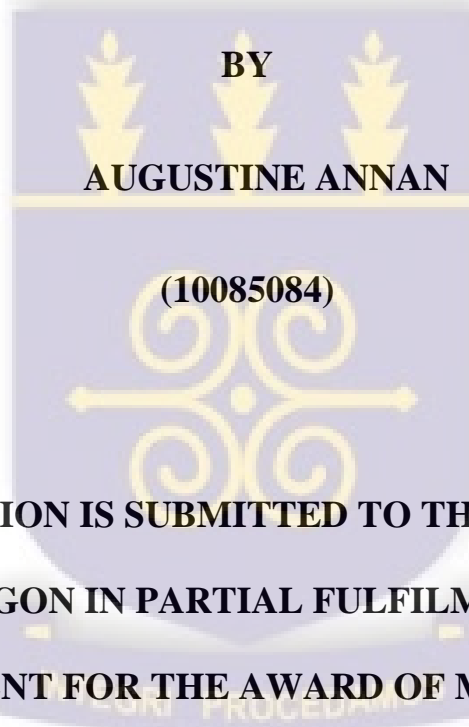


**UNIVERSITY OF GHANA - LEGON**

**GEOSTATISTICAL DETERMINATION OF SMALLEST MINING  
UNIT (SMU) FOR DAMANG MINE, GOLD FIELDS GHANA  
LIMITED-A CASE STUDY AT THE SADDLE PIT**



**BY**

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**THIS DISSERTATION IS SUBMITTED TO THE UNIVERSITY OF  
GHANA, LEGON IN PARTIAL FULFILMENT OF THE  
REQUIREMENT FOR THE AWARD OF MSC MINERAL  
EXPLORATION DEGREE**

**OCTOBER, 2014.**

## DECLARATION

I hereby declare that this dissertation is the writer's individual and authentic work with the exception of reference to previous work that has been duly cited. This work has neither in whole or part been submitted to this or any other university for a degree.

.....

(Student)

Augustine Annan

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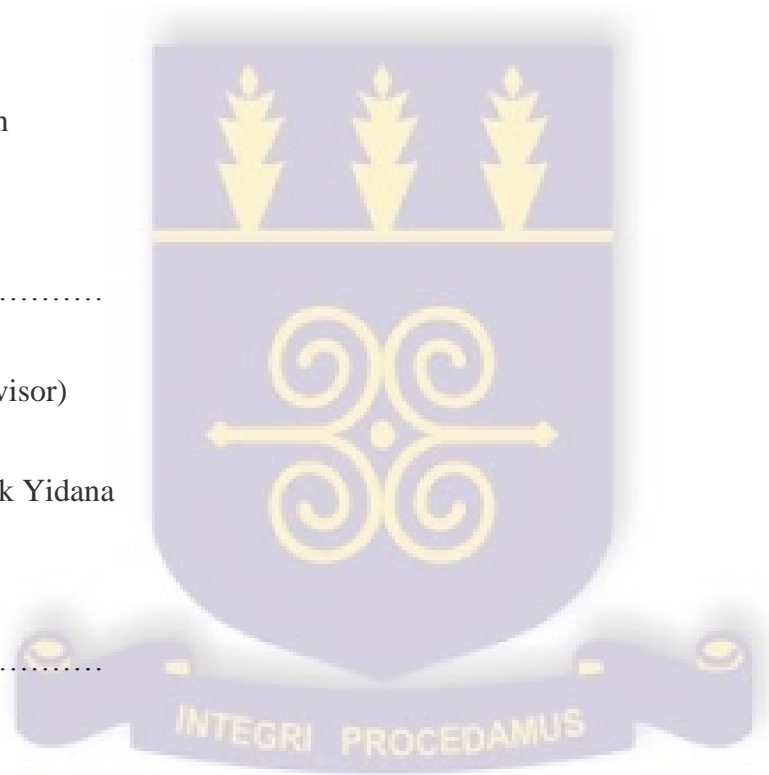
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## DEDICATION

“GREAT MINDS DISCUSS IDEAS; AVERAGE MINDS DISCUSS EVENTS; SMALL MINDS DISCUSS PEOPLE.” — ELEANOR ROOSEVELT

I DEDICATE THIS DISSERTATION TO GOD ALMIGHTY FOR HIS WISDOM,  
UNDERSTANDING AND DIRECTION GIVEN ME THROUGHOUT THIS MASTER’S  
PROGRAMME

AND

MY LOVELY WIFE MRS MILLICENT LAAKO OKLETEY ANNAN



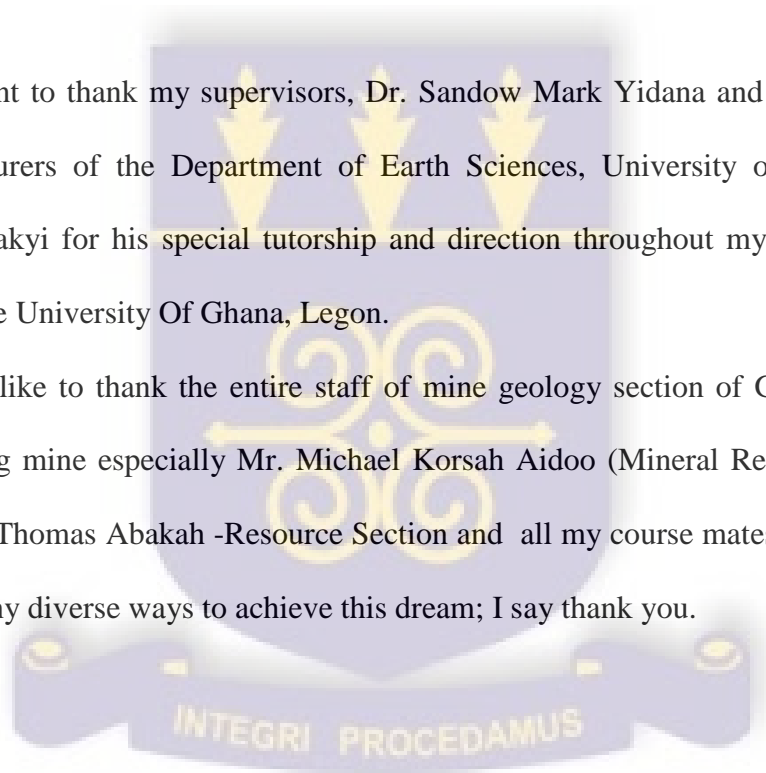
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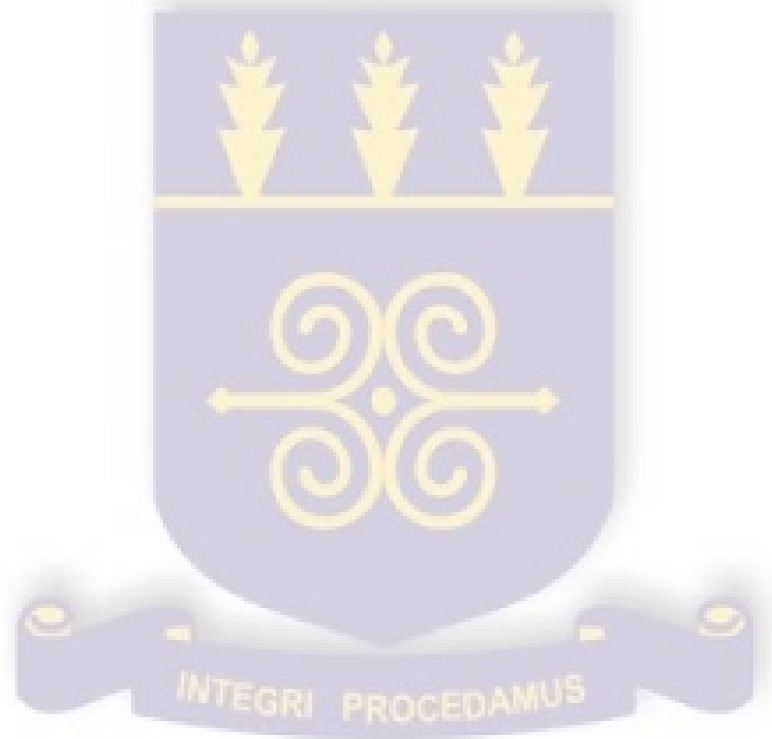
Finally I would like to thank the entire staff of mine geology section of Goldfields Ghana Limited, Damang mine especially Mr. Michael Korsah Aidoo (Mineral Resource Manager-Damang Mine), Thomas Abakah -Resource Section and all my course mates and friends that supported in many diverse ways to achieve this dream; I say thank you.



## ABSTRACT

Historically, the operations of Goldfields Ghana, Damang Mine had generated grade control models using smallest mining unit (SMU) of 2 m x 5 m x 3 m. Management has introduced the use of larger excavators example the Bucyrus RH 90C and RH120E O&K with bucket size 3.5 m hence, there is therefore the need to review the SMU being used for various pits for the Grade Control Modeling Procedures which will relate to the ability of the equipment to select ore or waste with minimal dilution. The calculation of mineral resources and ore reserves from a block model requires the choice of a block of SMU size. Each block is assigned a grade or a distribution of grades. The resources/reserves are calculated from these block values. This dissertation proposes the optimal SMU size to be used at the Saddle Pit. The style of Au mineralization at the study area - Saddle Pit is being hosted within narrow, (<0.5m) shallow- dipping, and laterally discontinuous quartz veins and requires an excavator with smaller bucket size to effectively mine. The orebody is hosted within two main lithologies namely; Huni Sand Stone and Phyllite of the Tarkwaian System. A geostatistical reserve estimation procedure (Ordinary Kriging) was implemented for a range of SMU block sizes (2 m x 5 m x 3 m and 5 m x 5 m x 3 m) and the SMU size that gave an acceptable tonnes and grade of Au was recommended. The optimal SMU sizes will yield an estimate of tonnes and grade of ore that is close to the actual production. The krig efficiency of both SMU (2 m x 5 m x 3 m and 5 m x 5 m x 3 m) Block Models falls within 45% and 81% whilst the regression slopes is within 0.7 to 0.99. Although both cases are within acceptable limit of 40% and 0.4 for krig efficiency and regression slope respectively, the 2 m x 5 m x 3 m proved to be better for the Saddle Pit. When no cutoff grade was applied, the undiluted 2 m x 5 m x 3 m grade control block model reported 2,454,458t @ 0.43g/ t which gave a metal content of 1,050,465.56g whilst the 5 m x 5 m x 3 m grade control block model reported

3,033,501t @ 0.28g/ t. which gave a metal content 843,851.6g. This indicated an extra metal of 206,614g representing 19.6% increment for the 2 m x 5 m x 3 m. The mining equipment needed to effectively mine the SMU 2 m x5 m x 3 m due to the style of mineralisation with minimum dilution is the Liebherr 984 with bucket width of 2.0 m. It is recommended that for the Saddle Pit area, SMU of 2m x5m x 3m should be used. The same analysis should be carried out at the main Damang CutBack Pit and Juno Pit to establish which SMU should be used as the style of mineralisation are associated with the banket conglomerate and Tarkwa phyllite with extensive quartz vein density.

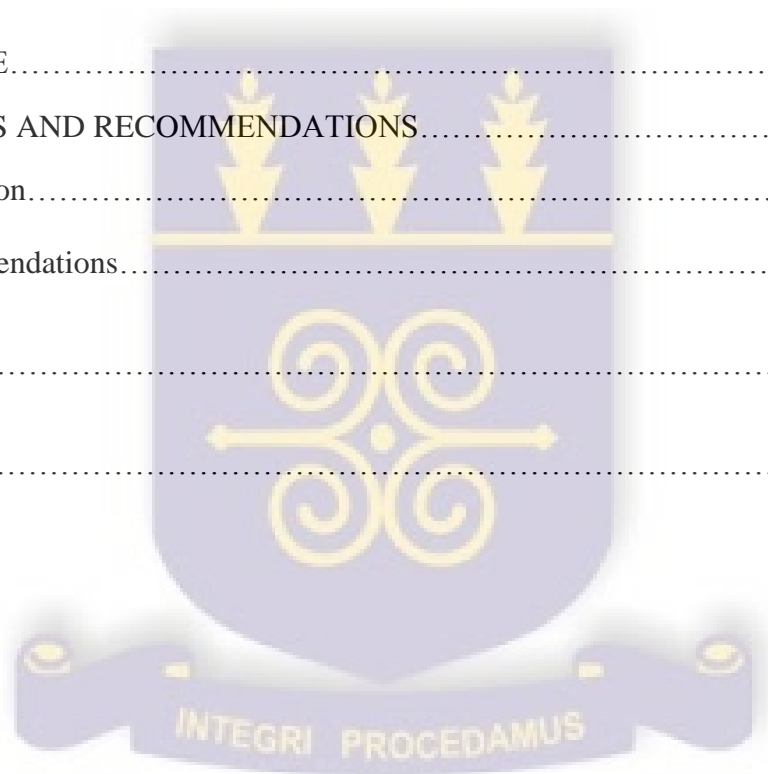


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## CHAPTER ONE

### INTRODUCTION

#### 1.1 BACKGROUND TO STUDY

Geostatistics is more widely accepted and applied method for calculating resources and reserves. Predicting block ore grades from a sample taken at point support was of great interest for D G Krige in the 1950s. Krige predicted areal gold deposit in South Africa based on large amount of data which exhibited strong positive correlation (Cressie 1990).

There is now widespread industry acceptance that mining optimisation studies are best built on resource models that predict the tonnes and grade that will be identified during mining. Ore loss and dilution vary with the scale of mining, notionally represented by the size of the smallest mining unit (SMU).

Most of the deposits in Ghana are estimated by the use of geostatistics for example the Manfo gold deposit in the Brong Ahafo Region. The operations of Goldfields Ghana, Damang Mine, have over the years applied geostatistics in the estimation of the reserves and resources. But there is the need to look at what SMU size is suitable for the various Gold deposits. This dissertation seeks to look at what factors of the estimation method (kriging) to look at when deciding on an SMU to use for a particular style of Gold mineralisation.

Saddle Pit is one of the open cut pits in the Damang concession. Aboso Goldfields Limited (AGL) is a Ghanaian registered Company that owns and operates the Damang Gold Mine which covers a total area of 27,174 ha located in the Western Region of Ghana. Goldfields Ghana Holding Limited holds 71.1% interest in AGL with IAMGold 18.9% and the Ghanaian Government the remaining 10%. Damang Mine manages five Prospecting Licenses as well as two Mining Leases namely the Damang Mining Lease and the Lima South Mining Lease, which equates to 8,111ha (Technical Report on Damang Gold Mine, 2009). The mining lease

was granted to AGL on April 19, 1995 for a period of 30 years, this was amended on 4<sup>th</sup> April 1996. The Prospecting Licence covers the Bonsa Forest, Subiri, Huni Valley, Saponso, Nyankumasi and Bonsa River (Competent Person's Report, 2011).

The Damang Gold Mine exploits oxide and fresh hydrothermal mineralization in addition to Witwatersrand Style palaeoplacer mineralisation. The hydrothermal mineralisation is located in the Tarkwaian sediments and is the only deposit of its kind, located on the eastern side of the Ashanti Belt. The main ore body is hosted by a north to north-westerly plunging antiform developed within Tarkwaian sediments, while all other known palaeoplacer mineralisation is located on the eastern and western limbs of the anticline (Junner et al., 1942).

The Damang plant currently processes 3.8 million tonnes per annum (Mtpa) from blend of approximately 5% oxide and 95% hard rock ore. The hard rock ore is sourced from various open pit mining operations (Huni, saddle and Juno pit) and from existing stockpiles, whilst the oxide is sourced from Lima (Paleoplacer style of mineralisation) and northern part of Huni (hydrothermal style of mineralisation) open pit. The study area currently supports the delivery of hard ore to the plant. The average tones processed (throughput) for the plant: 585tph (tones per hour), 12,800t daily (tones daily) and 365,000t monthly (tones monthly) (Competent Persons' Report, 2013).

AGL has a progressive reclamation plan, whereby, as mining areas become depleted they are immediately rehabilitated through contouring, and replacement of topsoil, seeding, planting, and fertilisation.

The target area for reverse circulation (RC) drilling is developed by reference to grade model that was developed with a software system called leapfrog also using results of previous grade control and exploration drilling to plan the Grade Control (GC) holes. Drilling spacing is 5m across strike and 8m along strike with closer or wider spacing determined based upon the degree of mineralization encountered on the bench above. Depth of each vertical hole is 24m

(at 3m operating height) for the next 8 benches.

Reverse circulation (RC) samples from grade control drilling programme in the chosen area are submitted to the Assay Laboratory for assaying. Quality assurance and quality control is done on the samples by introducing in the sample stream standards reference material, duplicates and blanks to check for accuracy, precision and contamination respectively.

Additional data validation is carried out on the grade control samples in relation to the collar of the hole, the depth drilled, failed sample batches (would be sent for re-assay) ,overlapping holes and missing samples are rectified.

## **1.2 PROBLEM STATEMENT**

Currently the Carbon in Leach (CIL) Damang plant is being fed with ore from the Saddle, Huni and Juno Pit. Saddle Pit (Project Area) is the second highest supplier of ore to the Plant aside Juno Pit. The style of mineralisation of these pits is structurally controlled hydrothermal sulphide mineralisation.

Saddle gold deposit is hosted by metasediments and metadolerites. Two types of gold mineralisation occur at saddle. The conglomerate units of the Banket Series Quartzite contain paleoplacer gold. However, at Saddle this type of mineralisation constitutes only a small proportion of ore as much of the gold appears to be largely associated with structurally controlled hydrothermal sulphide mineralisation. The hydrothermal gold is associated with pyrite and pyrrhotite disseminated throughout alteration selvages to quartz veins.

Grade control model is performed on assay, collar and survey data from reverse circulation drilling programme. A 3D grade control model is generated using the Resource Estimation System (RES) with smallest mining unit (SMU: 2 m x 5 m x 3 m, 2:X-axis, 5:Y-axis, 3:Z-axis).

The estimation is run using Ordinary Kriging (OK). Ore polygons are generated from the block model with their respective grades and tonnages.

The estimation of mineral resources and reserves from block model requires a choice of Smallest Mining Units (SMU). The resources and reserves are estimated from this block model which is assigned grade and tones. The optimal SMU will yield an estimation of tonnage and grade of ore and waste close to the actual production.

Smallest mining unit could be defined as the smallest volume of material on which Ore and waste classification is determined. In addition, SMU can also be defined as the block model size that would correctly predict the tonnes of ore and waste and their respective grades. (Sinclair et al., 2002)

The SMU depends on a number of different factors including the mining equipment size, the mining method to be used, the direction of mining and the type of ore body to be mined. These SMUs must be related to the ability of the mining equipment to select ore or waste efficiently with minimal dilution.

Thus a particular SMU is chosen for the reporting on resources and reserves. This SMU is generated from block model from a grade control drilling programme. Block estimates is done to determine the amount of ore and waste probability based on the chosen SMU as different SMU would give different tonnes and grade of ore and waste

Historically, Damang mine had run grade control model using an SMU of 2m x 5m x 3m, but management has decided to introduce the use of larger excavators' such as the Bucyrus RH 90C and RH120E O&K. There is therefore the need to review the SMU being used for the Grade Control Modeling Process which will relate to the ability of the equipment to select ore or waste with minimal dilution.

This project seeks to determine the optimal smallest mining units (SMU) needed for actual production by generating a grade control model using SMUs of 2 m x 5 m x 3 m and 5 m x 5 m x 3 m and comparing the tonnages of ore at various cutoff grades as well as the Kriging efficiency and regression slope of the Ordinary Kriging Estimation method to be used.

### **1.3 OBJECTIVES**

The objective of this project includes the determination of the best Smallest Mining Unit that would yield estimates of tonnes of ore and waste, and grade close to the actual production and which equipment's sizes that can best be used to mine with minimal dilution.

### **1.4 THE STUDY AREA**

The study area is located in the Ashanti Belt in the Western region of Ghana, which runs from Axim in the south to Konongo in the north, over a distance of some 240km. It is located some 40km northeast of Tarkwa with good access roads and an established infrastructure. The property is some 280km by road west of the capital, Accra, and 140km by road from the port of Takoradi on the Atlantic coast. The region is densely populated and serviced by well developed electrical and communication networks.

The Saddle gold deposit is hosted by metasediments and metadolerites of the Lower Proterozoic Tarkwaian System (Junner et al. 1942). The Tarkwaian comprises predominantly clastic metasedimentary rocks confined to a north-easterly trending synclinorium. The Abosso-Damang area lies close to the eastern margin of this structural basin, an area which features a number of major regional fold structures. One such structure is the Damang Anticline which hosts the Damang gold deposit and a number of other prospects.

Recently acquired aeromagnetic information and ongoing exploration work around Damang indicates that the geology is more structurally complex than has been previously proposed.

Mine mapping shows that the Damang Anticline structure is a tight, asymmetric fold, plunging at about 15 degrees to the NNE. The near-vertical western limb of the anticline is dismembered by a major fault, locally called the Damang Fault. This is almost certainly a steep, north westerly verging reverse fault with at least 1500 metres of displacement across it. The eastern limb of the anticline has suffered about almost two kilometres sinistral displacement (Brabham, 1998).

Tarkwaian metasediments are cut by dolerite dykes throughout the Abosso-Damang area. The intrusion of these dykes post-dates the main folding event but pre-dates the Damang mineralising event (Sharpe, 2005).

The Saddle ore body is located in Tarkwaian rocks on the eastern limb of the Damang Anticline. This district-scale structure is cut by a WNW-verging thrust fault. The Huni Sandstone on the western limb of the anticline whiles the Banket Series Quartzite and the Tarkwa Phyllite on the eastern limb of the anticline. The anticline plunges at about 15<sup>0</sup> to the NNE. In the Damang mine area (Figure 1.1), extensive metadolerite sills intrude the Tarkwaian sequence (Sharpe, 2005).

The conglomerate units of the Banket Series Quartzite contain paleoplacer gold similar to the Tarkwa, Iduapriem and Teberebie mines near Tarkwa. As mention earlier on in the Problem Statement, the paleoplacer style of mineralisation at the saddle pit forms only a small proportion of ore as much of the gold are largely associated with structurally controlled hydrothermal sulphide mineralisation with pyrite and pyrrhotite disseminated throughout alteration selvages to quartz veins. The veins form a sheeted system in which most veins are flat-dipping and arranged in ore zones that dip to both the east and northwest. Volumetrically, the quartz vein comprises less than 1% of ore and individual veins rarely extend more than 6-8 metres (Technical Report on the Damang Gold Mine, Ghana 2004). Fig1.1 shows the location of Damang Gold Mine and Fig 1.2 indicating the As-built shell of the Saddle Pit Project area.

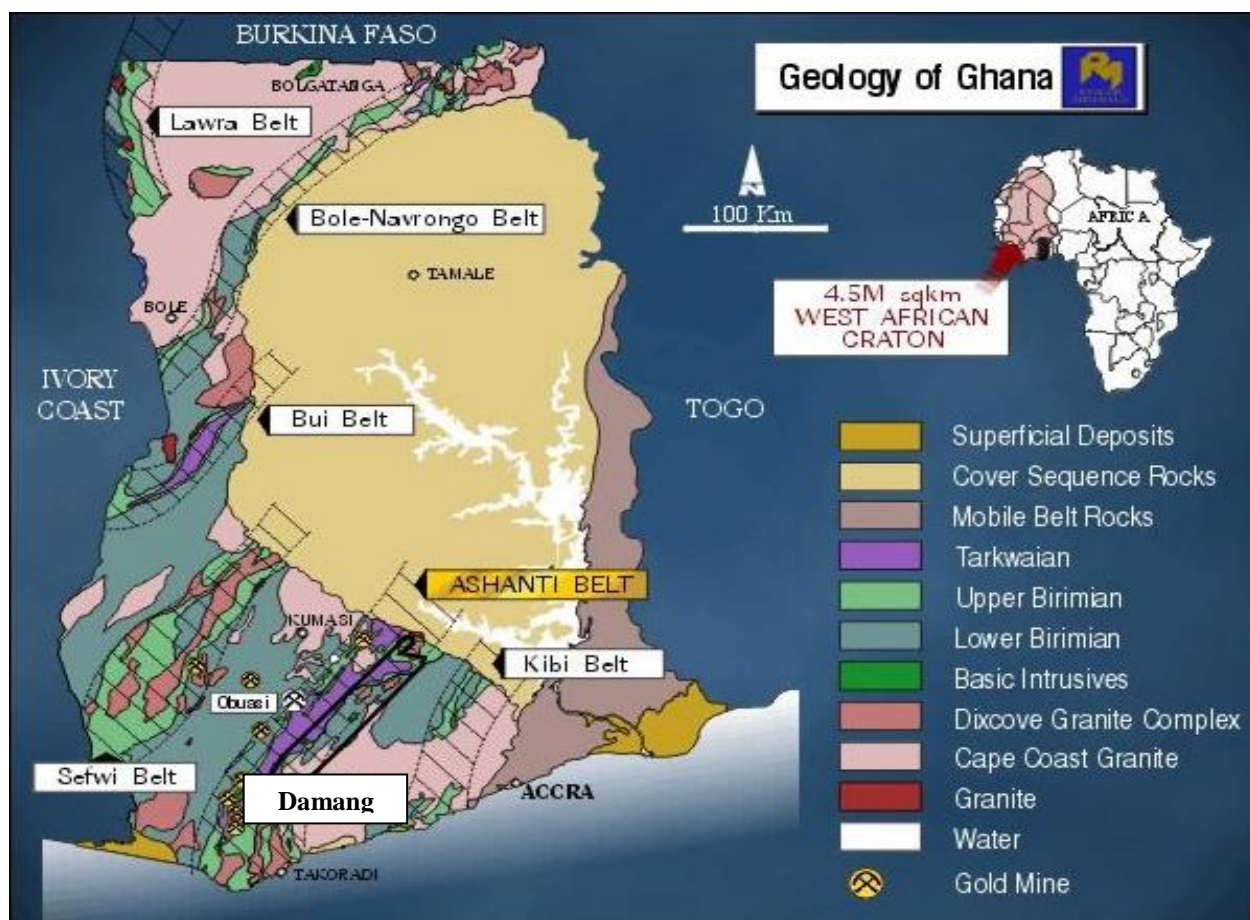


Fig1.1 Location of Damang Gold Mine (Source: White et al., 2010; modified from Pigois et al., 2003)

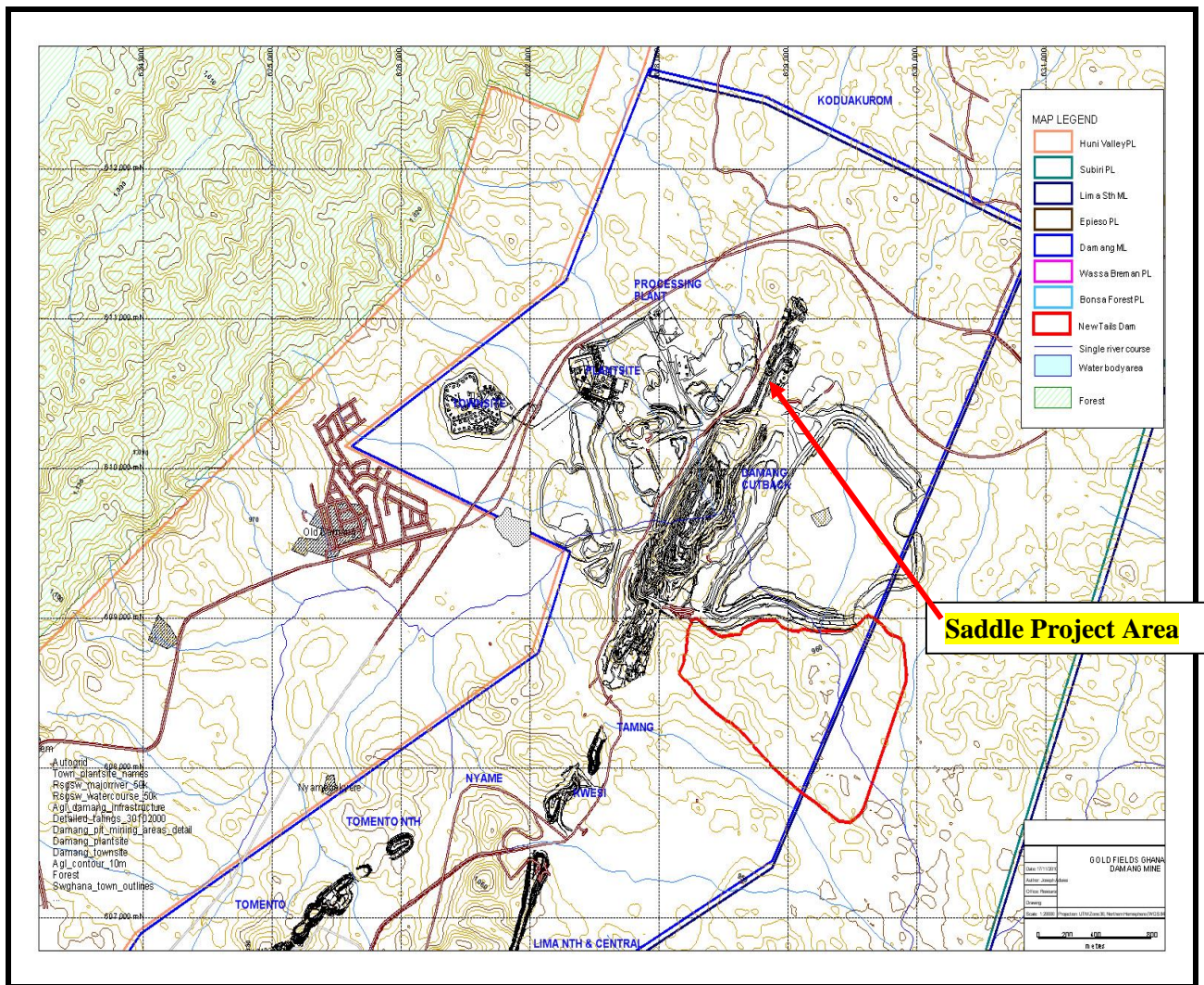


Fig 1.2 Indicating As-built of the Saddle Project Area

## 1.5 GEOGRAPHY

The study area has numerous NNE trending ridges and low hills with elevation of about 175-185m ASL and the adjacent valleys are in the range 70-80m ASL. The flat-topped ridges commonly have laterite cover and are remnant of a regional erosion surface. In the north part of the district, there is quite prominent set of hills with summits commonly just over 300mASL and this elevation level marks an even older, probably early tertiary erosional surface (Addo, 2009).

Most of the area is part of the Bonsa River Drainage system, a major tributary to the nearby Ankobra River system a somewhat smaller tributary of the Ankobra.

The vegetation is a mixture of tropical rain forests and semi-deciduous forest. Deforestation due to subsistence farming by the local population has altered the vegetation in the environs of the mines to secondary forest, scrub and cleared land. No primary forest is found on the concession. Extensive subsistence farming occurs throughout the area with cocoa, plantain, pineapple, cassava, maize, yam and some oil palm and coffee being the principal crops.

## **1.6 CLIMATE AND ENVIRONMENT**

The study area features quite high rainfall, with average monthly temperatures between 21°C and 32°C, is characterised by two distinct rainy seasons from March-July and September-November. Average annual rainfall near the site is 2030mm, whereas in the central and northern parts of the district it probably drops to about 1500-1750mm/yr (Addo, 2009).

There are no specific emissions of gaseous or particulates in the study area, but there are local fugitive emissions (e.g. road dust and bush burning). The area is under the influence of the seasonal dust-laden Harmattan winds. This particulate pollution occurs principally during the three months of the dry season, from December to February, affecting air quality and visibility. Results from the AGL dust deposition-monitoring programme from October 1996 to 2008, show that dust levels both at the Damang mine site and in nearby communities, are within the IFC World Bank Environmental Guidelines for Precious Metal Mining, (Addo, 2009).

Damang is in the rain forest belt of south-western Ghana. Some areas near the mining area that are composed of secondary forest at various stages of development, but in general, the area has been extensively disturbed by subsistence agriculture, by timber felling and by both artisanal and commercial mining activities. The Bonsa River Forest Reserve and the Ben West Forest Reserve, respectively to the southeast and northwest of the Mining Lease Area, have been degraded in the past, and are no longer primary forest areas.

As with the flora, the fauna has been extensively disturbed. Hunting and the destruction of habitats have reduced the numbers of larger mammals to very low levels. The 1996 update of the environmental baseline report noted a number of rare or uncommon bird species, though the majority of these were identified in the adjacent Bonsa River Forest Reserve. This assertion (extensive disturbance) was also confirmed by the Ghana Wildlife Society's biodiversity survey carried out in the Damang / Bonsa Forest Reserve areas.

The Damang Mining Leases (ML1409/96 and ML316/2006) lie within the Bonsa and Huni sub-basins of the Ankobra River main drainage basin. At the Damang mine site, the general flow of the rivers is from the west to the east, towards the Bonsa River. The main rivers in the area are the Tamang, Beni and Ayaasu, and their tributaries. As part of the mine development programme, the rivers crossing the mine site were re-routed to the north and south of the main pit. The southern portion of the Mining Lease drains to the Bonsa River and west into the Huni River. The Ankobra River flows south emptying into the Bay of Guinea, Atlantic Ocean.

The occurrence of groundwater in the concession is generally related to the nature of the geologic formations underlying the area. The total weathered zone consists of decomposed, to highly weathered rock material, overlying the fresh rock formations.

Groundwater recharge, estimated at 3% to 5% of the total annual rainfall, occurs mainly through rainfall and infiltration, and discharge is via springs created through fractures and the streams in the area, (Addo, 2009). There is significant variability in the degree of permeability, and recharge varies from place to place depending on the local geology and topography. The predominant land-use in the area is for subsistence farming and the production of cocoa and oil palm, with a lesser amount of mixed production farming.

## CHAPTER TWO

### LITERATURE REVIEW

#### 2.1 REGIONAL GEOLOGY

The Damang concession is located in Palaeoproterozoic rocks of the Man-Leo Shield, a large area of continental crust that stabilized at about 1850Ma. The terrain features several linear NNE trending belts of predominantly igneous rocks separated by broad basins of granitic rocks and fine-grained metasedimentary rocks. The setting is similar to other Proterozoic terrains of the world and also to many Archaean cratons. The belts of igneous rocks almost certainly represent a series of island arcs that were separated by ocean basins (Davis, et. Al., 1994).

Palaeoproterozoic rocks of Ghana feature two principal sequences: Birimian metavolcanic and metasedimentary rocks of the Birimian Supergroup and overlying clastic metasedimentary rocks of the Tarkwaian System.

The Tarkwaian Group which unconformably overlies the Birimian Supergroup comprises of a thick sequence of folds, faults and metamorphosed sandstones, conglomerates and shales of Palaeoproterozoic age and is found in the Ashanti and Bui Belts of southern and western Ghana respectively (Junner et al., 1942).

The Tarkwaian units are characterised by lower intensity metamorphism and the predominance of coarse grained, immature sedimentary units.

The Tarkwaian system is considered to be of shallow water continental origin derived from the Birimian and associated granite (Kesse, 1985). The rocks were deposited in elongated intra-cratonic basin bordered by granite-greenstone belt of the Birimian system. The sediments were deposited in high-energy alluvial fans entering a steep-sided basin filled with fresh

water. They consist of coarse poorly sorted immature sediments with low roundness typical of braided stream environment.

The Birimian Supergroup with approximate age 2170 Ma (Olson, et al, 1992) comprises thick, isoclinally folded metasediments formed in distal portions of broad marine sedimentary basins; volcanoclastic units interbedded with clastic and chemical metasediments and pyroclastics; and mafic volcanic flows/subvolcanic dykes with pyroclastics and interbedded clastic and chemical metasediments which have been metamorphosed to greenschist or amphibolite facies. Table 2.1 shows the stratigraphy with approximate thicknesses and lithological descriptions of the study area after Junner et al., 1924.

Table 2.1: The Stratigraphy and Lithologies of the study Area (Junner et al., 1924)

<b>Group</b>	<b>Series</b>	<b>Thickness</b>	<b>Lithology</b>
<b>Tarkwaian (2132 – 2096 Ma)</b>	Huni	1,370m	Fine grained massive quartzites with no significant mineralisation
	Tarkwa Phyllite	120 – 400m	Fine grained chloritic siltstones, mudstones and schists with no significant mineralisation
	Banket	120 - 160	Well sorted conglomerates and quartzites with clasts generally considered to be Birimian in origin and containing significant gold mineralisation
	Kawere	250 - 700	Poorly sorted polymictic conglomerates and quartzites with no significant mineralisation
<b>Major Unconformity</b>			
<b>Birimian (2170 Ma)</b>	Birimian		Meta-volcanics, volcanoclastics and sediments

Ages of detrital zircons from Tarkwaian metasediments cluster between 2194Ma to 2155 Ma with the youngest at 2132Ma (Davis, et. al., 1994). Recent U-Pb dating of zircons from samples of Tarkwaian System rocks in the immediate vicinity of Damang mine define a maximum age of sedimentation of 2133Ma ± 4Ma (Pigois, et.al., 2004). U-Pb dating of

hydrothermal xenoliths related to gold mineralisation at Damang yields an age of 2058Ma +/- 5Ma for gold mineralisation.

The age of Tarkwaian System rocks is similar to the age of clastic rocks of the Guyana Shield that host palaeoplacer deposits such as the Jacobina paleoplacer deposit in Brazil (Dallmeyer & L  corch  , 1990).

Fig 2.1 illustrates the South West geology of Ghana which includes the Ashanti belt and the Sefwi-Asankragua belt.

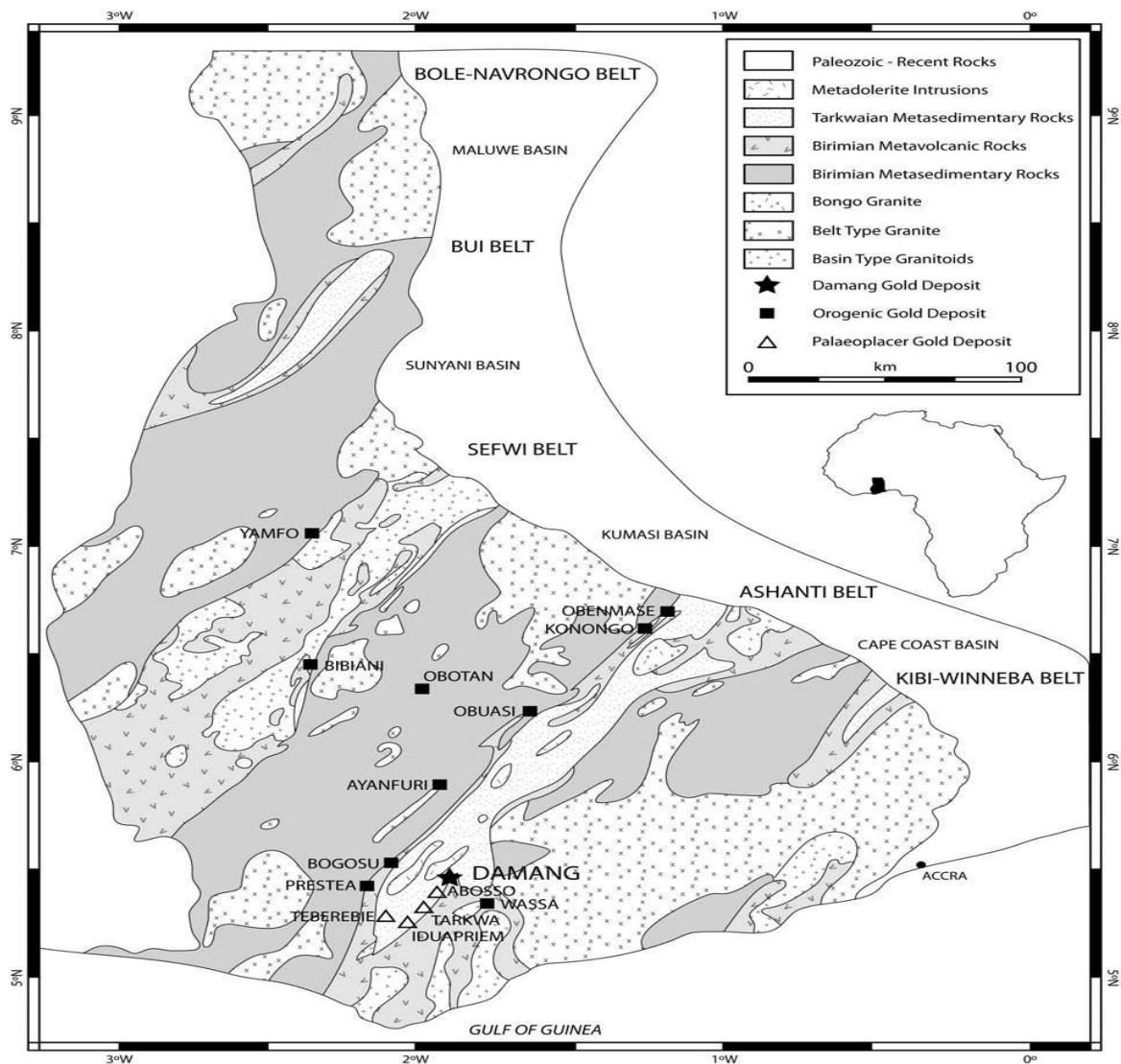


Figure 2.1: Geology of the Ashanti Belt, Southwest Ghana (after Junner et al., 1942)

## **2.2 LOCAL GEOLOGY**

The stratigraphy at Damang is primarily through the Tarkwian group and comprises a large – scale fining upwards sequence of clastic sediments (sediments consisting of broken fragments derived from pre-existing rocks) interrupted by up to four major gold-bearing quartz pebble conglomerate horizons.

The sequence unconformably overlies a mixed Birimian Supergroup basement comprising volcanoclastic deposit and minor fine-grained, clastic sediments and blackshale.

The entire region is intruded by a number of igneous intrusions, the most common being dolerites that occur as narrow dykes and sills-like bodies along contacts on either side of the Tarkwa Phyllites, which is a fined grain pelitic unit in the upper Tarkwian. The second intrusive body, the so called ‘Diorite Porphyry’ occurs sporadically along the boundary between the Birimian and the Tarkwaian.

### **2.2.1 Paleoplacer Mineralisation**

There are three gold- bearing conglomerate horizons recognised on the western limb of the Damang anticline. That is from the footwall to the hanging wall.

Mineralization at Tarkwa occurs within oligomictic, well sorted quartz-pebble conglomerates that are similar to the Witwatersrand conglomerates. However the conglomerates developed at Damang contain sub-rounded to angular clasts and display poorer sorting compared to the conglomerates at Tarkwa. The original mineral assemblages have been extensively altered by a low to medium grade metamorphic overprint. The low grade metamorphic minerals include chlorite, albite, muscovite and quartz.

In contrast to Precambrian quartz pebble conglomerates in the Witswaterstrand, the Tarkwaian conglomerates contain lesser sulphides and a higher proportion of haematite and magnetite. Gold occurs in a free state with these heavy minerals in the matrix of the quartz pebble conglomerates and is typically concentrated at the base of such units. Apart from very small

amounts of sulphide minerals observed near fault zones, there are no significant sulphides associated with the conglomerates. Reefs on the east limb of the Damang anticline feature significantly higher gold grades than the poorly mineralised sandstone units which separates' the reefs.

### **2.2.2 Hydrothermal Mineralisation**

Hydrothermal mineralisation occurs in pyrite and pyrrhotite alteration selvages, which are usually less than 1m wide and located immediately adjacent to en-echelon quartz veins. Gold is also associated with accessory vein minerals such as carbonate, muscovite, tourmaline, ilmenite and apatite. These alteration zones are often linked, and may result in significant volumes characterised by intense veining and gold mineralisation. The Damang Technical Report July 2004 has it that, the morphology of the vein arrays resembles thrust fault arrays, with dominant sole and roof thrust faults representing shallowly inclined veins and steeper veins representing frontal and lateral ramps within the thrust system. The haematite alteration which occurs is overprinted by quartz-sericite-sulphide (QSS) alteration spatially related to the quartz vein system. This QSS alteration occurs as haloes around the quartz vein that may extend to approximately 1m from the vein. Where the veins are closely enough to each other, the alteration haloes may overlap, resulting in volumes of pervasively altered lithologies around densely spaced quartz veins. Gold mineralisation is spatially associated with the quartz vein and associated alteration envelopes.

### **2.2.3 Structure Ore Control**

There are two dominant deposit scale structural features at Damang namely the NE-plunging Damang anticline and the Damang Fault, which is steeply east-dipping reverse fault that cuts through the western limb of the Damang anticline, sub-parallel to the west high wall.

Both structures formed during a long history of progressive NW-SE compression based on detailed structural studies (Brabham, 1998).

High-grade hydrothermal gold mineralisation (>1.0g/t) hosted in arrays of sub-horizontal extensional veins occur in conjugate east and west dipping vein arrays. However, majority of the higher grade arrays are west-dipping. This is based on the distribution of gold grades in vertical grade control drilling. The extensional veins and vein arrays are interpreted to have formed late in the deformational history of the deposit, following a steeply-dipping fold limbs during progressive NW-SE shortening (Damang Technical Report July 2004).

In addition there is a distinctive set of early sub - vertical quartz –carbonate veins in the Saddle Pit, which are cross cut by sub-horizontal ore-stage veins and veins arrays. The sub- vertical orientation of the early vein set, suggest that their formation is associated with axial-planar fractures that developed during folding. This support the interpretation that, the dominant structural feature in the Damang deposit formed as a result of regional progressive NW-SE compression and the formation of the Damang anticline.

In relation to metamorphism, there is textural evidence which suggests that peak metamorphism post-dates dolerite intrusion and predates gold mineralisation. General absence of biotite indicates that peak metamorphic temperature did not exceed 420<sup>0</sup>C. The metamorphism reached Upper Greenschist to Lower Amphibolite facies and resulted in the development of garnet porphyroblasts in sandstone, fine-grained metasediment and dolerite; staurolite porphyroblasts in fine-grained metasediment and less commonly in sandstone; rare kyanite in fine-grained metasediments and actinolite, epidote, ilmenite and rare biotite in dolerite (Junner et al., 1942).Table 2.2 Gives the description of the Saddle Pit Geology.

Table 2.2: Description of the Saddle Pit Geology (Source: Competent Persons' Report on the Mining and Assets of Gold Fields Ghana Abooso Goldfields Ltd, Damang Mine, 2005)

Lithological Unit	Symbol	Description
Siltstone <b>Huni Sandstone equivalent</b>	<b>PSLS</b>	Siltstone and fine-grained sandstone composed of quartz, biotite and muscovite. Contains interbeds of (a) pink and grey coarse-grained sandstone, which occasionally coarsens to poorly sorted, matrix-supported wacke with sparse angular lithic and vein quartz clasts, and (b) fine grained laminated metasediment.
Metasediment <b>Tarkwa Phyllite equivalent</b>	<b>PSLP</b>	Well laminated, fine grained semi-pelitic to pelitic iron-rich metasediment with accessory staurolite, tourmaline and garnet.
Metadolerite	<b>PMDO</b>	Massive, medium to coarse-grained ophitic igneous rock composed of roughly equal parts of actinolite, quartz and albite with traces of carbonate, K-feldspar, epidote, chlorite and opaques (magnetite, ilmenite, rutile and leucoxene).
Undifferentiated grey sandstone <b>Banket Series</b>	<b>PSSU</b>	Medium to very coarse grained, cross-bedded quartz sandstone with accessory sericite, kyanite, staurolite and garnet. Commonly shows haematite-magnetite-ilmenite heavy mineral bands.
Metadolerite	<b>PMDO</b>	As for PMDO above
Conglomerate and pebbly wacke <b>Banket Series</b>	<b>PSSC</b> & <b>PSSG</b>	Lenticular, poorly sorted, matrix supported polymict conglomerates within beds of gritty sandstones. Forms distinct channel deposits of sub-rounded quartz cobbles.
Undifferentiated grey sandstone <b>Banket Series</b>	<b>PSSU</b>	As for PSSU above
Siltstone <b>Huni Sandstone equivalent</b>	<b>PSLS</b>	As for PSLS above

### 2.3 HISTORY OF MINERAL RESERVE ESTIMATION

Since the start of the modern mining era, there has been an increasing use of sample data to estimate the amount of material mined or to be mined. The early reserve estimations arose out of this grade control function, where, for example, underground face samples were used to delineate a mining block. It was only in the middle part of the 20th century that exploration and grade control drill holes started to be used to define a form of mineral inventory. The earliest methods used were all variants of the polygonal estimator, and typically, dilution and recovery were built into the estimation process to define an Ore Reserve (Glacken *et al*, 2001).

As the importance of planning and scheduling of an operation's mineral endowment became apparent, the use of all types of sample data to define Ore Reserves increased. However, there was little regulation of the reporting standards and these tended to vary widely. The methods used to estimate resources also developed, and in the 1950s and 1960s there were a number of attempts to fit simple mathematical models (such as multiple linear regression) to the sample data and thus derive simple statistical parameters such as the mean and variance of the underlying population. In South Africa some success was achieved by fitting lognormal distributions to mine data (Krige, 1978).

These provided for the first time the ability to generate a model of the ore body comprising a large number of orthogonal, similarly-sized blocks. Accompanying this block modelling ability the packages generally offered a variety of resource estimation techniques, including polygons (more strictly a nearest-neighbour interpolator), inverse distance estimation, and simple or ordinary kriging. Although much work was carried out in developing mining geostatistics in the 1960s and 1970s in France and in North America, kriging was not routinely used as a resource estimation technique until the end of the 1970s. The method gained a somewhat tarnished reputation due to its poor application by inexperienced

practitioners, and inverse distance estimation, along with the various varieties of polygonal estimation, established themselves as the methods of choice (Glacken *et al*, 2001).

The mining software packages continued to develop in power and sophistication during the 1980s, and great advances were made in the visualisation and modelling of complex geological domains. It was during this time that the first resource estimates *sensu stricto* were generated. These were then modified by the mining design and other factors to yield Ore Reserves. Following its initial poor reputation, kriging techniques continued to be used increasingly, especially the non-linear approaches such as indicator kriging (Journel, 1983). However, a large number of resources were still estimated using non-geostatistical techniques.

## **2.4 METHODS OF MINERAL RESERVE ESTIMATION**

Mineral resource estimation practice is concerned with the determination of two important parameters. These are the volume or tonnage of ore and the mean grade. The practice involves the manipulation of grade and volumes of actual localised samples to predict the grades and volume or tonnage of a larger block. The process employs two basic concepts (David, 1977): The concept of extension and the concept of error of estimation.

For the concept of extension, the attributes of a sample are extended to the block to be estimated. Based on this principle, reserve estimation methods may be grouped under the main heading of traditional or classical methods and distance weighting methods.

As stated by David (1977), due to the simple fact that samples and blocks are not strictly equivalent, there is an error involved in any estimation procedure. The estimation of this error is to provide estimates that will yield the smallest deviation from reality is what is practiced under the concept of error of estimation. Here, Kriging attempts to model this error. Methods for resource estimation are generally divided into (1) traditional methods such as geometric,

polygonal and inverse-distance-weighting and (2) kriging.

#### 2.4.1 Geometrical Methods

These include cross-sectional, triangular and polygonal methods. In these methods, the grade of a portion of a drill hole is assigned to a block.

For cross-sectional method, ore zones are delineated by drawing envelopes or perimeters based on a cut-off grade on regularly or irregularly spaced cross-sections across the deposit as depicted in Fig. 2.2. The area of ore in each perimeter is determined by using any of the following: Planimeter, Digitisers, counting of squares, Calculation using formulas such as Simpson's rule and prismoidal formula.

The individual areas are totalled for each cross-section and the volume computed using the distance between sections. The volume can be obtained by calculating the volume of influence of each cross-section using the formula:

$$V = A \left( \frac{d_{i-1} + d_{i+1}}{2} \right)$$

Where,  $V_i$  and  $A_i$  are the volume and area respectively of any cross-section, with  $d_{i-1}$  and  $d_{i+1}$  being the distances between the sections to the left and right of the section under consideration. The sum of the individual volumes so obtained gives the total volume of the deposit. The volume is then converted to tonnes by the application of the appropriate tonnage factor or density.

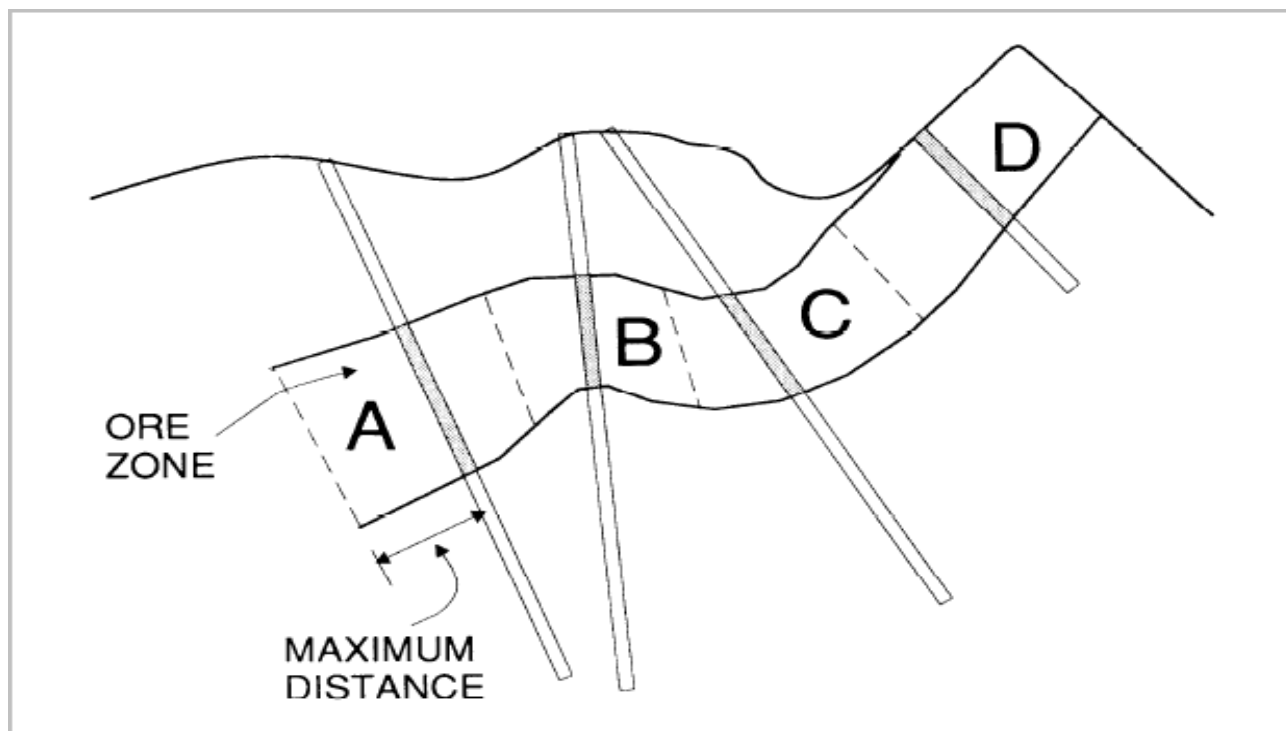


Fig. 2.2 Sample Cross-section Delineated for Reserve Estimation (Alan, 1996).

The mean grade of each section is calculated by weighting the sample grades by their lengths. The global mean grade is arrived at as a weighted mean of the grade of the sections by area, volume or tonnage using:

$$G_g = \frac{\sum(T_i + G_i)}{\sum T_i} \quad \text{And} \quad T_g = \sum T_i$$

Where,

$G_g$  = the global mean grade;

$T_g$  = the global tonnage;

$T_j$  = the tonnage of each cross-section; and

$G_j$  = the mean grade of ore material in each cross-section.

With the triangular method, each drill hole is positioned at the vertex of a triangle. The width and grade of a block of ore are taken to be equal to the average of its three vertices. This is a modification of the polygonal method in which the drilled area is divided into a series of triangles by connecting adjacent drill holes with the lines as illustrated in Fig. 2.3.

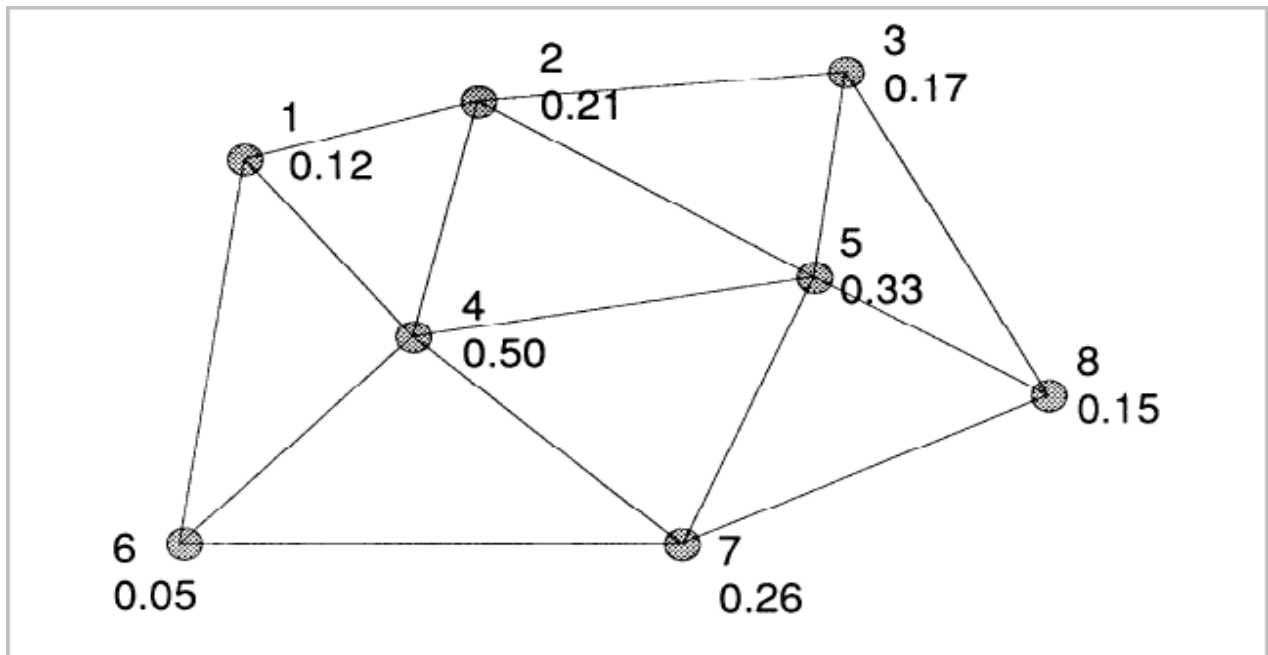


Fig. 2.3 Construction of Triangular Blocks for Resource Estimation

The method has the advantage that, by means of geometry and coordinates, areas are easily calculated. The grade above cut-off grade and the average grade are calculated for each drill hole. By weighting the average grade of each drill hole by the thickness, the grade and average thickness of each triangle is calculated. The volume of material within each triangle is evaluated by multiplying the area of the triangle by its average thickness. By applying tonnage factor or density, the tonnage of ore material within a triangle can be computed. The global grade,  $G_g$  and Tonnage,  $T_g$  of the deposit can be derived from the relations below:

$$G_g = \frac{\sum(T_i * G_i)}{\sum T_i} \quad \text{And} \quad T_g = \sum T_i$$

The  $G_i$  and  $T_i$  are the average grade and tonnage of each triangle respectively.

For the polygonal method, the assumption that the area of influence of any sample point or drill hole extends halfway to the adjacent sample point is made use of by constructing perpendicular bisectors to lines joining drill holes. These perpendicular bisectors are linked to form the sides of a polygon enclosing a drill hole as shown in Fig. 2.4

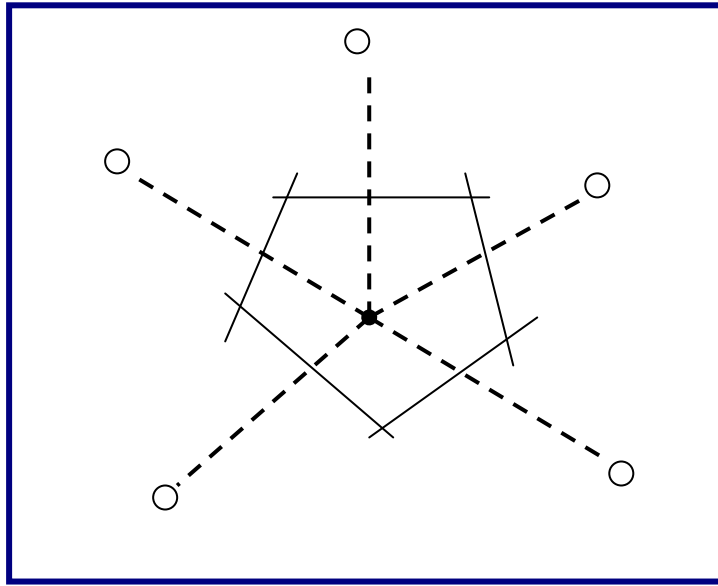


Fig. 2.4 Perpendicular Bisectors Linked to Form Sides of a Polygon

The average grade and thickness of each drill hole are assigned to the polygon enclosing it. The volume estimate within a polygon is established by multiplying the length of the ore intersection in the hole by the polygonal area. By applying the appropriate tonnage factor or density, the tonnage of material in each polygon is determined. The same relations are employed to compute the global grade and tonnage of the deposit being estimated.

#### 2.4.2 Distance Weighting Methods

These methods include Inverse Distance Weighting (IDW) and kriging. Under these methods, linear combinations of grades of surrounding samples are assigned to a block or to a point.

In the Inverse Distance Weighting (IDW) method, the grade of each sample is weighted by the inverse of the distance it makes with a reference point raised to a power. The value of the power applicable to most deposits is two (2) applied in the IDW method. The following formulae are applicable:

$$g_g = \frac{\sum \left( \frac{1}{d_i^n} g_i \right)}{\sum \frac{1}{d_i^n}} \quad \text{And} \quad \sum \left( \frac{\frac{1}{d_i^n}}{\sum \frac{1}{d_i^n}} \right) = 1$$

Where,  $g_i$  = the grade of sample point  $i$  at distance 'd' from a reference point 'g' and  $g_g$  = the grade of the block or point being estimated. As shown in Fig. 2.5, the grade of some reference point or block can easily be estimated using the above method from points of known grade should the coordinates of these points be known.

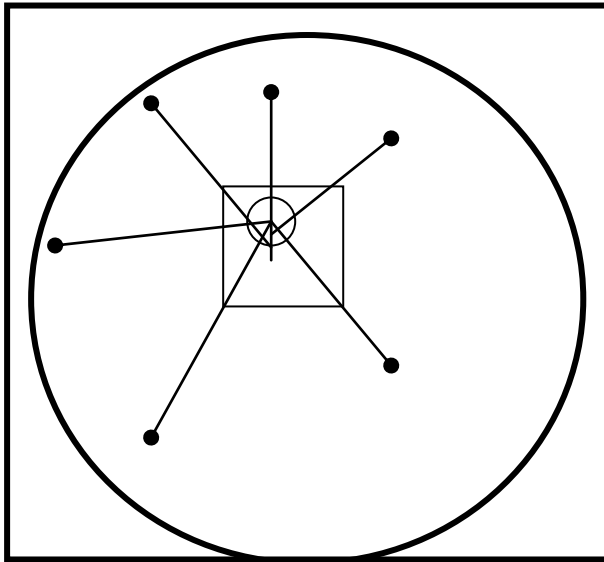


Fig. 2.5 Estimation of a Block using Inverse Square Distance method

## 2.5 ANALYSIS OF GRADE CONTROL DATA USE FOR SMU DETERMINATION

Computation of basic statistics and evaluation of grade distribution are first quantitative analysis of the geologic analysis and input to the ore block modelling. Important factors in these basic studies include.

- Detection of high-grade or low-grade outlier values.
- Evaluation of the favourability of different lithologies as host rocks.

- Differentiation of complex grade distribution into simple populations for ore block modelling.
- Identification of highly skewed and/or highly variable grade distribution that will be difficult to estimate.

The first step is the production of histograms and frequency distribution curves to ascertain the overall impression of the nature of the assay distribution. Of particular importance is the determination of the normality or approach to normality of the sample population under consideration, since, a normal distribution is one of the many distributions for which concise mathematical descriptions exist and has properties, which favour its use in theoretical approaches to estimation. For a normal population, the arithmetic mean or the median, i.e. 50% value, is regarded a good estimator of mean grade.

If the points obtained by plotting the cumulative distribution of the sample assay values on normal probability scale can be considered as lying on a straight line, the assumptions of normal distribution can be accepted (Rendu, 1981).

A distribution can be either symmetrical or asymmetrical with respect to some axis. It may be multimodal with modes belonging to the same mode that is poorly defined or may have distinct modes belonging to different populations with different means that were blended or mixed together by some geological process (Marsal, 1987).

The understanding of the shape of the distribution and the need to separate mixed populations for separate analysis is very paramount in establishing the best possible grade and tonnage estimates of mineralisation within an ore body. The visual detection and treatment of possible abnormally high or low sample values called outliers is enhanced under a histogram plot of the data. Due to the presence of these abnormally high or low values the arithmetic mean of assays may tend to overestimate or underestimate the mean grade of the deposit. Under this condition the distribution is said to be skewed.

## 2.6 GEOSTATISTICAL ESTIMATION OF GRADE CONTROL MODEL

In the early 1960's, George Matheron of Paris School of Mines(Matheron,1992) developed a general solution to the problem of local estimation that built upon an empirical solution developed by South African mining engineer D.G. Krige. To honour Krige's pioneering contribution in this field (Krige, 1951), Matheron named the new technique he developed *Kriging*.

Based on the error of estimation principle, geostatistics has been identified as the only method of estimation that recognises the spatial control or existence of trend in mineralisation. It offers a way of describing the spatial continuity and patterns in the variation of sample grades through a mineral deposit. Matheron(1962) defined geostatistics as the application of formalism of random functions to the reconnaissance and estimation of natural phenomenon. A natural phenomenon is characterised by the distribution in space of one or more variables called regionalised variables (Journel and Huijbregts, 1978). In addition, Barnes (1980) defines geostatistics as the application of the theory of regionalised variables to the study of mineralised volumes of rock. A regionalised variable is a variable whose magnitude depends on neighbouring values, which are distributed in two or three-dimensional space.

There is bound to be error in any reserve estimation, since they are realisation of values derived from manipulation of only few samples to represent a whole deposit. If this error can be found, the true value of the variable can be calculated. According to David (1977), the geological parameters that account for the presence of these errors, which must be quantified to develop a sound model for reserve estimation are:

- The continuity of the ore. If one knows the grade that exists at a given place and assumes that the grade at some distance is the same as this known one, then the higher the continuity the smaller the error.

- The zone of influence of a sample. This is highly related to the notion of continuity and is known by experienced geologists as the distance beyond which any prediction of grade is completely hazardous. This distance may vary according to the direction in which the prediction is to be made. This phenomenon is called anisotropy.
- Low-scale variations. In some mineralisation, small samples may exhibit very large variations over very short distances as in some gold deposits. To overcome this phenomenon one has to take large samples. The error will vary with the size of the sample as sample is not just grade but also a volume of material. The word "support" is used to designate the shape and volume of a sample.
- Homogeneity of the mineralisation. In a deposit where different areas of different geological characteristics exist, the error associated with one estimation procedure will vary if these geological conditions vary. It should remain the same whatever the location of the block if the area considered is homogeneous. Geologists usually recognise such homogeneous subsets in the deposit.

These geological parameters mentioned above which account for the error in mineral reserve estimation problems are accounted for in geostatistics under the semi-variogram by computing the variance of the error of estimation. According to Royle (1971), geostatistics has two objects. These are:

- To estimate the most likely value of blocks of ore, or the value of the whole deposit. .
- To estimate the errors of such estimates.

## **2.7 GEOSTATISTICAL ESTIMATION OF GRADE CONTROL MODEL**

### **PROCEDURES**

The procedure for making geostatistical mineral reserve estimation can be divided into two

parts (Clarke, 1980).

The first part consists of investigation and modelling of the physical and statistical structure of the ore body for which the estimation is being made. This is a very important step, since it is at this stage that the decision as to whether to use kriging in estimating the resources of the mineral deposit is taken. Concepts of continuity and structural variability of the deposit are embodied in what is called a semi-variogram model. This model acts as a qualitative summary of the available structural information, which is then channelled into the various estimation processes.

### 2.7.1 Variogram Modelling

Variography is a synonym for geostatistical structural analysis. Structural analysis is the procedure by which the structure of the spatial distribution of the regionalized variable is characterized through the semi variogram (Al-Hassan, 2004). It is basically a concept of analyzing differences between samples to be able to predict differences between estimates and true values. The semi –variogram,  $\gamma(h)$

$$\gamma(h) = \frac{1}{2N} \sum_{i=1}^N (f(x_i) - f(x_{i+h}))^2$$

Where, N is the number of experimental pairs  $\{f(x_i), f(x_{i+h})\}$  of data separated by a vector h.

A variogram summarizes the factors affecting accuracy as measured on specific supports. Samples represent average values measured over specified geometrical shapes oriented in directions. As the size increase it will tend to encompass more of the structures which give rise to variability. In particular it will tend to encompass the very small causes of variability that give rise to the nugget variance. The nugget variance is in fact inversely proportional to the volume of the sample. The nugget effect is the discontinuity of the variogram at the origin and it measures the importance of the random factor in the ore body (Annels, 1991).

### **2.7.2 Theoretical Semi-variogram Models**

The experimental variogram confirms only the mean spatial variation of certain definite distances. To make the variogram capable of estimating the variations of any other distance, it has to be modelled. The applicability of such a model is validated by using the parameters of the model to try to reproduce or estimate sample grades or values under the method of point Kriging or cross-validation. The actual sample grades or values are regressed on the estimated. The semi-variogram model is accepted if the coefficient of correlation is very close to one (1) and the points lie very close to the 45° regression line. Figure 2.6 show a summary of the few semi-variogram models that have been recognised.

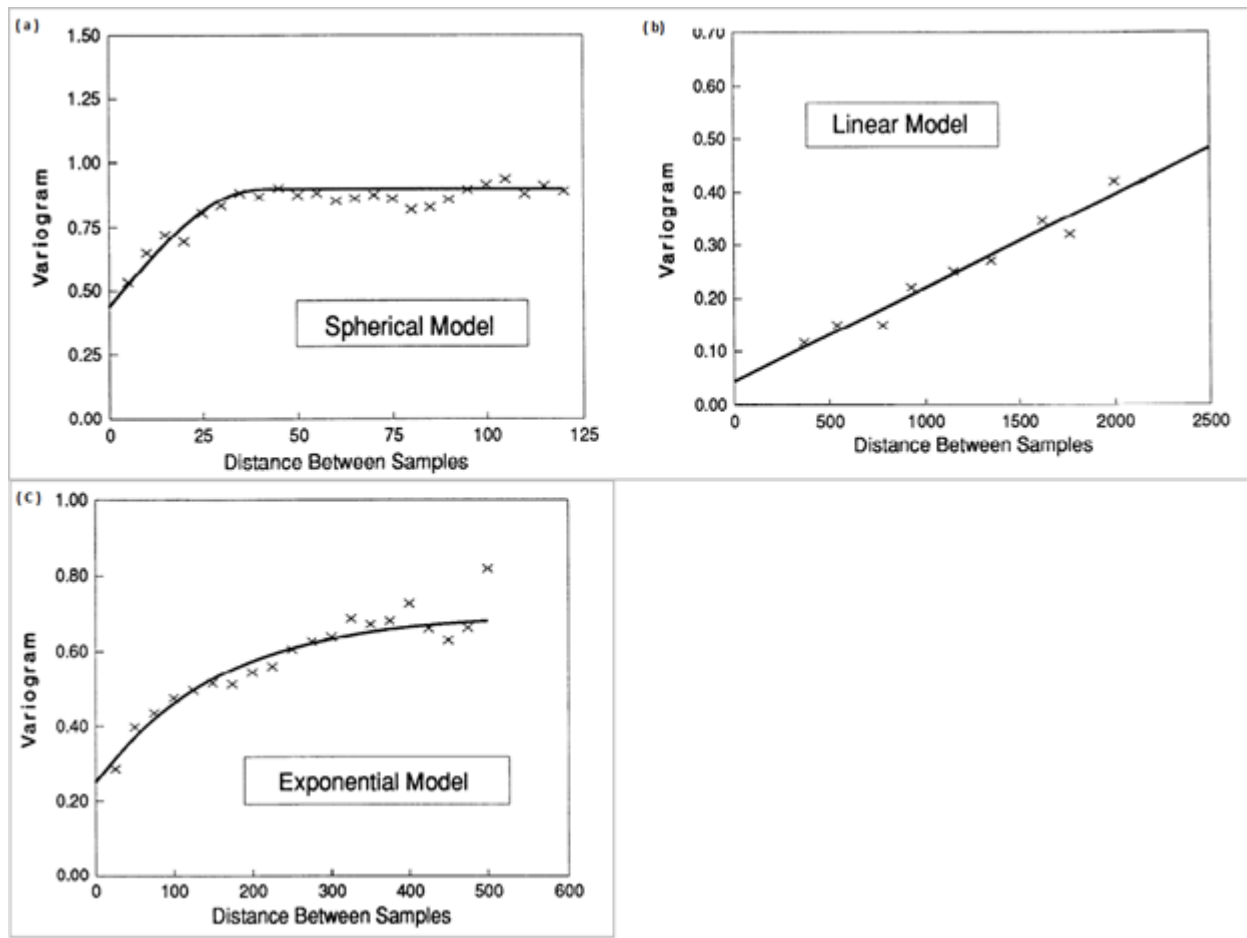


Figure 2.6 Illustration of semi variogram models

Among the models listed, the spherical model has been found to be the most commonly used for geostatistical mineral reserve estimation works (John, 2001). Beyond the range samples are no longer auto-correlated.

## 2.8 KRIGING METHOD

The kriging interpolator has been used over the years, this is due to the fact that the technique has been reliable and the most widely used in geo-resource studies and in environmental management. Kriging interpolation models have shown success in its application in many areas of research. These include predicting groundwater level, soil salinity, soil fertility property and areal storm pattern analysis (Yasrebi et al., 2008; Sokouti and Mahdian, 2009; Rabah et al., 2011). Kriging when applied to research data gives values which best represent

the distribution of the original data (Mohaghegh et al., 1997; NIH, 1998). In particular, kriging has been used extensively in the mining sector where it was first developed. Usually in reserve estimation, the choice of the most appropriate technique is critical towards determining to an appreciable degree of certainty the accuracy and reliability of the estimation.

Kriging is the geostatistical method of optimally estimating the grade of a point or a block of ore by assigning weights to sample grades or values within the range of influence. The weights assigned to the values are obtained by solving series of simultaneous linear equations involving parameters obtained from the semi-variogram model. Thus by employing the parameters of the semi-variogram which takes into account all the geological parameters responsible for errors in reserve estimation, the weights obtained under Kriging are deemed to give optimum estimation results.

The Kriging estimator is sometimes referred to as BLUE, in that, it gives the Best Linear Unbiased Estimate of a block and aims to minimise the estimation variance" of the block which thus make it an optimum estimator (AnneIs, 1991). The two linear techniques are simple and ordinary Kriging. The Advantages of Linear Kriging are: Very good in local and global estimates, Geological knowledge is captured in variogram, Statistical approach allows uncertainty to be quantified and ensures minimum estimation variance.

Disadvantages of Linear kriging are: Not easy to comprehend, computationally intensive Flexibility and power created by many parameters also create arbitrariness and more possibilities for error.

### **2.8.1 Simple Kriging**

Simple Kriging assumes that the randomized spatial function is stationary and that the mean is constant over the area of interest. This assumption is often unrealistic; the more robust Ordinary Kriging is thus frequently used instead. In Simple Kriging, all input points are used to calculate each output pixel value.

### **2.8.2 Ordinary Kriging**

In Ordinary Kriging the randomized spatial function is non-stationary and the mean varies over the area of interest. Ordinary Kriging amounts to re-estimating the mean at each new location. In Ordinary Kriging, you can influence the number of points that should be taken into account in the calculation of an output pixel value by specifying a limiting distance and a minimum and maximum number of points. Only the points that fall within the limiting distance to an output pixel will be used in the calculation for that output pixel value. (Burrough 1986)

There are two conditional bias used for Kriging optimisation (1)Kriging efficiency (KE) which measures the effectiveness of the Kriging estimate to reproduce the local Block grade accurately and(2) Slope of regression or conditional bias slope, which summarises the degree of over smoothing of high grades and low grades.

### **2.9 THE DETERMINATION OF SMALLEST MINING UNIT(S)**

Mineral reserve estimates is done to determine the amount of ore and waste probability based on a chosen SMU as different SMU would give different tonnes and grade of ore and waste.

The optimal block size for consideration is also a function of drill hole spacing. Also other factors which are considered are the geometry of the mineralisation and the drilling with respect to anisotropy as well as block size required to give a reasonable volume resolution. (Sinclair et al, 2002).

Ordinary Kriging is one of the linear methods used to determine the SMU for mining which gives an accurate tonnes of ore, waste and grade with reduce estimated error.

Several papers (Deraisme et al, 1998) and technical reports have indicated the use of ordinary kriging for the determination of SMU but did not indicate clearly the variations in the kriging

efficiencies, regression slope of the estimator as well as tonnages and grade when various SMU are used for a particular type of deposit.

Simpson (2004) used ordinary kriging for the evaluation of the Okvau Gold deposit in Cambodia where he used 5 m x 5 m x 5 m as the smallest mining unit. Gold mineralisation at Okvau is associated with pyrite and arsenopyrite veining within a fault zone system that transgresses diorite and hornfelsed lithologies. Moderate to high grade gold mineralisation is located within both the main shear and secondary linking shear.

Cube Consulting Pty Ltd of Perth, Western Australia, was contracted in 2013 by Medusa Mining Limited to undertake the resource estimate of the Bananghilig Gold Deposit. The Resource was estimated using Ordinary Block Kriging and Uniform Conditioning (UC). UC is a mathematical method that allows the discrimination of ore and waste at an assumed selective mining unit size within an estimated panel of significantly larger size. A Selective Mining Unit ("SMU") of 5 metres by 5 metres by 2 metres was evaluated using the Uniform Conditioning whiles for the Ordinary Kriged panels  $Y = 25$  metres;  $X = 25$  metres and  $Z = 5$  metres were used for the purposes of reporting recoverable resources.

Antona mining limited, West Perth Australia in 2011 evaluated the Little Eva deposit. The deposit is a copper-gold deposit style. Mineralisation is hosted by feldspar porphyritic intermediate to basic volcanics intercalated within a sequence of scapolitised calc-silicate metasediments. Mineralisation is largely hosted in volcanic rocks. Multiple Indicator Kriging (MIK) estimation was used to estimate copper and gold grades into parent blocks sizes 25 x 25 x 10 metres. The selective mining unit (SMU) size was defined at 6.25 x 6.25 x 5 metres where Ordinary Kriging was used.

## CHAPTER THREE

### METHODOLOGY

The Grade Control Model was run using the Datamine software and Resource Evaluation Systems (RES). The RES Front-End system, utilizes Microsoft Visual Basic which was developed by Charles Muller, a Goldfields Resource Modeling expert. Data from reverse circulation (RC) drilling program was used. Ordinary Kriging estimation was run on two different Smallest Mining Unit (SMU) i.e.  $2\text{ m} \times 5\text{ m} \times 3\text{ m}$  (2:X-axis, 5:Y-axis, 3:Z-axis) and  $5\text{ m} \times 5\text{ m} \times 3\text{ m}$  (5:X-axis, 5:Y-axis, 3:Z-axis).

The statistics and variogram of the sample stream were determined, this allowed the identification of high grade outliers. The grade values which were anomalous to the general population were cut back to the appropriate upper limit of the population.

Estimation of tonnes of ore at various cut-offs of the two smallest mining unit (SMU) was obtained. A plot of tonnes and grade at various cut-offs (Grade Tonnage Curve) was generated for the different SMU used. This would indicate the tonnes of ore at different cut-off grade as well as the kriging efficiency and the regression slope of the Ordinary kriging estimation which was used.

The ore being treated at the study area consists of fresh rocks of sandstone, phyllites, dolerite and weathered material such as laterite and saprolite.

Ancillary equipment includes bulldozers, graders, water trucks, and service truck vehicles, supporting the drill-and-blast and haulage operations through vehicle, road, and bench maintenance, dust and erosion control.

Fresh rock and transitional zones are drilled and blasted in 9 m lifts, with excavation in 3m flitches. To optimise ore fragmentation and blast control, blasting in fresh rock utilises both non-electronic and electronic detonators. Oxide material, which cannot be 'free-dug', is

blasted using lower powder factors. Waste material is hauled to planned dumps located close to the pit.

Load and haul carry out mining using a standard truck-shovel operation, with excavators in backhoe configuration, two Liebherr 984, Bucyrus RH 90C and RH120E O&K. The haulage fleet consists of five (5) Caterpillar Dozers and eighteen (18) 777F dump trucks, with an average payload capacity of 87 tonnes. With the current equipment fleet, the primary excavation rate is approximately 940 tones/hour and average loading time 6min 42sec. The excavators Liebherr 984 and Bucyrus RH 90C, RH120E O&K have respective bucket width of 2 m and 3.5 m.

### **3.1 DESIGN OF GRADE CONTROL HOLES**

The grade control holes were designed by the use of Minesight software (MS3D Version 7.0); this was developed by Mintec Company in 1970 by the Founder Fred Banfield in Tucson, Arizona, USA.

All exploration, resource and historical grade control holes drilled in the study area were used in the designing of the grade control holes. The purpose of this was to understand the area of mineralisation, the trend and also to reduce cost by increasing the distance between holes of areas where there is less mineralisation. The grade control holes were designed on a grid of 5m across strike and 8m along strike with a depth of 27 m plus 1 m sub-drill making 28 m. Pit designs were used to guide the holes taken in to consideration berms and ramps. The Design holes, the coordinates of the point was then handed over to survey department to peg them in the pit. Fig. 3.1 shows the grade control design holes and pit design.

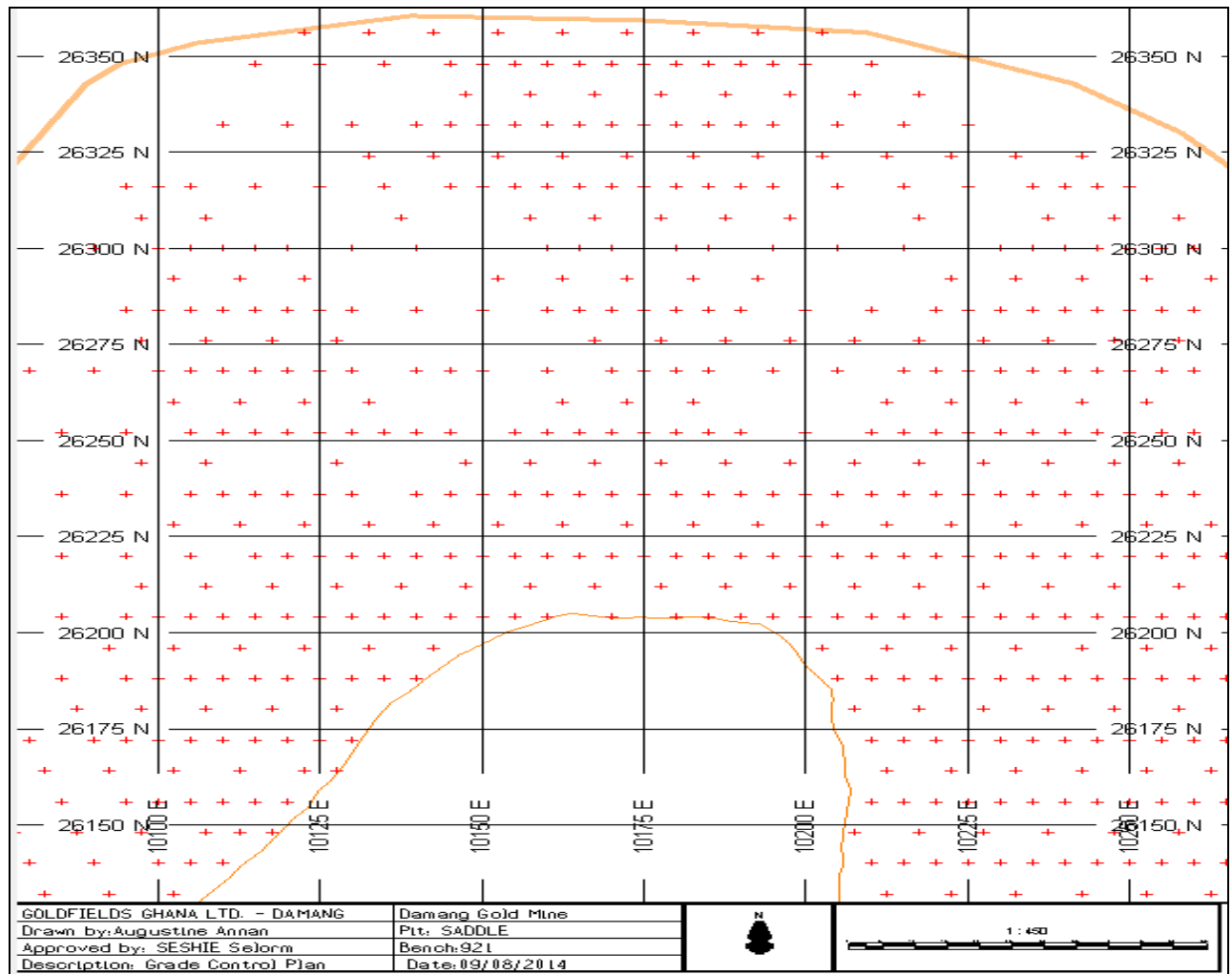


Fig. 3.1: Grade control design holes and pit design

### 3.2 DRILLING METHOD

The drilling in the study area was undertaken using Reverse Circulating (RC) rigs. The holes were drilled vertically to a depth of 27 m and 1m sub drill making 28 m. The 27m gives nine (9) flitches which are mined. During drilling RC samples were collected at 1.0m down hole intervals and weighed resulting in approximately 40-35 kilograms of samples using 5.5inches size drill bit. The samples were also logged to identify the lithology, alteration, stratigraphy and indicators for gold mineralisation. The first 1.0m from each hole was not sampled as the sub-drill of 1m had already been taken care in the previous drilling. This was done since most often than not the pit floors where Grade Control Drilling is to begin always has broken rocks which are not in-situ hence by drilling the sub-drill contamination of the first 1m does not

occur. Drill-hole collar locations after drilling were then surveyed and loaded in to acQuire™ Version 4.4.0.1, Data Base.

### **3.3 GRADE CONTROL SAMPLING**

Barnes (1980) has it that, a sample is a representative part of a single item from a larger whole, being drawn for the purpose of inspection or shown as evidence of quantity.

The (40 – 35) kg of samples for every meter which was obtained from the RC drilling was then poured on to a single stage mobile riffle splitter. The one meter sample (40 kg) was splitted using a riffle splitter to obtain 5 to 8 kg sample for gold (Au) analysis. After sample splitting, three different check samples were randomly inserted which included Blank samples, Standard samples and Field duplicate to check for contamination, accuracy and precision respectively of the SGS Assay Laboratory on site at Damang. After splitting every hundred (100) samples two standards was inserted at position 20<sup>th</sup> and 65<sup>th</sup>), two blanks was placed at 50<sup>th</sup> and 68<sup>th</sup> positions and four duplicates introduced at position 25<sup>th</sup>, 42<sup>nd</sup>, 55<sup>th</sup> and 85<sup>th</sup>.

In addition, a part of the 1.0 m sampling was taken, washed and placed on a plastic sheet where it was logged to indicate the variation in lithology, alteration and gold mineralization.

Fig.3.2 below shows a sample log sheet which was used for logging the RC chips.



Fig. 3.3 below indicates grade control drilling and sampling.



Fig. 3.3: Showing four (4) RC Rigs drilling, splitted samples and grade control crew.

A total of **1,189** reverse circulation and **8** diamond drilled holes were available for review and analysis at the project area. This drilling provided an overall total of **177,566** samples assayed for Au. The reverse circulation drilling was done on a grid of 5 m x 8 m while the diamond drilling was on a grid of 40 m x 20 m.

### **3.4 SAMPLE PREPARATION**

RC drill samples with a received weight up to a maximum of 10kg were dried at a normal 105<sup>0</sup>C and a record kept of the actual drying temperature. The entire sample was carefully jaw crushed and fine crush in a Keegor mill to a nominal 85% passing through -2mm with sieve testing carried out at every 20<sup>th</sup> sample. The entire crushed sample was carefully split to approximately 1.5kg using Jones riffle splitter. The 1.5kg split was pulverized in a LM2 mill to a nominal 90% passing 75-micron with wet sieve testing carried out at every 20<sup>th</sup> sample. Two analytical pulps of approximately 200g each were sub-sampled from the bulk pulp after

mat rolling and taking a multi-spoon scoop composite pulp sub-sample into a wire-tied sample packet. One wire-tied sample packet was retained for future reference.

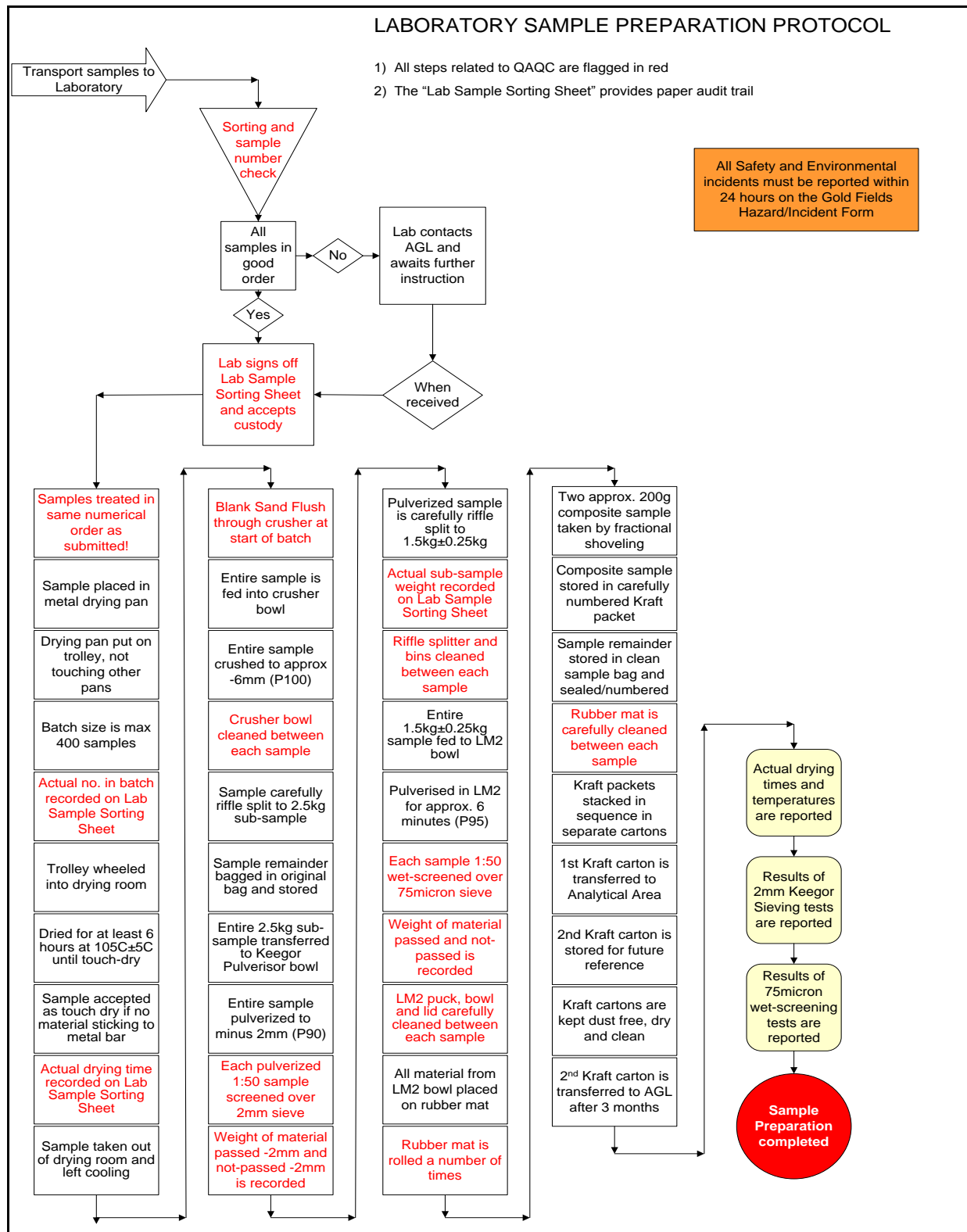


Figure 3.4: Sample Preparation protocol, (Source: Competent Persons' Report on the Mining and Assets of Goldfields Ghana Abooso Goldfields Ltd, Damang Mine, 2013).

### 3.4 ASSAY ANALYSIS

Assaying includes a variety of analytical techniques employed to determine the value of minerals contained in a sample. Most analytical laboratories practice one or more of the following techniques:

- Atomic Absorption Spectrometry
- Classical Volumetric and gravimetric analysis
- Coupled Plasma
- Leach Well
- Fire Assay
- Aqua Regia

In addition to the above listed methods, X-ray fluorescence spectrometry, radiometric analysis and Neutron activation analysis, which do not apply the destructive principle of sample preparation, are put to use by some analytical laboratories. Fire assays and aqua regia are predominantly used by the mineral industry for the analysis of Au (Moon et al., 2006).

In aqua regia, used by Goldfields Ghana Limited, Damang Mine, 800g of the pulverized samples was weighed into a 2 Litres bottle with a cap; this was followed by adding one Leachwell tablets (cyanide tablet) to every 800g of each sample. 800mls of water was then added to the 800g of each sample. The bottles were capped and shake thoroughly before packing it securely on a bottle roller machine.

The bottles were rolled on the roller machine for 4hrs. After the 4hr period the cap bottles with the samples were removed from the roller and made to stand for 4hrs until sufficient clear solution is available to obtain an aliquot.

Using a pipette, 20ml aliquot of each of the leach solutions were placed into test tubes. Using the calibrated dispenser, 4ml of 1% w/v Aliquot of Diisobutyl Ketone (DIBK) was also added. A Plastic/ rubber corks were applied on the test tubes which were shaken by hand for three

minute using foam backed board to clamp the rack. After mixing, sufficient time was allowed for the DIBK layer to separate from the cyanide layer. The cyanide layer of the samples were then analysed for gold (Au) by Atomic Absorption Spectrophotometer (AAS). Before the use of the AAS it was calibrated using standards. Reference standard samples and analytical control samples, especially prepared for the purpose, were used to serve as standard checks (Unpublished methodology SOX procedures at Damang, 2013).

Reference standard samples and analytical control samples, especially prepared for the purpose, can be employed to serve as standard checks (Unpublished methodology SOX procedures at Damang, 2013).

### **3.6 QUALITY ASSURANCE AND QUALITY CONTROL (QA/QC)**

QA/QC programmes are intended to monitor routine sampling and laboratory performance, in an effort to control the total possible errors in sample preparation and analysis.

When the assay results were received from the SGS Laboratory on site they were then loaded in to the Acquire drill hole database. A batch acceptance report was generated automatically by the Acquire drill hole database, and also rejection report when there were problems with the analysis in terms of QA/QC. The report detailed the results received for each standard, and shows any standard result which different from the nominal value by more than 3 standard deviations. If the QA/QC samples in a batch of samples did not pass the QA/QC, SGS Laboratory would be requested to repeat the analysis for that batch of samples.

A commercially prepared standard Rocklabs from New Zealand were used. The SGS Lab on site also performs internal QA/QC assays which provide secondary validation for the sample stream. Quality control-quality assurance reports were reviewed and the data was deemed suitable for any evaluation programme.

### **3.7 GRADE CONTROL MODELING PROCESS**

The system used for the grade control modeling process encompassed acQuire™, MineSight, Resource Evaluation Systems (RES) and Datamine Studio 3 Version 3.19 . The RES is a Front-End system as indicated earlier.

#### **3.7.1 Statistical and Spatial Analysis of Data**

The datasets obtained from the project area were loaded into the Acquire database which involves the Collar File, the Assay File and the Survey File of the holes drilled. Other parameters used include the cut off grade of Au mineralisation was at 0.3g/t for the block model and search parameters of 12m along x-axis, 18m along y-axis and 4m along the z-axis which depended on the drilling spacing. A geological mapping exercise was carried out to aid in the interpretation and geological domaining.

The project area has seven (7) geological Domains (PSSU\_W3B, PSLP\_E3B, PSLP\_E3A, PSLS\_ E3A, PSLS\_ E3B, PMDO\_E3B, and PMDO\_E3B) from three (3) lithological zones (Huni sandstone, phyllite and Dolerite intrusive) and two structural zones (3A and 3B) of which statistical analysis were done to assess directional continuity of the various domains and to establish a statistical relationship between the individual domains. Top-cuts were selected based on statistical analysis of the assay data for each domain, the following grade capping procedure was applied to the lithological Domains for the kriging and variography: Normal and Log-Normal probability plots in combination with histogram plots were used, and values that deviate significantly from the straight line were assumed to be outliers. These were investigated and top-cut where necessary, to manage the often excessive influence of extreme high grades.

#### **3.7.2 Planer and Down Hole Variograms**

Variogram calculations were undertaken first, primarily to get a quick overview of the typical

nugget-sill ratio's that can be expected for the major lithological / ore zones. Within the study area the variogram were calculated and developed using the 1.0m down-hole composite.

This grade range was determined from the in-situ probability statistics and associated analyses. These down-hole variogram studies also provided a starting point for the possible development of indicative “strike” variograms for the saddle area. Resource estimation system Minesoft was used in producing the variogram.

### 3.7.3 Ore Block Modeling

Ore block modeling was done for the two Block Model Sizes i.e. 2mx5mx3m and 5mx5mx3 m (where 3 m is the bench height to be mined). Table 3.1 gives the Block model parameters used for the two Block Model Sizes.

Table 3.1 Model Parameters

			<b>DISTANCE</b>	<b>NO OF CELLS</b>
<b>EASTINGS(m)</b>	<b>9600</b>	<b>10500</b>	<b>900</b>	<b>450</b>
<b>NOTHING (m)</b>	<b>23500</b>	<b>27500</b>	<b>4000</b>	<b>800</b>
<b>RL(m)</b>	<b>501</b>	<b>1101</b>	<b>600</b>	<b>200</b>

The models which would be created are geological model for interpretation, grade model for interpolation (proportional models) and krig model. The parameters of the geological model include dip of zones, vertical thickness, specific gravity, filled volume, void volume, dissolution factor and resource classification whiles the grade model involves local mean, number of samples, global mean, inverse distance value, ordinary kriged value, simple kriged value, minimum distance from sample, kriging variance for Simple Kriging and kriging variance for Ordinary Kriging. For the krig model the parameters are number of samples used, search volume, grade for ordinary kriging, grade for simple kriging, kriging efficiency and

regression slop.

This was followed by Grade estimation, done using Ordinary Kriging which was run by using the parameters of the variogram which includes the range, sill and the nugget.

The krig model has the following parameters, samples used, search volume, grade for ordinary kriging, grade for simple kriging, kriging efficiency and kriging regression slope which indicates the extent of conditional bias in the estimation.

The flow sheet diagram shown in Fig. 3.4 illustrates the steps taken to generate the grade control model for the saddle Pit project area.

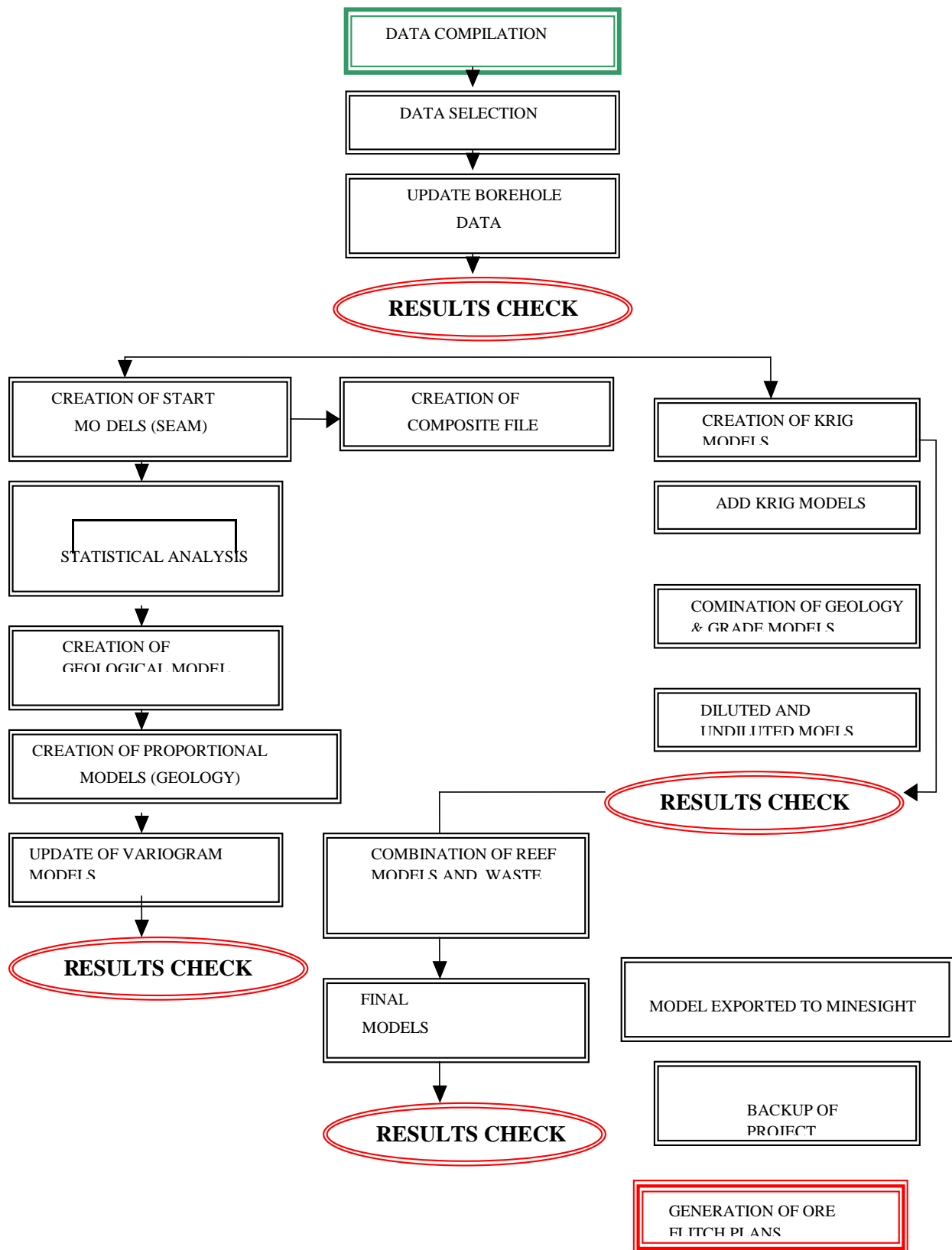


Fig. 3.4 Flow chart of grade control model process for the project area (Mamphey et al 2005)

## CHAPTER FOUR

### RESULTS AND DISCUSSIONS

#### 4.1 DOMAINING AND GEOLOGICAL STRUCTURES

The geological structures in the project area contributed a great role in the interpretation, lithological domaining and structural zones used in the estimation. This was obtained by geological mapping exercise carried out in the area.

Fig. 4.1 shows sub-horizontal quartz veins at the east high wall of the saddle pit. This vein was well developed in the Huni sand stone and dipping to the east with an approximate thickness of 10 cm. The vein can be traced for over 50 m and was typically irregular with subtle variation of orientation along its length as well as having high length /width ratios. Fig. 4.2 indicates sub-parallel quartz veins at the south wall of saddle pit. This (Fig. 4.2) quartz veins slightly dip to the west and occurs in the dolerite intrusive, they were thicker than those Fig 4.1 and had smaller length/width ratios.



Fig.4.1: Sub-horizontal quartz vein at east of saddle pit (East Dipping)



Fig.4.2: Sub-parallel veins at south of saddle pit (West Dipping)

Fig. 4.3 indicates a shallowly dipping ( $25^\circ$ ) thrust fault and accompanying flat lying extensional quartz veins which were seen in the phyllite. These quartz veins have extensive pyrite alteration halos which were responsible for much of the hydrothermal gold mineralisation at the saddle pit. These veins were formed during a long history of progressive NW-SE compression, based on detailed structural studies, The extensional veins and vein arrays are interpreted to have formed late in the deformational history of the deposit, following “lock-up” of steeply-dipping fold limbs during progressive NW-SE shortening (Brabham, 1998).

Fig. 4.4 indicates a narrow quartz vein in drilled core from saddle project area. This quartz vein and its associated pyrite were encountered at a depth of 244.35 m down which occurred within the phyllite.

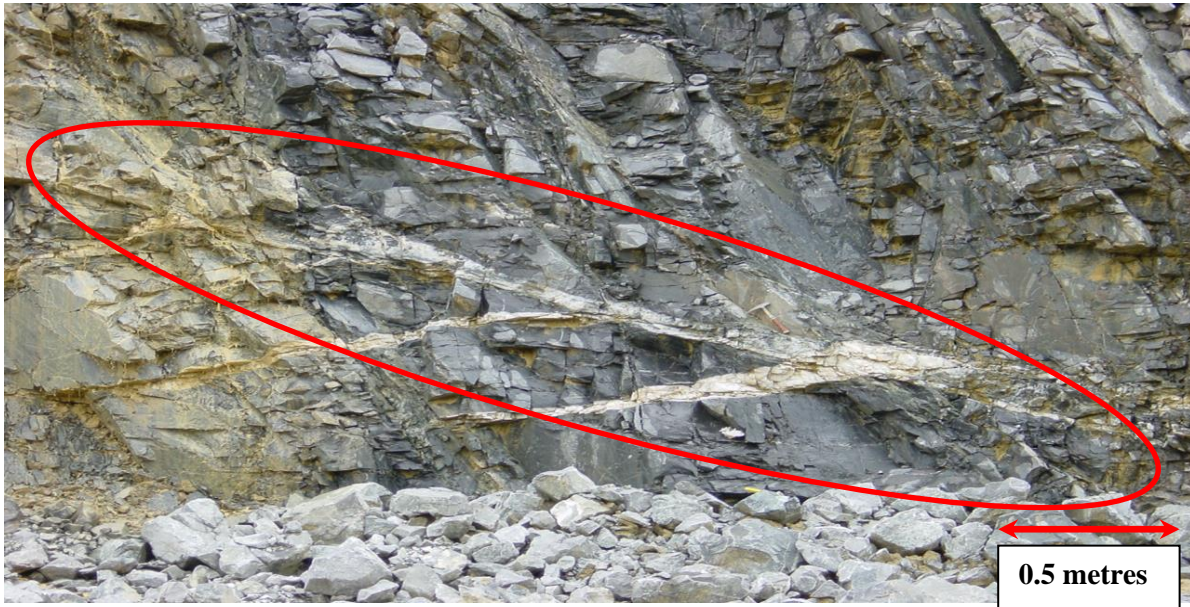


Fig.4.3: A shallow dipping thrust fault and accompanying flat lying extensional veins.

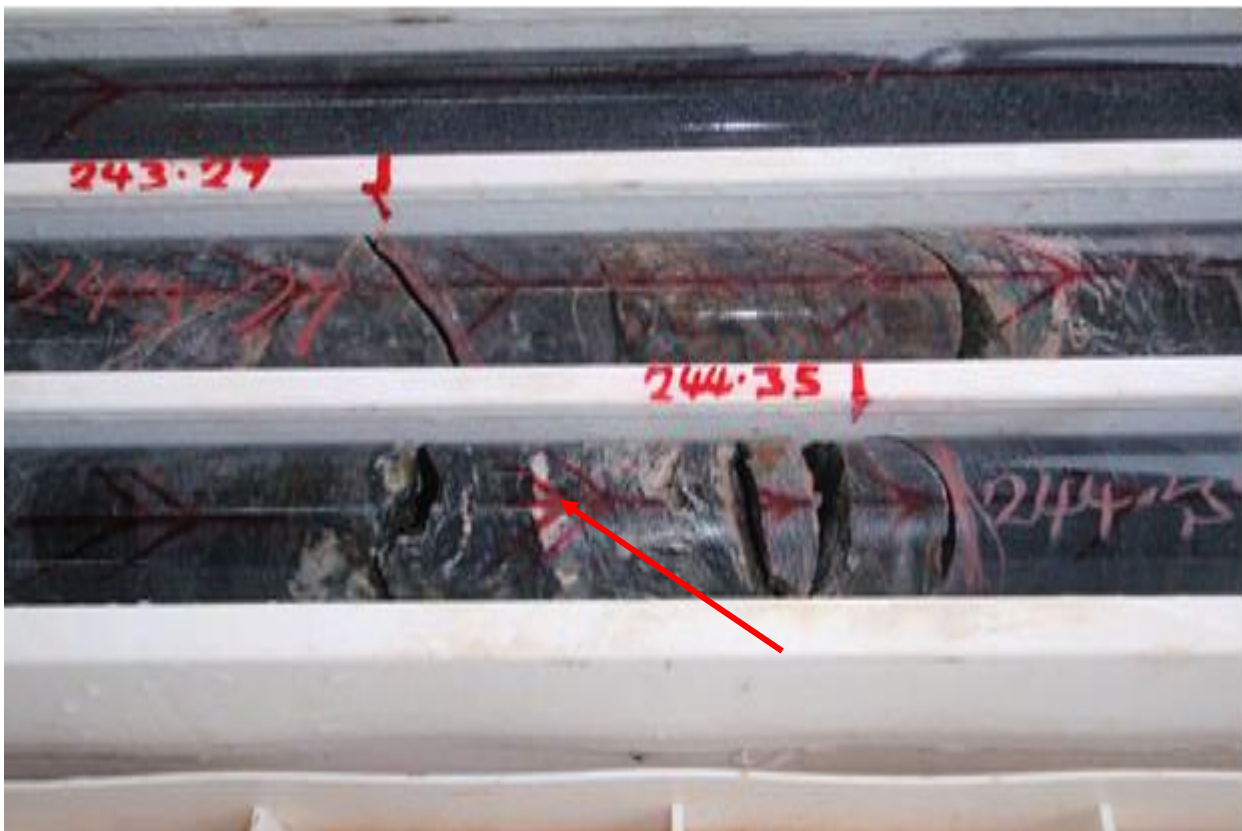


Fig. 4.4: Quartz vein in drilled core from saddle project area.

## 4.2 STATISTICAL AND SPATIAL ANALYSIS

The generation of statistics for the seven (7) geological domains from the two structural zones (3A and 3B) was important as this would provide information about the central tendencies and variability of the data as well as to identify whether populations in the various domains were significantly different.

Table 4.1 and 4.2 show the descriptive statistics for lithology (reef) that falls within the structural zones with the 2 m x 5 m x 3 m and 5 m x 5 m x 3 m block sizes respectively.

There was not much significant difference in the descriptive statistics parameters in terms of the minimum and maximum Au values indicated in Table 4.1 and 4.2. The PSSU\_W3B domain indicated in Table 4.1 gave a total number of samples of 210, with Au values between 0.005 – 41.9 with a respective average and standard deviation of 1.838 and 4.65531 indicating a coefficient of variation (CoV) value of 2.532 (Coefficient of Variation = Standard deviation/average). The coefficient of variation indicates the extent of variability of the data in relation to the mean of the population. When the value of the coefficient of variation is higher it means the data has high variability and less stability. On the other hand, when the value of the coefficient of variation is lower it indicates the data has less variability and high stability.

Therefore from Table 4.1 and 4.2, it can be seen that domain with higher standard deviation indicates higher CoV, this gives an indication of nugget effects and different populations occurring in the saddle project area. In addition, due to the hydrothermal style of mineralization occurring with quartz veins also influences the higher standard deviations as seen with domain PSLP\_E3B in Table 4.1.

Table 4.1: Descriptive statistics for all the lithology (reef) with the 2m x 5m x 3m Block Size

reef	parameter	domain	num records	total no samples	missing samples	samples_zero	minimum	maximum	average	variance	stdev	geomean	log mean	CoV
PSSU_W3B	AU	0	210	210	0	0	0.005	41.9	1.838	21.6719	4.65531	0.32	2.322	2.532
PSLP_E3A	AU	0	443	443	0	0	0.005	33.63	1.705	11.5238	3.39467	0.402	2.23	1.991
PSLP_E3B	AU	0	1158	1156	2	0	0.005	96.6	1.838	25.5681	5.05649	0.473	2.257	2.751
PSLS_E3A	AU	0	2339	2338	1	0	0.005	74.8	1.375	16.9674	4.11915	0.245	1.923	2.996
PSLS_E3B	AU	0	72	72	0	0	0.005	6.32	0.791	1.04212	1.02084	0.287	1.385	1.291
PMDO_E3A	AU	0	220	220	0	0	0.005	460	3.52	960.902	30.9984	0.354	2.343	8.807
PMDO_E3B	AU	0	998	998	0	0	0.005	54.7	1.428	10.6244	3.2595	0.42	1.628	2.283
W	AU	0	24002	23934	68	0	0.0045	35.9	0.077	0.3175	0.56347	0.017	0.037	7.344

Table 4.2: Descriptive statistics for all the lithology (reef) with the 5m x 5m x 3m Block Size

reef	parameter	domain	num records	total no samples	missing samples	samples_zero	minimum	maximum	average	variance	stdev	geomean	log mean	CoV
PSSU_W3B	AU	0	238	238	0	0	0.005	41.9	1.615	19.2092	4.38283	0.242	2.099	2.71
PSLP_E3A	AU	0	489	488	1	0	0.005	33.63	1.516	10.1846	3.19133	0.343	1.874	2.11
PSLP_E3B	AU	0	1290	1288	2	0	0.005	96.6	1.718	26.234	5.12192	0.355	2.159	2.98
PSLS_E3A	AU	0	2554	2553	1	0	0.005	74.8	1.265	15.577	3.94678	0.203	1.72	3.12
PSLS_E3B	AU	0	71	71	0	0	0.005	6.32	0.794	1.29888	1.13969	0.239	1.463	1.44
PMDO_E3A	AU	0	251	251	0	0	0.005	460	3.206	843.969	29.0512	0.302	1.976	9.06
PMDO_E3B	AU	0	1168	1168	0	0	0.005	34.9	1.155	6.26877	2.50375	0.269	1.633	2.17
W	AU	0	23380	23313	67	0	0.0045	35.9	0.076	0.30243	0.54993	0.016	0.036	7.2

Histogram plots were done to evaluate the normality and log normality of the Au population distributions. The main objective was to identify whether populations in the various zones were significantly different. It was also meant to assess the effects of the data distribution on

methods to be used for grade estimations, skewness of the data and presence of outliers of the distribution. Fig. 4.5 which shows histogram of Au values in structural block 3A (PSLP\_E3A) indicates that the distribution of the Au values were not normally distributed and are positively skewed. This can give rise to unstable estimate and interpretations as well as difficulty in determining the ideal central tendency of the data.

To obtain the normal distribution of the Au values the Log of the distribution was taken as indicated in Fig. 4.6 showing the normalization of the Au population i.e. Log Histogram of Au within structural Block 3A (PSLP\_E3A).

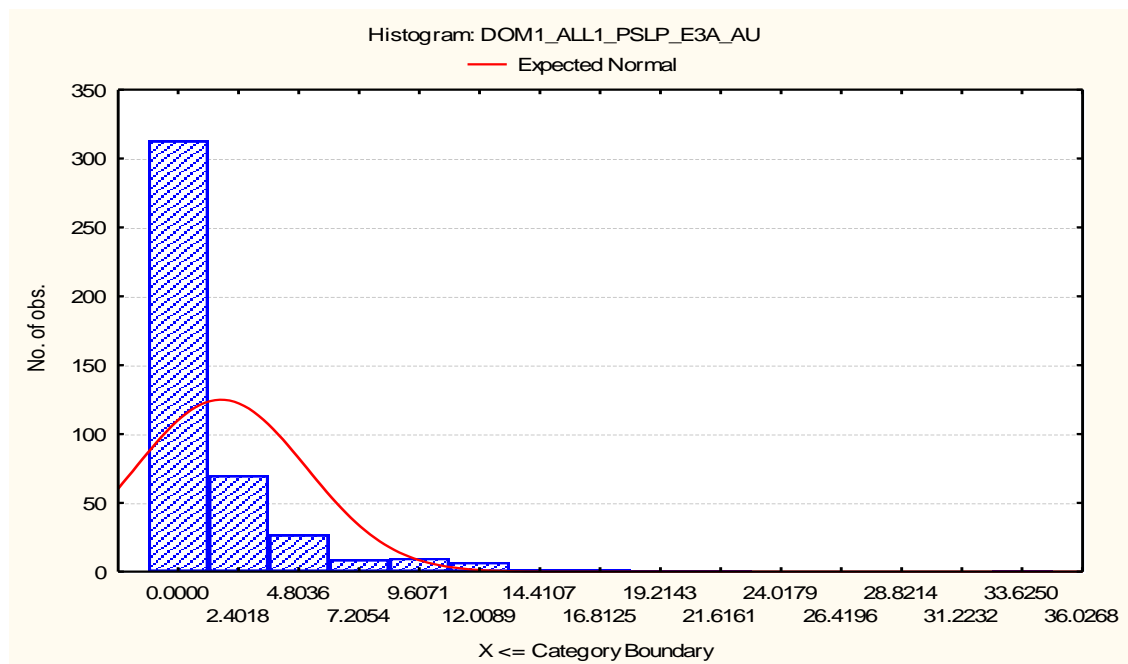


Fig. 4.5: Histogram of Au values that falls PSLP\_E3A

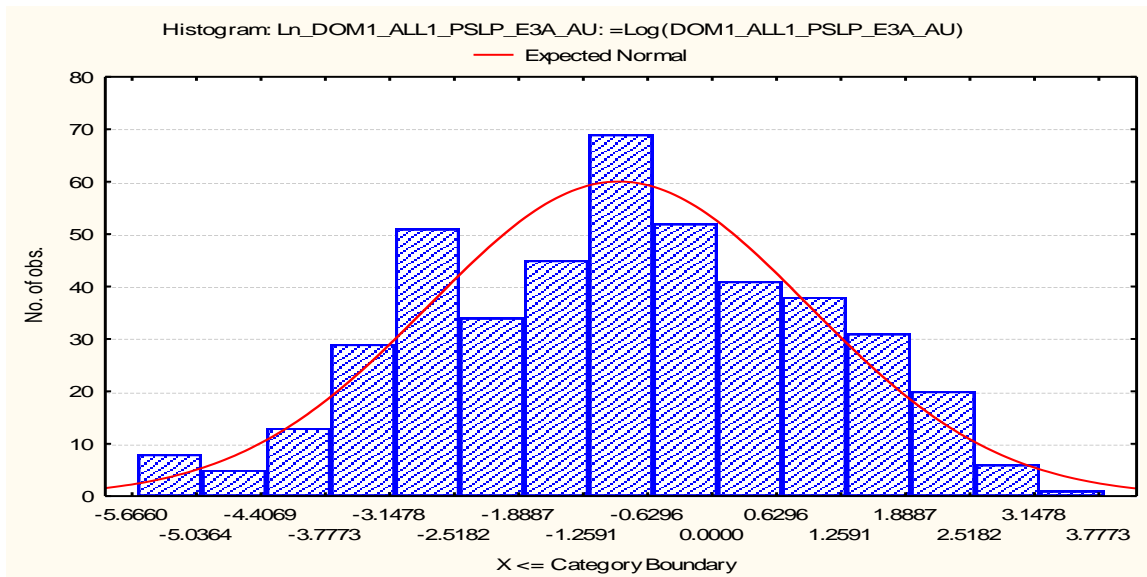


Fig. 4.6: Log Histogram of Au values that falls in PSLP\_E3A

### 4.3 DEALING WITH OUTLIERS

The normal probability plot and Log probability plots indicated the presence of some extreme values; hence top cut has to be applied to reduce smearing of Au when performing the krig estimate. Fig 4.7 which is the normal probability plot of Au values that falls in structural block 3A (PSLP\_E3A) was used for the krig top cut whiles Fig 4.8 representing the Log probability plots of Au values that falls in structural block 3A (PSLP\_E3A) was used for the variogram top cut.

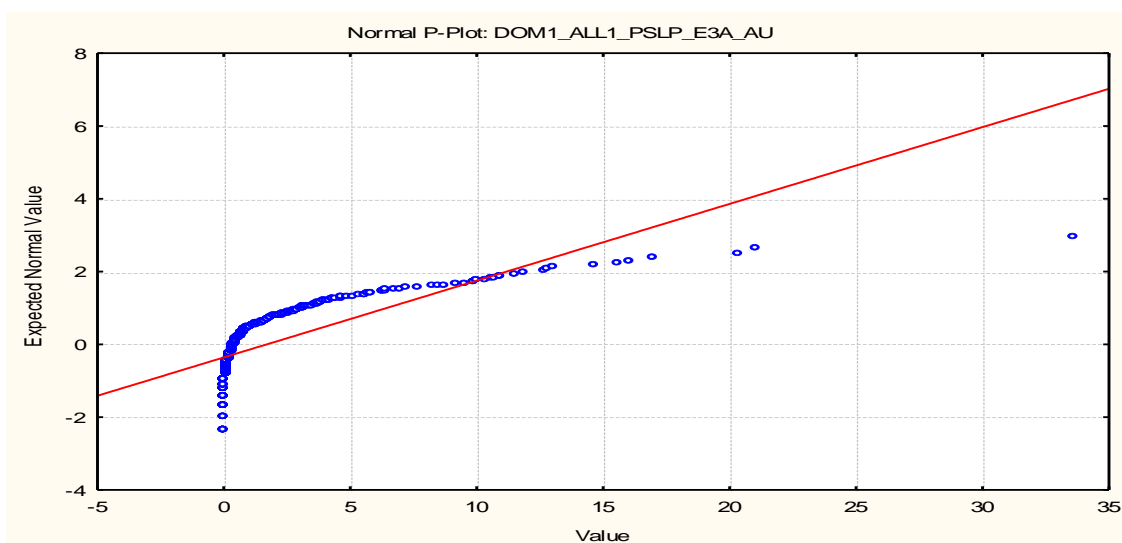


Fig. 4.7: Normal Probability Plot of Au values in PSLP\_E3A

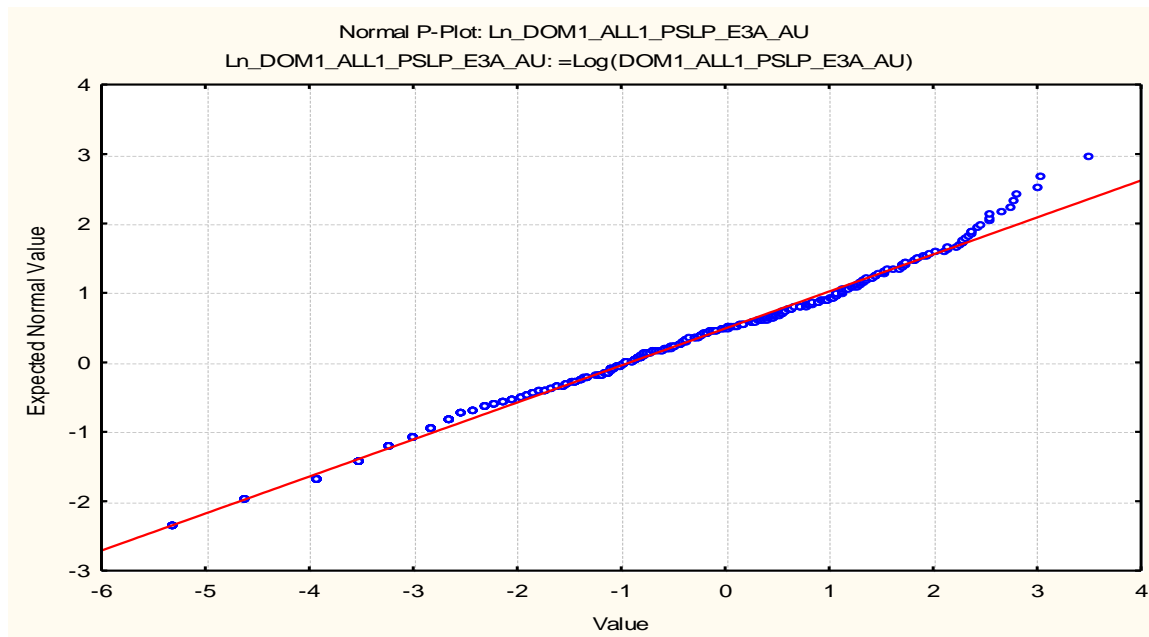


Fig. 4.8: Log Probability plot of Au values in PSLP\_E3A

The Krig top cut was placed at 17 and for the variogram top cut at 7. The other reef statistics and probability plots can be found in the Appendix.

#### 4.4 VARIOGRAMS MODEL PARAMETERS

A variogram is an increasing function of distance, implying that the difference between values measured at two distinct locations tends to increase as the distance between the location increases. The rate of increase of the variogram reflects the rate at which the influence of samples decreases with distance. The rate of increase may vary with direction (Dowd, 1988).

Experimental variogram calculations were done, firstly to give a theoretical variogram model which fit the Au mineralisation in the seven (7) domains of the saddle Pit. The experimental variogram gives an idea and also serves as a guide to how “strike” variograms for the project area should be modelled.

The search ellipsoid distances set for the block model interpolation process were directly derived from these variograms. The Saddle Pit area these were set to extend to at least two sections of grade control drilling spacing.

In addition, the nugget and sill values have a direct effect on the amount of block-to-block

smoothing of Au interpolation. The nugget variances as indicated in the figures below represent the variability of the sample data at the saddle pit. An increase in the nugget variance indicates high variability of the Au values.

Structural character of mineralisation at the project area showed anisotropy as indicated by the figure 4.9. For example Fig 4.9 exhibits geometrical anisotropy which has the same sill but different ranges the same observation is made with Fig 4.10. This implies the correlation is stronger in one direction than it is in other directions. Thus the variogram for the different populations has to be modelled separately to properly define the ore deposit in the saddle area. Using a single variogram to represent the populations would lead to a block model which does not represent the actual deposit.

Figures 4.9 to 4.12 presented show the planar variograms for the Au values in the domains of the Saddle Pit Project Area which was modelled.

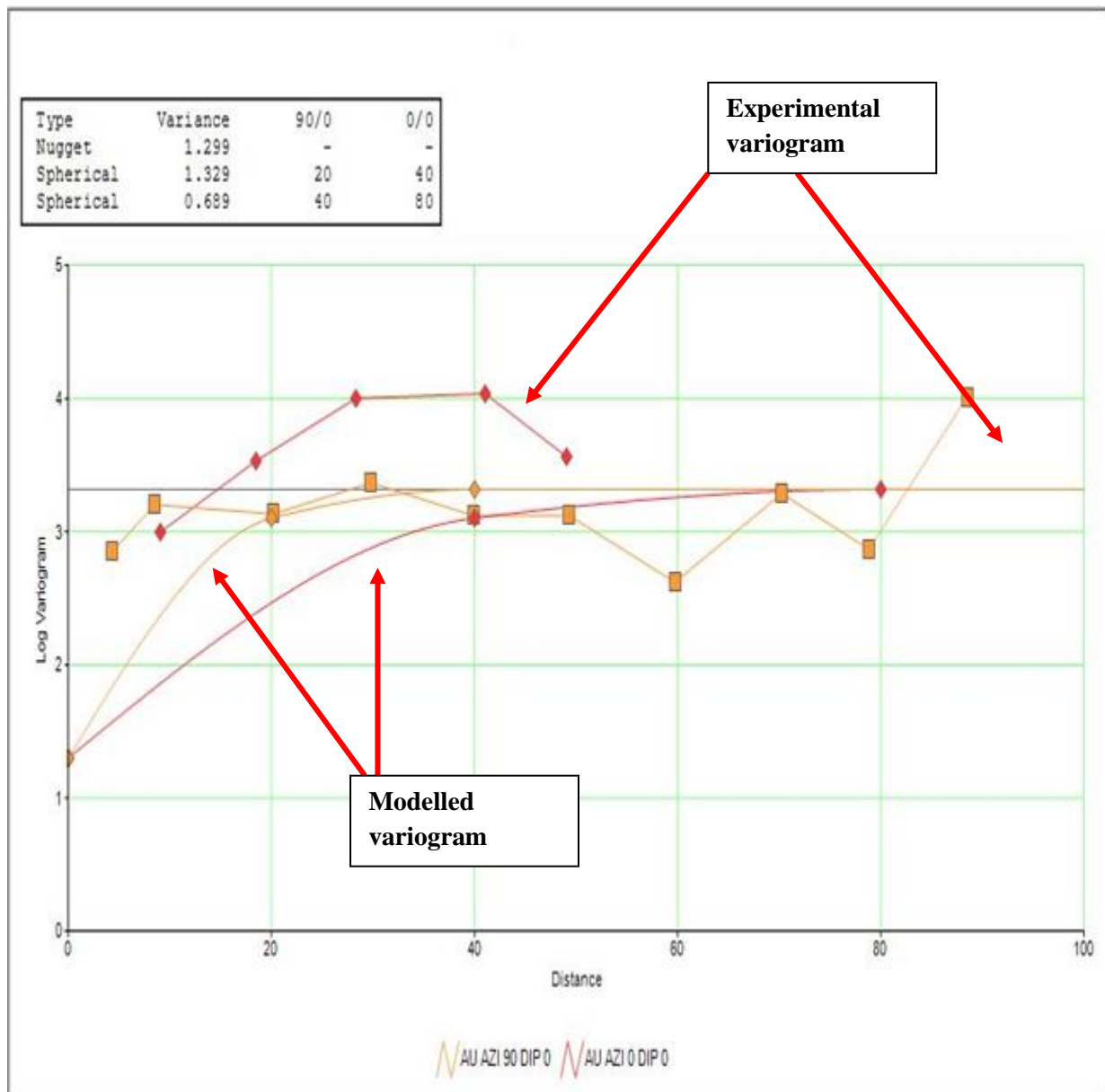


Fig 4.9: Planer Variogram for Domain PSLP\_E3A

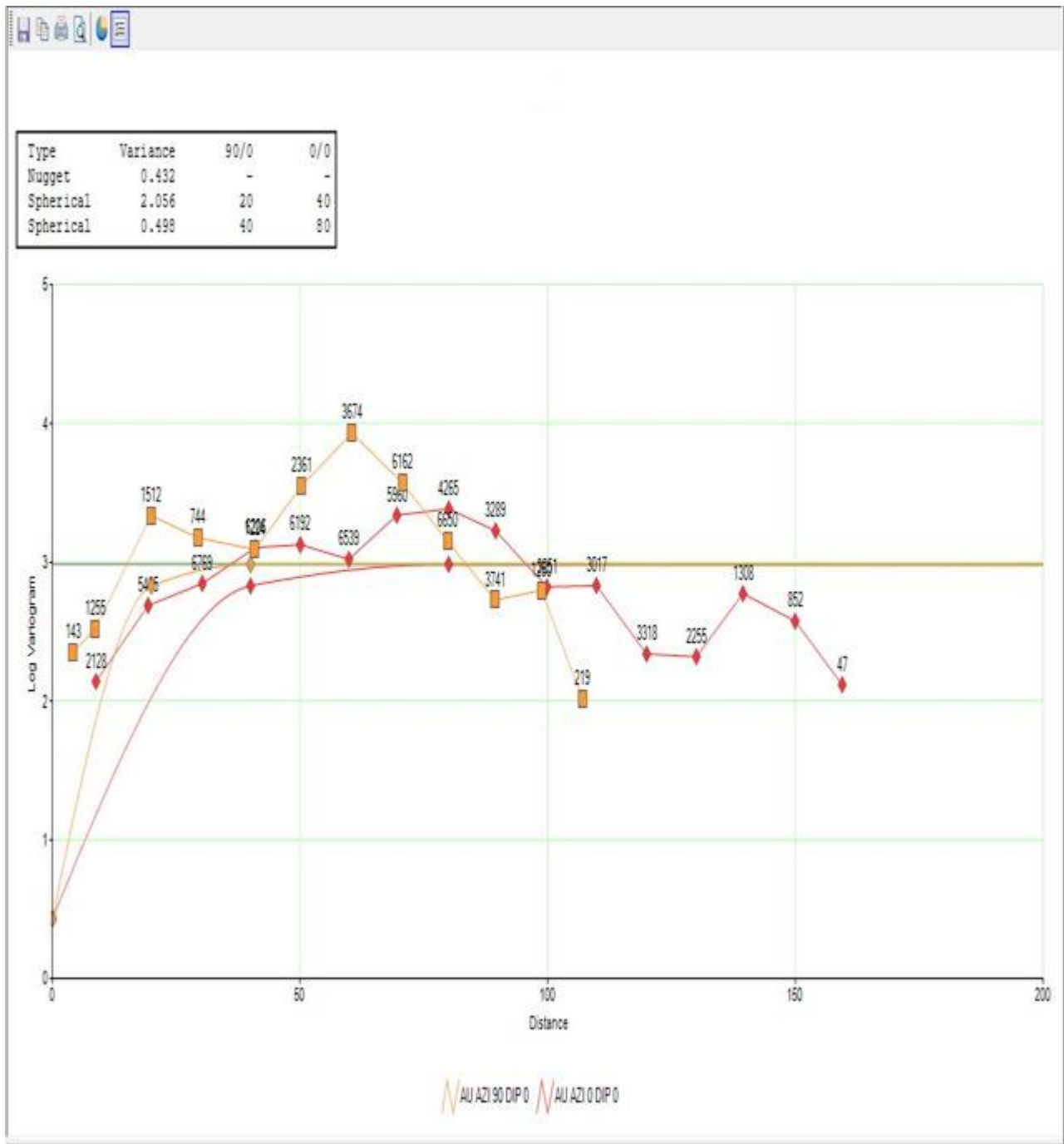


Fig 4.10: Planer Variogram for Domain PSLP\_E3B

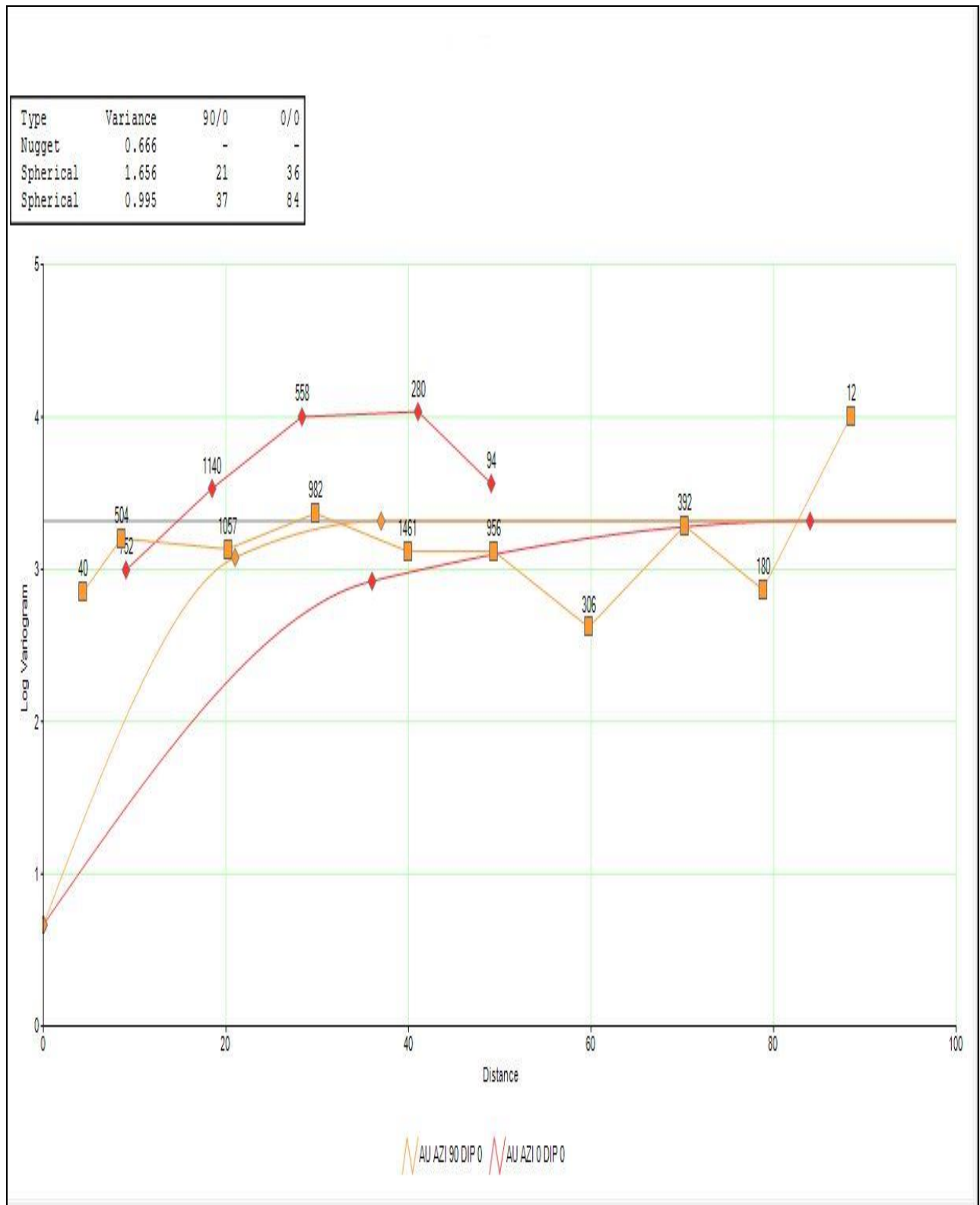


Fig 4.11: Planer Variogram for Domain PSLS\_E3A

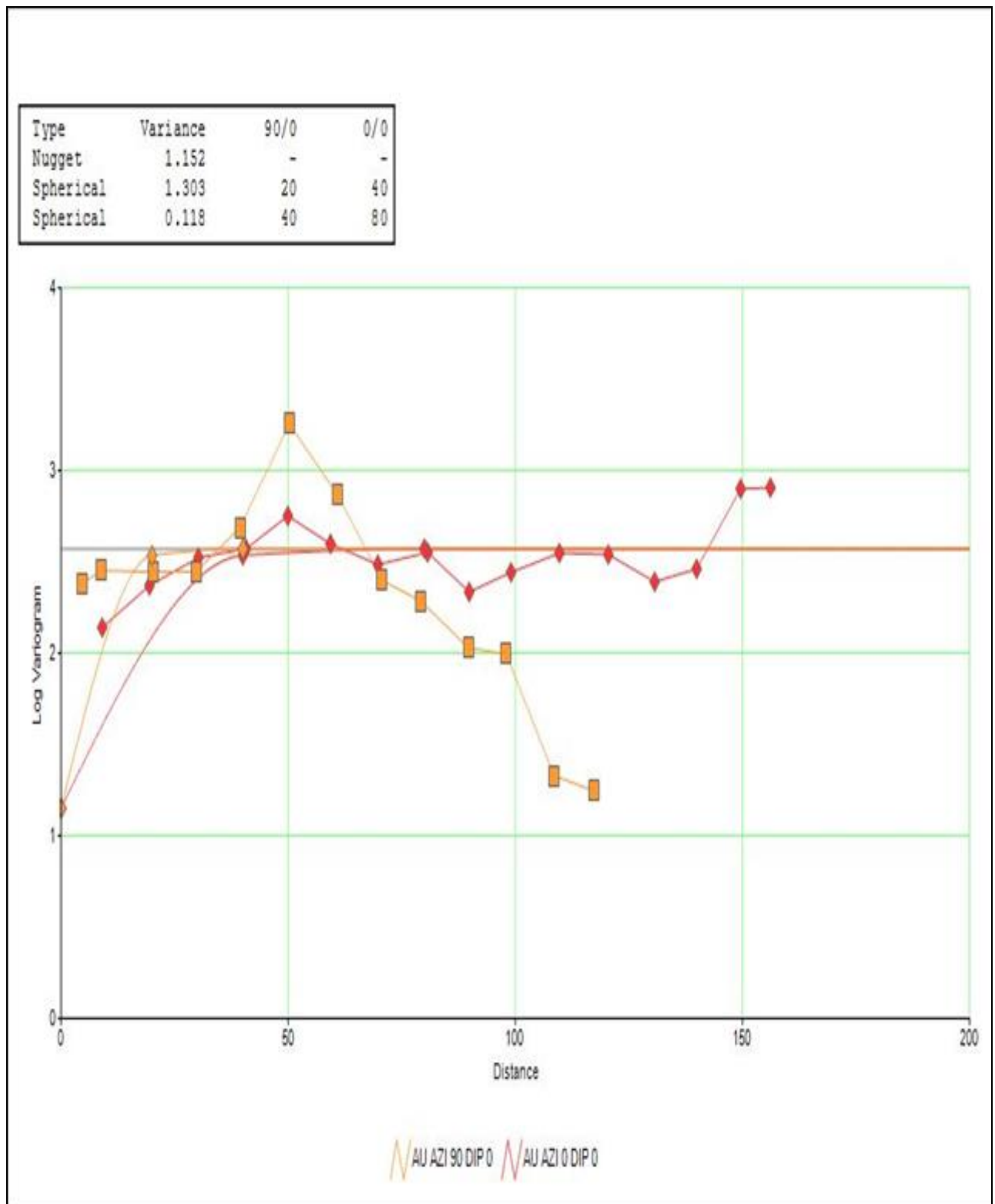


Fig 4.12: Planer Variogram for Domain PMDO\_E3B

Table 4.3 and Table 4.4 indicate variogram model parameters for the two Block Model Sizes 2m x 5m x 3m and 5m x 5m x 3m used respectively.

The two tables indicate the nuggets, the sill and the ranges used for Au mineralisation at the saddle pit area based on the modelled variograms.

Table 4.3: Variogram model parameter for 2m x 5m x 3m Block model Size

Reef	Parameter	Vrefnum	Sill	Angle1	Angle2	Angle3	Axis1	Axis2	Axis3	Nugget pe	Sill 1	Range1	Range2	Range3	Sill 2	Range1	Range2	Range3
PSSU_W3B	AU	11	19.57329	-15	40	0	1	2	3	39.755777	88.46018	20	40	9	100	40	80	9
PSLP_E3A	AU	11	12.10187	-15	20	0	1	2	3	39.755777	88.46018	20	40	9	100	40	80	9
PSLP_E3B	AU	11	26.95262	-15	-20	0	1	2	3	43.644601	94.68579	20	40	9	100	40	80	9
PSLS_E3A	AU	11	12.6418	-15	-20	0	1	2	3	44.737205	91.79819	21	37	9	100	36	84	9
PSLS_E3B	AU	11	2.462867	-15	-20	0	1	2	3	82.255423	97.36644	21	37	9	100	36	84	9
PMDO_E3A	AU	11	11.55158	-15	-20	0	1	2	3	45.338715	90.30576	20	40	9	100	40	80	9
PMDO_E3B	AU	11	17.29331	-15	-20	0	1	2	3	45.338715	90.30576	20	40	9	100	40	80	9
W	AU	11	0.00406	0	0	0	3	2	1	35.802459	100	8	25	10	100			

Table 4.4: Variogram model parameter for 5m x5m x3m Block model Size

Reef	Parameter	Vrefnum	Sill	Angle1	Angle2	Angle3	Axis1	Axis2	Axis3	Nugget pe	Sill 1	Range1	Range2	Range3	Sill 2	Range1	Range2	Range3
PSSU_W3B	AU	11	3.999572	-15	40	0	1	2	3	39.75578	88.46	20	40	9	100	40	80	9
PSLP_E3A	AU	11	6.597733	-15	20	0	1	2	3	39.75578	88.46	20	40	9	100	40	80	9
PSLP_E3B	AU	11	2.499284	-15	-20	0	1	2	3	45.99719	92.198	20	40	9	100	40	80	9
PSLS_E3A	AU	11	5.405896	-15	-20	0	1	2	3	44.73721	91.798	21	37	9	100	36	84	9
PSLS_E3B	AU	11	0.403559	-15	-20	0	1	2	3	82.25542	97.366	21	37	9	100	36	84	9
PMDO_E3A	AU	11	4.480481	-15	-20	0	1	2	3	45.33872	90.306	20	40	9	100	40	80	9
PMDO_E3B	AU	11	3.752543	-15	-20	0	1	2	3	45.33872	90.306	20	40	9	100	40	80	9
W	AU	11	0.151504	0	0	0	3	2	1	35.80246	100	8	25	10	100			

#### 4.5 KRIGING

In the estimation of the two Block Model Sizes, 2 m x 5 m x 3 m and 5 m x 5 m x 3 m the Ordinary Kriging (OK) used assumes that the local means are not necessarily closely related to the population mean and therefore uses only samples in the local neighborhood for estimation. The two parameters which Krig (1996) suggested whether the block size used for grade estimation is appropriate is Krig Efficiency (KE %) and Regression Slope (R) which can also be used to calibrate the confidence in block estimate. Perfect estimation would give KE 100% and  $R = 1$ .

Fig 4.13 and Fig 4.14 shows the Plan view of krig efficiency for the 2m x 5m x 3m and 5m x 5m x 3m Block Model Sizes estimated respectively. As indicated on the Legend, the efficiency of the kriging estimate is colour coded from 0% -100%. 100% kriging efficiency implies the kriging variance is smaller relative to the block variance. On the other hand when the kriging variance is higher than the Block variance (measures the degree of differences or variances between block grades and also dependent only on block size and will increase as the block size decreases) then the kriging efficiency would be low. Fig 4.13 has most of the kriged blocks in the range of 80% -100% as indicated in the red oval shape with extra kriged blocks closer to 100% giving a clear indication of higher kriging efficiency and low degree of smoothing for the 2m x 5m x 3m Block model Size .

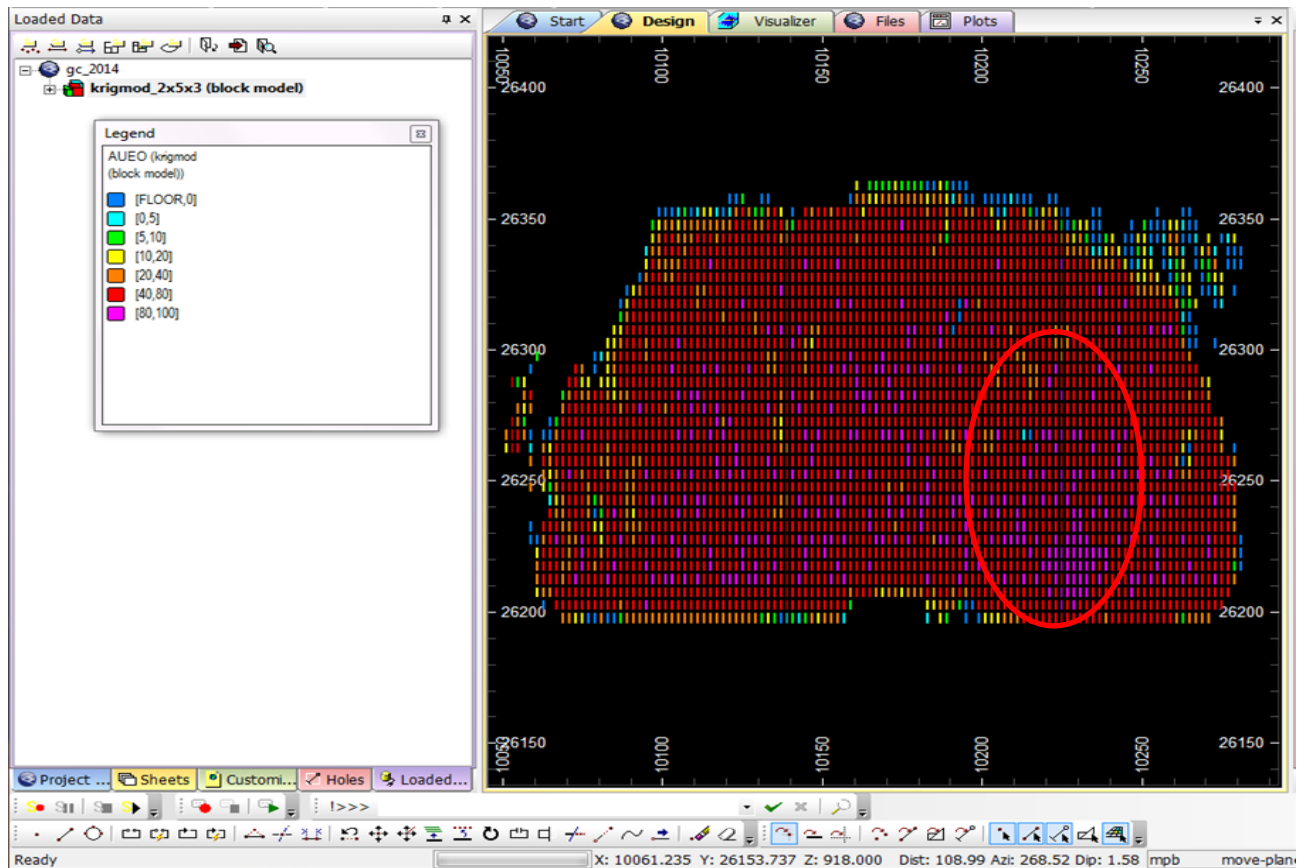


Fig. 4.13: Plan view of krig efficiency for the 2mx5mx3m Block Size

Fig.4.14 has most of the kriged blocks between 70% -90% as shown in the red oval shape which indicate a lower Kriging Efficiency. This implies that when the block size is increased the Block variance decreases leading to a lower kriging efficiency. Also due to the style of mineralisation at the project area in terms of discontinuous, narrow shallow dipping quartz veins, the spatial continuity of the Au values as suggested by the Variogram and the drilling spacing of the data, an increase in the block size would therefore leads to lower kriging efficiency.

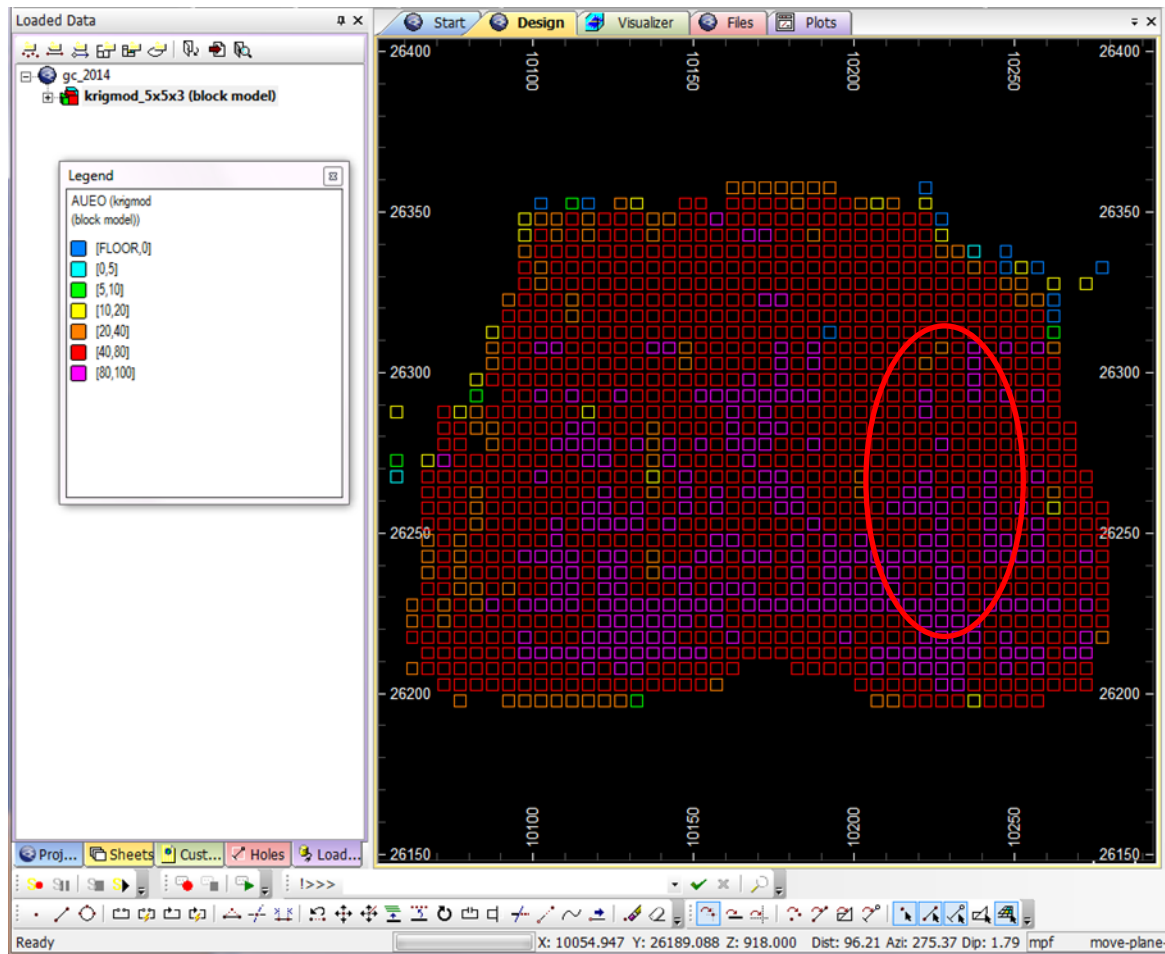


Fig. 4.14: Plan view of krig efficiency for the 5mx5mx3m Block Size

The slope of regression summarizes the degree of over smoothing of high and low grades based on the block sizes used.

However, slope close to one indicates the regression between the estimated and actual grades is very good, meaning there is limited over smoothing when smaller block sizes is used (i.e. 2 m x 5 m x 3 m). In this case the grade tonnage relationship above cut-off is realistic. Conversely, low slope values were obtained when the 5 m x 5 m x 3 m block size were used which indicated that there was over smoothing and hence a poor relationship between the estimated and actual blocks grades.

In addition a scatter plot of krig efficiency and the number of blocks of the two block sizes were plotted as indicated in Fig. 4.15. From the scatter plot the krig efficiency for the 2 m x 5 m x 3 m Block Size is between 50% - 81% while the krig efficiency of 5 m x 5 m x 3 m Block Size is between 45% - 77%. Thus the Krig efficiency of the 2mx5mx3m is slightly higher than that of the 5mx5mx3m. This implies that when the drilling spacing was maintained and the block model size is increased the kriging efficiency decreases.

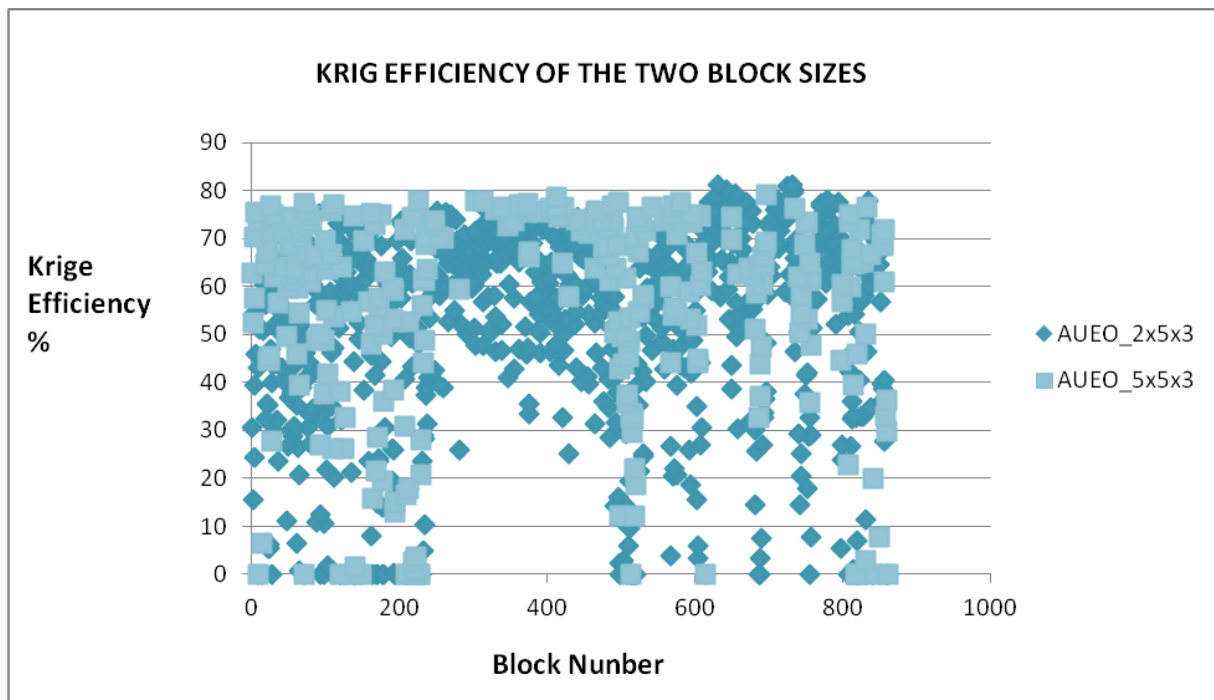


Fig. 4.15: Shows a scatter plot of krig efficiency of the two Block Sizes

The regression slope of the 2m x 5m x 3m and 5m x 5m x 3m block sizes is between 0.8 – 0.99 and 0.7 – 0.98 respectively, indicating a slight higher regression slope for 2m x 5m x 3m than 5m x 5m x 3m block sizes as shown in the scatter plot of Fig. 4.16. This gives an indication that the regression slope which is dependent on the block variance and krig variance would decrease when the block size is increased.



Fig. 4.16: Shows a scatter plot of regression slope of the two Block Sizes

It can therefore be said that with the current grade control drilling spacing at the project area (Saddle Pit), it is therefore prudent to use the 2mx5mx3m Block Sizes as the efficiency of the kriging estimate as well as the regression slope are much higher than that of the 5m x 5m x 3m block sizes.

The details of the Kriging estimation run for the saddle pit hydrothermal deposit is shown in the tables 4.5 and 4.6.

Grade estimation

Reef - PSLP\_E3A

Parameter – AU

Number of domain perimeters: 1

Number of soft domain perimeters: 1

Number of dip domain perimeters: 1

Table 4.5: Statistics for estimated cells (2mx5mx3m Block Size)

GM_AU :	1.609386
Parameter	AUOK
Total number of blocks	1144
Total number of estimated blocks	1144
Minimum value	0.1093
Maximum value	7.4229
Mean	1.7053
Variance	1.621074
Weighted Mean	1.7053

Grade estimation

Reef - PSLP\_E3A

Parameter - AU

Number of domain perimeters : 1

Number of soft domain perimeters: 1

Number of dip domain perimeters: 1

Table 4.6: Statistics for estimated cells (5mx5mx3m Block Size)

GM_AU :	1.455067
Parameter	AUOK
Total number of blocks	410
Total number of estimated blocks	410
Minimum value	0.1241
Maximum value	8.5394
Mean	1.563
Variance	1.48086
Weighted Mean	1.563

Figure 4.17 and 4.18 shows plan view Block Models of the 2 m x 5 m x 3 m Block Sizes and 5 mx5 mx3 m generated from the project area respectively. Figure 4.17 shows extra higher grade block sizes in the range of 0.25g/t - 1.5g/t as indicated in the red oval shape based on the legend. This gives an indication that 2 m x 5 m x 3 m Block Sizes gives extra kriged blocks at a higher Au values. On the contrary, Figure 4.18 shows grade blocks which ranges from 0.25g/t – 1.0g/t as shown in the oval red shape based on the legend, given an indication that the 5 m x 5 m x 3 m Block Sizes gave lower Au values and fewer kriged blocks.

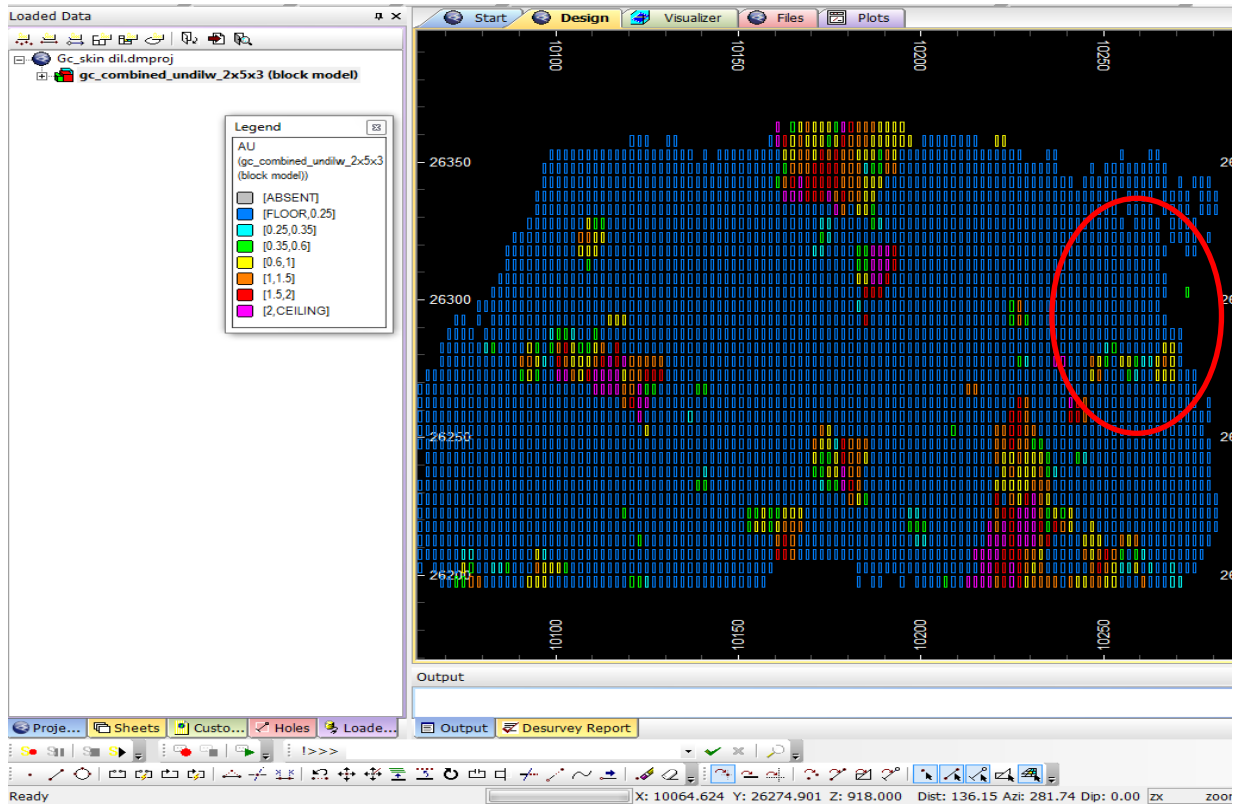


Fig 4.17: Plan view of Grade Control Model showing the 2mx5mx3m Block Size

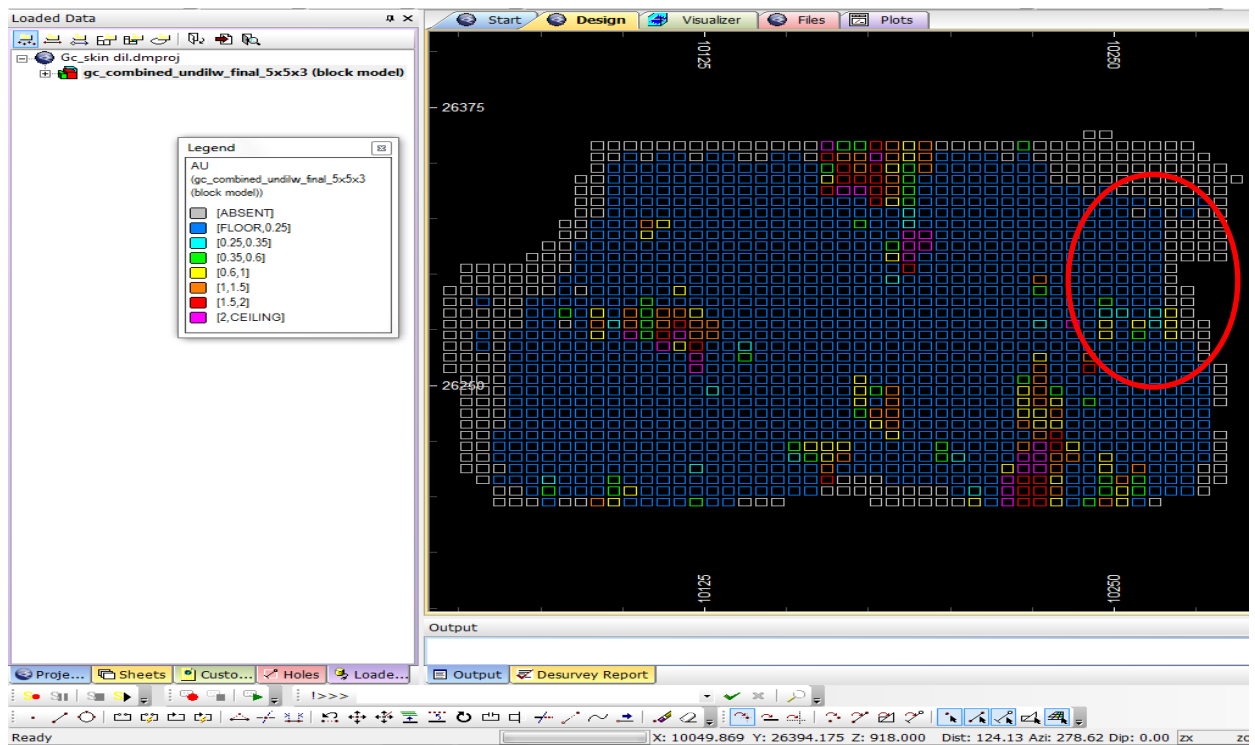


Fig. 4.18: Plan view of Grade Control Model showing the 5mx5mx3m Block Size

#### 4.6 TONNES AND GRADE REPORTING

Ordinary kriging methodology was used in the generation of the Grade Control Model for the two Block Model Sizes (2mx5mx3m and 5mx5mx3m) in the study area (Saddle Pit). This was based on the statistical characteristics of the data, the spatial anisotropy of the data as well as reduction in error variance (Dowd, 1992). The mineralisation style also contributed significantly since the ore body at the saddle pit area were structurally controlled as indicated by narrow shallow dipping quartz veins.

Table 4.7 and Table 4.8 show the various tonnages and grade at a cut-off grade between 0 and 1 at intervals of 0.2 for the two block model sizes .These values were used in plotting a grade tonnage curves as indicated in Figure 4.19 and Figure 4.20 for the 2 m x5 m x 3 m and 5 m x 5 m x 3 m block model sizes respectively.

From Table 4.7, the 2m x5m x 3m at a cut-off grade of 0g/t gave 2,454,458t @0.43g/t with metal content 1,050,465.56g which is more than the 843,851.6g obtained from the 5m x 5m x 5m Block Model Size with 3,033,501t @0.28g/t at the same cut-off grade of 0g/t indicated in Table 4.8.

Table 4.7: Grade Tonnage Curve Data for 2mx5mx3m Block Model Size

<b>2mx5mx3m Block Model Size</b>			
<b>Cut off grade</b>	<b>Tonnes</b>	<b>Ave Grade g/t</b>	<b>Metal (g)</b>
<b>0</b>	<b>2,454,458</b>	<b>0.43</b>	<b>1,050,465.56</b>
<b>0.2</b>	<b>643,751</b>	<b>1.54</b>	<b>989209.02</b>
<b>0.4</b>	<b>541,027</b>	<b>1.77</b>	<b>959288.39</b>
<b>0.6</b>	<b>461,920</b>	<b>1.99</b>	<b>920006.80</b>
<b>0.8</b>	<b>381414</b>	<b>2.26</b>	<b>863206.98</b>
<b>1</b>	<b>310656.9</b>	<b>2.57</b>	<b>799884.84</b>

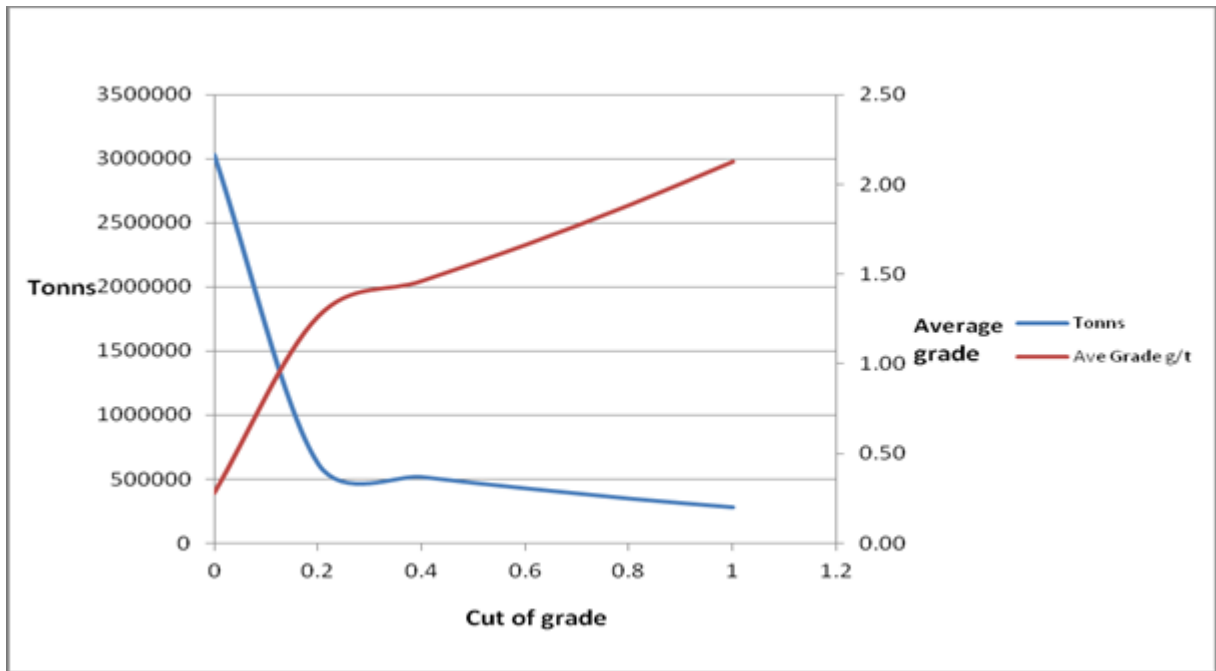


Fig. 4.19: Grade tonnage curve for 2mx5mx3m Block Model Size

Table 4.8: Grade Tonnage Curve Data for 5mx5mx3m Block Model Size

5mx5mx3m Block Model Size			
Cut off grade	Tonnes	Ave Grade g/t	Metal(g)
0	3,033,501	0.28	843,351.6
0.2	621129	1.26	784432.8
0.4	515603.25	1.46	753662.2
0.6	426513.75	1.66	709460.9
0.8	346919.25	1.89	654198.7
1	278649.75	2.13	593071.3

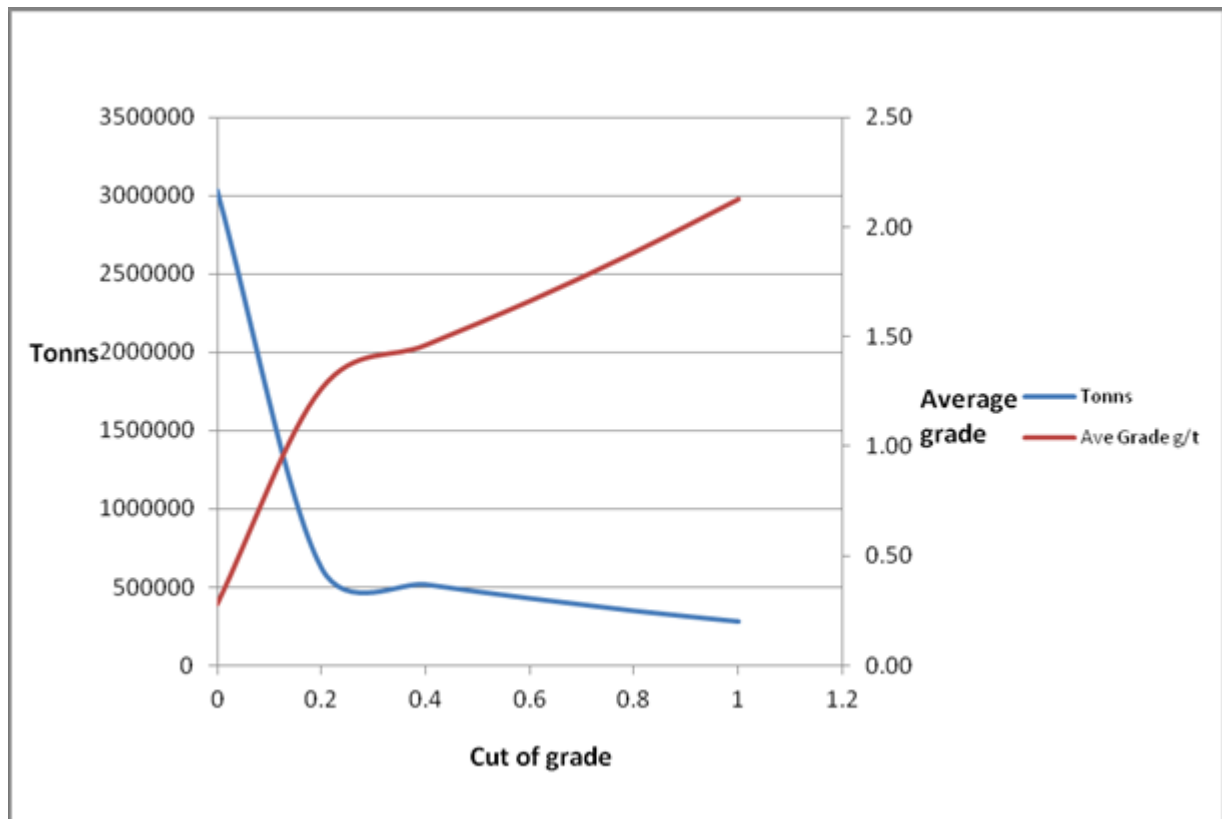


Fig. 4.20: Grade tonnage curve for 5mx5mx3m Block Model Size

#### 4.7 EQUIPMENT SIZES AND DILUTION

The two major excavators used in the project area are the RH120E O&K and Liebherr 984 having respective bucket widths of 3.5 m and 2 m as indicated in Fig 4.21 and Fig 4.22

Dilution is waste that is not segregated from ore during mining, thus decreasing the grade of the ore and increasing the tonnage. Ore losses are due to the inability of mining methods to mine accurately and to segregate waste from the block sizes. Dilution is most significant in deposits with sharp contacts between high-grade ore and waste.

Therefore using the excavator bucket width of 3.5 m to mine the 2 m x 5 m x 3 m block model size would increase dilution as it would be difficult to mine waste effectively from a 2 m x 5 m x 3 m block model size due to the size of the bucket width but 3.5m bucket width can mine the 5 m x 5 m x 3 m block model size with minimal dilution. On the other hand the 2.0m

excavator bucket width would be able to mine both the two block model sizes: 2 m x 5 m x 3 m and 5 m x 5 m x 3 m. But since the kriging efficiency, the regression slope and the metal content obtained are higher for the 2 m x 5 m x 3 m than that of the 5m x 5 m x 3 m block size, the Liebherr 984 with bucket width of 2.0 m should be used in the mining as this would minimise dilution.

In addition the style of Au mineralization at the saddle pit which are hosted within narrow, (<0.5m) shallow- dipping, and laterally discontinuous quartz veins requires an excavator with smaller bucket size to effectively mine.

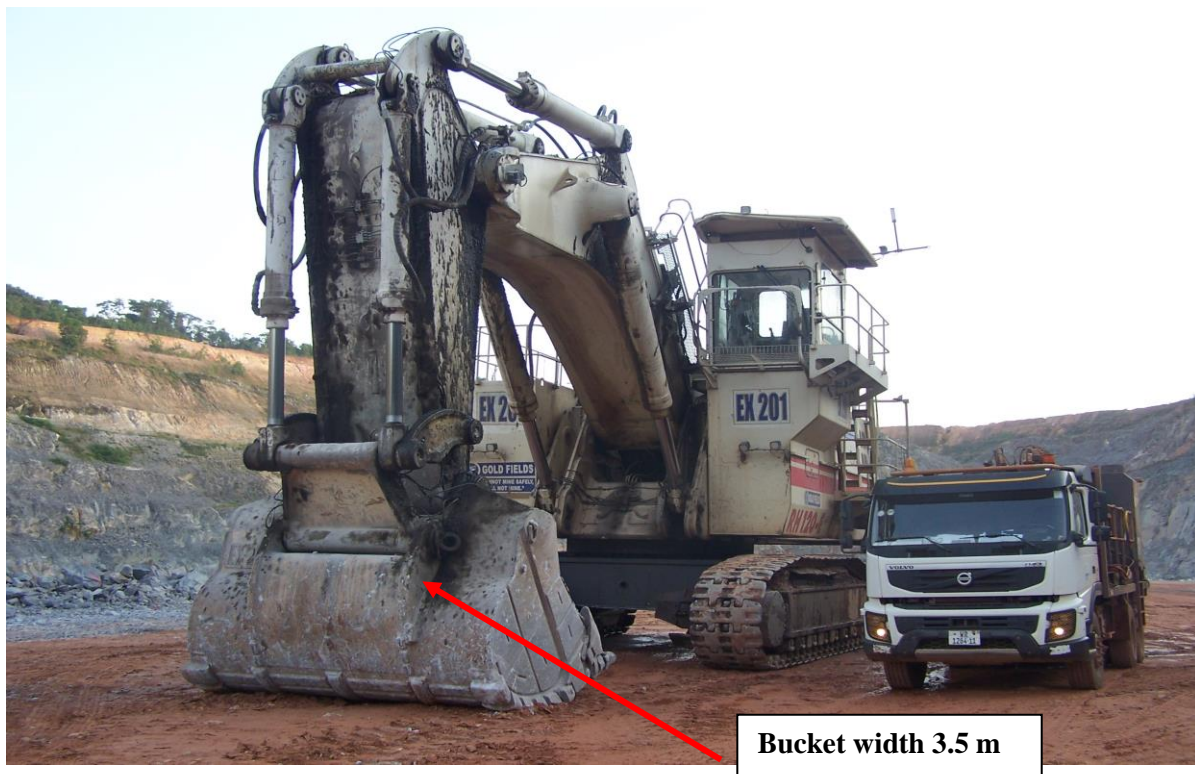


Fig.4.21:RH120E O&K Excavator



Fig.4.22: Liebherr 984 Excavator

## CHAPTER FIVE

### CONCLUSION AND RECOMMENDATION

#### 5.1 CONCLUSION

The main objective of this thesis is to generate grade control model using Smallest Mining Unit (SMU) of 2m x 5m x 3m and 5m x 5m x 3m and estimate the tonnes and grade of ore for the Saddle pit as well as to determine which excavator size be used during mining to reduce dilution.

In summary four lithological zones and seven domains were identified based on their geological and statistical characterizations within the project area. The lithologies are Huni sandstone (PSSU), Dolerite Intrusive (PMDO), Phyllite (PSLP) and Siltstone (PSLS).

Ordinary Kriging methodology was used in the estimation process.

From the estimation the following conclusions can be made:

The krig efficiency of 2m x 5m x 3m and 5m x 5m x 3m was respectively 50-81 % and 45-77 % while the regression slope respectively was 0.8 – 0.99 and 0.7 – 0.98 implying that when the grade control drilling spacing is maintained and the SMU is increased there would be decrease in the efficiency and the regression slope of the estimator.

The variogram model parameters for both block model sizes did not really change much in respect to the sill and the nugget values.

At a cut-off grade of 0.0g/ t the 2mx5mx3m block size gave 2,454,458t @ 0.43g/ t with metal content of 1,050,465.56g and the 5mx5mx3m block size gave 3,033,501t @ 0.28g/ t. with metal content of 843,851.6g. This gives an extra metal content of 206,614g representing 19.6% increment for the 2m x 5m x 3m. This implies that with smaller Block size there would be a decrease in tonnes but increase in grade as well as the metal content.

The SMU to be used at the saddle pit should be 2m x 5m x 3m as well as the usage of a smaller excavator Liebherr 984 due to the style of mineralisation ( Au hosted within narrow, (<0.5m) shallow- dipping, and laterally discontinuous quartz veins) and the block model size.

## **5.2 RECOMMENDATIONS**

The following recommendation(s) should be looked at:

- There is the need to look at the effects of internal dilution for the kriged block sizes to be mined.
- There is the need to carry out the same exercise at the Damang main pit and Juno pit as the style of mineralisation is different compared to that of the saddle pit which has shallow-dipping discontinuous quartz vein.

## REFERENCES

- Borrough P.A. (1986), Principles of Geographical Information Systems for Land Resources Assessment. Oxford University press, 194 pp.
- Annels, A. E. (1991), *Mineral Deposit Evaluation, A Practical Approach*, Chapman and Hall, London, England. 436 pp.
- Addo, B. (2009), *Annual Environmental Report*, Abosso Goldfields Limited. 64pp
- Addo, E., 2011. *Evaluation of the Huni pit reserve: an application of kriging methodology*. MSc Thesis, Dept. of Earth Science, Univ. of Ghana - Legon, 110pp.
- Al-Hassan, S., 2004. "Mineral Resource Evaluation", *MSc Lecture Notes*, Mining Engineering Department, University of Mines and Technology, Tarkwa 66 pp.
- Annels, A. E., 1991. *Mineral Deposit Evaluation, A Practical Approach*, Chapman and Hall, London, England, 436 pp.
- Anon, 1999. *The JORC Code: Australian Code for Reporting of Mineral Resources and Ore Reserves*, Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia, 16 pp.
- Brabham, G., 1998. The regional geological setting and nature of the Damang stockwork gold deposit – a new type of gold deposit in Ghana. MSc Thesis, Dept. of Geology and Geophysics, Univ. of Western Australia, 101pp.
- Barnes, M. P., 1980. *Computer-Assisted Mineral Appraisal and Feasibility*, AIME, New York, pp. 51-55.

Cole, G. and Mamphey, J., 2004. "From Exploration to Design: Optimizing Damang's Future", *Proceedings from the 3<sup>rd</sup> Annual Mineral Resource and Reserve Technical Conference*, West Wits, RSA, pp. 19-21.

Dowd, P. A., 1992. "Geostatistical Ore Reserve Estimation - A Case Study in a Disseminated Nickel Deposit", in *Case Histories and Methods in Mineral Resource Evaluation*, Geological Society Special Publication, London, (Annels A. E. ed.) No. 63, pp. 243-255.

Hustrulid, W. A., 1982, "Ore Reserve Calculation" in *Underground Mining Methods Handbook*, Readdy, L.A., Bolin, D.S. and Mathieson G.A., Society for Mining Engineers of AIME, New York, p. 29.

Koch, S. G. and Link, R. F., 1970, *Statistical Analysis of Geological Data*, John Wiley and Sons Inc., New York, pp. 205.

Tunks J. A., Selley D., Rogers, J. and Brabham, G., 2004. "Vein Mineralisation at the Damang Gold Mine, Ghana: controls on mineralization" *Journal of Structural Geology*. Elsevier, pp. 1257-1273.

The Damang Mine Mineral Resource Team (2010), "*Competent Persons' Report on the West African Mining and Exploration Assets of Damang Mine*", pp. 144.

White, A. J. R. et al., 2010. Portable infrared spectroscopy as a tool for the exploration of gold deposits in tropical terrains: a case study at damang deposit, Ghana. Society of economic geologists inc. Special publication 15, pp. 67-86.

Mamphey, J., Assibey-Bonsu and Muller, C., 2005. *Resource Modelling and Estimation Methodology for Mineral Resource Management at Abooso Goldfields Ltd.* Unpublished methodology for SOX procedures at Damang, pp. 3-40.

Mamphey, J, *Geostatistical Evaluation of a Mineral Deposit- A Case Study at Goldfields Ghana, Damang Gold Mine*, pp. 156

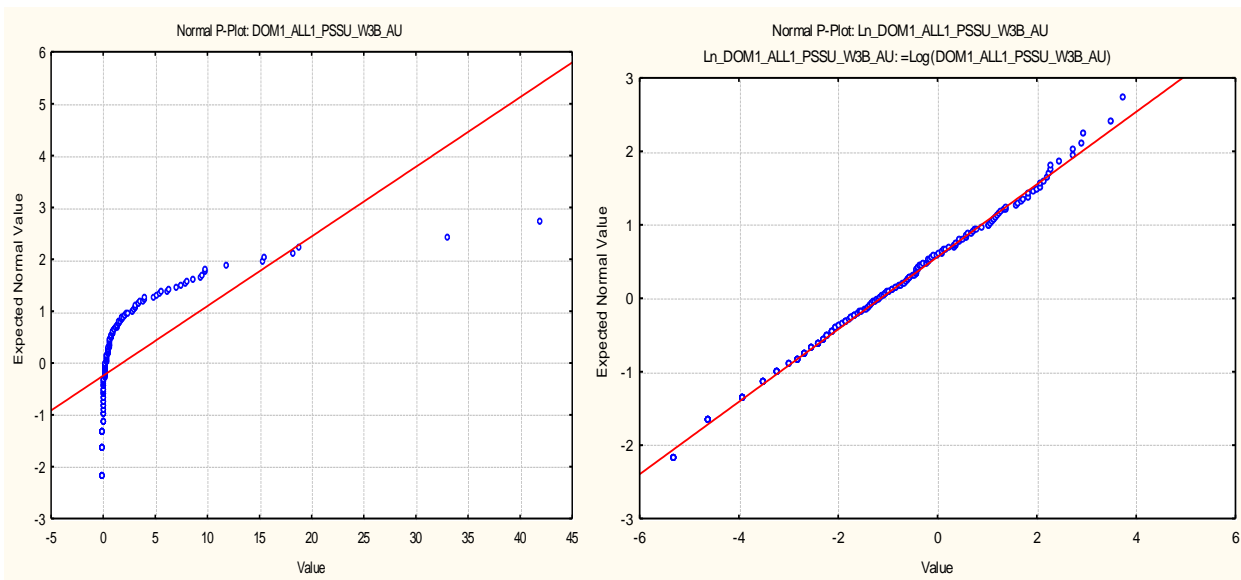
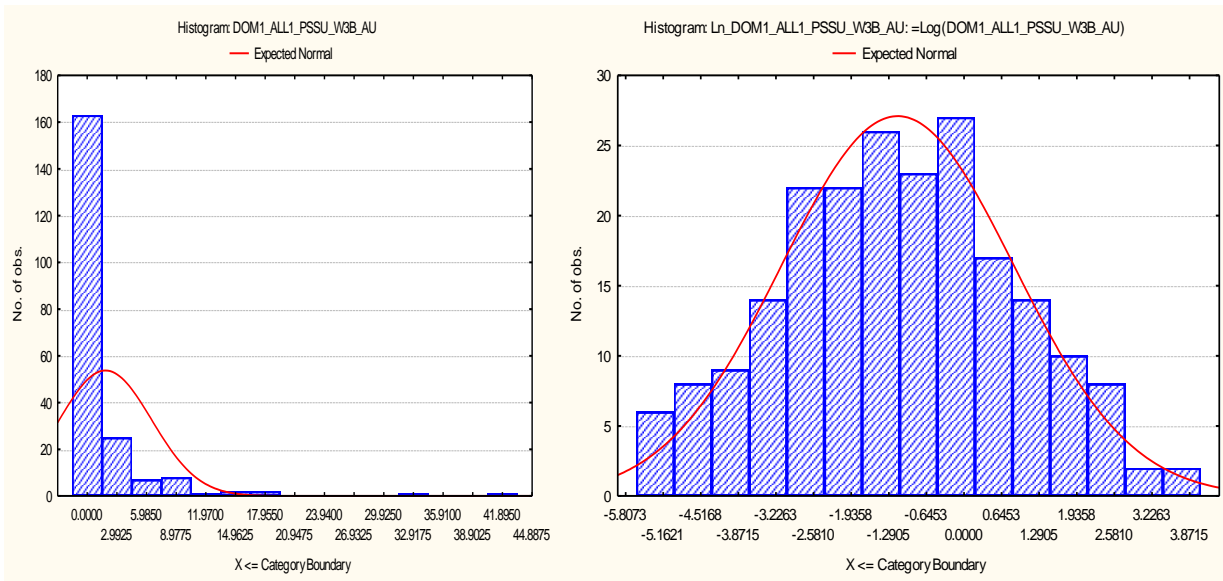
Rendu, J. M., 1981. *An Introduction to Geostatistical Methods of Mineral Evaluation*, South African IMM (Publishers), Johannesburg, 84 pp.

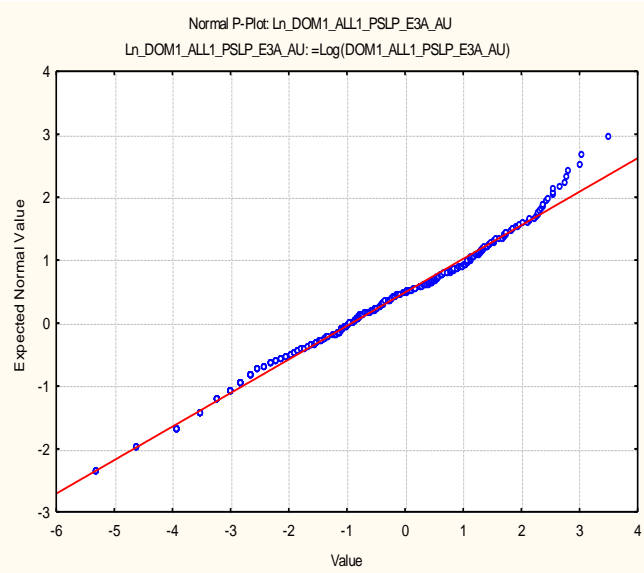
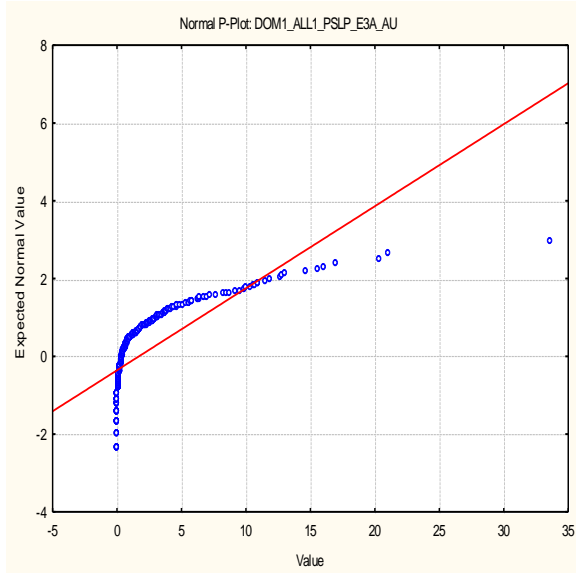
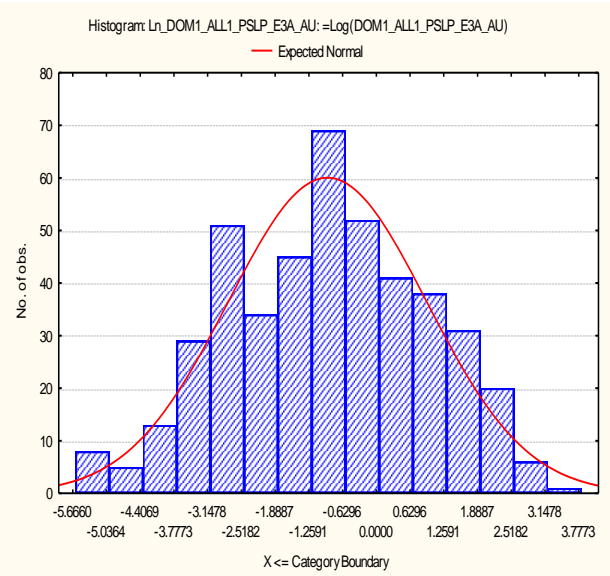
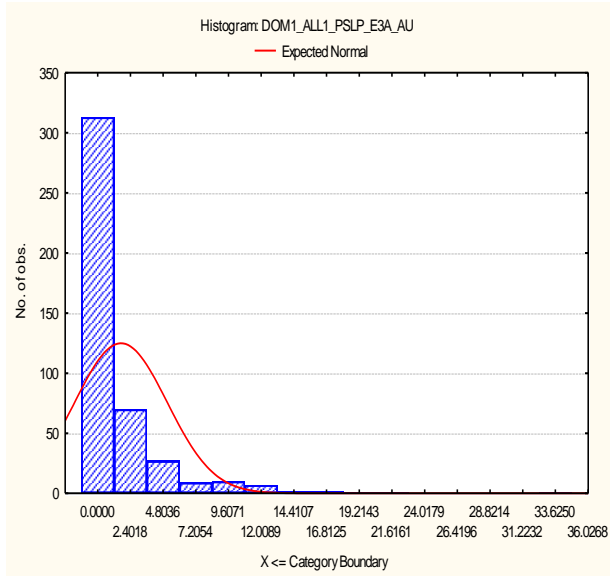
Royle, A. G., 1971. "A Practical Introduction to Geostatistics", *Course Notes*, University of Leeds, Dept. of Mining and Mineral Sciences, Leeds, Unpublished, p.89.

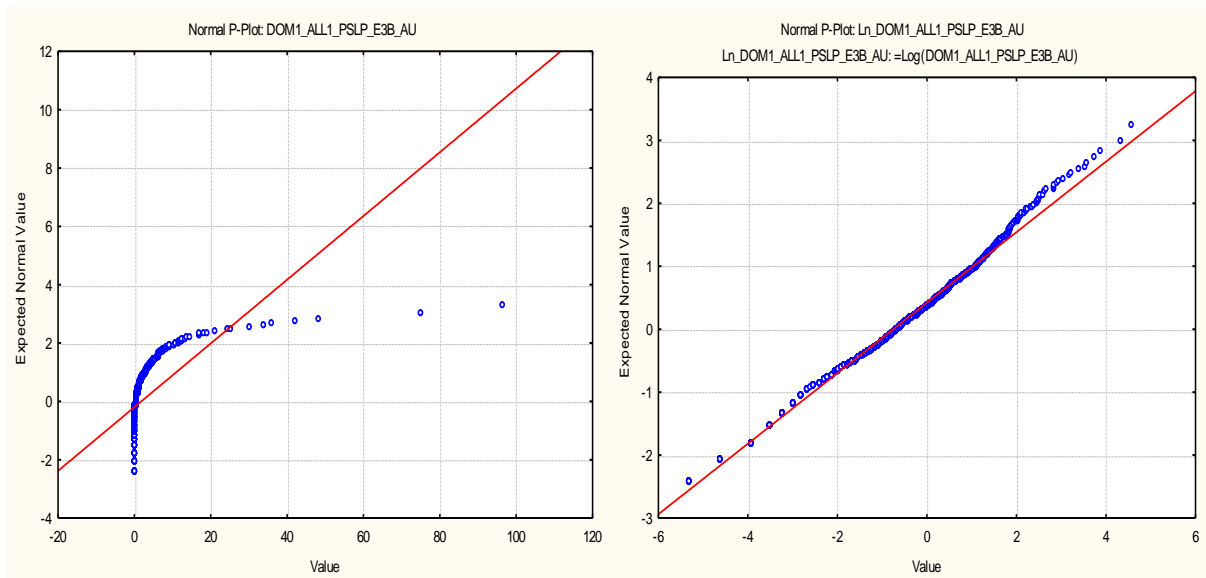
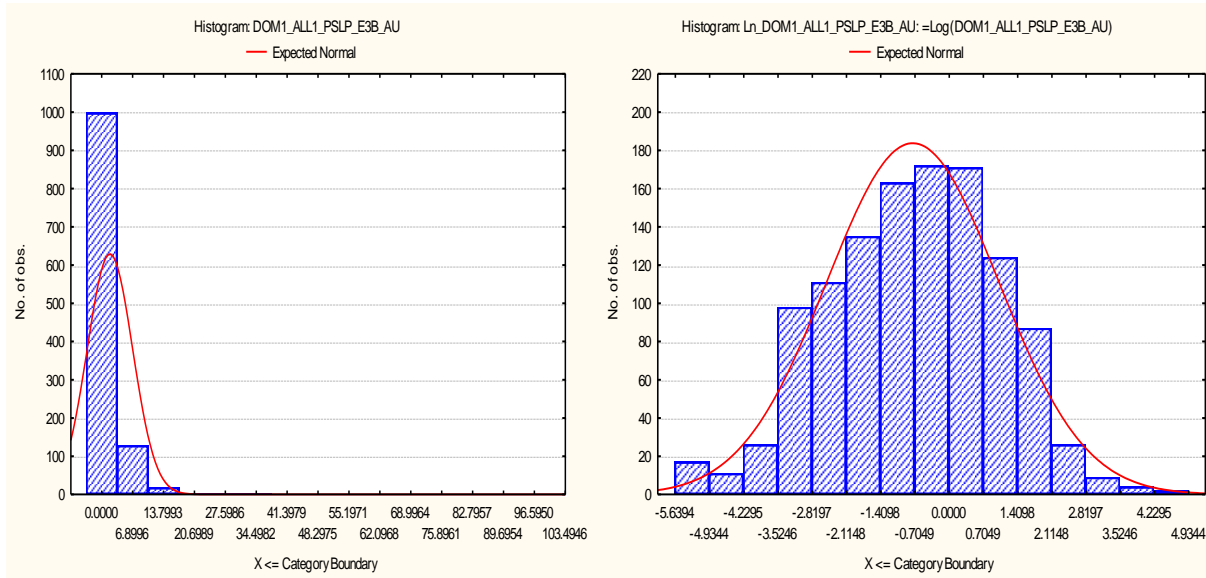
RSG Global, 2005. *Drilling, Sampling and Assaying Procedures Manual*, pp. 80

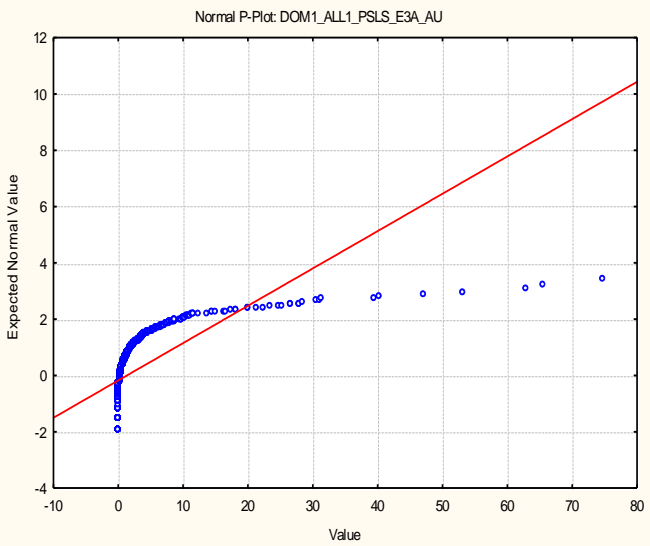
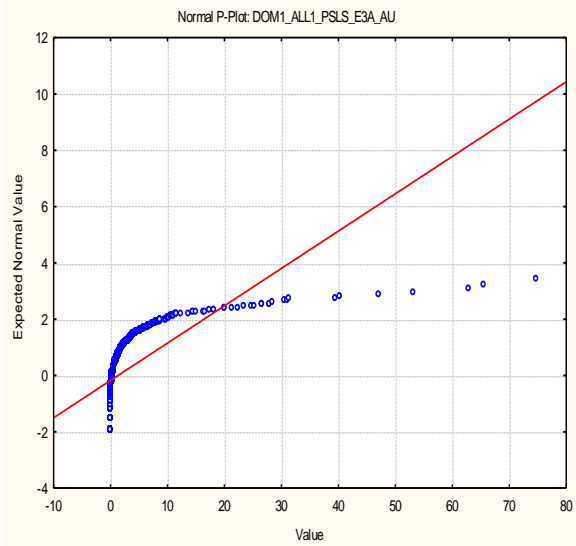
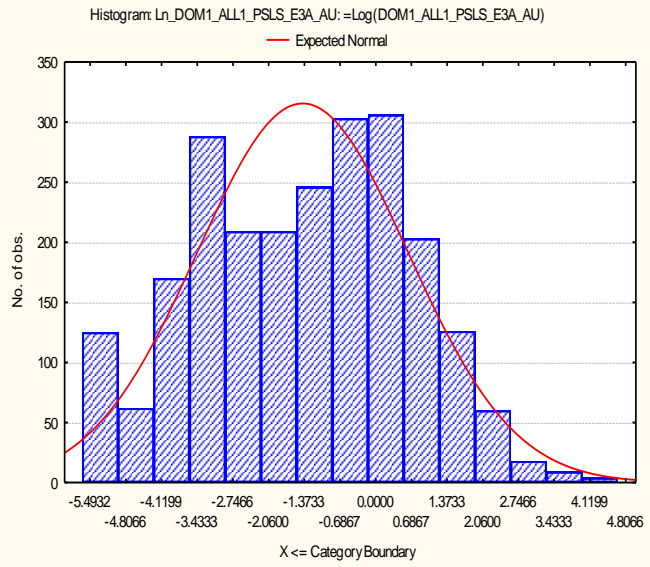
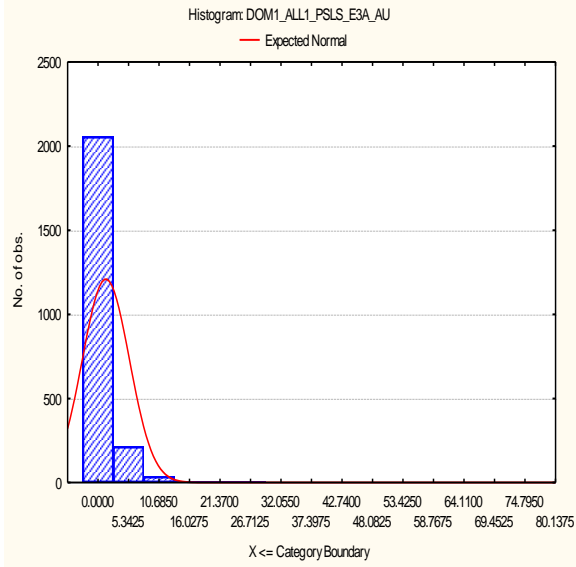
**APPENDIX**

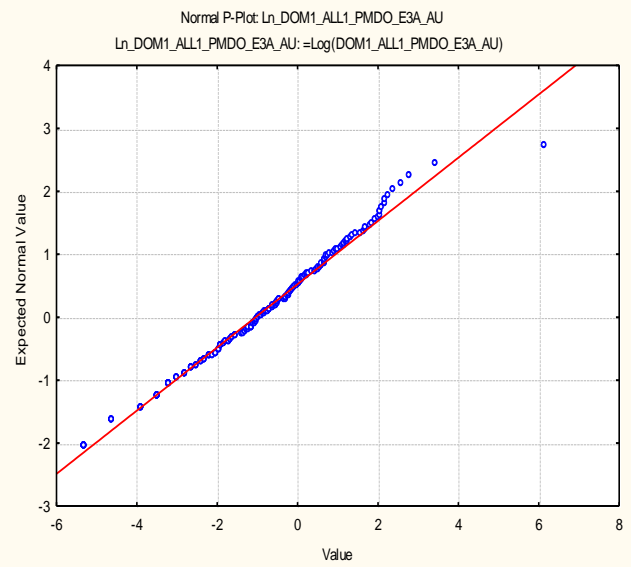
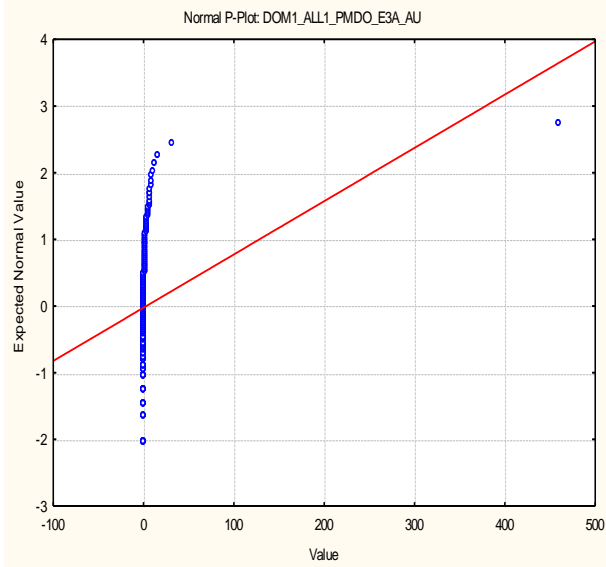
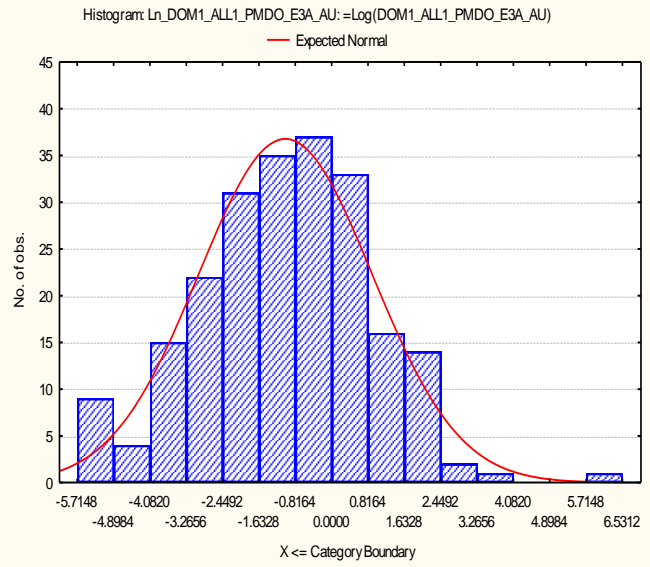
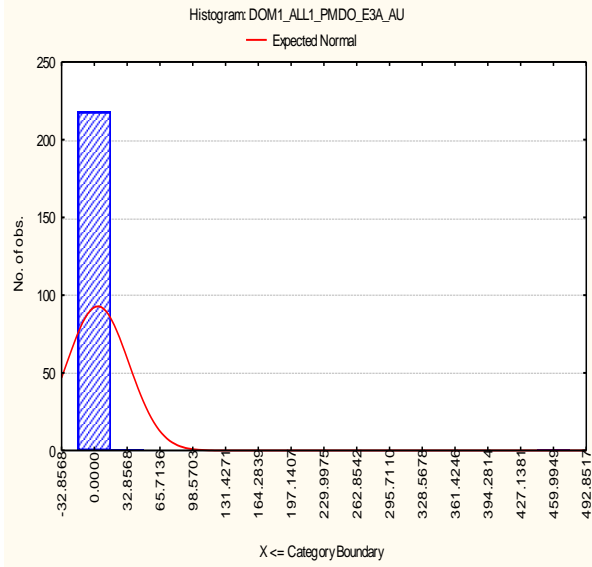
**HISTOGRAM, NORMAL AND PROBABILITY PLOTS FOR DOMAINS (PSSU\_W3B, PSLP\_E3A, PSLP\_E3B, PSLS\_E3A, PSLS\_E3B, PMDO\_E3A, PMDO\_E3B AND W) FOR THE 2M X 5M X 3M BLOCK SIZES**

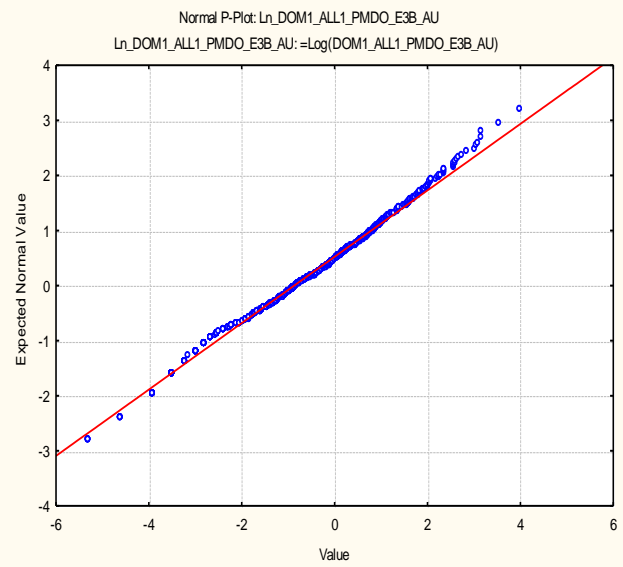
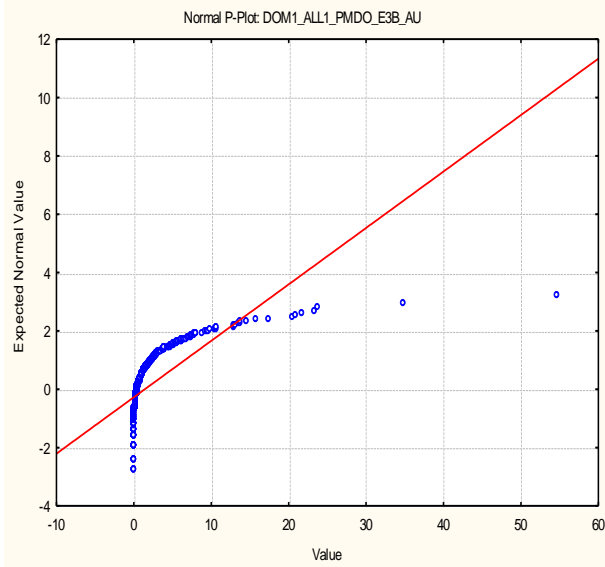
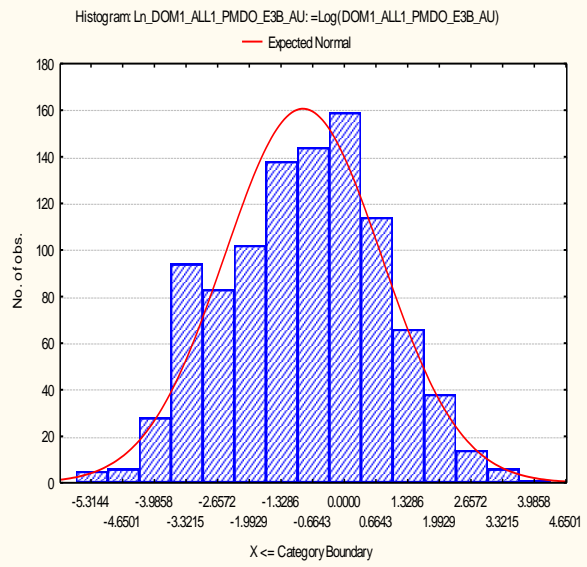
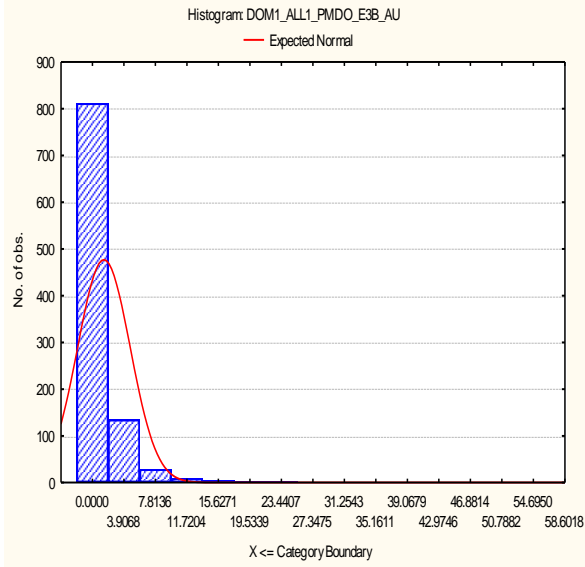


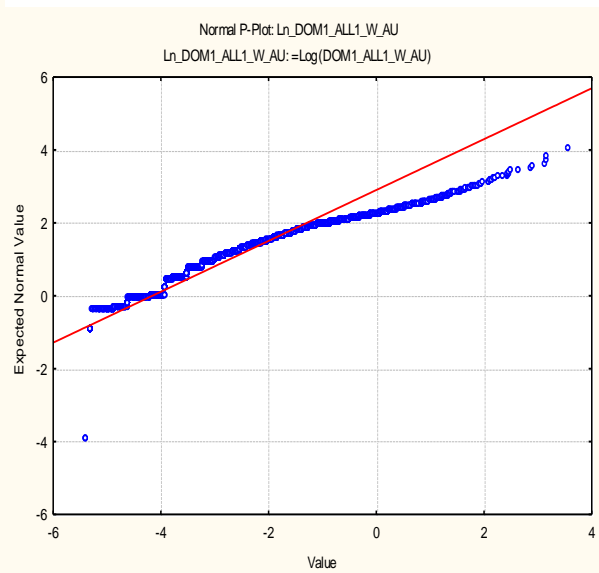
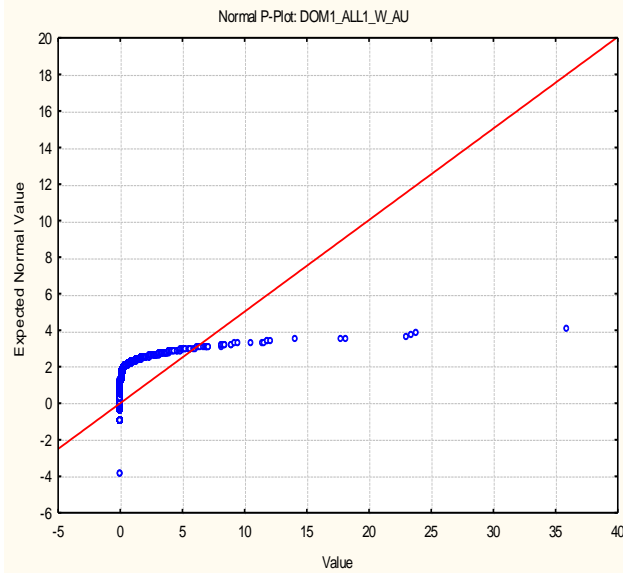
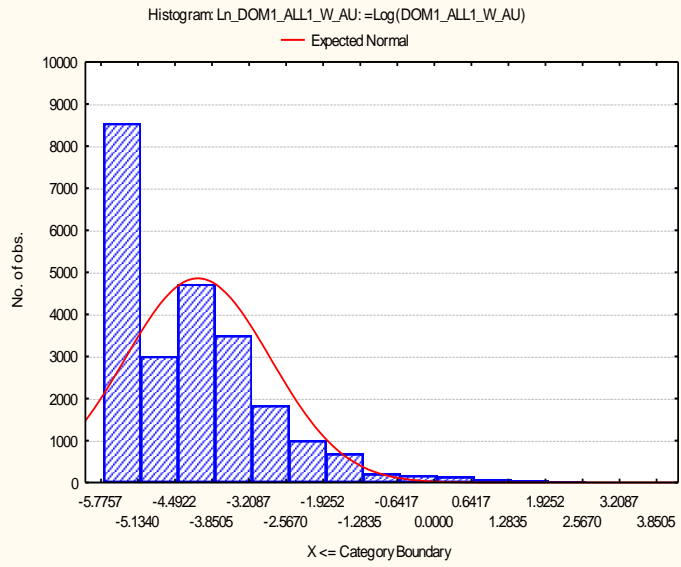
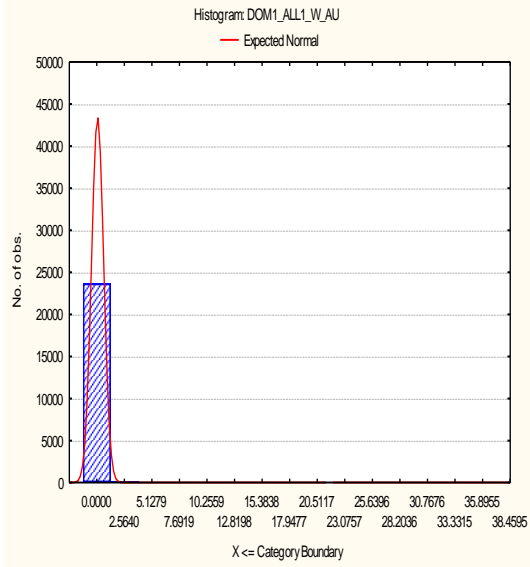












**HISTOGRAM, NORMAL AND PROBABILITY PLOTS FOR DOMAINS (PSSU\_W3B, PSLP\_E3A, PSLP\_E3B, PSLS\_E3A, PSLS\_E3B, PMDO\_E3A, PMDO\_E3B AND W) FOR THE 5M X 5M X 3M BLOCK SIZES**

