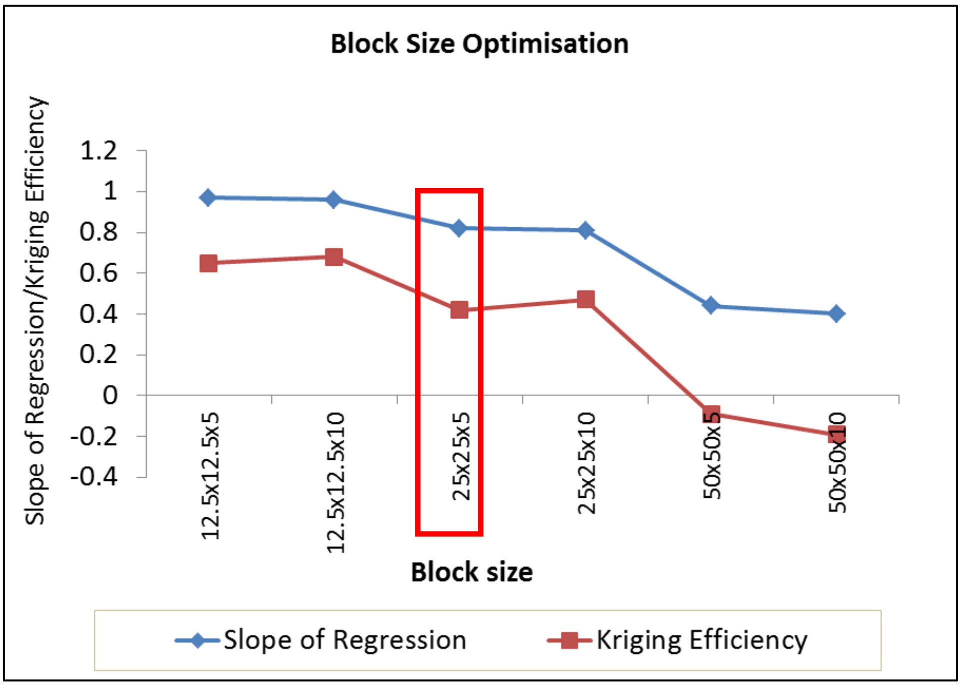

Figure 43: *Block size optimisation*

The optimal model block size was 25m x 25m x 10m. This is the same block size that was used for the previous model.

The results showed that as the test block increased in size, the block variance decreased between blocks but the variance inside the blocks was high due to the volume variance relationship.

When the test blocks were smaller, the variance within the blocks was lower because the samples were close together and less variability existed.

At smaller block sizes (of around 12.5m x 12.5m and 15m x 15m) although more optimal than the larger 25m x 25m or 50m x 50m blocks, are not useful for Uniform Conditioning.

For optimal UC results, larger block sizes are recommended to ensure that a sufficient amount of SMUs (of size 10m x 10m x 3.33m) occur in the estimation panel. As a result, the 25m x 25m x 10m block size was selected.

Negative weights are due to the “screening effect” caused by samples lying between a sample and the point to be estimated (**Table 26**).

Kriging sets weights less than zero to respect the rule that the sum of weights must equal to 1.

By doing so the estimates aren’t constrained i.e. estimates larger than the largest sample value or smaller than the smallest sample value can be reported and therefore accounts for possible extreme values that may be present in the data.

Table 26: *Negative weights reported*

Block Size	% Negative weights
12.5 x12.5 x 5	-4.1
12.5 x12.5 x10	-1.8
25 x 25 x 5	0
25 x 25 x10	0
50 x 50 x 5	0
50 x 50 x10	0

5.6. DISCRETISATION

Discretisation is used in block Kriging for calculating point-block average values. The covariance between blocks is calculated for a range of discretisations.

The optimum number of discretised points should be compatible with the dimension of the block in units of composite-length in the direction approximately parallel to the drilling. The optimum discretisation size tested in the GSK was found to be 5m x 5m x 5m.

5.7. SEARCH RANGE

The search neighbourhood was optimised based on a simulated drilling grid that approximated the drill spacing where less data was available i.e. 25m x 25m. The search ranges selected were not limited to the variogram ranges since the relative nugget effects play a more important role in the estimation.

When a variogram range approaches zero (pure nugget effect), a larger neighbourhood is required for good estimation. At pure nugget, there is no correlation between any two points in the domain therefore the sample grades are uncorrelated with the true block grade. The search is then expanded to the entire domain to find the maximum number of samples to produce a solution with the minimum estimation variance.

When the relative nugget effect approaches zero and long ranges (relative to block size) are involved, the closest samples are more correlated than those far apart to the true block grade. The neighbourhood is restricted to the nearest samples to produce a solution with the minimum estimation variance.

For the search distance optimisation, the wider spaced theoretical sample spacing of 25m (X) x 25m (Y) x 10m (Z) was used together with the optimised block size; a discretisation of 5m (X) x 5m (Y) x 5m (Z) and a minimum of 10 and maximum of 180 samples for EZONE 1 and 2 and a maximum of 80 samples for EZONE 3.

In closely spaced areas (at around 12.5m x 12.5m sample spacing) the search neighbourhood tends to be constrained by the maximum number of samples whereas in wider spaced areas (at around 25m x 25m) the search neighbourhood is constrained by the search ellipse thus substantiating the need to test at wider drill spaces.

In the selection of an appropriate neighbourhood (**Figure 44**), the percentage of negative weights was closely monitored. Less than 1 to 2 percent negative Krige weights was considered acceptable. The results showed that an adequate amount of samples would likely be located (assuming a wider spacing of 25m x 25m) within a 120m x

90m x 7msearch ellipse (around 243 samples) and that no negative weights are likely to be encountered.

Some classifications are based simply on the number of samples found within the search ellipse for a given block. This method is not particularly rigorous since it does not take into account the anisotropy or the relative spatial location of the samples. This means a block with a cluster of data nearby could be classified equally with another where the same number of samples is evenly distributed.

The number of samples per block can, however, be useful in screening out blatantly unreliable areas based on very little data. An octagonal or quadrant search filter can be imposed in order to correct for clustered data (by restricting the maximum number of samples used within individual octagonal quadrant search areas).

This was not deemed necessary for the search optimisation since Kriging is an effective technique to decluster data.

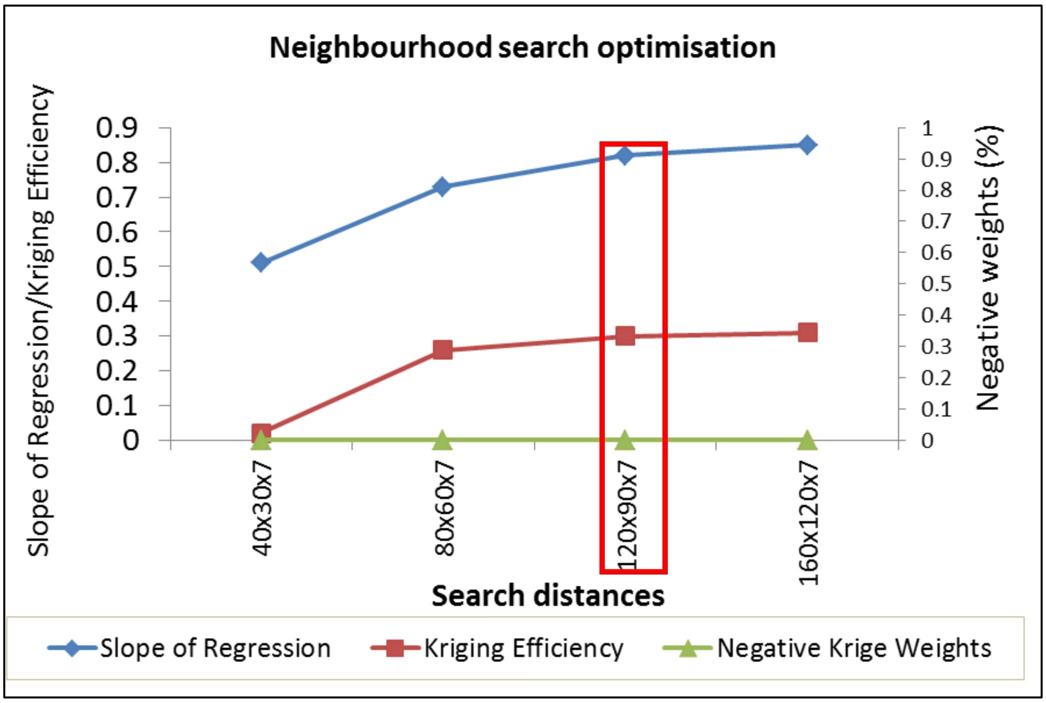

Figure 44: *Neighbourhood search optimisation*

The new optimised KNA results are presented in **Table 27**:

Table 27: *New QKNA results*

EZONE	Model Block Size	Discretisation	Search Distance	Sample max. (test)	Sample Min. (test)
1	25 x 25 x 10	5 x 5 x 5	120 x 90 x 7	180	10
2			120 x 90 x 7	180	10
3			110 x 55 x 55	80	10

6. GRADE ESTIMATION

The choice of estimation method applied depends on the appropriateness of the method to the deposit's geology and the available data.

The FE2 deposit lends itself well to the use of Ordinary Kriging (OK) as an estimation technique since it has performed well on historical models for the FE2 deposit and it minimises error variance (Kriging variance) since the technique is based on the configuration of the variogram and the data and not on the data used to make the estimates. In addition, the data did not show any significant trends at the scale of modelling and the variograms were characteristic of local stationarity. The estimation is based on the following Assumptions:

- The sample values are measured precisely and are reproducible
- The sample values are measured accurately and represent the true value at the location
- The samples are collected from a physically continuous, homogenous population of all possible samples and the phenomenon measured at the sample locations also exist at all un-sampled locations within the zone of interest (no sudden change in characteristics)
- Values at un-sampled locations are related to values at sampled locations

Kriging provides the best estimate since it provides the smallest standard error; narrowest confidence interval and most confidence (lowest risk) (Bertoli et al., 2003).

The block model grades were estimated using Ordinary Kriging in Datamine® Studio 3. Datamine® Studio 3 was also used for data manipulation, earlier statistics, block modelling, validation and reporting. The block model and estimation parameters are summarized in the sections to follow.

6.1. BLOCK MODELLING

The estimates were kriged (Ordinary Kriging) into a sub-celled block model. 25mE x 25mN x 10mRL (meters relative sea level) was the final model block size which was sub-celled into 5m x 5m x 1.67m (for rock-type filling) and 2.5m x 2.5m x 1.67m (for Ore zone filling) units to best represent the wireframe volumes and contacts. Block model parameters are shown in **Table 28**.

Table 28: *Block model parameters*

Model Setting	Value
X origin	213,875
Y origin	1,537,212.5
Z origin	-250
Block size in X direction	25
Block size in Y direction	25
Block size in Z direction	10
Number of blocks in X direction	124
Number of blocks in Y direction	152
Number of blocks in Z direction	45

6.2. BOUNDARY ANALYSIS

Boundary analyses were undertaken to determine whether or not the grade variations across the mineralisation-waste boundaries were “hard” or “soft”.

“Hard” boundaries are defined by abrupt changes in grade whilst “soft” boundaries allude to more gradual changes. The results of the boundary analyses are expressed as threshold searches measured in distance (m) away from the contact.]

Where “hard” boundaries are applied, the distance away from the contact is zero and no samples falling outside the domain were used in the estimation of that particular domain.

In the case of “soft” boundaries, where the grade change is gradational, samples falling outside the domain may be used for the estimation of that domain.

The boundary analysis result is presented below and was run using the Bloy® Geostats kit. The results indicated that hard boundaries were to be applied for the estimation.

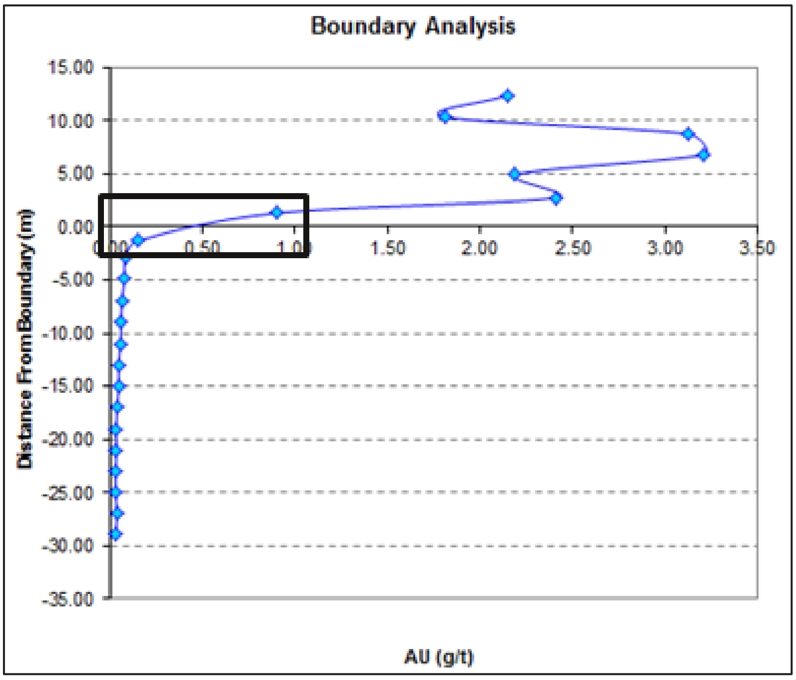

Figure 45: *Boundary analysis – Ore vs. waste*

6.3. SEARCH STRATEGY

The estimation and search parameters used for Ordinary Krige estimation are based on the results presented in **Table 27.** These parameters were optimised by means of a Kriging Neighbourhood Analysis whereby both Kriging efficiency (KE) and slope of regression

(PSlope) were used to investigate conditional bias for a given set of estimation parameters (**see Chapter on KNA**).

These search districts were examined against the resultant block model and were later expanded to ensure that any blocks in the model that were un-estimated would be estimated. This situation arises since the QKNA is based on a theoretical grid and not on the actual sample data on which the estimate is based.

The final search parameters applied for the estimation are tabulated in **Table 29**, below:

Table 29: *Estimation and search parameters*

EZONE	Discretisation	Block size	Rotation			Search Radii			No of samples	
			Z	X	Z	X	Y	Z	Min	Max
1 (Soft Ore)	5x5x5	25x25x10	90	130	0	130	55	25	10	180
2 (Hard Ore)	5x5x5	25x25x10	90	130	0	130	55	25	10	180
3 (Waste)	5x5x5	25x25x10	90	120	0	110	55	55	10	180

6.4. GRADE CAPPING

Unusually high grade samples (also called extreme values) can result in overestimation of a resource.

Top-capping is applied to the composited rather than the uncomposited data to avoid smearing of high grades and therefore a better estimate.

Histograms, log probability plots and mean and variance plots are analysed to determine which grade cap is the most appropriate per domain.

Estimates are sensitive to grade capping therefore attempts were made to keep the amount of values capped to no more than 0.6% of the total dataset based on AGA estimation guidelines.

The top capping results are described in **Table 30**. The variability in each domain (as expressed by the CV) decreased slightly after capping.

Table 30: *Top capping*

				Before		After	
EZONE	**Cap value**	**No. of samples capped**	**Percentage capped**	**Mean**	**CV**	**Mean**	**CV**
1	17.3	8	0.5%	1.69	1.6	1.65	1.4
2	12.0	1	0.6%	1.45	1.6	1.42	1.5
3	0.6	95	0.6%	0.05	2.1	0.05	1.9

6.5. GRADE CUTTING

Top cuts (99th percentile or above) are generally applied to remove the extreme grade values from the resource database whilst including the high-grade assays below the top cut that are recognized as a real feature of the assay distribution (Pocock, 2001, citing Enterprise Metals, 1990).

The impact of applying top cuts was evaluated by Pocock (2001). The results showed that the application of a cut-off leads to changing the inherent characteristics of the data. Attempts to apply even very high cut-offs still reduces the variation in mean gold grade as well as the percent relative standard deviation (RSD) between datasets.

In addition, Pocock (2001) found that wide spaced drilling produced strongly biased datasets and showed sensitivity to extreme values and a disproportionate contribution of a few samples to the average grade of an estimate (especially in smaller data sets).

In the case of FE2, it was decided best to include the high-grade in the database because the histograms and variograms for each domain determined that they were reliable and should be used in the estimation.

6.6. BLOCK MODEL FIELDS

The block model fields are summarized below.

Table 31: *Block Model Fields*

FIELDS	DESCRIPTION
IJK	Block index
XC, YC, ZC	Sub-cell coordinates
XINC, YINC, ZINC	Size of sub-cells
EZONE	Estimation domain
HARD	Hard or soft flag (0 = Soft; 1 = Hard)
REDOX	Redox level indicator
ROCKTYPE	Material type
HVAL	Blastability indicator
DENSITY	Density
AU	Ordinary krige estimates
AU_KVAR	Kriging variance for Ore Estimation
ALT	Graphitic material indicator (0 = No graphite; 1 = Graphite)
DOM_GPH	Graphite Estimates
CLASS	Mineral Resource classification
NUMSAM	Number of samples used to estimate the AU block
Gxxx	SMU grade above cut-off (xxx refers to specific cut-off)
Pxxx	SMU proportion above cut-off (xxx refers to specific cut-off)
XMORIG, YMORIG, ZMORIG	Block model origin (bottom left corner)
NX, NY, NZ	Number of parent cells in X, Y and Z (25m x 25m x 10m)

Table 32: *Summary of EZONE (estimation domain) values*

EZONE	DESCRIPTION
1	Laterite and Saprolite Ore
2	Hard Ore
3	Waste

Table 33: *Summary of definition of the Redox zones (REDOX values) used in the model*

REDOX	DESCRIPTION
1	Oxidised Material (above Redox 2 surface - "Oxide")
2	Transition Material (between Redox 2 and Redox 3 surfaces – “transition”)
3	Reduced Material (below Redox 3 surface - "Sulphide")

Table 34: *Summary of definition of the ALT values used in the model*

ALT	DESCRIPTION
0	No graphitic material present
1	Graphitic material present

Table 35: *Summary of definition of the HVAL top and bottom limits used in the model*

HVAL	DESCRIPTION
0	Completely Soft Material (above hard/soft surface)
1	Completely Hard Material

**HVAL values range from 0 to 1 (a hardness probability)*

Table 36: *Description of ROCKTYPE codes used in the model*

ROCKTYPE	DESCRIPTION
1	Laterite/Clay
2	Saprolite (Soft Oxide)
3	Silicified Oxide
4	Soft Sulphide
5	Hard Sulphide
6	Blast Oxide

7	Blast Sulphide
8	Transition Material

Table 37: *Bulk densities assigned per Rock type*

ROCKTYPE	DENSITY(g/cm3)
1 – Laterite/Clay	2.11
2 – Saprolite (soft Oxide)	1.97
3 – Silicified Oxide	2.05
4 – Soft Sulphide	2.46
5 – Hard Sulphide	2.70
6 – Blast Oxide	2.05
7 – Blast Sulphide	2.70
8 – Transition Material	Soft (hard=0) 2.21 Hard (hard=1) 2.46

Table 38: *Description of classification codes used in resource model*

CLASS	DESCRIPTION
0	Undefined – not present in the model
1	Measured – not present in model
2	Indicated
3	Inferred
4	Blue Sky Tangible

7. UNIFORM CONDITIONING

"*Uniform Conditioning (UC) provides a method for creating a resource model that is representative of the variability of the deposit for a defined selective mining unit (SMU), which if used for mine planning and reserve calculation can increase the confidence in the resulting reports and mine plans*" - dataminesoftware.com (2015).

In mining, resources are estimated into larger mine planning blocks called panels but are mined as selective mining units (SMU). SMUs are defined as the smallest volume of material on which the decision between Ore and waste is based. The estimation of recoverable resources therefore depends on the volume on which the Ore waste decision is made i.e. the support effect (Neufeld, 2005)

The aim of UC is to estimate the tonnage and the metal content of blocks inside a panel conditionally to the sole panel grade, which is estimated assuming local stationarity (e.g. ordinary Kriging) (geovariances.com, 2015)

This non-linear estimation technique was used to estimate the recoverable resources for EZONES 1 and 2. For the reporting the distribution of grades above various grade cut-offs.

Consider one large block (**Figure 46**): The average grade of one large block is the average of the grades of smaller blocks within that large block and the distribution of grades inside the large block is less variable than that for the smaller blocks as defined by the volume-variance relationship.

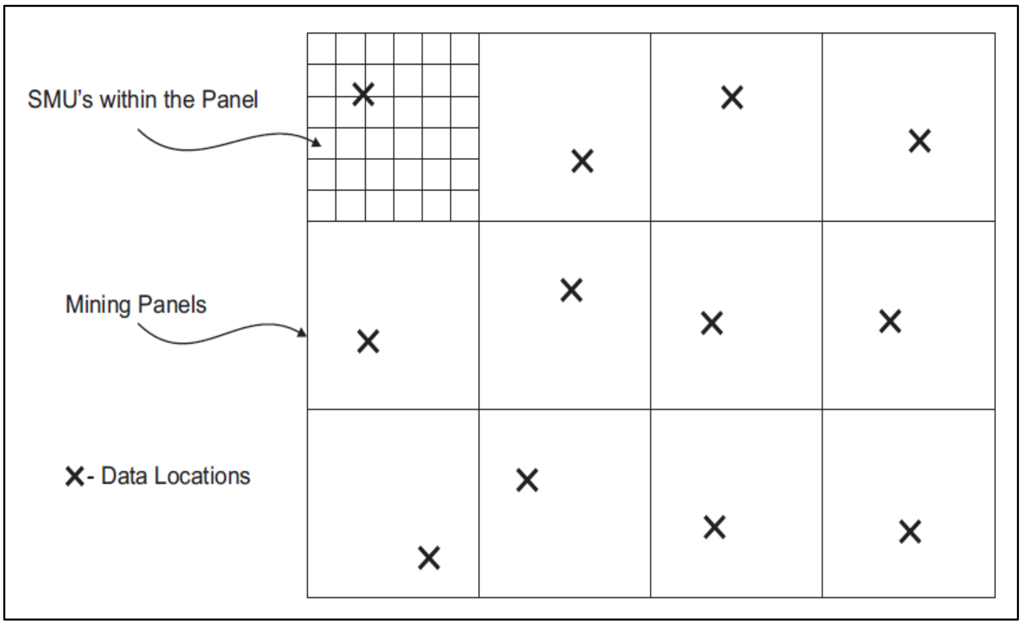

Figure 46: *Setting for Uniform Conditioning*(Neufeld, 2005, pp. 4)

UC is based on the premise that if a robust kriged estimate of a panel grade (one large block) is known, then the result can be conditioned to the SMU (small block) scale.

Since samples, composites, panels and SMUs are different in terms of scale, they cannot be directly compared to each other. Moving from one volume scale to another is known as a change of support. If the grade of a panel is known (estimated), the distribution of the SMUs inside the panel can be calculated using the anamorphosis function that converts input data into a normal score distribution. During the anamorphosis, the real grades are compared to the normal scores using Hermite polynomials (Neufeld, 2005)

Recoverable resources are calculated using the SMU distribution that has been derived from the panel estimate and the change of support model. Uniform conditioning uses the discrete Gaussian model to accomplish the change of support (Neufeld, 2005)

The detailed mathematical calculations underlying the process will not be presented below, as it is beyond the scope of this study, but more information can be found in the thesis by Neufeld (2008).

UC is based on the following assumptions (Neufeld (2005) and (VIZI, 2008)):

- SMU grades and the panel grades are bivariate normal. If the normal score panel grades are known, then we know the mean and variance of the normal scores of the SMU distribution i.e. If the panel grade is known (estimated using Ordinary Kriging) then the distribution of SMUs within that panel is also known
- The SMU distribution can be determined from the estimated panel grade and the change of support

However, limitations exist:

- Panels with the same estimate have the same grade and proportion curves, regardless of the surrounding data.
- It is unknown where the high or low grade SMU's are located within the panel

7.1. UC PROCESS

The general way to implement the UC process for a mining project is discussed by Neufeld (2005) and is summarised below. For a concise and complete understanding of the underlying uniform conditioning theory the reader is directed to the thesis by Neufeld (2005) as it is beyond the scope of this thesis.

- **Estimating the panel grades** – Making a robust estimate of the panel grade. Widely spaced data introduces errors when estimating small blocks or smaller SMUs, therefore block Kriging into larger mining panels is necessary. Using the fact that Kriging is based on a linear system of equations, the linear average of the grade within the blocks can be estimating by exchanging the point to point covariance factor in the equation with a point to block factor.

- **Fitting a Discrete Gaussian Model (DGM) to the data** – the minute sample collected for assaying is only representative at the scale it was collected at. The discrete Gaussian model allows for a change of support to be made from the sample to block scale and controls the shape and variability of the distribution at the larger scale

- **Determining the change of support coefficients** – there is often insufficient data to determine the distribution of grades for volumes larger than the point samples. The variance of the block is then calculated from model variograms using dispersion variance theory that relates the point sample support with larger supports.

- **Transforming the panel estimates** – Since panel estimation is based on the original grade data, the data needs to be transformed to Gaussian space in the process of panel anamorphosis.

- **Calculating the proportion and quantity of metal above different cut-offs** – once the panel grade is known, the SMU distribution within that panel can be calculated. The panel gets the average grade of the SMUs within it and variance is based on the change of support coefficients calculated earlier. The proportion and quantity of metal above a particular cut-off grade is calculated to define the recoverable resources.

The SMU size assumed for the FE2 UC process was 10x10x3.33 and was based on the selectivity achievable with the current mining equipment. Given the panel size of 25x25x10, there were about 18 SMUs in each panel. The ideal number of SMUs in a panel is considered to be at least 15 (to obtain a reasonable distribution of SMUs in the panel). This Block Anamorphosis is calculated on a small support equivalent to the SMU, and the mesh of the input grid has to be a multiple of this support (representing panels).

A tonnage adjustment factor was applied and was based on a volume representing half the SMU size. It was expressed as a percentage of the panel size (2.7%). Any proportions smaller than this percentage were removed as they would not be practically recoverable (these volumes would be too small to mine with the selected equipment).

For FE2, the following steps were followed in the ISATIS geostatistics software package to generate the recoverable resource model using the Uniform Conditioning technique:

- **Import composites and regularised panel estimates** – A prerequisite for data analysis is that all samples represent an equal volume i.e. support. When considering 3D data, regularisation is necessary to ensure that each point has the same importance before weights are assigned during Kriging. This is achieved by ensuring that the same sample volumes are used. Composited data was used due to the samples having equal lengths.
- **De-cluster the sample data and generate variogram models from the de-clustered data** - Experimental variograms were calculated for variables transformed into Gaussian space for simulation purposes, and for Gaussian anamorphosis modelling with block support correction. The weights were declustered to bias in over and under sampled areas.
- **Transform the data –** The data was transformed to normal space using Gaussian Anamorphosis Modelling. The distribution of point scale data in Gaussian space is required for the change of support.

Hermite Polynomials were selected for multivariate scenarios and to account for the information effect at the GC sample spacing of 12.5x12.5.

- **Perform a change of support** (on the data) – the distribution of the SMU was computed from the point distribution to be used for the UC. .
- **Run the UC estimation** – This was run using the block anamorphosis function (with information effect), the krige value and the dispersion variance (the block variance) as inputs. The grade distribution for each variable was then estimated inside the panel.
- **Add the SMU estimates to the kriged block model** – The SMU estimates were then joined to the Kriged block model in Datamine® Studio 3.

The SMU model was represented by the fields Gxxxx and Pxxxx which was the grade (G) and proportion (P) of a block above a particular cut-off grade (xxxx). For example, P050 was the proportion and G050 the grade of the block above a cut-off grade of 0.5 g/t Au.

The results of the Uniform Conditioning (UC) are presented below and are shown with the krige results and the theoretical grade-tonnage curves (BA with and without the information effect - IEF).

Overall, the UC curves compared well with the theoretical curves – especially in the well informed domains and the degree of selectivity achieved appeared reasonable.

The krige curves are shown in black; the UC curve in blue and the theoretical curves in red and green (without and with the information effect respectively). UC was not applied to the waste zone (dummy UC values were placed in the grade and proportion fields).

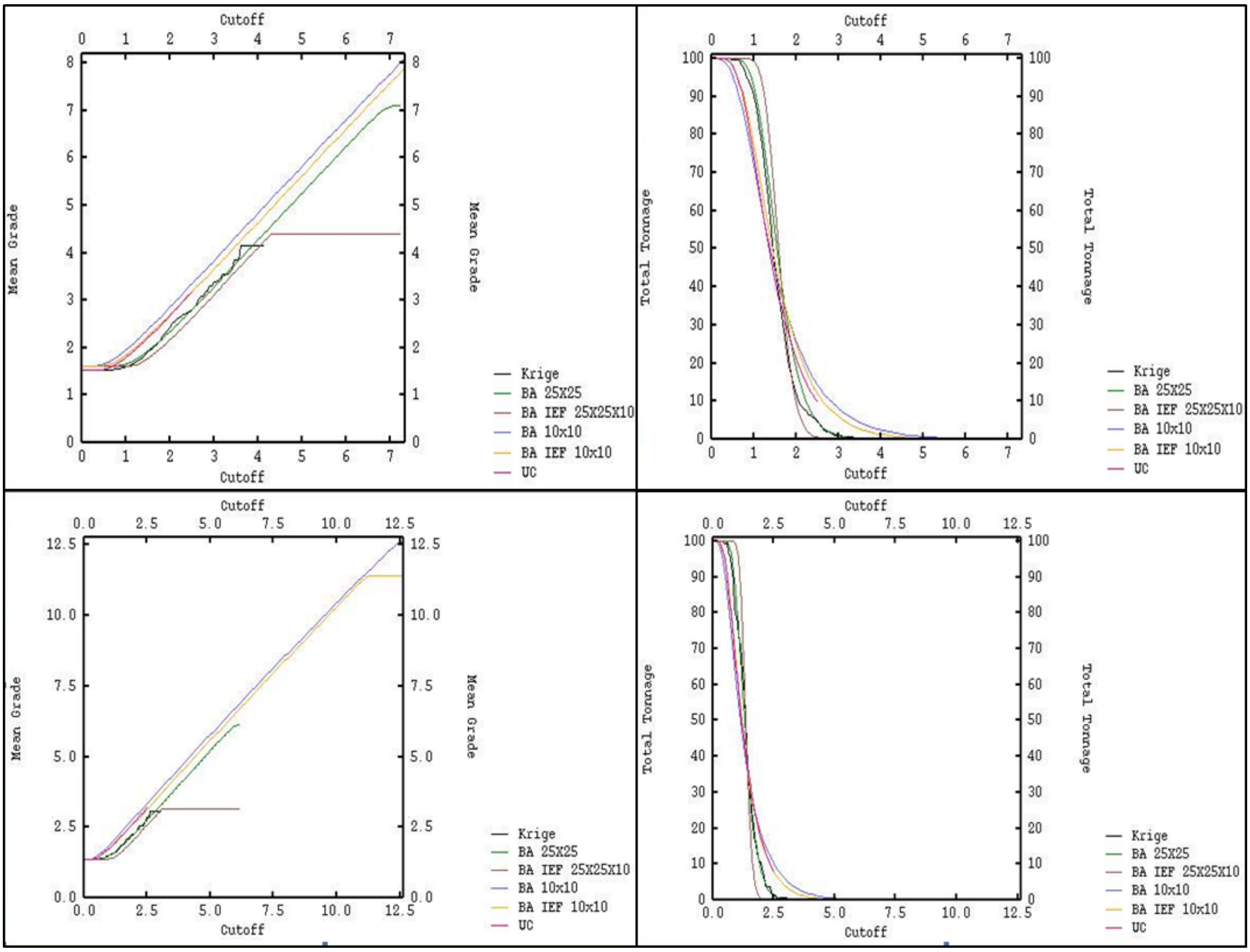

Figure 47: *UC grade-tonnage curves shown with krige and theoretical curves*

8. MODEL VALIDATION

The block model estimates were validated as follows:

- Visually comparing the model estimates against input grades
- Comparison of the global and input means
- Sectional plots of number or composites, model grades and composite grades
- Grade-tonnage curves

Globally the domain means for the reefs are within 10% of the input data means which shows that the estimates are representative of the input data and thus acceptable.

A comparison between global and input means is presented in **Table 39** below.

Table 39: *Global mean comparison*

	Drillholes				Blocks				
EZONE	No Samples	Min	Max	Mean	Volume	Min	Max	Mean	% Difference
1	3344	0.00	17.30	1.71	1,356,271	0.42	4.16	1.68	-2%
2	168	0.00	14.00	1.37	111,375	0.46	3.06	1.39	2%
3	19281	0.00	0.60	0.07	52,870,844	0.00	0.25	0.04	-36%

Slice plots were generated for each EZONE also to compare the global mean model block grade with the input data (drillholes) mean grade.

A swath plot is a graphical display of the grade distribution from series of bands/swaths that are generated in several directions through the deposit.

The input sample composite averages and calculated block model grades were calculated on the easting (vertical N-S slices) that correspond to the to the dimensions of the block model block which was 25m thick.

The purpose is to compare the input sample data with the resulting block model data to ensure that no gross over or under estimation occurs.

The slice plots follow in **Figure 48** and the results are discussed below.

8.1. DISCUSSION OF VALIDATION RESULTS

The northing composites generally compare well for EZONES 1 and 2. Local over and under estimation is may be attributed to the estimation process and the selection of the composite level relative to the parent block centroids.

Other deviations occur due to reduced tonnages at the edge of the deposit as well as differences in grade in lower grade areas - generally at the flanks of the deposit where the density of drilling decreases and material is classified as Inferred Mineral Resource.

In the case of EZONE 1, where block estimates are higher than the composites, this is possibly due to the orientation of the Kriging ellipse against the orthogonal nature of the slice plots.

Overall the block averages follow the general trend of the input sample data.

On a local scale, the model does not provide reliable estimates of grade, but on a larger scale, it represents unbiased estimations of the grade distributions based on the underlying sample data.

EZONE 3 showed that the sample grades were in cases higher than the block estimates which may suggest a bias since the grades were extremely low as it is a waste zone.

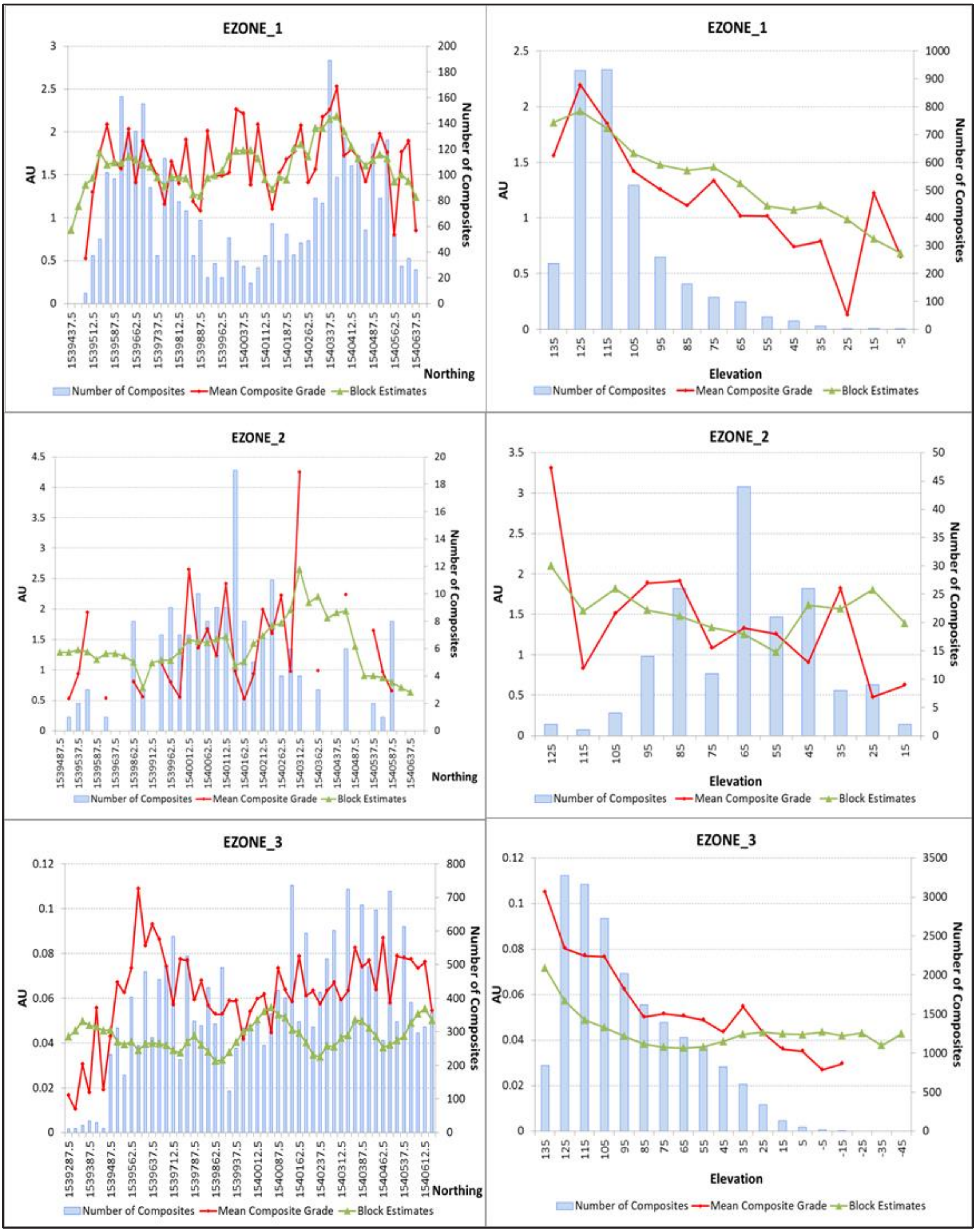

Figure 48: *Sectional validation plots*

9. MINERAL RESOURCE CLASSIFICATION

9.1. GUIDELINES

The Mineral Resource was classified in accordance to the South African Code for the Reporting of Exploration Results, Mineral Resource and Mineral Reserves (SAMREC) and Joint Ore Reserves Committee (JORC) Code.

Classified tonnage and grade estimates are for use in mining investment decisions to assess relative risk and allow interested parties to make a judgment on the relative worth of the statement of the Mineral Resource and Ore Reserves.

Due to these economic consequences the quality, quantity and continuity of the geological data informing the estimates must be thoroughly assessed as done in previous chapters to ensure reliable classification (**Figure 49**).

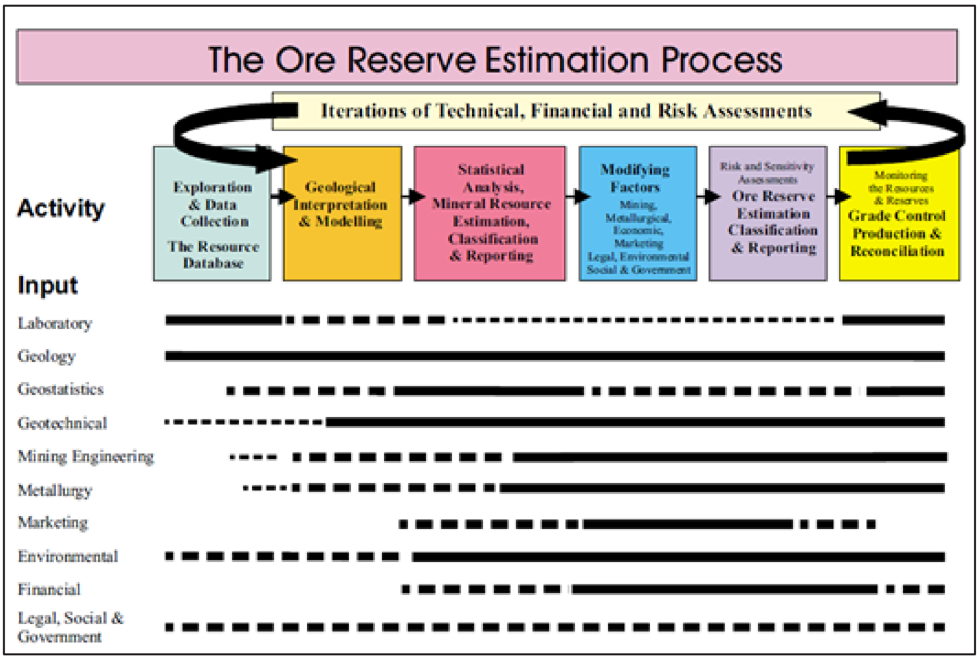

Figure 49: *Ore Reserve Estimation Process* (Appleyard, 2001, pp. 4)

9.2. CLASSIFICATION

The framework for classifying tonnage and grade estimates according to differing degrees of geosientic confidence and economic evaluation is depicted in **Figure 50** (SAMREC Code, pp. 9).

In general, to move a Mineral Resource from infered to measured, the level of confidence should increase. One way, to increase the confidence is to use estimates based on optimised drill hole spacing. Drillhole spacing exercises are carried out to see at which spacing the confidence is the highest.

AGA's bases its classification on the Mineral Resource and Reserve Committee's guidelines. The metal content above the Ore cut-off is measured to an accuracy of 90% confidence at 15% error over a period of 3 months for Measured and for 1 year for Indicated.

The 15% error 90% confidence method is adapted from Anglo American and the idea is to estimate the average grade above cut-off with less than 15% relative error and 90% confidence.

In mining terms this would mean that one out of a possible ten blocks (or production panels/time period) would have a relative Kriging error in excess of 15%.

- Indicated Mineral Resource: One year's production should meet the criteria (i.e. for ten year's production one year would be expected to have an error in excess of 15%).
- Measured Mineral Resource: One month's production should meet the criteria.

The drillhole spacing selected for the classification of the FE2 Mineral Resource is taken from similar studies based on historical reconciliation for the mine.

A drill spacing of 25 m by 25 m was considered sufficient to classify the Mineral resource as Indicated and 50 m by 50 m as Inferred.

Areas covered by greater spacing was considered to be Blue Sky potential (not an official Mineral Resource Category, but used for internal purposes to estimate possible mineralisation potential). No Measured Resource was defined.

The classification criteria are based on studies completed for other, similar Sadiola deposits (such as FE3 and 4) - an updated classification study to confirm its suitability is recommended.

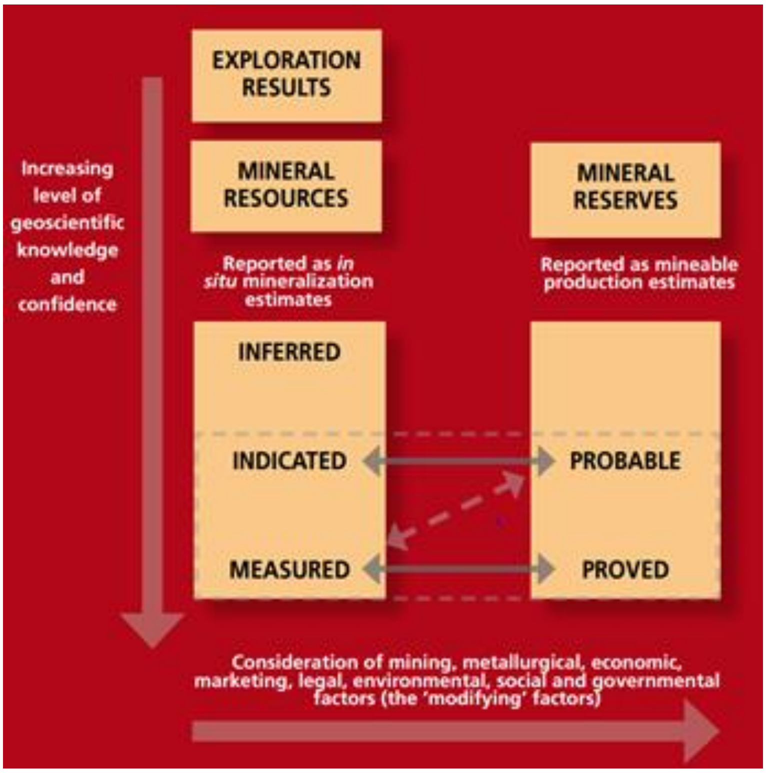

Figure 50: *Relationship between Exploration Results, Mineral Resources and Mineral Reserves* (SAMREC Code, 2012, pp. 10)

The Mineral Resource was subdivided and reported into the Inferred, Indicated and Blue Sky Categories using the technique outlined below:

- Classification strings and wireframes were modelled to include those areas occurring within a 25m x 25m drill hole spacing
- Samples occurring within the wireframes and within a mineralised reef were classified as being an Indicated Mineral Resource

- Those samples occurring inside the mineralised reefs but outside the classification wireframe were classified as Inferred if they were estimated.
- The samples that occurred inside the mineralised reefs, but which had not been estimated were assigned a global mean and classified as being a Blue Sky Mineral Resource.

All mention of cut-off grades in this chapter refer to the economic cut-off grade that is calculated by AGA financial analysts and is above the scope of work of this dissertation.

Sections through the Mineral Resource model are shown below, indicated areas that have been classified.

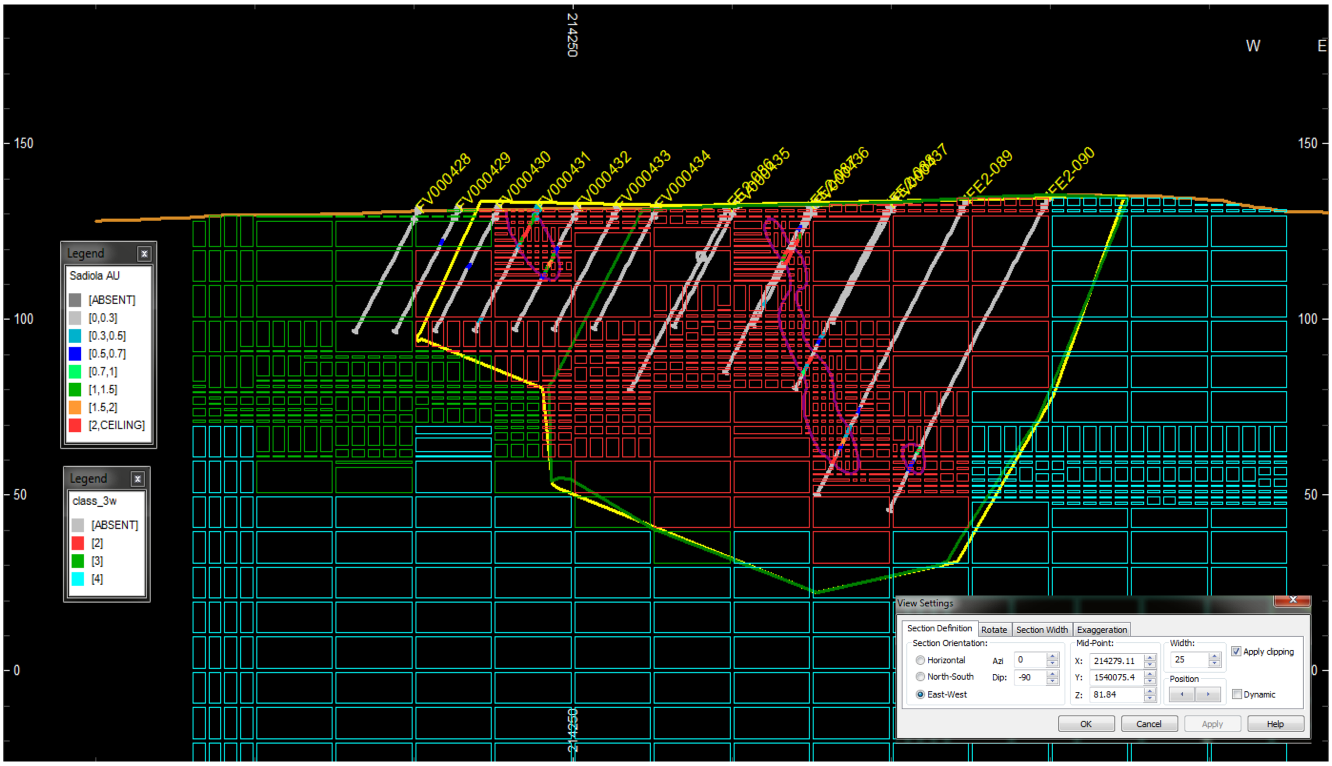

Figure 51: *Section through the Mineral Resource model (looking north) shows the Indicated (red), Inferred (green) and Blue Sky (blue) classes in relation to the mineralisation (Purple) and drillholes informing the estimate*

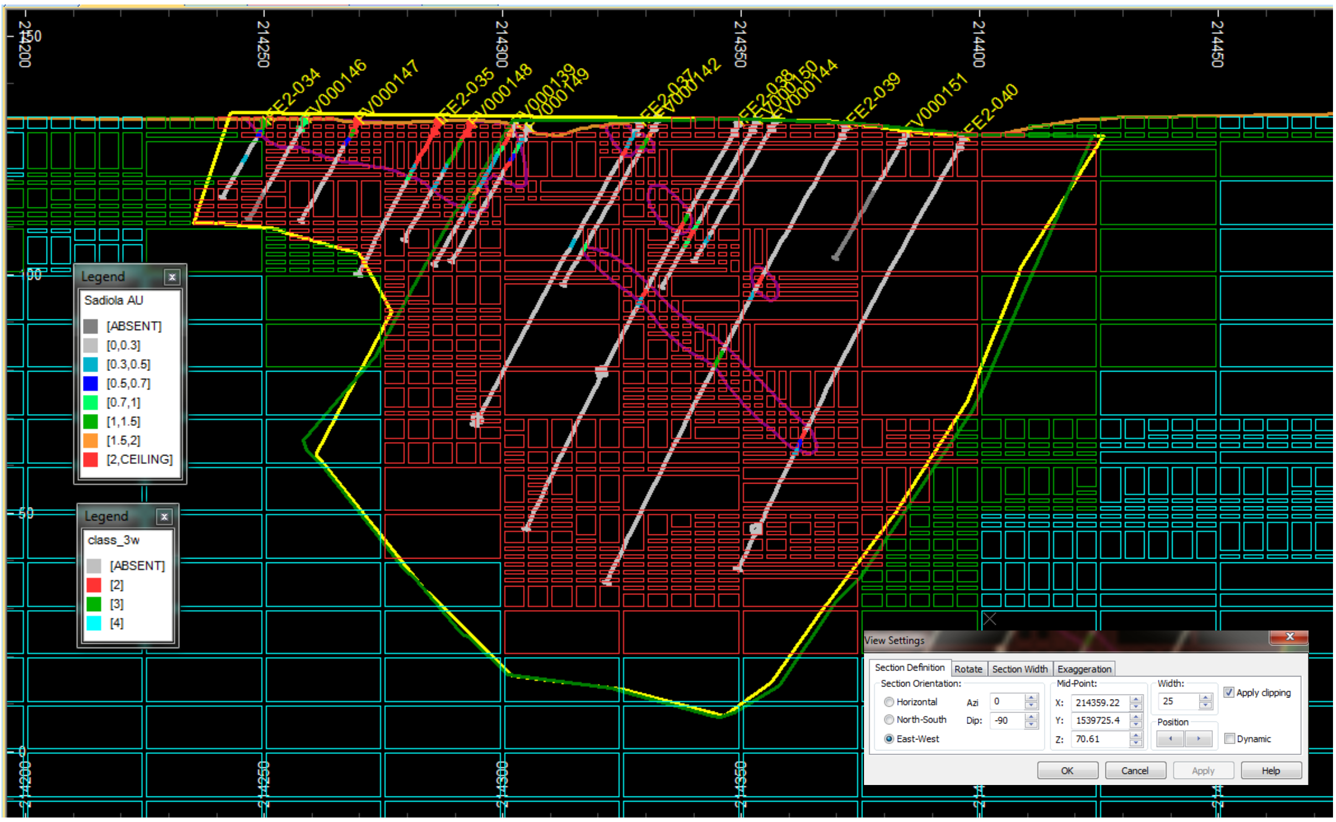

Figure 52: *Section through the Mineral Resource model (looking north) shows the Indicated (red), Inferred (green) and Blue Sky (blue) classes in relation to the mineralisation (Purple) and drillholes informing the estimate*

9.3. BLUE SKY ESTIMATES

Both the waste blocks and the blue sky Ore estimates were assigned "dummy" UC values by assigning the krige grade to every UC grade field and a proportion of either 0 (grade not above particular cut-off) or 1 (grade is above particular cut-off) to the proportion fields. The mean grades assigned to the blue sky estimates are summarised below.

Table 40: *Mean grades assigned to blue sky estimates*

EZONE	AU (g/t)
1 (Lat and Sap Ore)	1.68
2 (Hard Ore)	1.39
3 (Waste)	0.001

9.4. MINERAL RESOURCE TABULATION

The FE2 Mineral Resource is reported below in accordance with the guidelines of the JORC Code (2012 Edition) above Rock-type variable cut-off grades (Mineralised Waste cut-off grades). The Mineral Resource was reported inclusive of Ore Reserves and within the Business Plan (BP) 2015 $1,600/Oz optimised pit shell.

Table 41: *FE2 Mineral Resource Reported by Rock-type, March 2015*

Rock type	**Cut-off grade g/t**	**Indicated**			**Inferred**		
		Tonnes '000 t	Au g/t	Au metal Ounces	Tonnes '000 t	Au g/t	Au metal Ounces
Laterite & Clay	0.75	167,654	1.86	10,008	4,669	1.71	257
Oxide Saprolite	0.70	1,264,238	1.94	78,814	13,551	1.71	744
Siliceous Saprolite	0.75	1,635	1.88	99	-	-	-
Sulphidic Saprolite	1.05	564	2.07	38	-	-	-
Hard Sulphide	1.05	1,299	2.06	86	-	-	-
Blast Oxide	0.75	10,868	2.37	829	-	-	-
Blast Sulphide	1.10	-	-	-	-	-	-
Transitional	0.75	28,434	2.05	1872	-	-	-
Total		**1,474,693**	**1.94**	**91,746**	**18,220**	**1.71**	**1,001**

Notes:

1. *Mineral Resource reported above Rock type variable cut-off grades.*
2. *Mineral Resource reported inclusive of Ore Reserves.*
3. *Mineral Resource reported within BP2015 $1,600/Oz optimised pit shell.*

10. MODEL RECONCILIATION

Model reconciliations quantify the differences between the new and previous model. The same modelling and estimation methodology was applied to both models.

10.1. COMMON VOLUME COMPARISON

These differences are determined by comparing a common volume between the two models i.e. the old versus the new model. Even though the models being compared, cover a common volume (within an optimised shell), the commonality can be affected when a category is excluded from reporting.

In the comparisons presented below, it should be noted that Blue Sky is often excluded and only the Inferred and Indicated Mineral Resource considered during reporting which would lead to tonnage differences at zero cut-off (an uncommon volume).

The previous Mineral Resource model (2014) was compared with the updated Mineral Resource model (2015) within a common volume i.e. within the BP2015 $1,600 Mineral Resource shell and the $1,200 Mineral Reserve design (and below the topography as no mining has taken place at FE2) to quantify the differences as a result of the model update.

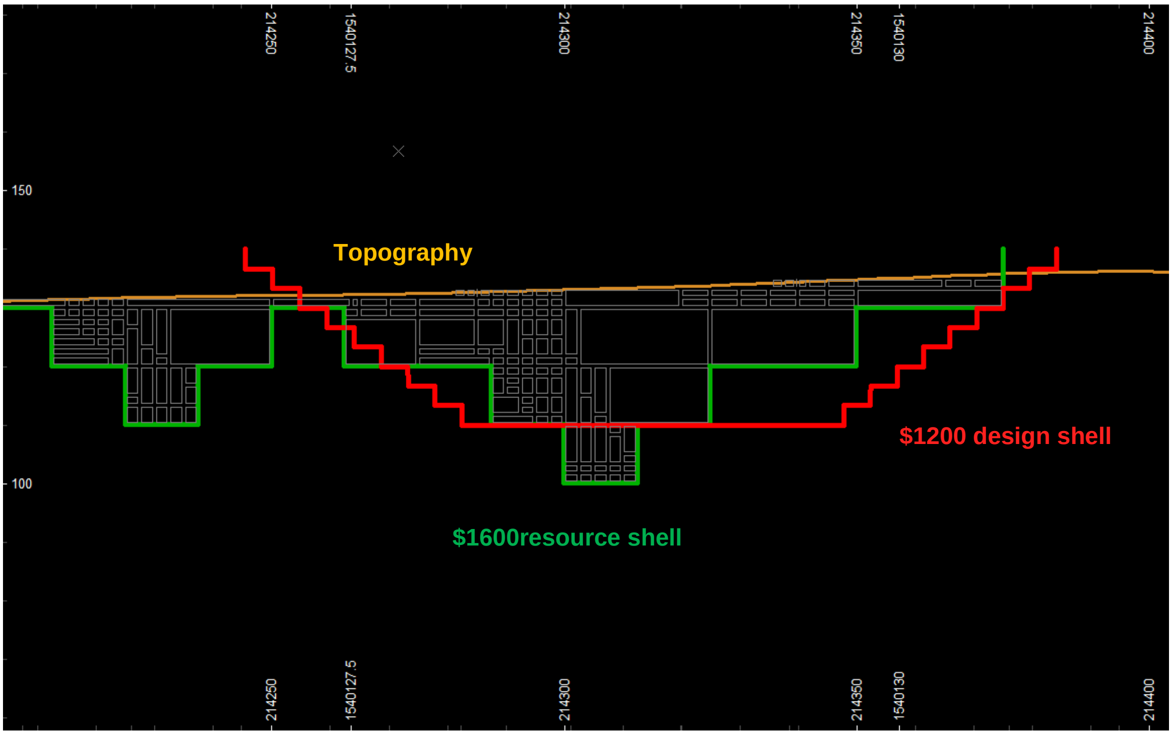

Figure 53: *Section (looking north) through the 2014 Mineral Resource model showing the common volume (grey blocks) used for the reconciliation*

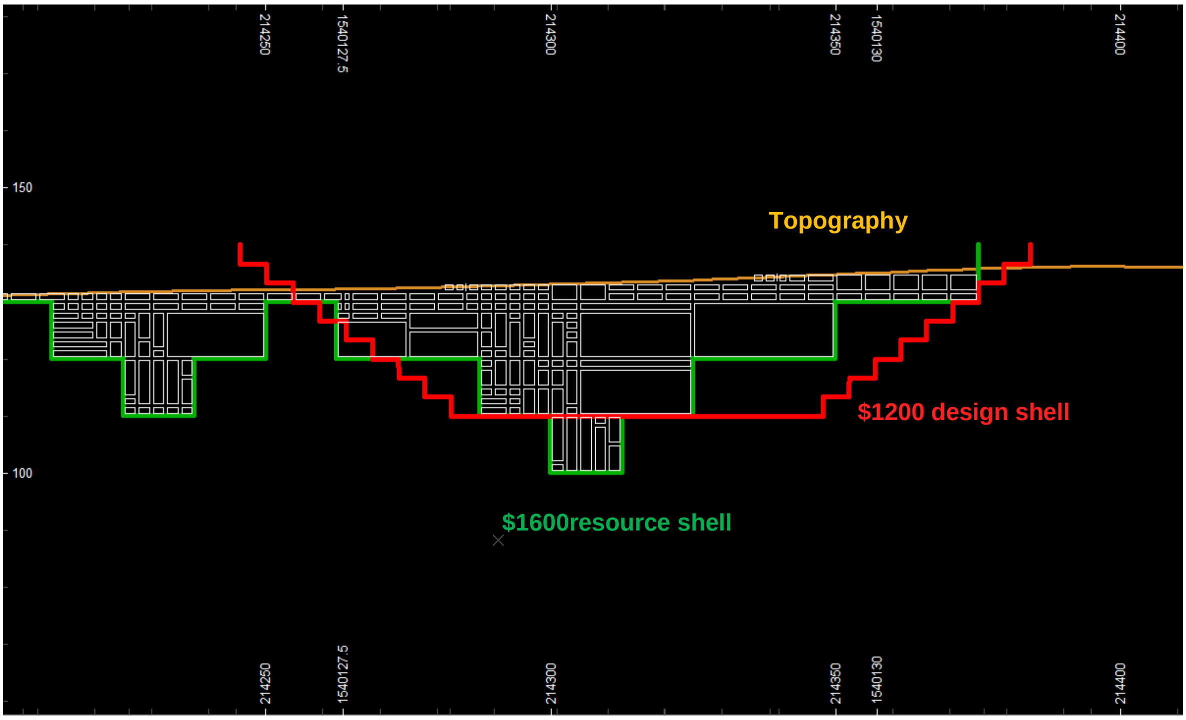

Figure 54: *Section (looking north) through the 2015 Mineral Resource model showing the common volume (white blocks) used for the reconciliation*

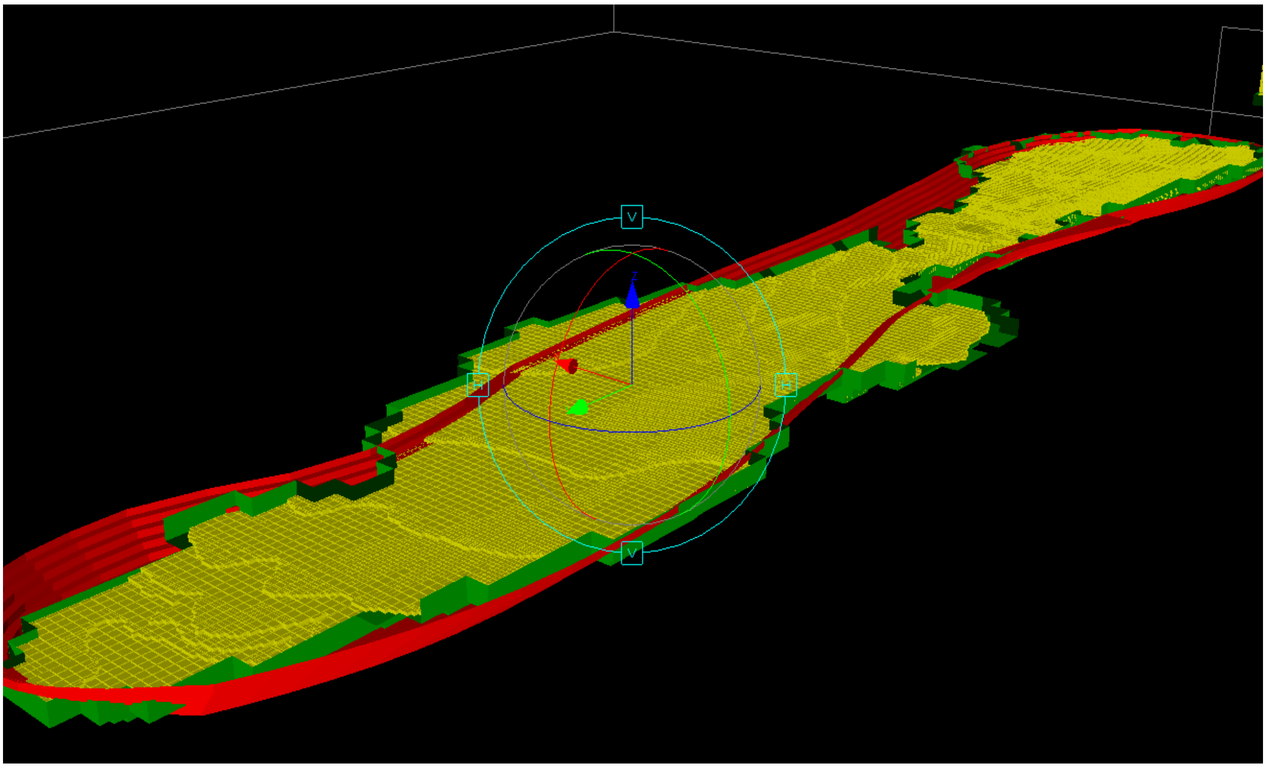

Figure 55: *3D view showing the $1200 design shell (red) and the $1600 Mineral Resource shell (green) and the common volume (yellow blocks) between them*

The mineralised waste cut-off grades (BP2015) were used for the comparison and the results presented in **Table 42.**

Table 42: *Economic Cut-off grades for BP2015 – FE2*

Laterite	Saprolite	Siliceous Oxide	Saprolite Sulphide	Hard Sulphide	Intermediate Oxide	Intermediate Sulphide	Transitional
0.75	0.70	0.75	1.05	1.05	0.75	1.10	0.75

The grade-tonnage curve for the two models within the common volume is presented in **Figure 56** (it included both the Indicated and Inferred Mineral Resource).

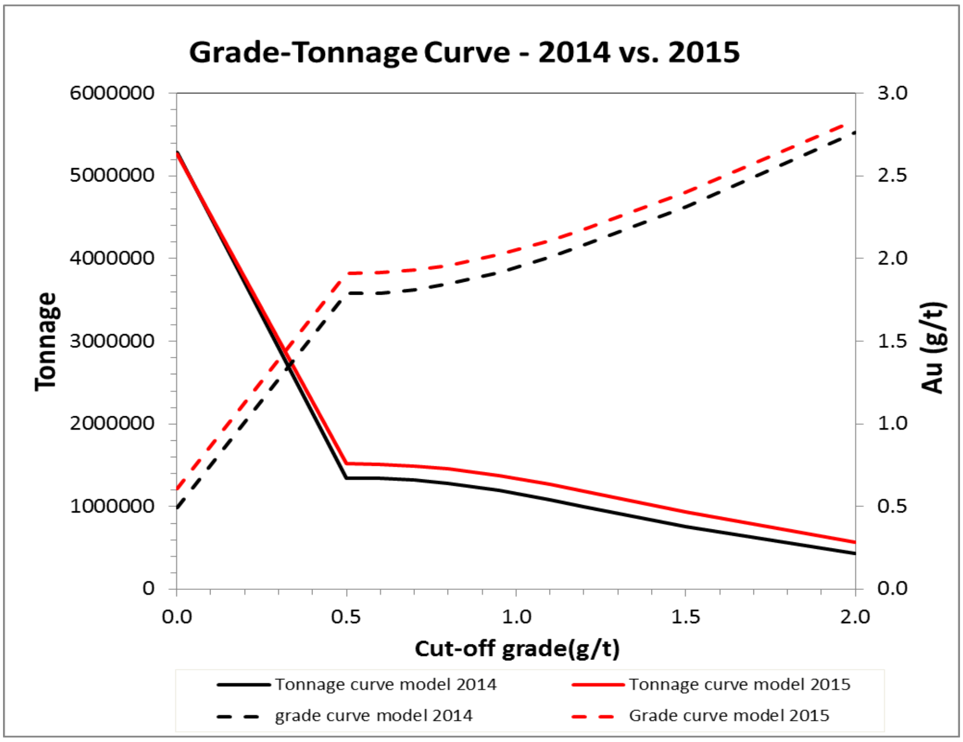

Figure 56: *Grade-Tonnage Curves: 2014 vs. 2015 models within BP 2015 $1,600 shell*

The comparison is presented in table format below. The new model has higher grade and higher tonnes than the previous model with new Indicated and upgraded Inferred category Oxide Ore Mineral Resource being added.

Table 43: *Tabular comparison – 2014 RM vs. 2015 RM within BP2015 $1,600 shell*

2014 RM				2015 RM			Percentage difference (2015-2014/2015)		
CUT OFF	TONNES	AU(g/t)	AU(g)	TONNES	AU(g/t)	AU(g)	TONNES	AU(g/t)	AU(g)
0.00	5,287,628	0.39	2,615,876	5,264,353	0.61	3,213,192	-0.4%	36%	19%
0.50	1,344,015	1.79	2,407,476	1,519,868	1.91	2,900,692	12%	6%	17%
0.60	1,342,682	1.79	2,406,813	1,512,756	1.92	2,897,212	11%	6%	17%
0.70	1,321,959	1.81	2,394,531	1,495,519	1.93	2,886,741	12%	6%	17%
0.80	1,281,864	1.85	2,366,270	1,463,474	1.96	2,863,876	12%	6%	17%
0.95	1,194,976	1.92	2,292,450	1,379,763	2.02	2,793,250	13%	5%	18%
1.10	1,085,301	2.01	2,181,600	1,271,267	2.11	2,684,482	15%	5%	19%
1.20	1,003,303	2.08	2,087,991	1,191,388	2.18	2,593,280	16%	4%	19%
1.50	760,924	2.31	1,761,104	938,545	2.40	2,253,928	19%	4%	22%
2.00	429,336	2.76	1,186,709	566,615	2.84	1,608,342	24%	3%	26%

An overall comparison of the two models by Mineral Resource category and Rock-type is presented below (still within the $1,600 Mineral Resource shell and above the Mineralised Waste cut-off grades).

To further simplify, some of the Rock-types were combined (presented in the second set of tables). The comparisons show an increase in the hard and soft Indicated Oxide material (of about ten thousand ounces) and a decrease in the hard and soft Inferred Oxides (of about two thousand ounces).

These changes are mostly ascribed to the changes in the interpretation of the mineralised envelopes; the estimation search parameters and the general increase of the bulk densities assigned to the various Rock types.

Table 44: *Model reconciliation by broader material types: 2014 vs. 2015 MW cut-off grades*

	2014			2015			
	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Ounce difference (2014-2012)
Indicated							
Soft Oxides	1,460,037	1.75	82,351	1,431,892	1.93	88,822	6,471
Hard Oxides	4,126	1.57	208	12,503	2.31	928	720
Transitional	9,188	1.86	550	28,434	2.05	1,872	1,872
Soft Sulphides	-	-	-	564	2.07	38	-512
Hard Sulphides	-	-	-	1,299	2.06	86	86
Total Indicated	1,473,350	1.75	83,109	1,474,693	1.94	91,746	8,637
Inferred							
Soft Oxides	53,988	1.67	2,894	18,220	1.71	1,001	-1,893
Hard Oxides	-	-	-	-	-	-	-
Transitional	-	-	-	-	-	-	-
Soft Sulphides	-	-	-	-	-	-	-
Hard Sulphides	-	-	-	-	-	-	-
Total Inferred	53,988	1.67	2,894	18,220	1.71	1,1001	-1,893

Table 45: *Model reconciliation by Mineral Resource category: 2014 vs. 2015 at MW cut-off grades*

Year	Category	Rock type	Cut-off grade	Tonnes	Aug/t	Au metal (Oz)
2014	**Indicated**	**1**	0.7	87,582	1.87	5,267
		2	0.7	1,372,455	1.75	77,084
		3	0.7	4,126	1.57	208
		4	0.8	-	-	-
		5	0.85	-	-	-
		6	0.7	-	-	-
		7	0.85	-	-	-
		8	0.7	9,188	1.86	550
	Total Indicated			**1,473,350**	**1.75**	**83,109**
	Inferred	**1**	0.7	18,548	1.66	990
		2	0.7	35,440	1.67	1,904
		3	0.7	-	-	-
		4	0.8	-	-	-
		5	0.85	-	-	-

		6	0.7	-	-	-
		7	0.85	-	-	-
		8	0.7	-	-	-
	Total Inferred			53,988	1.67	2,894
	Total Blue Sky			3,708	1.39	166
2015	**Indicated**	**1**	0.75	167,654	1.86	10,008
		2	0.70	1,264,238	1.94	78,814
		3	0.75	1,635	1.88	99
		4	1.05	564	2.07	38
		5	1.05	1,299	2.06	86
		6	0.75	10,868	2.37	829
		7	1.10	-	-	-
		8	0.75	28,434	2.05	1872
	Total Indicated			1,474,693	1.94	91,746
	Inferred	**1**	0.75	4,669	1.71	257
		2	0.70	13,551	1.71	744
		3	0.75	-	-	-
		4	1.05	-	-	-
		5	1.05	-	-	-
		6	0.75	-	-	-
		7	1.10	-	-	-
		8	0.75	**-**	-	-
	Total Inferred			18,220	1.71	1,001
	Total Blue Sky			174	1.68	9,419

10.2.MINERAL RESOURCE RECONCILIATION

The Mineral Resource reported from the previous model (the 2014 model) is compared with the Mineral Resource reported from the updated model as at March 2015. The models have been reported above separate Mineralised Waste cut-off grades and within separate $1,600/Oz optimised pit shells (BP2015 reporting).

Table 46: *FE2 Mineral Resource Reported by Rock type, December 2014*

Rock type	Cut-off grade g/t	Indicated			Inferred		
		Tonnes '000 t	Au g/t	Au metal Ounces	Tonnes '000 t	Au g/t	Au metal Ounces
Laterite & Clay	0.75	80.7	1.83	4,759	18.8	1.68	1,018
Oxide Saprolite	0.7	1,187.0	1.78	67,963	36.0	1.67	1,932
Siliceous Saprolite	0.75	2.1	1.80	120	-	-	-
Sulphidic Saprolite	1.05	0.8	2.02	52	-	-	-
Hard Sulphide	1.05	1.6	2.10	105	-	-	-
Blast Oxide	0.75	-	-	-	-	-	-
Blast Sulphide	1.1	-	-	-	-	-	-
Transitional	0.75	32.7	2.11	2,217	-	-	-
Total		**1,342.5**	**1.80**	**77,492**	**54.9**	**1.67**	**2,950**

**Mineral Resource reported above Rock type variable cut-off grades.*

**Mineral Resource reported inclusive of Ore Reserves.*

**Mineral Resource reported within BP2015 $1,600/Oz optimised pit shell.*

Table 47: *FE2 Mineral Resource Reported by Rock type, March 2015*

Rock type	Cut-off grade g/t	Indicated			Inferred		
		Tonnes '000 t	Au g/t	Au metal Ounces	Tonnes '000 t	Au g/t	Au metal Ounces
Laterite & Clay	0.75	167,654	1.86	10,008	4,669	1.71	257
Oxide Saprolite	0.70	1,264,238	1.94	78,814	13,551	1.71	744
Siliceous Saprolite	0.75	1,635	1.88	99	-	-	-
Sulphidic Saprolite	1.05	564	2.07	38	-	-	-
Hard Sulphide	1.05	1,299	2.06	86	-	-	-
Blast Oxide	0.75	10,868	2.37	829	-	-	-
Blast Sulphide	1.10	-	-	-	-	-	-
Transitional	0.75	28,434	2.05	1872	-	-	-
Total		**1,474,693**	**1.94**	**91,746**	**18,220**	**1.71**	**1,001**

**Mineral Resource reported above Rock type variable cut-off grades.*

**Mineral Resource reported inclusive of Ore Reserves.*

**Mineral Resource reported within BP2015 $1,600/Oz optimised pit shell.*

11. DISCUSSION AND RECOMMENDATIONS

Major adjustments made since the previous update and any other key issues regarding the updated model are summarised below.

Changes to the mineralised envelope interpretation

Mineralised envelopes were re-interpreted using Leapfrog® software's grade interpolation technique. In some places the intepretations remained similar; in others it was narrowed (to remove internal waste samples) or steepened (to better fit the overall mineralisation trend).

The weathering and hardness surfaces were updated based on the advanced grade control drilling and interpretations improved on where necessary.

Update to variograms and estimation parameters

The variograms and search parameters were updated as a result of the changes to the mineralisation interpretation. Estimation was by Ordinary Kriging into 25m x 25m x 10m panels followed by a Uniform Conditioning exercise assuming an SMU size of 10m x 10m x 3.33m.

Top capping

For the updated model, around 0.3-0.6% of the krige and UC values was top capped to remove the influence of extreme/outlier grades on the estimates.

The top caps applied were not the same as those used previously.

Rock type and density updates

The FE3 deposit densities were previously applied to the FE2 deposit (as they were considered similar in mineralisation style). These densities were revised using density probe measurements where applicable. In comparison with the 2014 densities, most of the densities have remained the same or have decreased slightly. The Soft Oxide density probe measurements were significantly higher than in 2014 and were thus unchanged until further work is done to confirm the density.

Table 48: *Previous densities compared with updated densities*

ROCKTYPE	DENSITY - FE2 2014 (t/m3)	DENSITY - FE2 2015 (t/m3)
1 – Laterite and Clay	2.22	2.11
2 – Saprolite (Soft Oxide)	1.97	1.97
3 – Silicified Oxide	2.16	2.05
4 – Soft Sulphide	2.46	2.46
5 – Hard Sulphide	2.70	2.70
6 – Blast Oxide	2.16	2.05
7 – Blast Sulphide	2.70	2.70
8 – Transitional material	Soft (hard=0) 2.21 Hard (hard=1) 2.46	Soft (hard=0) 2.21 Hard (hard=1) 2.46

Exclusion of RAB holes

RAB holes were included in the previous model but removed for this update. On a local scale, the previous model showed that were RAB holes were included slight over-estimation below the mean and under-estimation above the mean resulted.

General observations

- The computer software is a tool for estimation and modelling and the output must always be checked after each step; always make sense, and be managed

- Checking the data continuously helps identify and differentiate between manageable risks and critical ones

- All input data must be validated and be as accurate as possible

- The estimation method selected should not cause unnecessary smoothing

- Model block sizes and search strategies should be optimised in a KQNA exercise

- A good understanding of the geology and geometry of the Ore body is essential. Experience, training and teamwork is important to capture the detail required

- A good understanding of the waste contained in the Ore body is equally important because it affects mining costs (mining method, selectivity and processing)

- Producing a Mineral Resource model requires planning in advance to ensure that enough time is spent on each phase in the Mineral Resource estimation process

- Classification expresses the confidence of the data in terms of how much data is available in the area of interest and how it relates to the true mineralisation. As contentious a topic as Classification is, common sense should always take precedence.

- Good reconciliation between estimates and raw data shows that the choice of estimation method is appropriate.

- If the metal content is consistent with previous results but large differences in grade and tonnage occur – reconciled tons is not a good tool to measure how good the estimation was.

- Grade is the most important – more tons mined at a lower grade might have negative economic implications

Recommendations

- The Domaining applied to the data was grade-based (0.32g/t cut off). In gold deposits, low grade cut offs give poorly structured variograms due to the mixing of external waste and Ore as well as internal waste. At higher cut-offs clearer structures can be observed. The Domaining could therefore be applied at different cut-offs and then variography applied

- The Drillhole type should be coded into the data i.e. Exploration and Grade Control

- The drillhole data errors were corrected in the Datamine® files used for the model update, but will also need to be corrected in the master Century Systems database.
- The classification criteria used for the FE2 Mineral Resource model were based on studies completed for other, similar Sadiola deposits (such as FE3 and 4) - an updated classification study to confirm its suitability is recommended.
- Soft Oxide density probe measurements were significantly higher than in 2014 and were thus unchanged until further work is done to confirm the density.
- The generally accepted test to validate Mineral Resource models is with the grade control results. Carras (2001), however states that this view is flawed since Ore bodies are mined to maximise the present value and mining is not executed in the same way that the Ore body was estimated. Perfect reconciliation instead is an indication that all the factors in the system are well-tuned and by implication production targets can be met whilst allowing operators some flexibility in production.

- The software used in the estimation process should be compared against similar software in the industry to single out the one that provides the most accurate results
- Further work should be carried out to assess the effect of top cuts and top caps on the resulting Mineral Resource models
- Further work is required on boundary analysis going forward as in reality the Laterite and Saprolite are very different, despite the results of the statistics suggesting that they are similar. It needs to be investigated whether the Laterite was transported or not and if the mineralised zones match. These zones can therefore be combined for variography but not for estimation in future.
- The holes that showed the greatest discrepancies (FV000001; FV000110; AFE2-169; AFE2-170 and AFE2-243) were then checked against the latest LIDAR surface. The collars still did not honour the topography and were excluded from the estimation. At the time of Ore modelling however, the latest LIDAR had not been provided and these holes were used to guide the Ore wireframe.
- Further work is required to understand how to best model the extend of the graphite alteration
- The classification criteria are based on studies completed for other, similar Sadiola deposits (such as FE3 and FE4) - an updated classification study to confirm its suitability is recommended.

12. RISK ASSESSMENT

Mineral Resource evaluation risks arise due to the probability of estimates failing (Appleyard, 2001).

Technical investigations are carried out as a basis for Mineral Resource bankers to construct financial models upon. Since these models dictate what amount of money is available for the project, it must be based on the highest standard of information (Amos, 2001).

All estimates are not the same as the real value. Errors and therefore risks are inherent to any estimation. Kriging as an interpolation method provides a single estimate for a particular grade at an un-sampled location based on the variogram.

The models are therefore "smoother" than reality since extreme grades are removed for the variography. The interpolation is independent of the actual grades but dependant on the variography which outlines the grade similarities between samples at varying distances.

For a full description on why risk assessments are important, the reader is referred to the paper by Amos, Q.G., 2001, Resources and Risk – A Lender's View.

Previous risks that were outlined based on the geology and estimation process were readdressed in this update:

Geological Risks

- Bias on the Grade/Tonnage Curve- No matter how accurate the estimation process, it will never be exactly equal to the real value of the estimated block.

As a result – blocks that are estimated to be just below cut-off might in fact actually be Ore based on its true value. This block is sent to waste. The same can occur for blocks estimated to be just above cut-off but are in fact truly waste.

This block is sent to the plant. This can cause discrepancies between predicted and recovered grade and tonnage (Clark, n.d.)

- The volume variance relationship - causes a second problem. The variance of the block estimates is generally larger than the variance of the true block values. Since the grade tonnage curve is based on estimates, it becomes bias toward lower tonnage and higher average grades (Clark)

Table 49 presents a checklist of assessment and reporting criteria based on the JORC code. No major risk was identified.

Table 49: *Risk Analysis*

Sampling Techniques and Data	
Criteria	Comment
Sampling techniques	Chip samples were collected over 2 m intervals and split using a two tiered stacked riffle splitter (Jones riffler). Samples were crushed on site by using a conventional jaw crusher before submission of an approximate 2-3 kg of sample to SGS Kayes for Fire Assay. Samples from the last phase of drilling, Advanced Grade Control carried out during Q4 2014, were assayed at the SSEMOS on site assay lab.
Drilling techniques	Grade control and Reverse Circulation holes were included in the estimate. Rotary Air Blast samples were excluded due to possible bias and contamination associated with the sampling technique
Drill sample recovery	Sample recovery was in line with industry standards.
Logging	Logging of the RC chips was acceptable. Logs were received for Rock type, alteration type, alteration intensity, hardness and Redox.
Sub-sampling techniques and sample preparation	At the laboratory, the samples were dried (typically for 8 hours), then passed through a jaw crusher which reduced the maximum size to <6 mm. A riffle splitter was used to reduce the sample size to 500 g which was then pulverized for a minimum of 3 minutes in a Labtech LM2 chrome steel pulveriser. Depending on the lab and material type, 30 or 50 grams of material were extracted for analysis. The gold analyses were by traditional Fire Assay followed by Atomic Absorption Spectroscope.
Quality of assay data and laboratory tests	The routine insertion of QC materials is incorporated into the FE2 sample streams and regular audits and job observations are performed to monitor quality. The QC material comprised Certified Reference Materials (CRMs), blanks; field and pulp duplicates and pulp reject repeats from previous sample submissions. Audit programs were run in addition to the normal QC insertions and monitoring undertaken by the assay laboratory. The CRMs are commercially certified standards prepared and supplied by Rocklabs Limited for a variety of gold grade ranges
Verification of sampling and assaying	The performance of the certified reference materials (CRMs) was very good. In all, a total of 178 standards from 3 different grade ranges (low, medium and high) were inserted in to the batches of samples submitted for assay. The duplicates (field duplicates and pulp repeats) and blanks (coarse and pulverised) also passed the QAQC process. The assay data used in the modelling and evaluation of FE2 was deemed precise (repeatable) and accurate (unbiased).

	Zero assays were found and queried with site personnel. The zero values were the result of voids in lithology and lost sample and therefore set to absent in the database. (-) value replaced and by 0.1g/t Au in accordance to the assay value for all samples within BHIDFV000602. (0) value was changed to 1.71g/t Au in accordance to the assay value for all samples within BHIDFV000741
Location of data points	Visual checks showed that a few collar positions did not honour the topographic surface. Leapfrog mining software was used to make adjustments to collar depth values so that any collars not lying on the topography were superimposed onto it. A new collar table was produced with depth values that reflected the topography. The results were analysed and small differences in the topography (1-2 m) due to vegetation were ignored. The holes that showed the greatest discrepancies (FV000001; FV000110; AFE2-169; AFE2-170 and AFE2-243) were excluded from the estimation
Data spacing and distribution	Exploration hole spacing was at 25m x25m and Advanced Grade Control holes at 12.5m x12.5m. Holes were composited to 2m. The data was adequate for defining the spatial continuity of the resource.
Orientation of data in relation to geological structure	The average drilling direction was a bearing of 270 degrees. Some hole inclinations were -90 degrees which was not optimum to capture the geological structure. This however did not appear to bias the result
Sample security	Samples are bagged in the field and transported directly to the laboratory
Audits or reviews	A QAQC report Is produced for all sampling. The database is also audited.

Estimation and Reporting of Mineral Resources	
Criteria	Comment
Database integrity	All drillhole and surface sampling data for FE2 has been validated. Drillholes were checked for zero or missing collars; errors in collar positions; duplicated collars and coordinates; interval errors (missing intervals, overlapping intervals, negative length intervals); zero grade values; long sample lengths and visual inspected (holes terminating mid mineralisation-domain, collars not sitting in correct position, sampling gaps in the reef, drillhole deviations). The data is stored in a sequel database using an AGA customised Century Systems. The following checks are in place to ensure

	that the integrity of the database is not compromised: • Field logs are captured into Excel templates and signed off before loading into Fusion. Key sections include: Collar, Survey, Meta Data, Sample information, Sample QA/QC insertions and geological coding. • Survey collar positions are updated in the database and plotting positions verified. • Geologists check on down-hole surveys and drilling methods and that all other related columns in the database such as drillhole depths, drill rigs and the reason for drilling are correctly populated. • Quality Control samples and standards are captured. Sample numbers are checked. • lithological information is checked • Assay results are received in electronic format from the laboratories and loaded directly into the database. A random check of 10% of the data is done by the project geologists to confirm the validation. The database is backed up as part of the mine's IT protocol and a copy stored off site.
Site visits	A site visit was carried out a few months before this evaluation which include a visit to some of the AGC drilling sites at the FE2 deposit.
Geological interpretation	The FE2 mineralisation is controlled by a combination of lithology, structure, weathering and alteration. Wireframes were generated for the topography, weathering surfaces, hardness boundaries, graphite alteration and mineralisation. The composited drill data was used to generate the updated grade envelopes in Leapfrog software in an attempt to separate out higher grade areas in order to estimate them separately (0.32g/t threshold). Some manual refinement of the envelopes was required to fine-tune the result. This manual adjustment involved using “dummy” high or low grade points to either extend mineralisation where connectivity was less than desired or restrict mineralisation where it was more than desired (or there were unreasonable extensions beyond data support - often termed Leapfrog “blow outs”). The previous and updated mineralisation interpretations were carefully assessed and compared well with each other in terms of the geological controls governing the mineralisation
Dimensions	The resultant mineralisation interpretation strikes approximately N-S (for 1.1 km) and dips at 50 degrees towards the east
Estimation	The wireframe interpretations were imported into Datamine Studio 3 software and

and modelling techniques	used to code the drillhole samples according to mineralisation, lithology, weathering and structure. Samples within the mineralised envelope were deemed "Ore" and those outside, "waste. Mineralized domains were identified on the basis of logged samples and grade continuity, and were guided by the previously modelled mineralized wireframes. The spatial limits of these domains were examined through a boundary analysis. Supervisor (v8) geostatistical software was used to calculate and model the variograms. The results of the QKNA exercise informed the selection of model block sizes, maximum and minimum sample numbers and the dimensions of the search neighbourhood defined. During the QKNA exercise, the neighbourhood was optimized to ensure the best regression statistics in order to reduce or eliminate conditional bias. The Kriging efficiency was compared against the block variance. If the Kriging variance is low compared to the block variance, the degree of smoothing is minimised and the grade tonnage relationship is best reflected. Kriging provides the best estimate since it provides the smallest standard error; narrowest confidence interval and most confidence (lowest risk).
Moisture	Dry bulk densities were used
Cut-off parameters	The cut-offs were provided by site
Mining factors or assumptions	Some issues were found with the mining shells provided. In some instances the $1200 design shell lay above the $1600 resource shell. These were assessed and did not affect the results. The conversion of the resource model to a reserve model will be done by the SEMOS mine planning team. The methods and process is similar to the other satellite pits on the lease.
Metallurgical factors or assumptions	A graphite wireframe was created using the interpolation technique in Leapfrog mining. The modelling was based on the logged alteration codes in the drillhole file. Graphite alteration reduces metallurgical recoveries. Indicator Kriging was also used to estimate graphite and was flagged in the model with a field. The estimated graphite was visually compared with the field ALT and did not capture the extent of the graphite alteration as well as the graphite wireframe. Samples of the Ore body were submitted to the onsite laboratory for recovery studies using the bottle roll method. The recoveries of the Oxide Ore are in line with what has been observed on the other deposits mined on the concession.
Environmental factors or assumptions	Environmental rehabilitation costs will incorporated in the pit optimisation process by the SEMOS mine planning team .

Bulk density	Density measurements for 2015 were carried out for the FE2 pit from the advanced grade control holes (using downhole density probe). What method was used for density (attach procedures to appendix)
Classification	Classification strings and wireframes were modelled to include those areas occurring within a 25m x 25m drill hole spacing. Samples occurring within the wireframes and within a mineralised reef were classified as Indicated Resources. Those samples occurring inside the mineralised reefs but outside the classification wireframe were classified as Inferred if they were estimated. The samples that occurred inside the mineralised reefs, but which had not been estimated were assigned a global mean and classified as Blue Sky Resources. The metal content above the Ore cut-off is measured to an accuracy of 90% confidence at 15% error over a period of 3 months for Measured and for 1 year for Indicated. The 15% error 90% confidence method is adapted from Anglo American and the idea is to estimate the average grade above cut-off with less than 15% relative error and 90% confidence.
Audits or reviews	Model reconciliations were run to quantify the differences between the new and previous model.
Discussion of relative accuracy/ confidence	The resource model is compared against the grade control model on site using BGK. If discrepancies occur, these are then updated in the model.

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APPENDIX A

QA/AC REPORT

SADIOLA MINE
FE2 EXPLORATION AND ADVANCED GRADE CONTROL DRILLING QAQC REPORT

Report date :January 2015

TABLE OF CONTENTS

1. INTRODUCTION

The QA/QC measures utilised during the drilling of 1074 drillholes at Project FE2 (FE2) incorporated the routine insertion of QC materials into the sample stream as well as regular audits and job observations. QC material comprised Certified Reference Materials (CRMs), blanks, field and pulp duplicates and pulp reject repeats from previous sample submissions. These programmes were run in addition to the normal QC insertions and monitoring undertaken by the assay laboratory. The CRMs are commercially certified standards prepared and supplied by Rocklabs Limited for a variety of gold grade ranges.

The analytical technique utilised was fire assaying on 30gm aliquots. Aliquot size delivered to the laboratory was reduced from 90 g to 30 g in 2014. The reason for the mass reduction was to eliminate possibility of trial run by the laboratory and to reduce assaying cost.

The drilling programs covers the period 1998 – 2015. Recent 2015 exploration drilling consisted of 24 sterilization drillholes. The count of drillhole types by project is as follows:

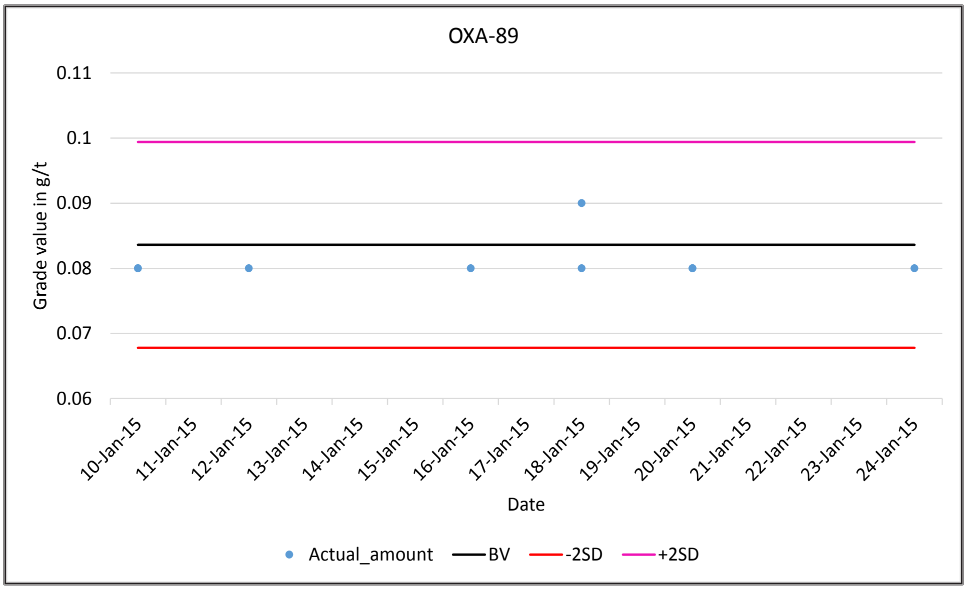
Project	DD	RC	RAB	Total
Exploration	3	487	168	658
Advanced grade control	0	416	0	416
Total	3	903	168	1074

Only drillholes that were drilled in the years 2013-2015 have associated QAQC data in the database hereby used in the report.

2. STANDARDS

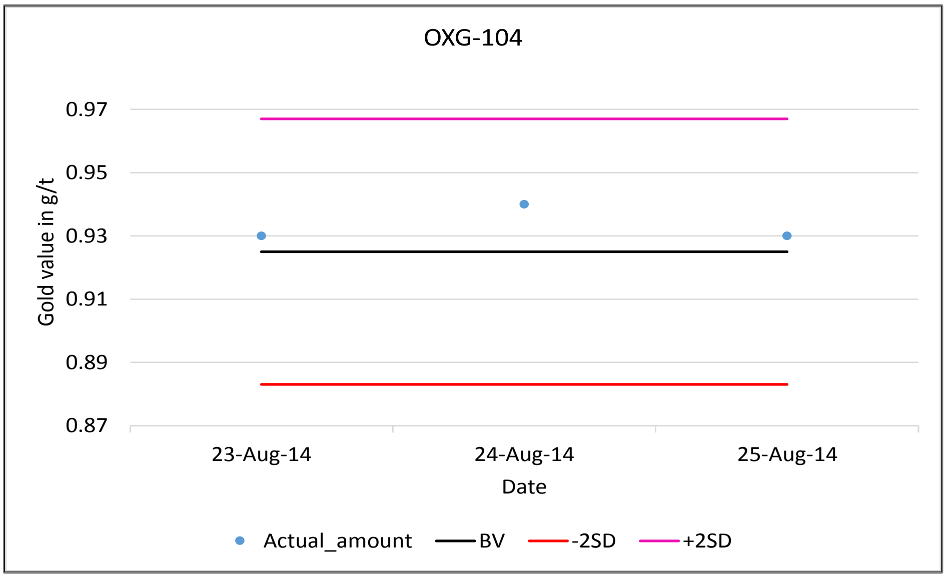
QC results were monitored immediately results were received and sample submissions falling outside the acceptable limits (± 2 standard deviations) were investigated and resubmitted for re-assay if necessary

OXA-89



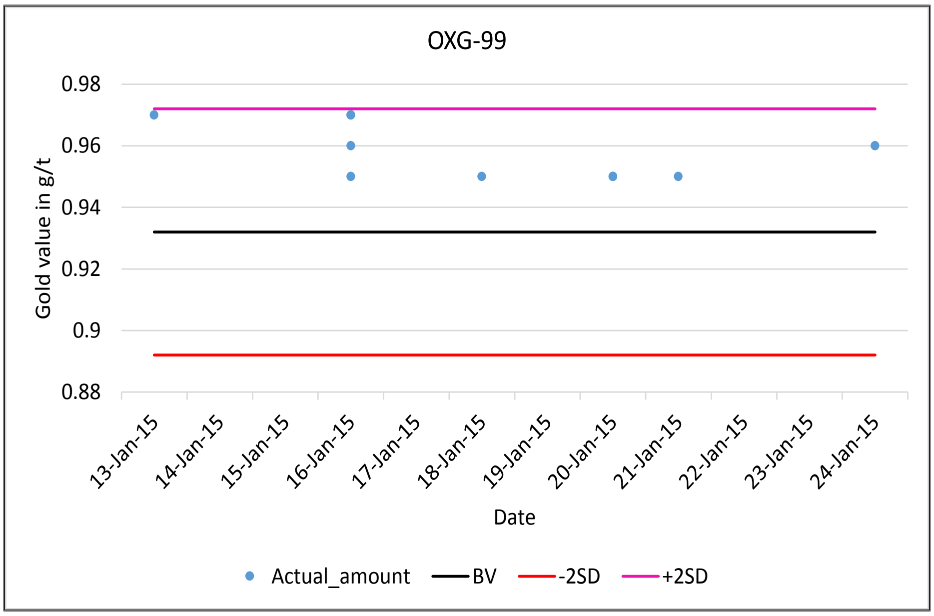
All the 9 OXA-89 standards utilized in FE2 drilling passed QAQC test.

OXG-104



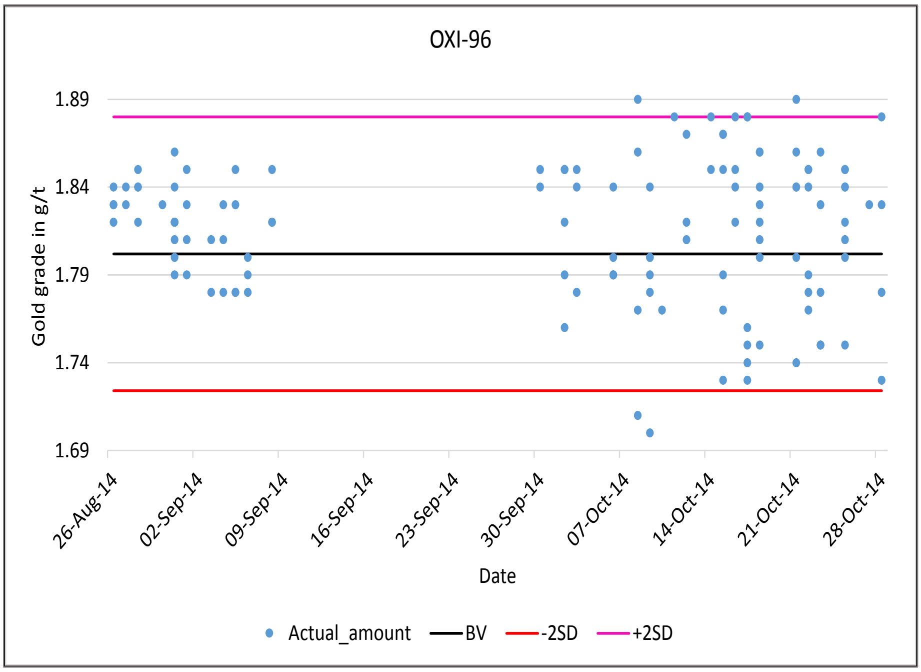
All the 3 OXG-104 standards utilized in FE2 drilling passed QAQC test.

OXG-99



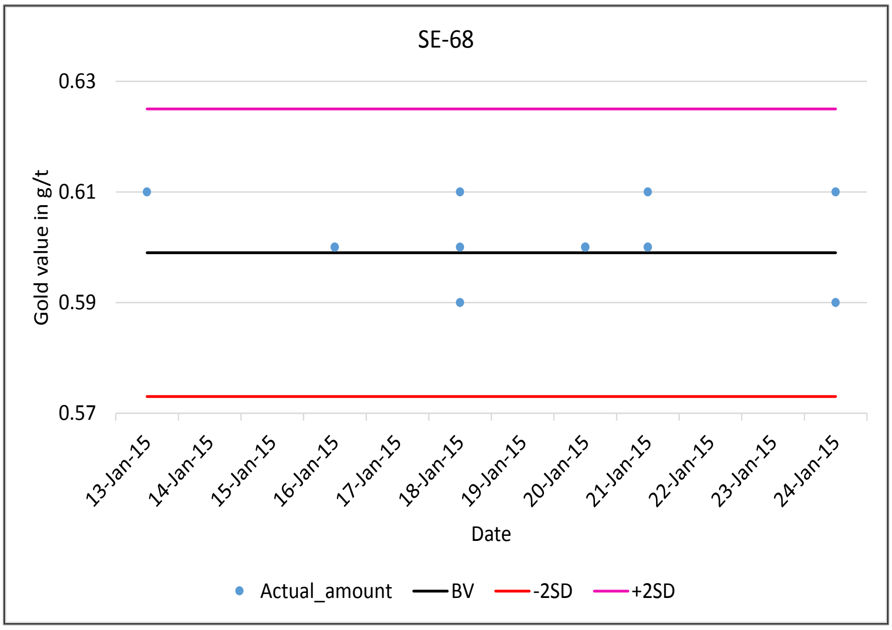
All the 10 OXG-99 standards utilized in FE2 drilling passed QAQC test.

OXI-96



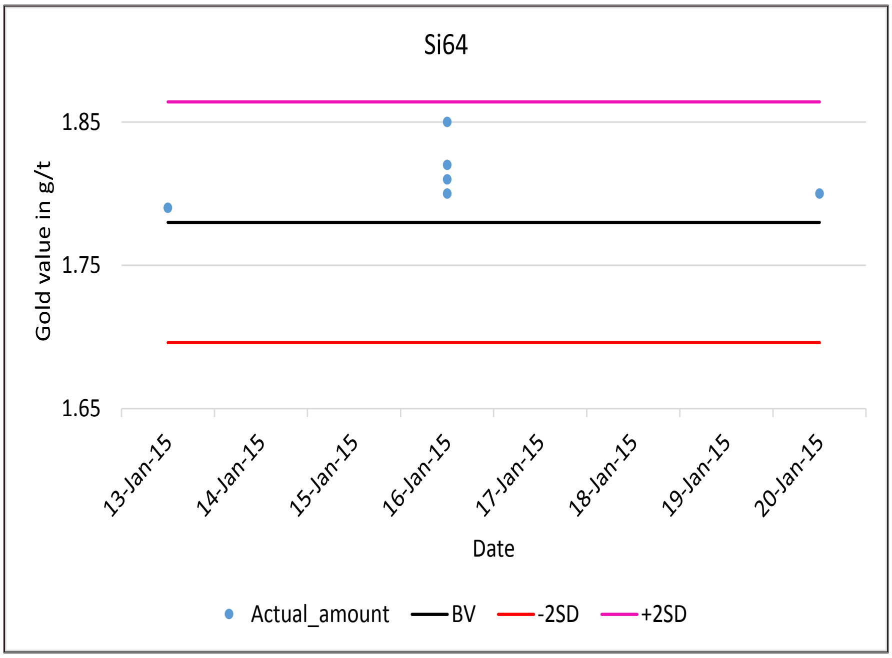
4 out of the 124 OXI-96 standards utilized in the FE2 drilling failed QAQC test.

SE-68



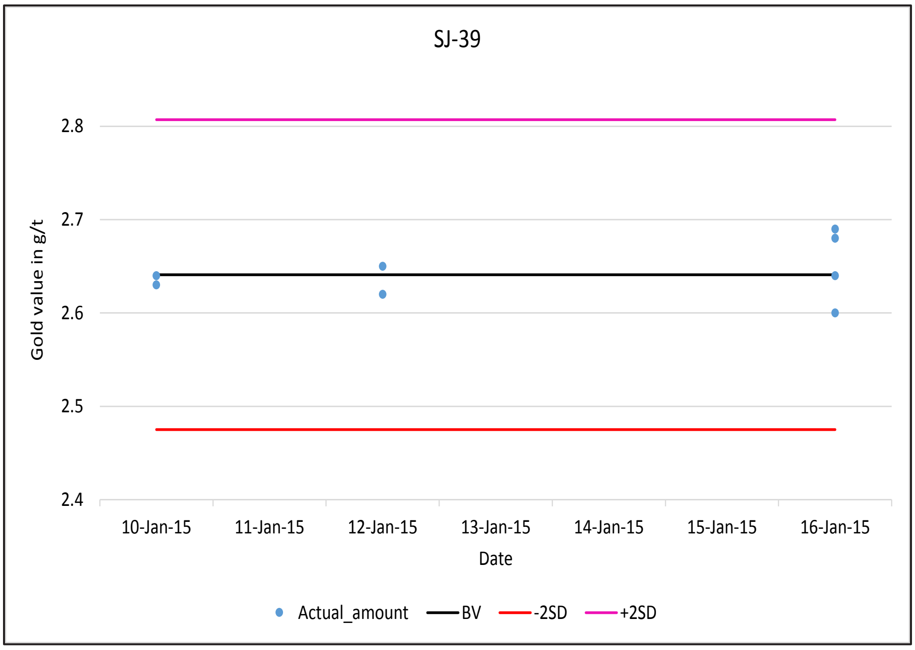
All the 14 SE-68 standards utilized in the FE2 drilling passed QAQC test.

SI64



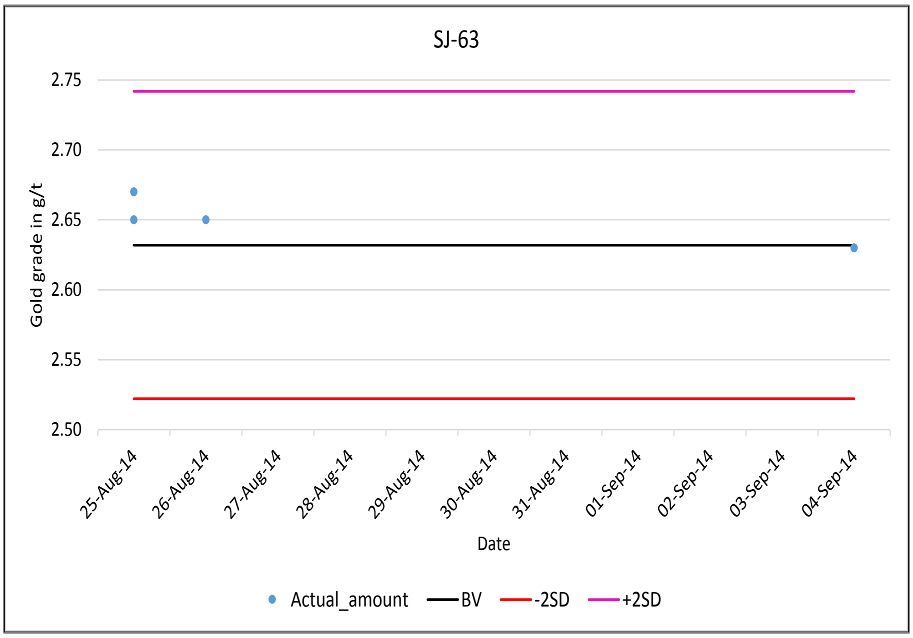
All the 6Si64 standards utilized in the FE2 drilling passed QAQC test.

SJ-39



All the 8SJ-39 standards utilized in the FE2 drilling passed QAQC test.

SJ-63



All the 4SJ-63standards utilized in the FE2 drilling passed QAQC test.

BIAS ESTIMATE TABLE

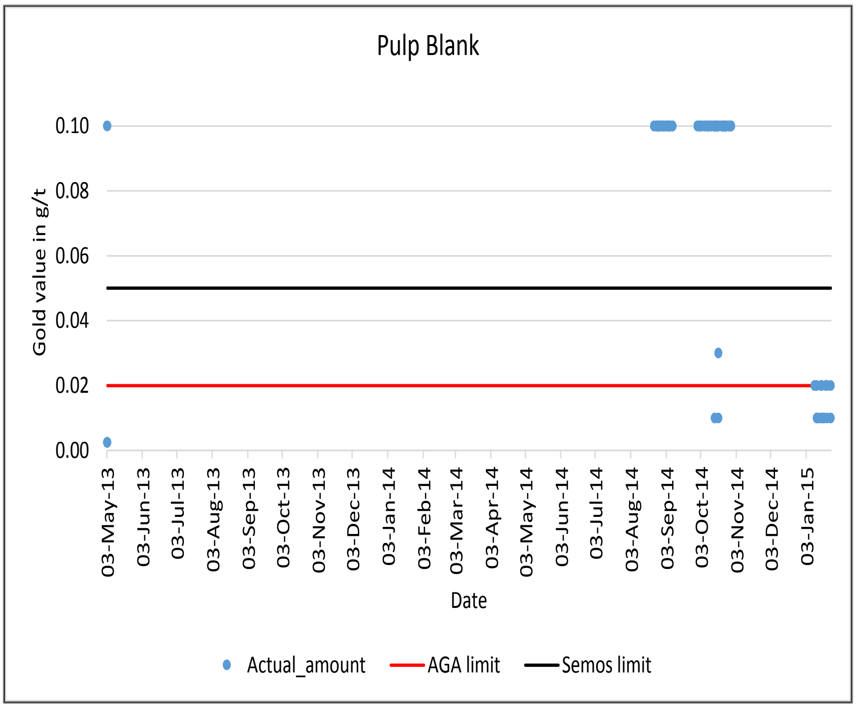
FE2						
STANDARD NAME	LAB MEAN (g/t)	BEST VALUE(g/t)	BIAS %	TOTAL SUBMITTED	% FAILED	COMMENT
OXA-89	0.0811	0.0836	-2.99	9	0	Acceptable
OXG-104	0.9333	0.9250	0.90	3	0	Acceptable
OXG-99	0.9580	0.9320	2.79	10	0	Acceptable
OXI-96	1.8162	1.8020	0.79	124	3	Acceptable
SE-68	0.6014	0.5990	0.40	14	0	Acceptable
Si64	1.8111	1.7800	1.75	6	0	Acceptable
SJ-39	2.6437	2.6410	0.10	8	0	Acceptable
SJ-63	2.6500	2.6320	0.68	4	0	Acceptable

By rule of thumb, acceptable bias should be ≤ 5%. Overall good accuracy performance measured for standards utilized on the FE2 project.

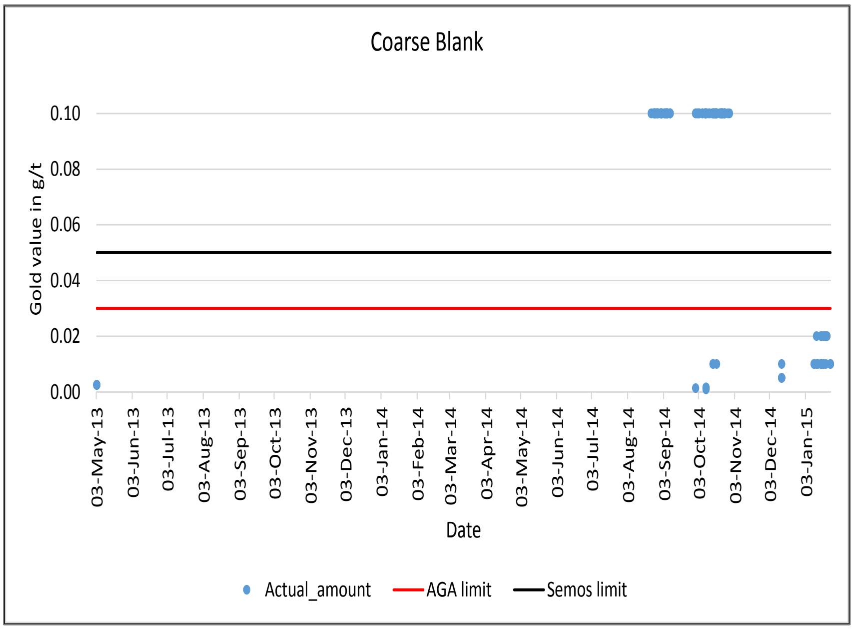
3. BLANKS

Blank insertions for all the FE2 sample submissions are sourced from the barren Sourokoto sandstone located near the SEMOS village. The blank material from this area has been used historically and proven barren during this period. Lots (1kg) were crushed to <6mm and inserted into FE2 drilling sample streams together with 30g pulverised materials.

PULP BLANK



COARSE BLANK



Assay results values of samples with assay values ≤ 0.05 ppm were being rounded up to 0.1 ppm by the BLOY LIMS query software. The situation did not impact other results.

Apparently, the problem was known and a solution has been offered by the release of an updated software. Unfortunately, the updated software was not implemented on the grade control computer. Corrective action taken by updating the software to appropriate version.

Sample swapping or contamination not identified from the performance of the coarse and pulp blank results.

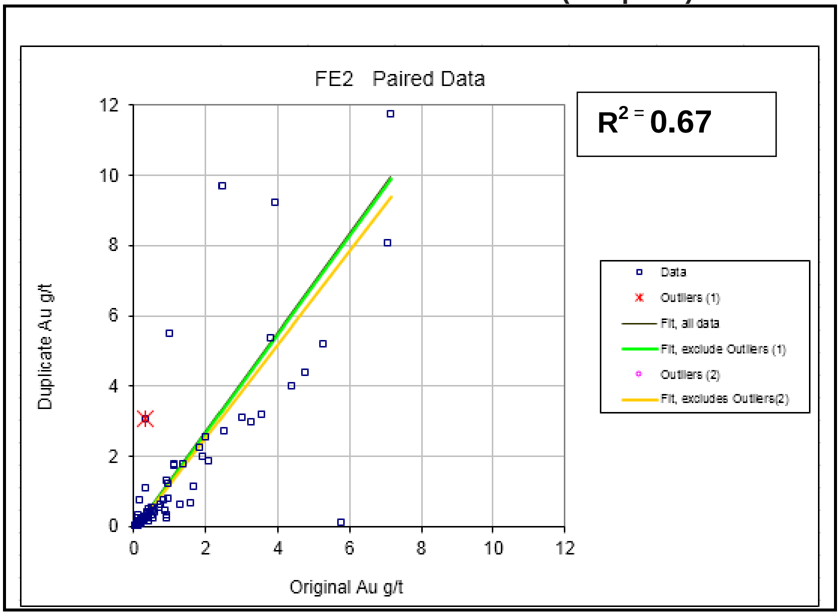
4. DUPLICATES

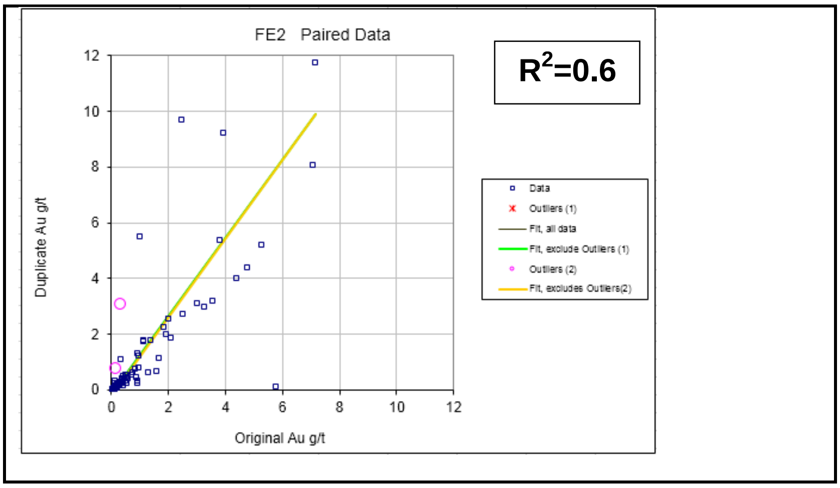
367 paired data of coarse (rig) duplicate samples available for data analysis.

SCATTER PLOT - RIG DUPLICATES

Due to limited number of paired data available; linear relationship is investigated below for the entire datasets without detection limit values and outliers removed.

All data without detection limit values (170 pairs)





1.4% Outliers removed (168 pairs)

Outliers were removed using the AGA outlier tool. The estimated linear relationship is 0.69 which is far from the ideal distribution value of one (1).

5. RATIO OF QAQC MATERIALSTO PROJECT SAMPLES

Proportion of QC-material to project samples tabulated below.

SAMPLES AND QC MATERRIAL SUBMITTED – FE2			
CRM or SRM	Number submitted	% of Samples submitted	Comment
Standards	178	<1	< QAQC Rev 1.05 guideline of 2014 recommended level of 2%
Pulp blanks	390	1	Within QAQC Rev 1.05 guideline of 2014 recommended level of 1%
Coarse blanks	247	<1	< QAQC Rev 1.05 guideline of 2014 recommended level of 1%
Field duplicates	367	1	Within QAQC Rev 1.05 guideline of 2014 recommended level of 1%

Check assays	NA	NA	included in the 2013 annual check assay program
Project samples excluding RAB samples	28024	% estimates excludes RAB samples and based on GC recommended levels.	
Project samples	29989		

Pre 2013 drillholes do not have associated QAQC data stored in database.