UNIVERSITY OF GHANA

EVALUATION OF KOBADA NORTH GOLD RESOURCE, MALI, USING TWO DIFFERENT DRILL DIRECTIONS

BY

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DECLARATION

This dissertation is an outcome of a research undertaken by James Yaw Nkansah in the Department of Earth Science, University of Ghana, Legon. Under the supervision of: Dr Johnson Manu and Dr Sandow Mark Yidana.

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

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DEDICATION

This thesis is dedicated to God Almighty for His unending Love through His Son and my Saviour Jesus Christ, without Him I am nothing.

This thesis is also dedicated to my family, my wife Sarah and my daughter Maame Somuah-Nkansah.

To my grandmother and mother who through their toys made sure I had a strong educational footing. To my sisters Lucy, Linda and Deborah, I say I very much remember your love and sacrifices.

I will lift up mine eyes unto the hills, from whence cometh my help.
My help cometh from the LORD, which made heaven and earth - ***Psalm 121: 1-2***

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ABSTRACT

The 22.5 km^2 Kobada project which is located in the Kangaba Cercle of the Koulikouro region of Mali, is owned and operated by African Gold Group Inc. (AGG), a Canadian junior exploration company (Fig. 1.1 shows perimeter for 22.5 km^2; fig. 1.9 shows perimeter for consolidated permit of 215 km^2).

Notably the permit hosts a soil anomaly greater than 500 ppm arsenic that extends over 12 km of strike length and one kilometre of width, which is offset to the west of extensive artisanal surface saprolite and placer gold mining activity. Most importantly just about two kilometres of this strike length host the Kobada gold zone referred to as "Kobada Zone 1" which has over 1.6 million oz (oxide) gold in 59.1 million tonnes grading 0.87 g/t at the 0.3 g/t cut-off and still expanding in both directions along strike and at depth.

Historically, the major drilling direction in the Kobada zone 1 have been on azimuth 290°. However recent structural investigations indicate that whilst the Kobada Shear Zone exhibits a N20°E (easterly dipping) strike, the mineralisation most actually occurs in quartz vein swarms with an average strike orientation that is E-W (azimuth 190°-200°), confined within the boundaries of the Kobada Shear Zone. There is an area in the northern most 400m of the current resource where it is covered with drill information of both orientations by which the project sets out to test which drilling direction will yield the most gold resource.

Gold mineralisation has been constrained by the development of wireframes modeled at a 0.2 g/t Au lower cut-off grade at an average dip of 74° respectively for both drilling directions.

For subsequent analysis and estimation, two geostatistical domains, the saprolite (modelled at an average dip of 74°) and the duricrust domain (a roughly flat lying horizontal hard pan on top of the saprolite) have been used.

Variography was completed on the 2m composite for the duricrust and saprolite domains for each drilling direction.

Using the ordinary kriging methodology, drilling to the west is relatively continuous and hence more viable than drilling south, with the west yielding in excess of 15,000 ounces more in the various respective grade cut offs.

Geological and structural models have been recommended as well as closing in the drill spacing to help establish more confidence in the resource estimate.

TABLE OF CONTENTS

Contents **Page**

CHAPTER TWO

CHAPTER FOUR

CHAPTER FIVE

LIST OF FIGURES

LIST OF TABLES

CHAPTER ONE

INTRODUCTION OF THESIS

1.1 BACKGROUND

African Gold Group Inc. (AGG) has several contiguous concessions situated in Mali, West Africa of which the Kobada Gold Prospect is an integral part and the most advanced. The Kobada prospect is comprised of 22.55 km^2 of land located in proximity to Kobada (a small artisanal mining settlement) in the Kangaba Cercle of the Koulikouro region of Mali (Figure 1.1).

Notably the permit hosts a soil anomaly greater than 500 ppm arsenic that extends over 12 km of strike length and one kilometre of width, which is offset to the west of extensive artisanal surface saprolite and placer gold mining activity. Most importantly just about two kilometres of this strike length host the Kobada gold zone referred as "Zone 1" which has over 1.6 million oz (oxide) gold in 59.1 million tonnes grading 0.87 g/t at the 0.3 g/t cut-off and still expanding in both directions along strike and at depth.

Until recently, the major drilling direction in the Kobada main zone was on azimuth 290° and all pre-2011 drilling campaigns have been drilled on this azimuth. The resource estimates that have been done have been based on drilling with this azimuth.

However, recent structural investigations indicate that whilst the Kobada Shear Zone exhibits a N20°E (easterly dipping) strike, the mineralisation also occurs in quartz vein swarms with an average strike orientation that is E-W, confined within the boundaries of the Kobada Shear Zone. These vein swarms are envisaged to be extension fractures or tension gashes within a strike slip shear zone. If the hypothesis is right that the E-W is the main mineralisation event then the 290° azimuth drilling lies parallel to the main event and will not give a true reflection of the mineralisation. Due to this structural interpretation the 2010-2011 drilling campaign which was directed at the N and S extensions of Zone 1 was carried out with southerly directed (azimuth 190°)

drill holes. There is an area in the northern most 400 meters of the current resource where it is covered with drill information of both orientations (Figures 1.2 & 1.3). Management of AGG therefore would like to compare the two drilling directions in this area of Zone 1 to see if there could be any significant difference in grade and tonnage between the 2 sets of data.

1.2 PROJECT SETTINGS

1.2.1 Location and Accessibility

The Kobada project is located approximately 125 km south of Bamako, the capital city of Mali It is part of the Kangaba Cercle within the Koulikouro region. The project which covers km^2, is bound in close proximity to the Niger River to the north and is bound to the east by the Fié River. The international border with the Republic of Guinea is located in close proximity to the south (Figure 1.1)

The property is accessible from Bamako, the capital partly by both bituminous and dirt road access. Access is gained by following the main paved highway South to Sikasso for 80 km to the Selingue road junction. The Selingue junction is 60 km long, paved and is in good condition. After crossing the hydroelectric dam, there is 52 km of laterite track leading to Kobada village located within the property. This section of the road is in poor shape, but can be travelled by 2-wheel drive vehicles in the dry season (November to July). During the rainy season, 4 wheel drive vehicles are required. However, vehicular crossing of the Fié River during the peak rainy season (August - September) is rendered impassable as water level can be as high as two meters above the bridge deck and may cover about two kilometers stretch of the access road. The concession can only be accessed by small canoes (Brian and Lafleur, 2008).

1.2.2 Climate

In southern Mali, the rainy season extends from June through October, generally in the form of thunderstorms with an average of one storm per day during the peak of the season. The countryside becomes lush and green with grass attaining 2 meters in height. By December, the vegetation dries up and the grass is burned by the villagers. The land returns to its brown and yellow semi-desert state. Temperatures can easily reach over 40°C during the day in the dry season. Average annual rainfall for this part of the country is about 1,550 mm (Brian and Lafleur, 2008).

1.2.3 Local Resources

Local villagers are available and keen to work as labourers. Trades persons such as carpenters, masons, welders, mechanics, etc. have businesses in Selingue and are readily available for hire. Educated and qualified personnel can be recruited in Bamako and in some of the villages within the Kobada vicinity (Archambault, 2006).

1.2.4 Infrastructure

The property is approximately 50 km away from the Selingue Hydroelectric Dam . The town of Selingue, possesses numerous amenities including a hospital, a gas station, a hotel and restaurant, and numerous small shops. In the property area, the main industries are small-scale cotton growing, fishing, and artisanal gold mining (Archambault, 2006).

1.2.5 Physiography

A series of lateritic plateaus characterises the area and they are deeply cut by a well defined drainage system. Elevations in the project area vary between approximately 350 meters and 430 meters. A few kilometers to the northwest, the Niger River is prominent north-easterly flowing water system that drains most of north-western Africa. Most of the drainages on the property flow to join the Niger River.

The vegetation is that of savannah, a sparsely forested plain on the margins of the tropics where the rainfall is seasonal (Lalande, 2010).

1.3 EXPLORATION HISTORY

"Between 1982 and 1988, the area has been the focus of numerous exploration campaigns by the BRGM (Bureau de Recherches Géologiques et Minières). In 1982, a regional soil geochemical survey spaced 1,500 by 500 metres outlined the Kobada gold and arsenic anomaly. In 1983, the grid was tightened to 500 by 100 metres and 200 by 100 metres. During 1986 and 1987, these surveys were followed by detailed grids at 200 by 50 metres and 100 by 50 metres over the anomalous portions of the 1983 grid. A ground geophysical survey was completed by the geophysical department of the BRGM in December 1987. It comprised of: 12 line-km of resistivity and induced polarization (IP); 24 line-km of spontaneous polarization survey, all with readings every 10 metres; and 1,200 line-km of magnetometer surveying. The geophysical surveys were followed by seven diamond drill holes (KOBC1-KOBC7) totalling 913.4 metres.

In 1995-1996, La Source conducted a 50 hole, 4,803 metre reverse circulation (RC) drilling program (KBD01 - KBD50). However, information pertaining to ownership and work completed on the property between 1997 and 2000 was not readily available.

COMINOR acquired the property in 2000 and the Arrêté was issued in January 2001.

Fieldwork by COMINOR started in 2002 with a ground magnetometer survey in October, followed by a RC and AirCore drilling campaign in November and December. The magnetometer survey covered approximately 4 km2 with 21 lines (1,500 metres length) and 8 lines (1,000 metres length) at 100 metre line space and stations at 25 metres. The RC drilling totalled 1,350 metres in 19 holes (KBRC051-KBRC069) varying from 60 to 78 metres in depth, from which 1,336 one metre samples were collected (Lalande, 2010).

Mali
International boundary
Region boundary
National capital
Region capital
Railroad
Road
Track
ALGERIA
MAURITANIA
TOMBOUCTOU
KIDAL
GAO
MOPTI
KAYES
KOULIKORO
SEGOU
SIKASSO
NIGER
BURKINA
SENEGAL
GUINEA
BENIN
Bamako
Kayes
Koulikoro
Segou
Mopti
Kidal
Gao
Niamey
Ouagadougou
Tessalit
Bourem
Ménaka
AGG-Kobada

- 14 holes (1,134 metres) near the eastern outline of the prospect. The holes were disposed along two sections oriented 344° and 323°; and,
- 30 holes (2,190 metres) at a distance of 0.8 to 2.5km to the southwest of the prospect, disposed along three sections oriented 326°, 325° and 336°. The holes were planned to be 80 metre deep with a 50 metres spacing and a –55° dip.

A total of 5,205 samples were collected with a one-metre interval with the exception of holes KBRC094, 099, 101 to 104, and 111 to 120 where every other sample was analyzed due to historical poor results in this area.

This drilling was followed by soil geochemical surveys, resistivity and additional magnetometer surveys, and geomorphological mapping.

The soil survey was conducted in the north and the south of the Kobada prospect and samples were analysed for gold only:

- In the north, 260 samples on a grid spacing of 200 by 100 metres and covering approximately 4.5 km², and later, detailing over 8 existing lines with 42 samples on a grid spacing of 200 by 50 metres. Samples were analyzed for gold only.
- In the south, 220 samples were collected at a grid spacing of 200 by 50 metres.

In April 2004, a resistivity geophysical survey was conducted in three blocks of 11 lines each, at the Kobada prospect area, and a fourth block to the northeast. A ground magnetometer survey was performed at the same time in order to extend the 2002 survey towards the south to the limit of the concession (this is as presented by Lalande, 2010).

An additional RC campaign totalling 1,820 metres in 23 holes (KBRC135-KBRC157) targeted the northeast area of the property. The holes were oriented 293° on sections 2200N, 2400N, 2600N and

295° on section 3400N. They were planned at a dip of -55° and a spacing of 40 metres. A total of 1,804 samples were collected.

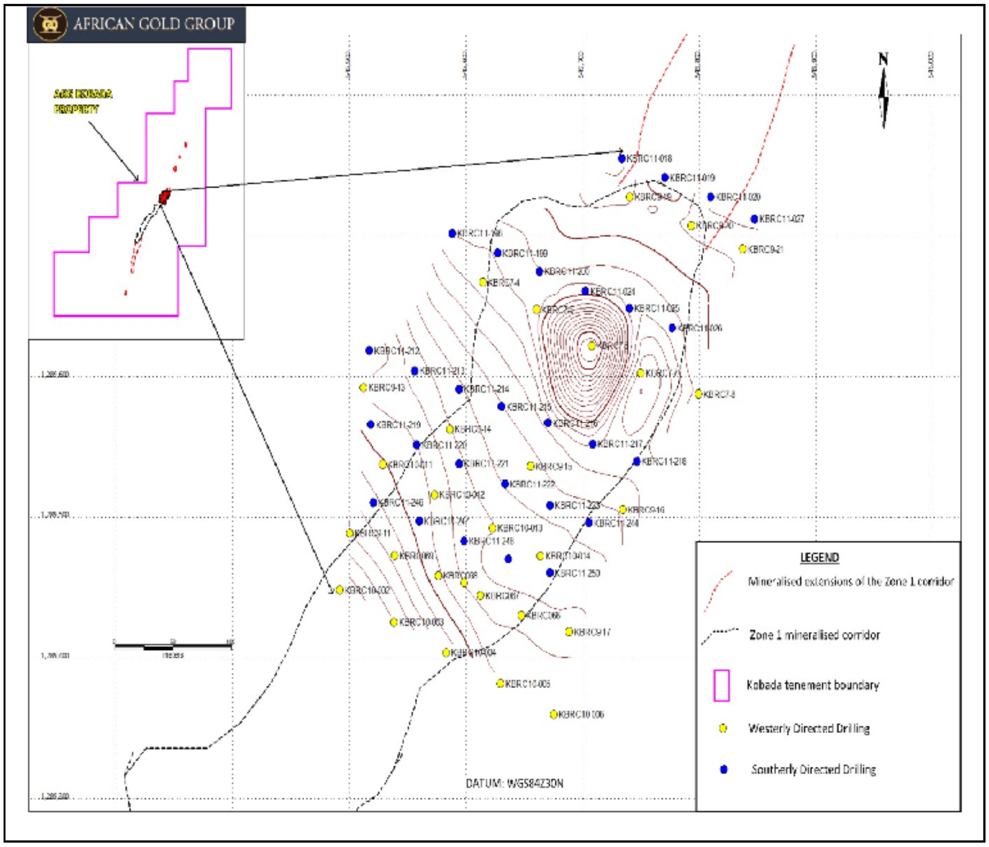
No further work has been conducted on the concession until AGG's work in 2005."

In late 2005, AGG retained the services of CME & Company, Vancouver, Canada to:

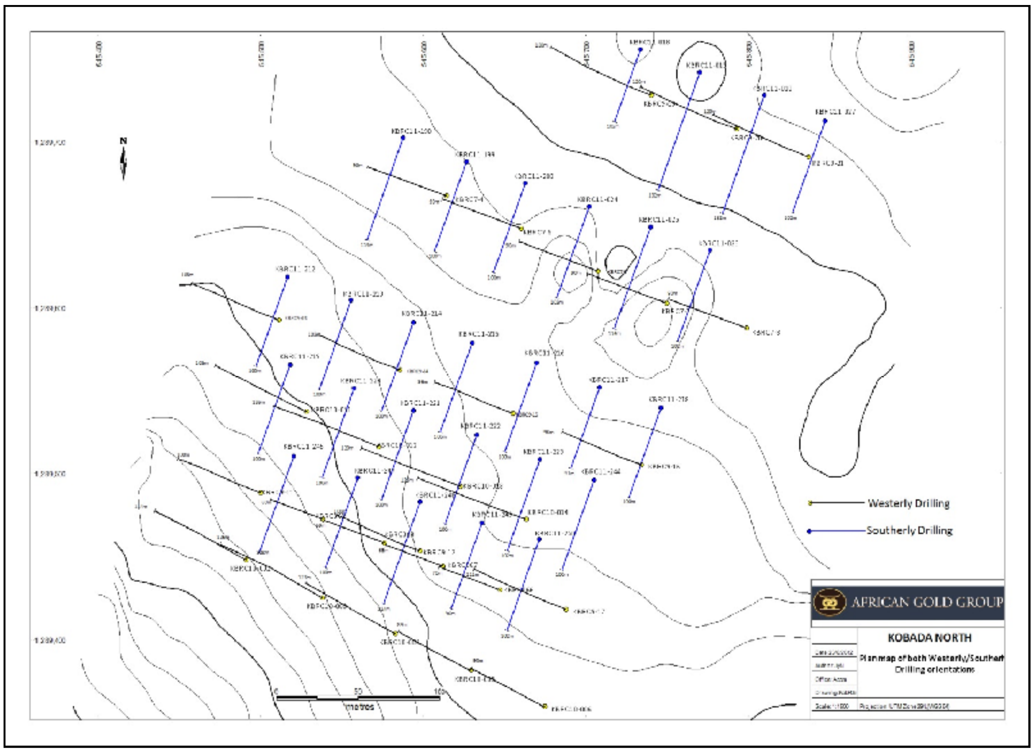
1. Compile all historical work on the Kobada Project.
2. Verify the historical data by Cominor through re-sampling rejects and re-analysis for gold and multi-elements.
3. Carry out a differential GPS survey to accurately locate the historical drill holes on the property.
4. Twin some historical holes and test mineralisation at depth, below the oxidation in Zone 1 and exploratory holes in the Northeast Area of the permit. A total of 10 holes were drilled for 1,519m with 1,148 samples collected.

These tasks confirm the validity of Cominor's data bases and demonstrated continuity of the mineralisation in Kobada Zone 1 prospect into lower saprolite to vertical depths of 100m and into bedrock to 125m depths.

The services of CME & Company continued until 2008 when AGG's mainstream exploration team took over and yet sourcing the services qualified industry consultants to help develop the Kobada gold potential up to this present (Lalande, 2010).



AFRICAN GOLD GROUP
N
LEGEND
Mineralised extensions of the Zone 1 corridor
Zone 1 mineralised corridor
Kobada tenement boundary
Westerly Directed Drilling
Southerly Directed Drilling
DATUM: WGS84Z30N


Westerly Drilling
Southerly Drilling
AFRICAN GOLD GROUP
KOBADA NORTH
Planmap of both Westerly/Southerly Drilling orientations

1.6 DESCRIPTION OF THE STUDY AREA

1.6.1 Birimian Geology and Mineralisation

As mentioned in the introductory text, the Kobada prospect is comprised of 22.55 km^2 of land located in proximity to Kobada (a small artisanal mining settlement) in the Kangaba region of Mali The area is extensively characterised by intense artisanal mining activity. Just to the South of the AGG camp at 2000S (local grid) a folded quartz vein up to 2m thick outcrops, and the laterised metasediments surrounding this vein preserve much of their Birimian structure. From here to about 3800S, essentially across the headwaters of the Kobada River which has eroded most of the second age laterite cap and mottle, it proved possible to trench into the saprolite where structural data is preserved. Also in the area 2150S to 2400S, artisanal miners have been producing gold from quartz veins in the saprolite. These artisanal pits, the fold outcrop and the trenches were all sampled and mapped and have lead to the following synopsis of the structure and the mineralisation of the Zone 1 ore body at Kobada (Downing, 2010).

There is no easy visible recognition of the occurrence of gold in the Kobada deposit. It can be said with reasonable confidence that gold is associated with quartz veining and mineralised blocks do contain quartz veins, but not all quartz veined blocks contain gold. In the mineralised areas gold occurs in the veins, the selvage of the veins and in the surrounding saprolite. At the present time, it has not been possible to determine consistent grade variations in these various rock types in mineralised sections, but it must be noted that the "orpailleur" (artisanal) in Zone 1 are mining and producing their gold from the quartz veins, they do not process the saprolite they extract whilst pursuing the quartz.

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

The stereogram plots of several hundred measured quartz veins show strikes and dips of all orientations and angles, however three broad populations can be determined. Figures 1.4 and 1.5 are stereogram plots of quartz veins that were also assayed in an effort to determine if specific orientations of veining contained the gold (Downing, 2010).

(1) A N20°E population (Figure 1.4a and 1.4b) which is parallel to the regional foliation and is remarkably consistent at between N10°E and N35°E dipping between 60° and 70° to the East**.** These veins range from 5mm to 1m in width and are often sheared, strongly brecciated and cemented with iron and manganese oxides. In the area of the fold outcrop these veins appear to be mylonitised in places. Quartz veins of this orientation are mineralised but tend to be of low grade, 0 to 1ppm Au with a few <10ppm occurrences.

(2) An E-W population (Figure 1.5a) which includes strikes between N45°E and N135°E but with a concentration between N80°E and N110°E. Probably about 65% of these veins dip to the North at between 60° and 90°, the rest dip steeply to the South. Veins of this population are very common and range between 1mm and 50cm in width, pinch and swell, are quite discontinuous and can be sygmoidal. Occasionally they form stockwork zones up to 3m in width. Almost invariably these veins display a fracture cleavage, and when the regional foliation is visible in the saprolite, this cleavage can be seen to be the same fabric refracted by the quartz vein. The fracture cleavage is commonly stained with red iron oxides, and often the white to translucent grey of the quartz is not visible until the fractured fragments of the vein are broken open. These veins are also often surrounded by limonite rinds which vary from 5mm to 10 cm in width and are often wider than the veins themselves. Sometimes there is no oxide staining and the veins are white-grey in colour. In a few locations the E-W veins are seen to be folded in open folds (Figure 1.6). The E-W veins clearly cross cut the foliation parallel veins and are not as intensely deformed as the latter. As a population, these veins appear to be well mineralised with grades ranging from 0 to 26ppm Au (within the mineralised envelope), and it is these veins that are the target of the orpailleur working in Zone 1 (Figure 1.7). It is thought that these veins may have formed as extensional fractures and tension gashes in the Kobada shear zone.

(3) A sub-horizontal population of veins (Figure 1.5b). Due to the nature of low angle planes and the difficulty of measuring them in a trench environment, this population of 0° to 30° dipping structures has strike directions at all points of the compass, and as yet no preferential orientation has been determined, but mapping and analysis is continuing. These veins vary from 1mm to 10cm in thickness and often form short and sygmoidal features which occur in stockworks, (Figure 1.8) and ladder vein systems. They can form long continuous features which clearly cross cut all other structures seen in the project. The sampling of these low angle structures has so far shown them to be poorly mineralised, 0 ppm to 2 ppm Au within the mineralised envelope, and the stockwerk zones have proven to be barren of gold. The model outlined below, casts this set of veins as tension gashes and extension fractures formed in a reverse faulting environment (Downing, 2010)..

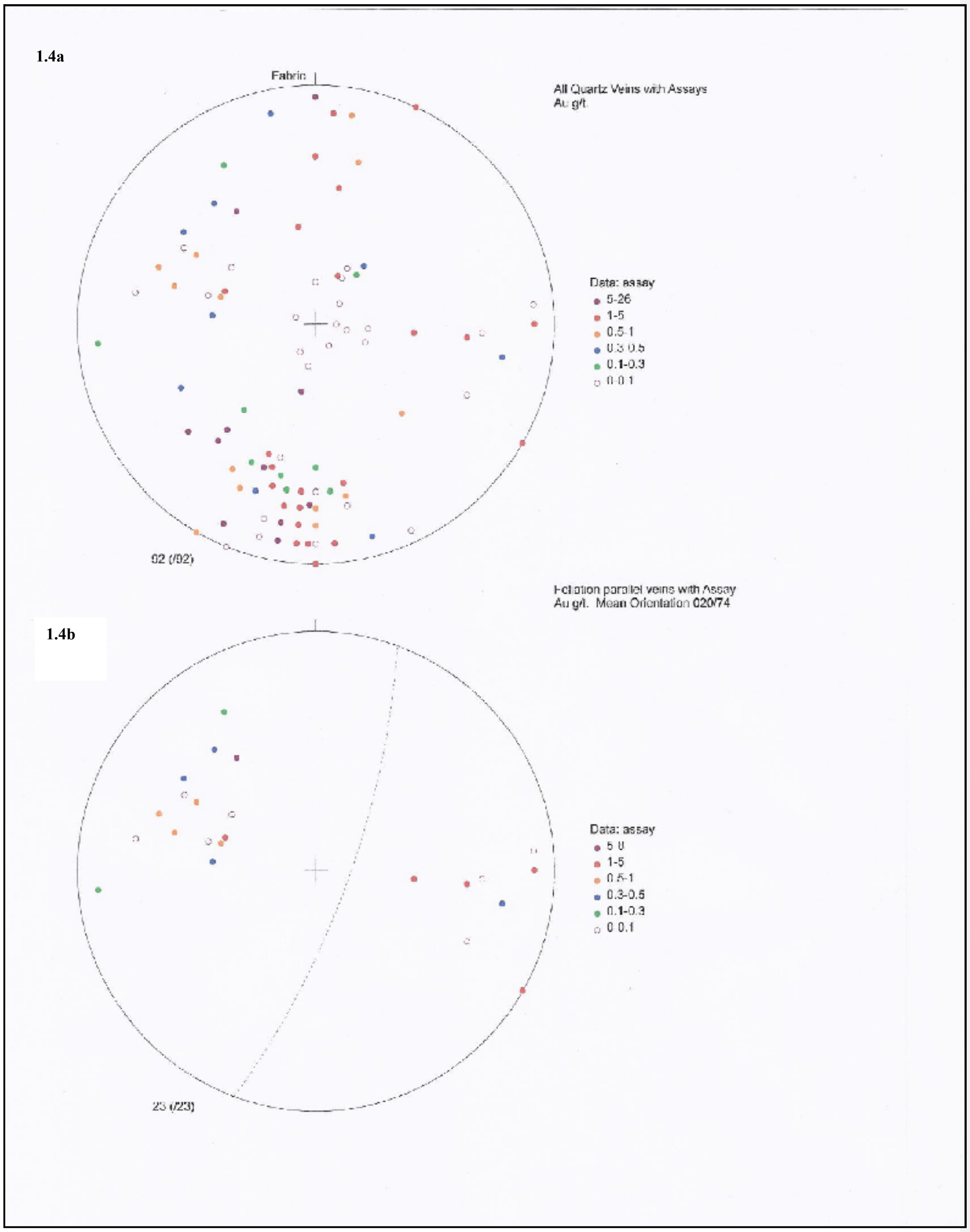


Figure 1.4: Stereographic projection of quartz veins with assay values. Top Figure 1.4a, all quartz veins V1, V2, and V3. Bottom Figure 1.4b, V1 quartz veins parallel to foliation S1.

3a

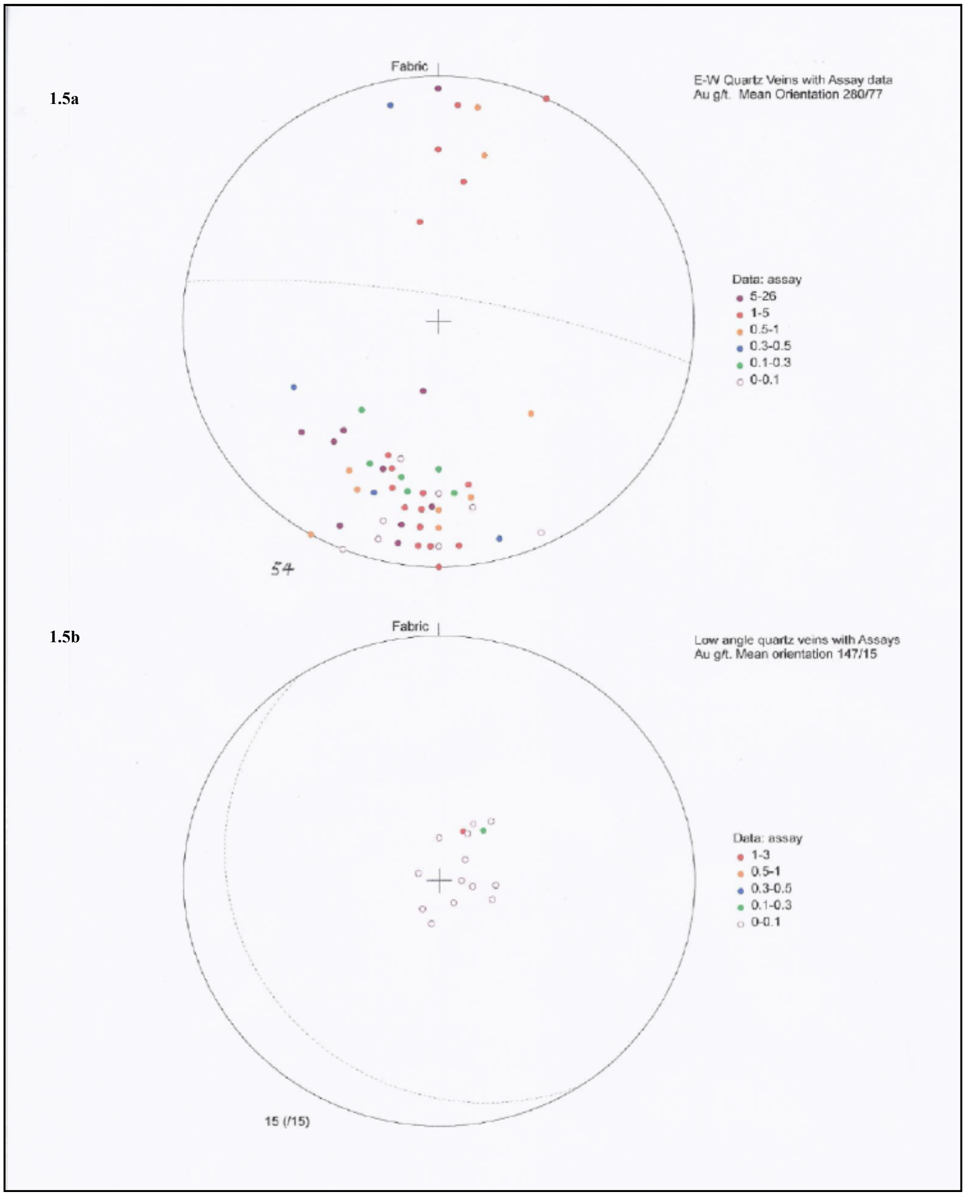
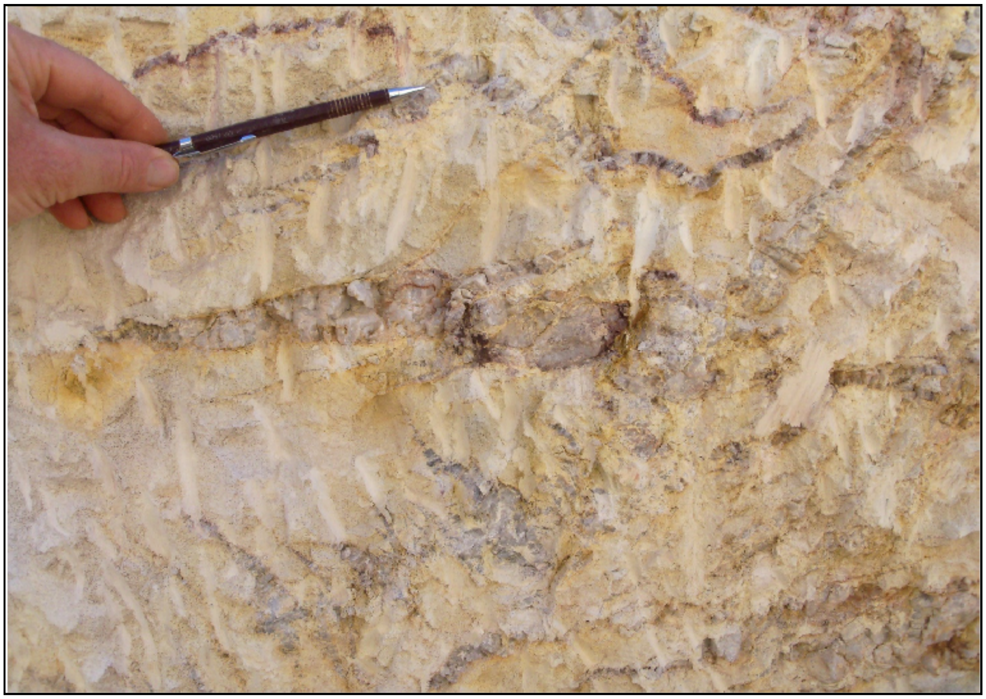


Figure 1.5: Stereographic projection of quartz veins with assay values. Top Figure 1.5a, E-W quartz veins V2. Bottom, sub-horizontal quartz veins V3.


35800


KD024



The crystalline rocks of the West African Craton in Mali are covered by sediments of the Taoudeni basin. These sediments, mainly of Proterozoic to Carboniferous age, are gently folded and slightly metamorphosed during the collision between the West African Craton and the Tuareg Shield (Bronner et al., 1980). The Tuareg Shield is formed by the terrains affected/reworked by the Pan-African orogeny between 600-650Ma (Kennedy, 1964). The sedimentary cover of this shield comprises Paleozoic and Tertiary marine and continental sediments of the Iullemeden basin, to the east and southeast of the Adrar Mountains and the Gao Graben. Crystalline rocks of the West African Craton occur in two large areas in Mali: Bougouni (40,000 km^2) and Kenieba (20,000 km2), respectively in the south and south-west of the country, and in several small inliers. In the Bougouni area, the granite-gneissic rocks to the west of Bagoe River yielded U/Pb ages on zircon of 2136 +/-6 Ma and 2152+/-16Ma, (Liegeois, 1990). The Bougouni Basin contains four Birimian belts. The eastern-most Bagoe belt, hosting the Syama gold deposit, is probably the best studied. It consists mostly of epi-metamorphosed sediments (schists, greywackes), flanked on both sides by metavolcanics (andesites, basalts). In the Kenieba area, granites are few and the area is composed of metasediments: schists, greywackes, minor marbles and several horizons of tourmalinised (gold-bearing) sandstones (Bessoles, 1977).

1.7.1 Evolutionary Models

The lithostratigraphy of the Birimian volcano-sedimentary deposits is the subject of some debate due to the poor quality of the outcrop in most parts of West Africa. Two proposed evolutionary scenarios are outlined below.

(i) Milesi et al. (1989), defined a classical succession found in Ghana. It consists of two major units that were successively emplaced over a time span of about 40 Ma. The first unit (B1) contains locally developed volcanosediments and minor tholeiitic volcanic rocks. This unit was affected by the three main Eburnean tectono-metamorphic phases (D1 to D3). The second unit (B2) consists of volcanic and associated volcanosedimentary rocks and intercalated deposits of fluvio-deltaic units. The second unit was only deformed by the last Eburnean tectono-metamorphic phases (D2 and D3).

The first deformation (D1), around 2,100 to 2,090 Ma, marked an interruption between the sedimentation of the B1 unit and volcanic areas of the B2 unit. Locally, D1 was accompanied by the emplacement of syn-kinematic granitoid rocks. The D1 deformation was responsible for the thrusting of the lower Proterozoic succession over its Archean gneissic basement (Bessoles, 1977).

The present day configuration of the Lower Proterozoic succession and part of the Archean nucleus, is dominated by the effects of two successive Eburnium transcurrent tectonic phases (D2 and D3) that were also responsible for the rise of granitic mantle diapirs between 2080 and 1945 Ma. This tectonism resulted in N-S sinistral strike slip faults, northeasterly directed thrusting in Ghana with its associated D2 folds, and finally ENE-WSW strike-slip faults and associated D3 folds. This last phase indicates a change in the axis of cratonic convergence.

(ii) In contrast to the above discussion, Hein et al. (2004) and more recently Hein (2010) present more recent work on the Boromo-Goren Greenstone Belt (BGGB) of Burkina Faso, this forms part of the Proterozoic Birimian Domain of the West African Craton. The Goren segment of this greenstone belt trends SE across the Yatenga, Bam, and Sanmatenga districts of Burkina Faso for a distance of approximately 140 km, and constitutes an arcuate extension to the predominantly northerly trending, 500 km long, Boromo segment of the belt (Bessoles, 1977).

In 1996-1998, the BGGB became the focus of a study by Randgold Resources and North Ltd. In 2007 the area was revisited in a study sponsored by Riverstone Resources Inc. In the course of these studies, the geophysical signature, lithologies and structure of the Goren segment were documented. The results showed the region could be divided into two domains, a western domain, where the dominant trend is north-easterly and an eastern domain, where the dominant trend is NW-NNW. The boundary between these two domains is a geophysical lineament interpreted to be an east dipping shear zone. Contrary to data from Milesi et al (1989), Feybesse et al (1990), and Hottin and Ouedraogo (1992), which suggest an unconformity occurs between the Lower and Upper Birimian in the study area, a discrete unconformity was not found. The field studies have established broad stratigraphic relationships between at least three conformable successions of Birimian units, and temporal structural relationships between regional and local structures, metamorphism and granite emplacement. In the BGGB, three deformation events, D1-D3 are recognized.

D1 is characterized by the development of NW-trending isoclinal folds and reverse dextral shear zones, steeply NE dipping and SW verging; a pervasive schistosity is developed. This event is termed the Tangaean Event and it can be correlated across NE Burkina Faso.

The D2 event resulted in the progressive development of NNE to NE-trending macroscopic to mesoscopic folds and a penetrative axial planar cleavage (S2), which was followed by the formation of dextral and sinistral-reverse shears and a pervasive schistosity (S2-C). D2 in the BGGB corresponds with the Eburnean Orogeny at 2130-1980 Ma, as described by Feybesse, et al, (2006).

The deformation D3 is recognised throughout the BGGB. It is characterised by the formation of kink and chevron folds (F3), and/or a crenulation cleavage (S3), that are hosted by narrow WNW-trending shear zones. These formed during a period of north-south shortening termed the Wabo-Tampelse Event that post-dates the Eburnean Orogeny (Hottin and Ouedraogo, 1992).

The Proterozoic tectonic evolution of the Boromo-Goren Greenstone Belt may be described by four events:

1. Lithologies record the deposition of volcanic, sedimentary and turbiditic rocks in three conformable successions. These successions indicate a variable interplay between marine-deltaic sedimentary processes and bimodal volcanic-pyroclastic activity and indicate that deposition occurred within a marginal marine setting adjacent to an emergent volcanic centre (island arc setting). The conformable nature of the successions does not provide evidence for separating the Birimian rocks into two separate formations. Deposition is thought to have occurred between 2238Ma and 2170Ma.

2. The formation of NW-trending isoclinal folds and reverse dextral shear zones by NE-SW shortening (F1), the Tangaean Event that preceded the Eburnean Orogeny.

3. The Eburnean Orogeny (2130-1980 Ma) a period of NW-SE crustal shortening resulted in regional metamorphism to greenschist facies the D2 development of folds (F2), NNE to NE-trending shear zones and a shear cleavage (S2-C), and the emplacement of pre- to syn-tectonic granite and granodiorite-tonalite plutons (2190-2108 Ma). Contact metamorphism was to hornblende-hornfels facies. The development of quartz veins (V2) in the NE-trending shear zones occurred early in D2 before development of a crosscutting C-S foliation.

4. A period of N-S shortening, the Wabo-Tampelse Event that postdates the Eburnian, formed mesoscopic to microscopic WNW-trending folds (F3) and a westerly trending crenulations cleavage (S3). The general trend and orientation of D3 structures indicate a net dextral reverse sense of motion, with transport to the north (Junner, 1940).

1.8 LOCAL GEOLOGY

The Kobada group of properties lie within the Bougouni Basin. The basin is composed primarily of sedimentary rocks with minor tholeiitic volcano-sedimentary intercalations. These sediments were deposited in a broad trough during the early Proterozoic Birimian period (2200 Ma to 1800 Ma). The Bougouni Batholith, a large felsic intrusion, occupies the central part of the basin.

A north-south band of biotite and/or amphibole granite of Eburnean age (Lower Proterozoic) occurs approximately 50 km to the east of Kobada (Figure 1.9). Small intermediate intrusions, mainly diorite to granodiorite, also occur within the basin.

The terrane in the area has been intensely lateritized and with the exceptions of the granitic rocks, the protolith is rarely identified on surface. Other than rare outcrops of quartz veins, there is no exposure of the Birimian units on the Kobada group of licenses. 1.5km west of the Kobada license there is a belt of massively bedded, medium grained, poorly sorted, moderately foliated greywacke exposed. It is not clear why this 4km long by up to 500m wide belt of rocks has escaped the deep weathering process.

Within the Kobada licenses, rare outcrops occur where resistive quartz veins and the iron oxide rinds that commonly mantle them protect the structure if not the mineralogy and chemistry of their host rocks. Drilling to date has intersected interbedded meta greywackes, meta siltstones, mudstones and phyllites.

So far, the meta sediments form the only known lithological units on the concessions and it has not proven possible to identify and trace marker horizons in drilling or trenching (Milesi et al (1989).

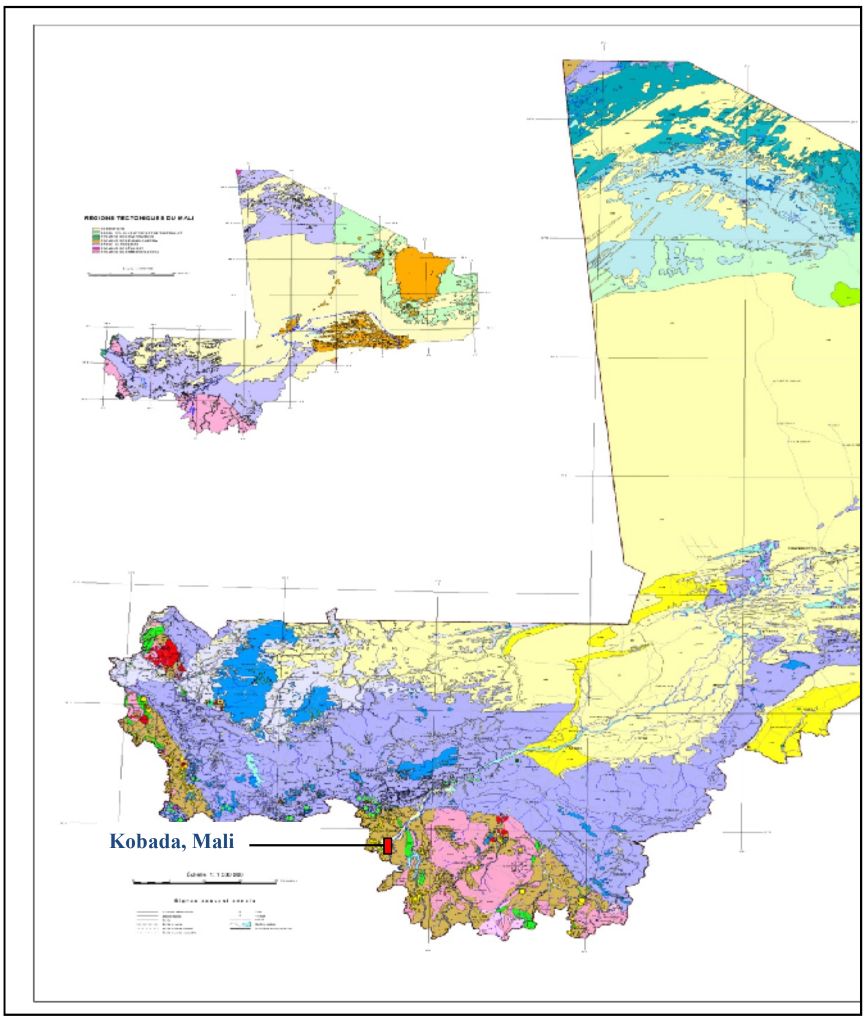


Figure 1.9: Simplified Geologic Map of SW Mali showing the location of the Kobada Licenses.

1.8.1 Laterites

Due to the occurrence of alluvial and eluvial gold deposits in the lateratised surface of the project area, it is worth discussing the weathering profile. The BRGM studied the laterite profile in some detail (Kaisin and Hanssen, 1993) and this author from the study of the eluvial and alluvial gold workings on the property has drawn several conclusions. The following discussion is a combination of both sources.

Large laterite plateaus cover most of the concession. The underlying mottle clay and saprolite is exposed below the plateau boundaries. Whilst the mottle is generally of a red and orange colour, the colour of the saprolite can be exceedingly variable. Yellow ochre is the most common colour, and zones of white saprolite caused by intense kaolinitisation, likely of hydrothermal origin, are not rare, but any colour of the rainbow can be found in the saprolite.

The BRGM mapped 3 generations of lateritic plateaus which cover the saprolite. The older, upper one, has been deeply eroded and is only preserved in isolated high topographic features.

However, this generation of laterite has not been identified on the Kobada properties.

It is a second, younger cuirass that forms the bulk of the plateaus that are found in the project area (Figure 1.10). This ferricrete can be seen to consist of pisoliths and blocks of older laterite as well as elements of saprolite, locally chips of saprolitized rock and quartz vein fragments that have all been eroded from the higher older cuirass. Most often these materials are very angular and have not been transported any great distance and are probably colluvial materials deposited at the margins of the old eroding cuirass plateau or materials that have only migrated vertically downward during a volume loss deflation process occurring at the top of the saprolite during leaching. This second age laterite cap and its underlying mottle zone is the thickest identified on the property, often 20m thick. The very hard iron cemented ferricrete or cuirass can be 6m thick and slowly softens with depth until the iron cement is replaced by clay in the mottle zone clay horizon. The base of the mottle clay often displays a ‘stone line’ of quartz vein clasts which mark the base of the deflationary structureless cover and the top of the leached but not disrupted saprolite. This ‘stone line' base of the mottle zone is often a very gold enriched horizon when in the vicinity of underlying vein mineralisation, and has often been the target of ancient artisanal miners who mined large caverns (up to 30m x 50m x 3m) at this level, now often identified on the surface as collapse structures. These eluvial gold deposits are rarely worked by modern orpailleurs. When asked to dig some pits in this very hard cuirass, the local labourers say it is too hard to penetrate. The ancient workforce was slave labour and could not refuse the very hard work. Many of the collapse structures form the densest woodland in the project area often with quite large, old trees, it is concluded here that the collapse and breaking of the cuirass makes a tree's quest for water easier. Occasionally in mined but not collapsed areas tree roots descend the old orpailleur holes and pass into the saprolite.

The present day drainage pattern exhibited on the property is the one that cuts down through this second phase laterite cap. It is within these present day drainage channels that the third phase and youngest ferricrete is developing. This ferricrete is generally composed of rounded materials with signs of water transport and is much softer and thinner than the second age cuirass, exhibiting weaker cementation. It is gravel layers in these cemented alluvial placers that form the main target of the modern orpailleur in the Kobada area. Mapping of these placers indicate the Kobada Shear Zone to be their principal source.


Small Rivers
Main Rivers
Alluvium
2nd Level Lateritic Cover

level now exposed, occurred at the ductile/brittle deformation crustal level. At this juncture, quartz vein introduction occurred. Deformation continued, and the early quartz veins were folded (F1) and the limbs of the folds sheared in the Kobada shear which is now oriented N20°E to form the sheared foliation (S1) parallel early veins (V1) described above. Progressive strike slip shear deformation with a subhorizontal principal extension direction resulted in the high angle E-W striking quartz filled extension fractures or tension gashes (V2), which accompany the introduction of significant quantities of gold, and also pyrite and arsenopyrite, though there is no direct correlation between gold and the sulphides. Deformation continued with a similar E-W principal compression direction, but now the principal extension direction rotated into the vertical or subvertical plane to produce reverse faulting or thrusting. This progressive deformation continued to cause movement on the earlier formed shear zones, the same penetrative foliation in the metasediments and the open folding (F2) and fracture cleavage (S2) formation in the E-W high angle vein set. Also at this stage, further hydrothermal fluids were introduced into the system and tension gash and extensional fracture veins formed with a subhorizontal orientation (V3). Only weak gold mineralisation was introduced at this late stage, and subhorizontal stockworks and tension gashes are barren (Downing, 2010).

Whilst evidence for the sense of shear is slight, a left lateral or sinistral sense of movement is postulated. This conclusion is based on the occurrence of a flexure in the Kobada Zone 1 deposit at about the 1900S area. This area coincides with thinner weaker mineralisation which could be related to a “pop up” or an orientation of compression where open space fracturing and hence space for mineralisation would be at a minimum. If the flexure is a “pop up” it implies sinistral shear (Downing, 2010).

CHAPTER TWO

LITERATURE REVIEW

2.1 RELEVANCE OF MINERAL RESOURCE ESTIMATION

Mineral Resource Estimation comprises all the activities which together make it possible for accurate prediction of grade and tonnage of a mineral resource. A resource estimate is based on prediction of the physical characteristics of a mineral deposit through collection of data, analysis of the data, and modeling the size, shape, and grade of the deposit. Important physical characteristics of the ore body that must be predicted include

(1) the size, shape, and continuity of ore zones,

(2) the frequency distribution of mineral grade, and

(3) the spatial variability of mineral grade.

These physical characteristics of the mineral deposit are never completely known, but are inferred from sample data.

The application of geostatistics in the mining industry has been particularly helpful in various ways providing estimates that assist in decision making and helps monitor and maintain the profitable continuation or otherwise of a mining operation. Unlike the traditional estimation methods such as polygonal estimates, inverse distance weighting etc., modern geostatistics provides a measure of accuracy in the form of kriging variance (Glacken, and Snowden, 2001). By this approach, decision makers are informed as to what degree of error is present in a particular estimate.

Generally, ore is mined as blocks and geostatistics has the capacity to estimate the average grade and tonnage of each block based on nearby samples. Although most mining companies usually want block estimates of their variables instead of contour maps, geostatistics can be used to generate estimated contour map of a deposit (Armstrong, 1998).

Modern mineral resource evaluation is a computer based process involving but not limited to geologic surveys, systematic surveys on large grids, sampling on small grids, quality assurance and quality control checks, classical statistical evaluations of data-sets, geologic framework modeling, geologic block modeling, spatial statistical evaluations, interpolation of grades, model validation, classification, resource inventory calculations, model documentation and the model is delivered to mining engineers for pit optimization and economic evaluations (Journel and Huijbregts,1978).

The process of estimating mineral resources can only take place after the resource geologist is convinced of the soundness of the fundamentals underlying the estimation process. Thus the database of sampling, density, and other quality data for both estimation and geological interpretation must have integrity and robustness; the geological data must be sufficiently complete for the definition of a geological model; the geological model itself must have internal consistency, should explain the observed arrangement of lithological and mineralogical domains, and should represent the estimator's best knowledge of the genesis of the mineral deposit; and the geological model should support the distribution of mineralization seen in the sampling (Journel and Huijbregts,1978).

It is only at this stage that a resource model may be generated. The resource estimation process involves the definition of mineralization constraints or geological domains, the statistical and/or geostatistical analysis of the sample data, and the application of a suitable grade interpolation technique. The final stage of the estimation process will be to classify the resource according to standards set by internationally accepted entities such as the Joint Ore Reserves Committee, (JORC Code), CIM Evaluation Committee (2010), South African Mineral Resource Committee (SAMREC), AUISIMM etc.

2.2 HISTORY OF RESOURCE ESTIMATION

The use of geostatistics in evaluating mineral resources and ore reserves began in the early 1960s to mid-1970s when computers became available. In the 1980s and 1990s when computers became common, much work was carried out in the field of geostatistics. However, geostatistics is most often applied to relatively larger and more uniform deposits (Sinclair and Deraisme, 1974).

Since the start of the modern mining era, there has been an increasing use of sample data to estimate (predict) the amount of material mined or to be mined. Early mining operations had little forward planning, and consequently little use was made of exploration or mining data to estimate resources or reserves, although there was extensive use of data for grade control and for monitoring of the mining operations. The early resource estimations which were in fact reserves, arose out of this grade control function, where, for example, underground face and raise samples were used to delineate a mining block. It was only in the middle part of the 20th century that exploration 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 without first progressing through the resource stage. 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 reserves 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 (Krige, 1978).

In South Africa some success was achieved by fitting lognormal distributions to mine data (Krige, 1978), but this failed to translate well to Australian ore bodies. With the advent of increasingly fast and reliable computers, mine planning packages, which incorporated resource and reserve estimation, were introduced from the late-1970s. These provided for the first time the ability to generate a model of the orebody comprising a large number of orthogonal, similarly-sized blocks. Accompanying this block modeling ability the packages generally offered a variety of resource estimation techniques, including polygons (more strictly 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 (David,1977).

The mining software packages continued to develop in power and sophistication during the 1980s, and great advances were made in the visualization and modeling 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. This polarization continues to the present day, although some form of block model is now accepted as the norm in the majority of estimations.

As computers continue to grow in power and sophistication, a number of very computationally-intensive techniques such as conditional simulation have now become common.

2.3 SYNOPSIS OF RESOURCE ESTIMATION TECHNIQUES

Resource estimation methods can be grouped under two main types. These are the manual methods and block modeling (which is generally computer based) methods. It should be noted that some of the manual estimation techniques can be implemented in computer based block modeling. In general, Resource estimation techniques range in complexity, roughly proportional to the amount of computation involved in deriving the estimate.

2.3.1 Manual Methods

Manual methods are the simplest techniques and involve assigning to an ore body intersection its own clearly defined area of influence, defined in relation to the other intersections. Such techniques include polygonal (area-of influence, or nearest-neighbor) and sectional approaches (Glacken and Snowden, 2001).

2.3.1.1 Polygonal Method

Polygonal methods are very simple estimation method (Figure 2.1). Polygons may be developed in the plane of mineralization, or may be projected onto a suitable horizontal or vertical plane with the corresponding geometric transformation. The most common versions of the polygonal approach are the nearest neighbor and the sectional area-of influence methods (Glacken and Snowden, 2001).

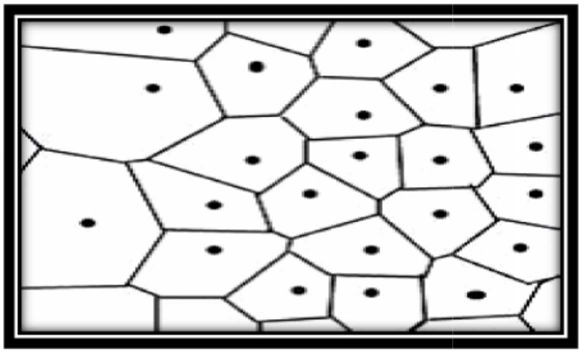


Figure 2.1: Nearest Neighbor Method

As discussed by Isaaks and Srivastava (1989), nearest neighbor in its simple terms, the grade at a given location is the grade of the nearest sample grade. The procedure is to draw polygons of influences of each sample. These polygons are drawn using the perpendicular bisectors of adjacent samples (similar to thiessen triangles). The estimated grade of the polygon is the grade of the central sample. This kind of estimates is strictly by assignment and there are no complexities in implementing it. Its advantages are simplicity and also as being the best declustered estimate, this makes it possible to use it as a check to other linear estimation methods. However, there are lots of disadvantages to using nearest neighbor estimates, notable among them are as follows;

1. Discontinuities between estimates (this makes mineralization spotty).
2. Only one sample is used per estimate and this makes it more variable and less reliable .
3. Estimates are too variable introducing conditional biasedness into the entire estimates.
4. It does not regard such geologic properties as nugget effect which means that for less variable commodities like iron ore the estimation process is similar to that of a high nuggety gold deposit.
5. Until recently this method was only achievable in 2D space.

2.3.1.2 Sectional Methods

The sectional method is achieved with polygons defined on sections (perpendicular to mineralization) extended orthogonally to the section plane halfway to the next section.

The procedure used to achieve resource estimates using sectional method are summarized in Figure 2.2, and briefly outlined in the following steps:

1. Compute weighted average grade of each mineralized drill hole intersection and draw polygons to capture the intercepts;
2. Project mineralized intercepts mid-way towards adjacent drill intersections on the same vertical section;

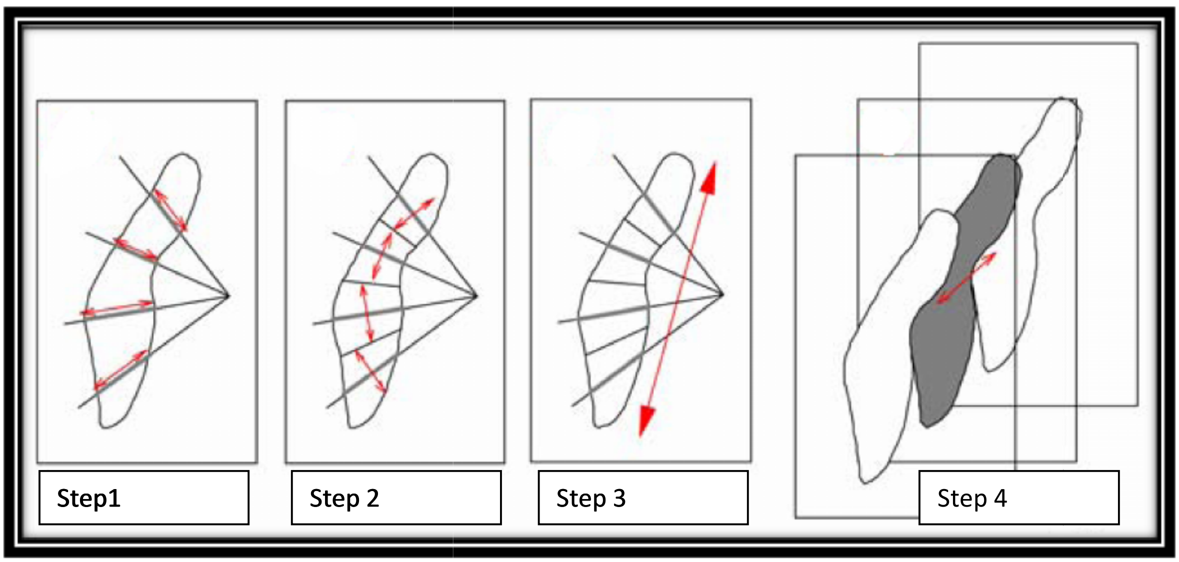


Figure 2.2: Sectional estimation method

3. Compute weighted average grade of mineralized zone within vertical section;
4. Project mineralized zone grade midway towards adjacent vertical sections;
5. Compute weighted average grade of 3D mineralized envelope.

The main advantage of sectional methods is that it can be done on intricate shapes. Some disadvantages on the other hand include:

2. It has some polygonal flavor since grades are projected half ways in the above step 2 and step 4, thereby inheriting the disadvantages of polygonal method;
3. No grade distribution within mineralized envelope is assumed, (Virley,2010)

Limited averaging of grades occurs in some techniques, not common nowadays, where, for instance, triangulations are constructed about the mid-point of intersections, with the grade of each triangle being the arithmetic mean of the grades at each of its vertices.

Other approaches include averaging values at the corners of other regular shapes or within grade contours (Glacken and Snowden, 2001)..

2.3.2 Block Modeling Methods

A block model is a series of blocks or cells that collectively define a larger zone. This zone could be geologic, geotechnical, metallurgical or any volume applicable to mining. The advent of block modeling has revolutionized the mineral resource industry and has been improved tremendously with increasing power of computers. Common practice is to define a block model, comprising a series of orthogonal cuboid blocks either of the same size or of subsets of a nominated 'parent' block size. The blocks are used to fill the various domains within which estimation would be performed. The advantages of using block models include but not limited to the following;

1. Block modeling approach provides the framework for a good local estimation, and that provides a model which lends itself readily to reserve estimation and mining selectivity.
2. A block model is a very efficient data structure in which large amount of information can be stored.
3. Very flexible construction methods allow creating models that accurately represents the geological and mining conditions.

4. Block models allow excellent visualization of geological zones or grade trends within an ore body.
5. Block models allow for the increase use of geostatistical methods to express grade distribution.
6. Rapid calculations between the values within variables allow effective resource/reserve estimates to be undertaken.

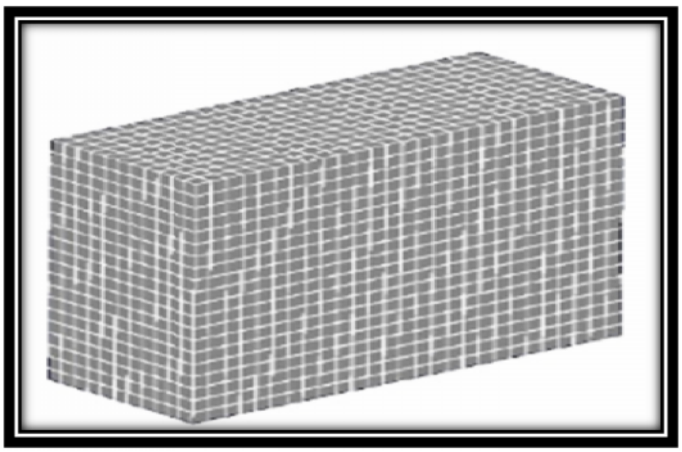


Figure 2.3: Regular Sized Block Model

The next step up in computational complexity is to apply some weighting function to grades surrounding a point or block to be estimated; Figure 2.3 shows a typical Vulcan 3D block model configuration (Guibal, 1997*)*.

This is attractive where a series of regular points or blocks need to be estimated in two or three dimensions for subsequent planning or manipulation. Estimation techniques that depend on block models can be grouped under the two main umbrellas; the linear and non-linear estimation methods. Linear estimation techniques are those estimation techniques which require some form of linear weighting of samples during the estimation process. In linear estimation, the weights are applied to the actual samples used to estimate the blocks or points and this makes them exact estimates i.e. they are located at where the estimate was done. Some common examples of linear estimation methods are; Inverse distance power weighting (IDW), Ordinary Kriging, and Simple Kriging common characteristic of all the linear estimation methods is that they are all linear combinations of the data (Guibal,1997).

2.3.2.1 Inverse Distance Weighting

The simplest weighting function in common usage is based upon the inverse of the distance of the sample from the point to be estimated, usually raised to the second power, although higher or lower powers may also be used (**Figure 2.4**). Such inverse distance techniques introduce issues such as sample search and declustering decisions, and cater for the estimation of blocks of a defined size, in addition to point estimates.

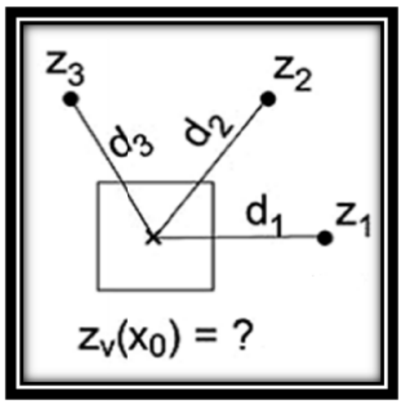


Figure 2.4: Diagram showing IDW Estimation method

Where $Z_v(X_0)$ is the estimated block or point using IDW to the power Θ and d are the distances of samples z(x) to the location being estimated.

As can be seen the weighing scheme is independent of the spatial relationship with sample values.

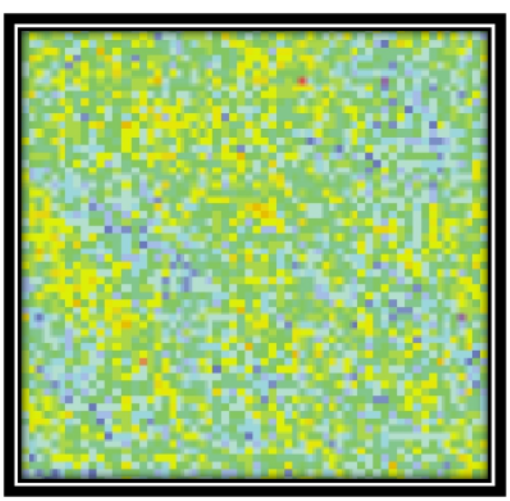
The Estimate of grade at location V that is $Z_v(X_0)$ given that the power is 3 is given below

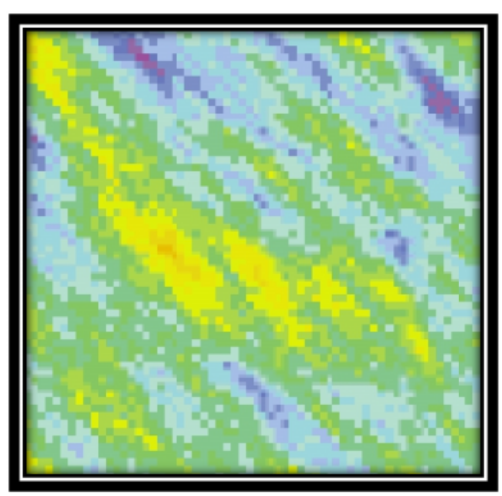
$$Z_v(X_0) = \frac{\left(z_1 . \frac{1}{d_1^3} + z_2 . \frac{1}{d_2^3} + z_3 . \frac{1}{d_3^3} \right)}{\left(\frac{1}{d_1^3} + \frac{1}{d_2^3} + \frac{1}{d_3^3} \right)}$$

The inverse distance method is much better than the polygonal method; however it does not account for geologic characteristics such as anisotropy and clusters which is usually common in mineral deposits (Glacken and Snowden, 2001).

2.3.2.2 Kriging Technique

Kriging is an estimator designed primarily for the local estimation of block grades as a linear combination of the available data in or near the block, such that the estimate is unbiased and has minimum variance. It is a method that is often associated with the acronym B.L.U.E. for *best linear unbiased estimator*. Ordinary kriging is *linear* because its estimates are weighted linear combinations of the available data, *unbiased* since the sum of the weights is 1, and best because it aims at minimizing the variance of errors. The conventional estimation methods, such as inverse distance weighting method, are also linear and theoretically unbiased. Therefore, the distinguishing feature of ordinary kriging from the conventional linear estimation methods is its aim of minimizing the error variance (Isaaks and Srivastava,1989).





Although the Kriging technique is a powerful geostatistical technique, the estimates are mostly too smooth if care is not taken during the interpolation process. Figure 2.5 above shows smooth grades of OK compared with IDW.

These methods all seek to utilize the spatial relationship between samples, as quantified by the semi-variogram, to provide weights for the estimation of the unknown point or block. The standard technique of geostatistics, named kriging by Matheron in honor of the South African mining engineer Danie Krige, has many varieties, but those most commonly used are the variants of kriging, the so-called linear kriging techniques (Guibal and Touffait,1982).

The evolution of geostatistics in the last decade and a half has seen the development of a range of non-linear kriging techniques, based upon non-linear transformations of grades. These include the commonly used methods of indicator kriging and the various flavors of uniform conditioning and disjunctive kriging. Non-linear kriging approaches seek to estimate a distribution of grades into each point or block, thus providing some measure of local uncertainty (Vann, 1998).

The most recent development, and the most complex and computationally intensive method used in resource estimation, is that of conditional simulation. This builds upon kriging and the use of stochastic (random) sampling approaches to provide, in theory at least, a full measure of uncertainty. Conditional simulation, while honoring data values locally, over comes many of the short comings of kriging methods. The negative aspects of complex approaches such as this are the time of computational power required for its implementation and the lack of simplicity, which hampers understanding and acceptance (Pan and Harris,2000)..

All methods described above have a number of common attributes; all must be based upon a carefully defined geological model, and all require extensive validation by the practitioner. A good overview of the various techniques is given by Carras (2001).

It should be noted that the resource estimation process, while often driven by the geoscientist, is a team endeavor, and should include contributions from mining, metallurgical and often the commercial disciplines. Another essential aspect of the process is quality control, including management of the estimation process and most importantly, management of the data sourced from the various disciplines (Journel and Huijbregts,1978).

2.4 THE RESOURCE ESTIMATION PROCESS

2.4.1 Geologic Framework and Domains

The resource estimation domains should honor the geology wherever possible, but where this is not achievable, some other form of domain boundary needs to be imposed. Typically this is a grade boundary defined by a cut-off grade which should bear some relation to the economics of the deposit to be evaluated, however preliminary the assessment is which needs to be made. Domains may be defined by a combination of statistical and geostatistical means, in addition to or instead of by a cut-off grade (Glacken and Snowden, 2001). Where grade alone is used to define the domain boundaries, then it is risky to use a cut-off grade too close to the overall economic cut-off of the deposit. If this is the case, the result is often the overestimation of grades within the domain, and the underestimation of grades outside the domain. Some deposits show a rapid change from ore to non-ore, so selecting a natural cut-off is relatively safe (e.g. Osborne copper-gold deposit, NW Queensland, and most of the Achaean lode gold deposits of the Yilgarn craton in Western Australia) as presented by Glacken and Snowden (2001).

These boundaries may be termed hard, and greatly facilitate resource estimation. The other extreme is the gradual or soft boundary, requiring much more careful treatment when estimating resources. Structurally complex deposits such as those at the Macraes gold mine, New Zealand, display a combination of hard and soft boundaries. The well-defined hanging wall of the shear zone is a hard boundary, but the footwall is gradational and areas of associated stock work mineralization also have soft boundaries. It is possible, in defining domains for resource estimation, to impose several types of boundary conditions (Glacken, 1996).

Soft domain boundaries allow grades from either side of the boundary to be used in estimating both domains, to varying degrees. Hard domain boundaries do not permit interpolation of grades across domains. One way soft domains are often used in estimation, data within a high-grade domain is not used to estimate within an adjacent low-grade domain, but estimation of the high-grade domain will use data from within the low-grade domain. Such one-way boundaries introduce conservative bias and avoid artificially sharp grade boundaries. More useful are partly soft domain boundaries, where only limited crossing of domain boundaries is allowed (Glacken and Sommerville, 1998).

Mathews et al., (1999) discusses the effects of domaining on resource estimation in the Cobar region of New South Wales. Finally, it is worth bearing in mind that several different types of domain may be used in the same region. A typical example is the use of oxide, transition, and primary domains to allocate specific gravity in an oxidized mineral deposit. Similarly, metallurgical domains may be used in addition to geological domains to define areas of differing metal recovery. Often one set of domains is used for estimation and another for mining/metallurgical purposes. As many different types of domain as are needed to define the resource and to provide information for reserve estimation should be used.

2.4.2 Assignment of domains

Once the geological model is as complete as the available data and knowledge of the setting and genesis of the mineralization allow, the data must be coded according to its domain. A domain in this context is defined in the loosest sense, and represents an area or volume within which the characteristics of the mineralization are more similar than outside the domain. (Glacken and Snowden, 2001).

Hopefully the geological modeling will have highlighted a number of domains, which should conform in some way with the geology wherever possible. In many cases, the geological units are the same as the mineralization domains, such as in many iron ore, sedimentary base metals or base metal sulphide deposits. In this case, the grade modeling is constrained entirely by the geologic model, and the resource grade model will be a true reflection of the geology. In many other cases the mineralization of interest does not entirely correspond to a geological unit, or transgresses geological units. This is a typical feature of structurally-controlled mineralization, such as shear-hosted gold deposits (e.g. Ahafo deposits in Ghana).

A number of deposits, such as those in the Callie and Granites ore systems of the Northern Territories of Australia, combine elements of structural and lithological control. A further class sees disseminated mineralization scattered widely throughout a limited range of rock types, with no clear geological points to the distribution of grades. Typical examples of this style of mineralization are the large porphyry-hosted copper and gold deposits, such as those at Batu Hijo being mined by Newmont and Grasberg being mined by Freeport McMoRan Copper and Gold Mining in Indonesia.

2.4.3 Exploratory Data Analysis (EDA)

Exploratory data evaluation is an essential part of every high quality, mineral inventory estimate. Efficient analysis involves a thorough organization of available quantitative data that forms the basis of the estimates and perhaps the formation of composites grades (Sinclair, 1999).

Statistical analysis may help decide the nature of the domain boundaries. The analysis should include studies of how grades change at domain boundaries. Once a series or several series of coherent domains have been defined, the numerical characteristics of the mineralization in each of these areas should be described. This will not only assist with the choice of a grade interpolation method, but will also highlight any special data treatments (such as grade cutting) which need to be applied. Where there are several minerals or variables of interest, statistical analysis will reveal any patterns or correlations between them which need to be taken into account (Glacken and Snowden, 2001).

Statistical analysis should take place within the domains defined either by the geological model or by other approaches. One prerequisite of data analysis is that the samples all represent an equal volume. This is called the support of the sample. The accepted way of ensuring equal support of all samples within a domain is to composite the samples (usually drill- or interval-related data) into equal lengths (Glacken and Snowden, 2001). There are various algorithms for compositing, but the practitioner should always check that the results are as expected. Another problem which needs to be tackled before data analysis can take place is preferential clustering of data, which not only affects statistical analysis but can also bias the variography. Since there is a natural tendency to drill more holes or take more samples of the higher grade portions of the mineralization, clustering or irregular representation of samples is a real possibility. Domaining might be necessary to correct this.

Alternatively, various declustering approaches are available to ensure that each sample represents an equal volume for statistical analysis. Failure to take heed of preferential clustering of data may lead to biased results off course interpolation algorithms implemented in most commercial Ordinary Kriging packages have embedded declustering technique such as quadrant and octant searches. It should be noted that, sometimes it may be desirable to treat different drilling types (e.g. diamond and reverse circulation) differently within the same domain (Vann, 2008).

If sampling is done on fixed lengths, then compositing should be on multiples of the original sampling interval to minimize unnecessary smoothing. Zonal compositing (that is, compositing within domains) prevents this in many commercial packages, as composites commence after a change in zone at an odd interval, not at the start of an original sample, especially where zones are not defined on sample intervals. The method of compositing adopted should take into account the style of mineralization and boundary requirements (Vann, 2008). For example in a narrow vein, hard boundary environment, it is essential to composite by zone to avoid over- or under-dilution. This is of less consequence where the boundaries are gradational. There are some schools of thought that believe that composites should always be equal to the height of the mining bench. There are advantages in employing bench compositing, in that the input data is fully diluted to the bench height. However, where drill holes are angled at various orientations with respect to the mining bench, this could result in different support lengths for different composites, particularly when drilling is orientated close to horizontal. Down hole compositing ensures that each sample represents the same support. Assay lengths should not be split into smaller composite lengths, as this results in an artificially low variance for the modified support as adjacent composites could be identical in value. Issues of bench compositing are starting to have less importance as true three-dimensional methods of resource estimation predominate, and in this paradigm each composite represents a point in three-dimensional space, irrespective of orientation (Glacken and Snowden, 2001).

Once compositing and declustering have been applied to the data, statistical analysis can take place. There are no recipes for this, but a good summary of statistical analysis techniques is given in Isaaks and Srivastava (1989). Desired outcomes from statistical analysis of the data would be:

- depiction of any trends evident within the domains;
- depiction of data distributions in both histogram and cumulative probability form;

- a decision as to whether a distribution-based interpolation technique may be used;
- definition of any data subsets within each domain;
- comparison of different sample types;
- establishment of the basis for any top cuts to be applied;
- choice of thresholds for indicator kriging, and
- Establishment of any relationships between multiple variables.

Statistical analysis may indicate that some domains have very mixed populations, e.g. excessively high coefficients of variation or multi-population probability plots. This may signal the need for more or different domains or, if it is not possible to separate out the populations, indicator techniques may be necessary for variography and kriging. Where there is a positive correlation between the bulk density of the ore and the grade of the minerals of interest, typically in massive sulphide deposits, the bulk density should be involved in the actual estimation process, in other words, sample intersections used for grade estimation should be weighted by density as well as by length. Another option which is often considered is the direct interpolation of bulk density along with the grade variable(s). This method of density weighting is used at many base metal deposits e.g. Kambalda, (Glacken et al, 1998).

2.4.4 Spatial Statistical Analysis (Variography)

As a precursor to any of the various kriging or conditional simulation techniques, spatial (geostatistical) analysis of the domained data – that is, the calculation and modeling of semi variogram is an obvious and necessary step (Vann, 2008).

However, the analysis of the continuity of data values in three dimensions is also a very useful precursor to almost any form of estimation, as it defines, at the very least, the classical 'range of influence' of the data. Knowledge of this can and should have a bearing on the choice of a suitable grade interpolation technique. For instance, the direction and magnitude of the ranges may be used to define grade search parameters or the maximum size of polygons of influence. The generation of semi-variogram and their subsequent modeling should reveal the structure of spatial continuity of the data, and should confirm geostatistically any geological trends previously modeled or noted. The practitioner should always seek a geological explanation for the principal directions revealed by semi-variogram analysis. Quite often such a step will reveal some subtle controls on mineralization not immediately evident in the geology. Geostatistical analysis will also reveal any anisotropy in the domains of the mineral deposit and seek to quantify the magnitude of that anisotropy (Isaaks and Srivastava, 1989). The anisotropy may be represented by the same total variation but at different ranges in the various directions (geometric anisotropy), or by different magnitudes of the variation in different directions (zonal anisotropy).

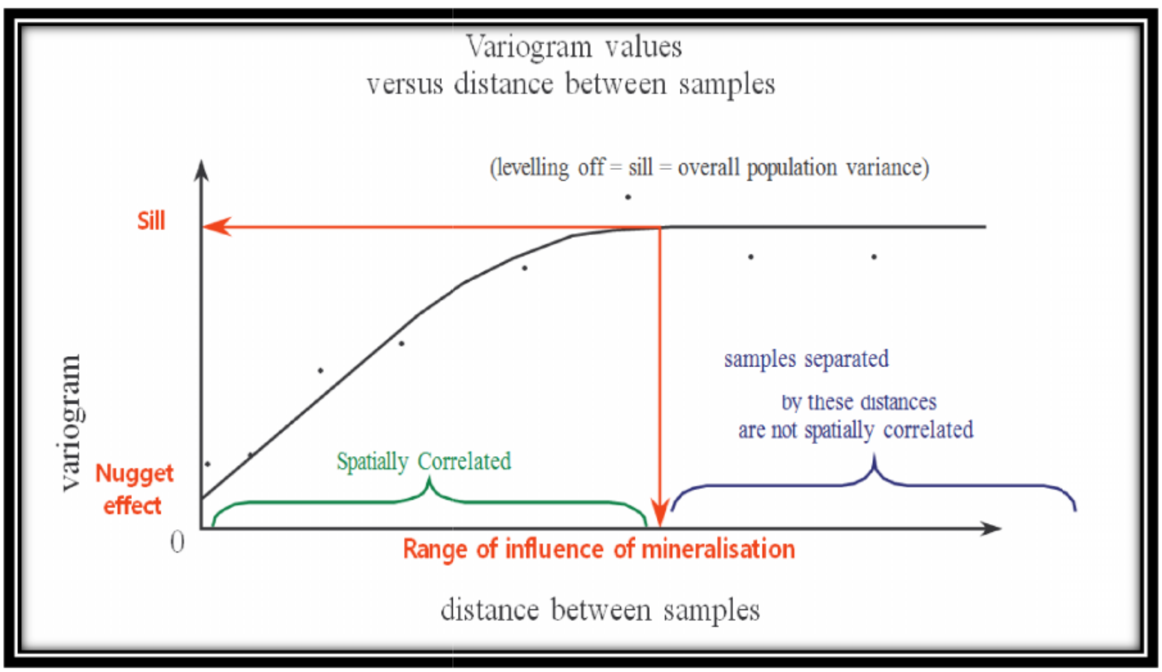


Figure 2.6: Showing Experimental semivariogram and spherical model (Snowden, 1999)

The analysis should also seek to verify the decision to use hard or soft domain boundaries, and inherent randomness in the data, in each direction. Although variography should not be used as a substitute for geological interpretation, it can indicate whether the geological model is appropriate (Isaaks and Srivastava, 1989). Indicator variography may demonstrate that the amount and direction of anisotropy varies with grade, e.g. high-grade veins may have a different orientation compared to the bulk of the mineralization. Finally, the definition and modeling of semi-variograms will help with the definition of the basic block size to be used in any block modeling techniques, and will provide information for the aggregation of grades into larger block sizes if required. Some Australian operators use variography to determine the range of influence as shown in Figure 2.6 and hence filter size for inverse distance interpolations. Geostatistical techniques for data analysis are presented by Coombes, 1997.

2.4.5 The volume model

Prior to grade estimation, it is necessary to convert the geological model and/or the domain model into a physical, usually three-dimensional, representation of the volume of mineralization to be estimated. Common practice is to define a block model, comprising a series of orthogonal cuboid blocks either of the same size or of subsets of a nominated 'parent' block size. This is usually a semi-automatic process, but generally requires a confining shape in which to generate the blocks. Typically this is given either by a three-dimensional enclosed solid or by a series of surfaces. These are generated by wire framing strings of points on section or plan, or by triangulation of a series of points and strings into a digital terrain model, more commonly termed a surface. Surfaces may also be interpolated from the raw data by a number of surface-fitting techniques.

Modern mining softwares have advanced to such a degree that almost all of the major packages provide moderate to excellent tools for defining both the three-dimensional shapes and for filling them with blocks. Key decisions for the practitioner include:

- How large should the blocks be compared to the data?
- What should the relative shapes of the blocks be in the three dimensional space?
- How complex should the wireframes be?
- Should sub-ceiling (sub-blocks) be introduced, or
- Will the extra resolution produce too many blocks that are unrealistic to the actual mining selectivity (Glacken and Snowden, 2001).

2.4.6 Unsmoothed grade estimation

As mentioned above, despite the speed and power of modern computers and the sophistication in estimation algorithms which this allows, many practitioners carry out grade estimation unsmoothed or polygonal techniques. The essential aspect of all of these algorithms is that each grade, or series of grades in a defined intersection, is allocated unaltered to a specific area of influence. The simplest means of allocating grades is via polygons of influence, which are generated by constructing the perpendicular bisectors between adjacent samples or intersections. These are generally constructed in two dimensions on composited intersections, and rarely in three dimensions. In the block modeling context the polygonal approach is represented by the nearest neighbour approach, in which each cell or block to be estimated assumes the grade of its closest sample within the defined domain. This in effect generates three-dimensional polygons. Although conceptually very simple, many resource and reserve systems based upon the polygonal approach are very complex in their treatment of individual samples and their geometrical relationship to the ore surface. Many systems involve projection onto both a horizontal or vertical plane and the subsequent geometric manipulation of areas and volumes. A very common adaptation of the polygonal approach is the cross-sectional resource estimation method. In this approach, sectional interpretations are constructed, generally orthogonal to the strike of mineralization. Each separate ore intersection on each drill hole is allocated its own volume of influence, which usually extends halfway to the next drill hole up and down dip, and halfway to the next section in each strike direction. An adaptation of the polygonal approach in narrow ore bodies is the use of accumulations (sometimes termed service variables). Metal accumulations are the product of grade and thickness. In this approach the accumulation and the thickness are estimated independently. The final grade is obtained at each point by dividing the accumulation by the thickness. The accumulation approach only works well where there is no correlation between grade and thickness, and also requires the spatial orientation of the samples to be taken into consideration as true (normal to dip) ore thicknesses are required. It is worth noting that the accumulation and thickness variables can be estimated using any technique, including geostatistical methods.

The advantages of the polygonal, nearest neighbor, and sectional methods are their simplicity and theoretical ease of application (although, as noted, the geometrical manipulations may be extremely complex). Another bonus is the speed of obtaining results.

The polygonal estimator also has the added advantage of being the perfect declustering technique for irregularly spaced datasets. However, there are a number of distinct disadvantages, including the lack of applicability of the method to thick, on-tabular bodies, and the assumption of an unrealistic model for grade variation. The major objections by advocates of geostatistical techniques include the issue of ignoring sample support (every sample has a different support, equivalent to the size of the polygon or the area of influence) and possible conditional bias (high-grade areas are overestimated and low grade areas are underestimated). This is thoroughly discussed by Glacken and Snowden, 2001.

2.4.7 Smoothed grade interpolation

Most resource estimation techniques, and most in common practice today, use some form of grade smoothing to interpolate values into a block based upon surrounding samples. These fall neatly into two categories – the non-geostatistical methods and the geostatistical methods.

The methods of grade interpolation use some relationship between the distance of a sample from the block centre and the weighting which it is given. The most commonly-used approach weights each sample by some power of the inverse of its distance from the block to be estimated, usually the second or third power. The power chosen is somewhat arbitrary, although it is well-known that the lower the power, the greater the smoothing of grades. Higher powers of inverse distance tend to approximate a nearest neighbor approach, with distal samples receiving almost no weight (Glacken and Snowden, 2001). The geostatistical approaches to grade interpolation all rely on some form of kriging, whereby the weights given to each sample are derived from the semi-variogram model, which defines the continuity of grades in two or three dimensions. These geostatistical methods may in turn be subdivided into three classes – linear kriging, non-linear kriging, and simulation. All geostatistical methods rely to a lesser or greater extent on the assumption of stationarity, which is seen as the decision to pool data within a given area or domain, and not as a hypothesis which can be proven or disproven (Glacken and Snowden, 2001).

Linear kriging techniques are the simplest to apply, and centre on simple or ordinary kriging and their variants. These techniques are generally based on classical parametric statistics, which are affected by the distribution of the grade population underlying the data. Ordinary kriging is more resistant to departures from the assumption of stationarity than simple kriging, and is optimal for normal or Gaussian distributions of data, although still effective in other circumstances. Non-linear techniques have gained in popularity in the last 15 years, and address some of the deficiencies of the linear techniques. All non-linear kriging techniques are based upon non-linear transformations of the sample data such as the natural logarithm, the Gaussian (normal scores) transform, or the indicator transform. The most widely-used non-linear techniques are the various flavours of indicator kriging, uniform conditioning and disjunctive kriging. A comprehensive review of most non-linear kriging approaches is given by Vann (1998).

The perceived advantage of non-linear kriging techniques is firstly that they are able to cope with highly-skewed or mixed distributions of data (in other words, data most commonly seen in practice) and secondly, that they are able to derive local distributions of uncertainty which lead to a practical estimate of resources above a range of cut-off grades. Such estimates are known as ‘recoverable resources’, and while representing the correct support for mining still need to be subjected to the reserve process. It is fair to say that indicator kriging is the most easily understood and the most commonly applied of all the non-linear kriging techniques; overviews of indicator kriging are given by Glacken and Blackney (1998) and Khosrowshahi et al (1998).

Practical implementations of non-linear kriging techniques are presented by Elliott et al (1997), Collins *et al* (1997), and Matthews *et al* (1999). Indicator kriging is a reasonably robust technique in that considerable changes in variogram parameters can be withstood before greatly affecting estimated grades. Some other geostatistical estimation methods may be much more dependent on the accuracy of the variogram models. As gold typically has difficult variography (because of mixed mineralization styles and often noisy sampling and assaying) robustness is desirable.

2.4.8 Conditional Simulation

Conditional simulation, which typically uses a kriging technique combined with a Monte Carlo sampling approach, has the potential to enhance most of the geostatistical approaches currently used. Conditional simulation produces a number of equally-likely pictures of the grade estimation, each of which honors the distribution and spatial continuity inherent in the input data. Conditional simulation also has the ability to reproduce the level of variability in the samples, in contrast to other smoothing methods which reduce the variability. Simulations allow the practitioner to choose a result which is commensurate with the objective of the investigation, and have the ability to incorporate the risk and cost profiles of the operation under investigation. An overview of conditional simulation for resource estimation is given by Khosrowshahi and Shaw (1997). Practical aspects are presented by Rossi and Alvarado (1998). Schofield (1993) describes a particular use of conditional simulation, that of identifying the optimum drilling density. Conditional simulation may be most useful where data are closely spaced (i.e. ample conditioning data) and may be seen as supplementing other estimation methods (i.e. by providing sensitivity analysis) rather than replacing them. The key aspect of simulation which differentiates it from estimation is the consideration of probability or risk (Dowd 1996) and (Glacken 1996). It is possible to derive the probability of exceeding any given cut-off grade, or to define true confidence intervals of a specified degree of certainty around any single figure. The risk or confidence is a function of the production unit; the chance of achieving target grades or blends is greater for large parcels of ore (such as six months production) than daily production units. In any case, however, simulation allows the quantification of that risk.

2.4.9 Volume-variance, sample search and block size issues

It is well-known that there is a volume-variance relationship, whereby grades based on a sample support are more variable than grades based on a block support, and moreover that the larger the block, the less variable, or smoother, the distribution of the grades. It is thus important to model the appropriate block size to achieve a representative distribution on which to the grade/tonnage relationship for a given mining scenario. Whatever the estimation approach adopted, all techniques which seek to interpolate grades into blocks (except the nearest neighbor method) depend on the sample search procedure. The methods by which samples are selected for subsequent weighting is critical to the process and in some cases (e.g. Carras, 2001) are of more consequence than the estimation algorithm itself. A good sample search plan should have some or all of the following features:

- Declustering of data via octant or quadrant selection;
- Restrictions on the number of samples from one drill hole;
- Minimum and maximum numbers of samples specified for search;
- Preferred search directions (i.e. anisotropic search); and
- Restriction of the influence of high-grade samples.

In common with the sample search plan are issues of block size for interpolation into block models. The block size to be used should ideally bear some relation to the mining equipment Planned or used, often referred to as the selectivity. The concept of the selective mining unit is the smallest parcel of ground on which mining decisions, such as the allocation to ore or waste, may be made. The block dimensions should also be considered in relation to the sampling grid; blocks that are too small will result in over smoothing of the sample data and subsequent very low precision results (Glacken and Snowden, 2001).

Over-smoothing results in conditional bias, where by high-grade blocks are underestimated and low-grade blocks are overestimated. A number of tools are available to decide on a suitable block size – these are discussed in some detail by Krige (1996).Under smoothing results in a form of conditional bias whereby high-grade blocks are overestimated and low-grade blocks are underestimated. Elliott *et al* (1997) describe the change in the grade/tonnage relationship depending on whether a model is based on kriged block estimates (smoothed) or raw assay data (unsmoothed).

Reconciliation of the estimates with the actual head grades achieved during mining, plus the eventual processing of low-grade stockpiles, illustrates the sensitivities of the volume-variance effect. Conditional simulation provides some useful tools to check or determine optimal block sizes and the degree of smoothing inherent in a resource. Small scale simulations on representative areas or domains allow the change of support to be determined more accurately and locally than using one of the global techniques.

2.4.10 Checking the resource

Validation of the resource estimate can be undertaken using various tools as presented by Coombes, 2008, these in include:

- Graphical plots of sections/plans showing input data and block grade estimates;
- Histograms of input and output grade distributions within domains;
- Input and output mean grades within domains and Contact plots; and
- Generation of resource estimates using alternative algorithms or approaches.

The graphical validation plots of the resource need to make geological sense and show accord with the geological model. If not, further work must be carried out to resolve any anomalies. Once again, the importance of clustering on data statistics should be taken into account as this can bias the mean of the input data. It is also important to note that the mean grade obtained can also be biased if the population is skewed in distribution, and the input mean grade needs to be corrected accordingly. This is typically done by applying a top cut.

If the data is perfectly lognormally distributed (i.e. the logs of the data show a normal or Gaussian distribution), the Sichel mean is a better estimator of the unbiased population mean. The best validation tool for any resource estimate is reconciliation with production. Comparisons between the grade/tonnage curves for the resource model and the grade control model within a given identical volume and with the production records for the corresponding mining period will give an operation the greatest confidence in the merit of the resource model.

2.5 MINERAL RESOURCE CLASSIFICATION

2.5.1 Mineral Resource

A 'Mineral Resource' is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge (JORC, 2004).

The final task in resource estimation is to classify the Mineral Resources in order of increasing geological confidence, into Inferred, Indicated and Measured categories (Figure 2.7). This not only imparts levels of confidence in the results, but also dictates which classes of Ore Reserve may be generated by subsequent modification of the resource figures. The task of resource classification under the JORC Code (JORC, 2004) is the duty of the Competent Person, and therefore is ultimately a subjective duty, but notwithstanding this, the resource classification should ideally be based upon as many objective factors as possible. Clearly those factors related to the data and their configurations are of most importance, but these may be tempered by overriding geological, mining or data-related issues.

At the simplest level, resources may be classified on the basis of the overall *drill spacing* within the various domains or regions of the deposit. This is easily understood, but may not take into account any anisotropy which exists.

Another approach is to consider the average distance from the block centre of those samples used to estimate that block, or simply the number of samples defined inside the search volume.

Another technique expands search volumes during multi-pass estimations, with those blocks being estimated by the most distal samples having the lowest-confidence resource category. One of the more useful criteria for resource classification is the kriging variance or error arising from the estimation.

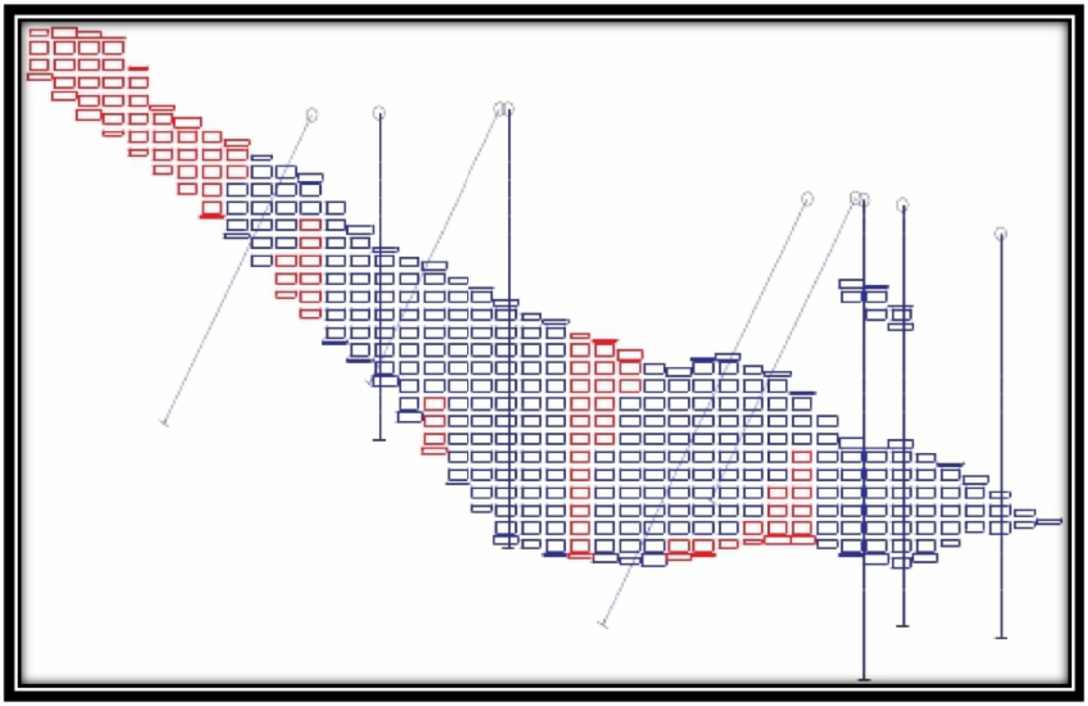


Figure 2.7: Drillhole section showing resource classification categories, Blue=Measured, Red=Indicated (Glacken and Snowden, 2001).

The kriging variance depends on the arrangement and continuity of samples around the block, and thus is a good indicator of overall sample spacing which takes anisotropy and sample clustering into account. Other numerical approaches include the regression coefficient and kriging efficiency measures proposed by Krige (1996). There is a move in some sectors of the industry towards the use of conditional simulation as a resource classification aid (Guibal, 1997). Since simulation quantifies the grade confidence, for a given block size, in addition to considering data position and continuity, it will provide more information to assist resource classification than numerical indicators such as the kriging variance alone. Any simulation-based resource categories must be tempered by detailed consideration of non grade factors, as with any classification. In most cases the resource classification will be based upon a combination of criteria, numerical and geological, with overall manual override by the Competent Person (Snowden,1996).

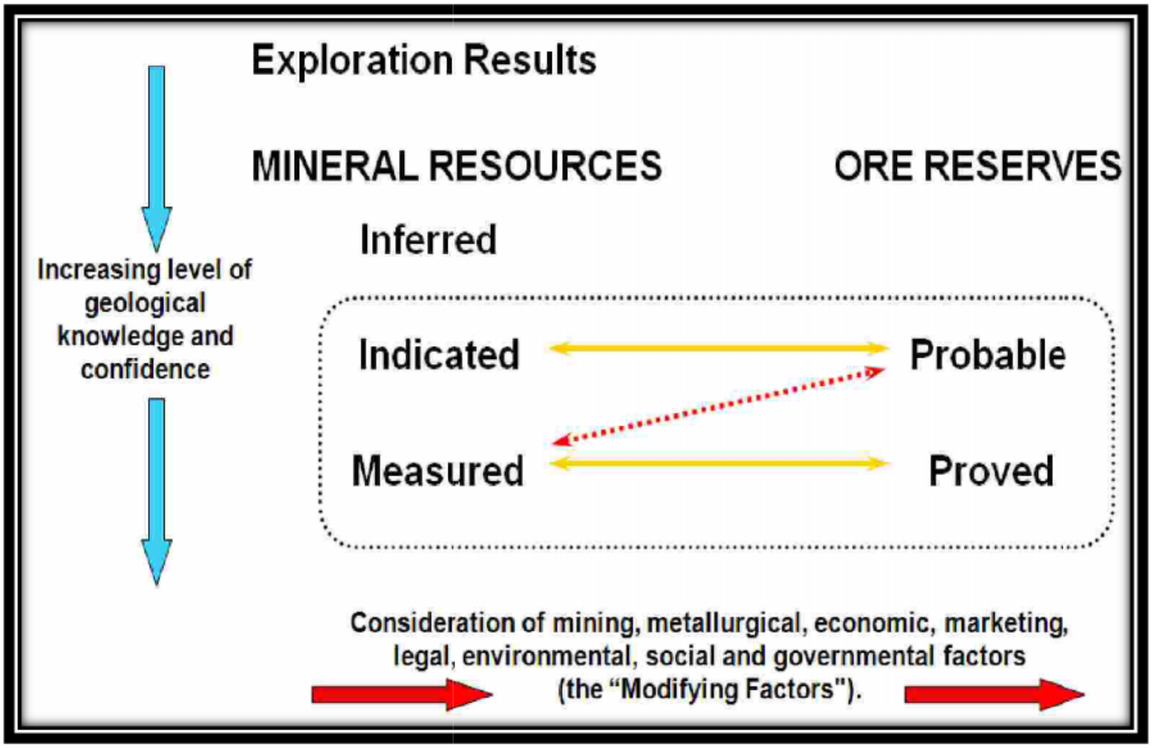


Figure 2.8: Relationship between Mineral Resources and Ore Reserves (JORC Code Review, 2011).

2.5.1.1 Inferred Mineral Resource

The Inferred category is intended to cover situations where a mineral concentration or occurrence has been identified and limited measurements and sampling completed, but where the data are insufficient to allow the geological and/or grade continuity to be confidently interpreted. Commonly, it would be reasonable to expect that the majority of Inferred Mineral Resources would upgrade to Indicated Mineral Resources with continued exploration. However, due to the uncertainty of Inferred Mineral Resources, it should not be assumed that such upgrading will always occur. Confidence in the estimate of Inferred Mineral Resources is parameters to be used for detailed planning. For this reason, there is no direct link from an Inferred Resource to any category of Ore Reserves (Figure 2.8).

Inferred Mineral Resource therefore is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low confidence level.

2.5.1.2 Indicated Mineral Resources

A mineral concentration may be classified as an Indicated Mineral Resource when the nature, quality, amount and distribution of data are such as to allow confident interpretation of the geological framework and to assume continuity of mineralisation. Confidence in the estimate is sufficient to allow the application of technical and economic parameters, and to enable an evaluation of economic viability.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource, but has a higher level of confidence than that applying to an Inferred Mineral Resource.

2.5.1.3 Measured Mineral Resources

Mineralisation may be classified as a "Measured Mineral Resource" when the nature, quality, amount and distribution of data are such as to leave no reasonable doubt in the estimate. In this regard the tonnage and grade of the mineralisation can be estimated to within close limits, and that any variation from the estimate would be unlikely to significantly impact potential economic viability.

This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit. Confidence in the estimate is sufficient to allow the application of technical and economic parameters and to enable an evaluation of economic viability that has a greater degree of certainty than an evaluation based on an Indicated Mineral Resource. Parameters such as which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and grade continuity.

2.5.2 Mineral Reserves

A 'Mineral Reserve' is the economically mineable material derived from a Measured and/or Indicated Mineral Resource. It is inclusive of diluting materials and allows for losses that may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction is reasonably justified. Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proved Mineral Reserves. Mineral Reserves are those portions of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Competent Person making the estimates, can be the basis of a viable project after taking account of all relevant modifying factors. Mineral Reserves are reported as inclusive of marginally economic material and diluting material delivered for treatment or dispatched from the mine without treatment.

The term 'economic' implies that extraction of the Mineral Reserve has been demonstrated to be viable and justifiable under reasonable financial assumptions (JORC, 2004).

The term mineral reserve need not necessarily signify that extraction facilities are in place or operative or that all governmental provisions and approvals have been received. It does signify that there are reasonable expectations of such approvals.

2.5.2.1 Probable Mineral Reserve

A 'Probable Ore Reserve' is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration/adjustments of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified in economic wise. As mentioned before with increasing levels of grade continuity and geological information gathering, Probable Ore Reserve has lower confidence than Proven Ore Reserve but the information gathered up to the level of Probable Ore Reserve is of sufficient quality to serve as the basis for a decision to develop a deposit (JORC, 2004).

2.5.2.2 Proved Mineral Reserve

A 'Proved Mineral Reserve' is the economically mineable material derived from a Measured Mineral Resource. It is estimated with a high level of confidence. It is inclusive of diluting materials and allows for losses that may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, including consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction is reasonably justified. A Proved Ore Reserve represents the highest confidence category of reserve estimate. The style of mineralisation or other factors could mean that Proved Ore Reserves are not achievable in some deposits (JORC, 2004).

CHAPTER THREE

METHODOLOGY

3.1 LOCAL GRID SYSTEM

Differences in the position of the local grid were noted from report to report over the years with the differences related to a change in the operator for field work. The actual location of the various grids is not given in reports.

It was decided in the course of the 2010 program to base the local grid according to the following 2 historical collars that were surveyed in 2007 using a differential GPS. The two starting points and their 2010 grid reference and UTM coordinates are as shown in Table 3.1:

Table 3.1: Local grid reference coordinates setup (adopted during 2011-2012 campaign)

Drill Collar	2010 Easting	2010 Northing	Longitude (m) Datum: WGS 84	Latitude (m) Datum: WGS 84	Elevation (m)
KB07-92	1500S	0 000E	545016.00	1289583.00	430.00
KBD31	2100S	0 000E	544818.37	1289021.37	415.393

The UTM coordinates are from the 2007 drill hole collars survey conducted by CME using a Trimble GeoExplorer XT rover in conjunction with a Trimble 4600 Base Station receiver set up in the AGG Kobada camp. The base station and the rover data were downloaded and differentially corrected using the Trimble GPS Pathfinder Office software. The base station is located at 11°39’47.686840”N latitude, 8°35’06.720973”W longitude and 442.230 meters elevation above sea level (CME, 2007).

3.2 HOLE PROPERTIES

All the holes used in this study are reverse circulation (RC) holes resulting from different drill campaigns. Though discrepancies may occur, 28 holes have been carefully selected in each drill direction without any special preferences to test the same area under this study.

3.2.1 Collar Locations

All the holes used in this study have been surveyed on the field during the 2011-2012 campaign with Novatel differential GPS (Flex Pack 6 DGPS using the OmniSTAR XP-G2 service), which gives ± 20 cm accuracy. To achieve this level of accuracy or better depends on the number of satellites that registers with the system at a particular time. Before subscribing to the Omnistar service drill holes were located/sited with hand held GPS with presumed accuracy of ± 3m also depending on the satellite availability. After drilling the actual drilled location is marked twice and the mean taken to be the coordinate location of the collar although such coordinate repeatability is highly unattainable. All holes are surveyed with a Flexit that measures downhole attributes such as azimuth, magnetic field, inclination and temperature at each depth selected. These measurements are taken at three planned depths: 6 meters, 65 meters and 130 meters for DDH. For RC holes, surveys are taken at 6m and at the EOH (max depth). These measures are taken by the driller, during the advancement of the hole.

3.2.2 Downhole Surveys

All holes are surveyed with the Reflex EZ-Shot instrument that measures downhole parameters such as azimuth, magnetic field, inclination and temperature at each depth selected. The first survey is taken at 6 meter depth for the aforementioned parameters but especially for the azimuth and inclination. The aim of this first survey is to check and verify the collar set up and to make the necessary adjustment if required in order to ensure the correct target intersection.

Surveys are then carried out at the end-of-hole (EOH) meter with a final bottom of hole survey. If the inclination of this bottom of hole survey deviated ≥5º from the collar inclination, additional surveys at selected depth intervals upwards are performed. This ensures better control of the planned drill path and subsequent intersection of the target area. Surveys are performed by the drilling contractor and are checked and confirmed by the geologist in charge of the drill site.

3.2.3 Topography

No topographic file was available for the study, however, a conventional topographic surface was generated using the collar elevations available at the area. To a larger extent it could serve as a valid surface having been surveyed with a differential GPS with accuracy of ±20 cm.

3.3 SAMPLE PREPARATION

3.3.1 Drill Site

Before drilling commences, the onsite geologist inspects the cyclone to ensure that it has been properly cleaned and is free of contamination. If contamination is observed, that material is manually removed, and the cyclone is cleaned with compressed air. During drilling, the driller lift the feed (load) after each meter drilled to flash out any chips/debris so as not to contaminate the next sample.

Regular inspection and cleaning of the cyclone is done by releasing compressed air to flash out water and other material which may contaminate the sample, typically during rod changes. To enhance the quality of sample recovery, the onsite geologist ensures that the drilling rate is reasonably steady and that the driller briefly lifts off the hammer and blows gently at the end of each sample length (especially in the oxide portion).

Sample numbers, hole numbers and metreage are written on each RC-sample bag. The on site geologist also writes the same information on the sampling sheet and crosschecks the information on individual RC-sample bags to ensure accuracy of sampling.

Samples are collected from the cyclone direct into a large polyethylene bag large enough to collect all sample drilled in 1m. The sample length is usually 1 m and is measured and marked on the drill rods with drill grease. The RC samples recovered are not split at the drill site, the bulk material from each meter is bagged and transported to the campsite for sample preparation where the first split is performed in a large rotary splitter.

3.3.2 Preparation Shed

Samples drilled from the various sites are shipped (transported) to the sample preparation shed situated in the AGG campsite.

Samples are arranged in sequential order and weighed. Two methods have been used in the sample preparation laboratory to ensure homogenized sample result. These are; one done with Jones riffle splitter and the other is roller mixing the whole sample in the roller mixing device set up at the laboratory. The westerly directed drilling all went through Jones riffle splitter method before the advent of the roller mixing which was brought in to help solve the issues of backlog at the field facilities and of high grade variability due to the high nugget effect (whether it solved the latter issue is not within the scope of this study). The 2011-2012 drilling campaign used the roller mixing during the sample preparations.

In the first method (Jones riffle splitter), the sample was poured into a large basin and subsequently poured into a riffle splitter which splits the sample into two equal halves. The parts were combined and passed through the splitter again to ensure thorough mixing of the sample. This process is repeated one more time after which the splitter is utilized to obtain a 3 kg sample which is sent to the lab. The remaining part of the bulk is sent to the sample storage. After each split, compressed air is used to clean the riffle splitter and the large basin. In the second method, the roller mixing device is comprised of two metal H-beams (height 2 meters) clamped together with freely moving cylindrical metal tubes (ball bearings at each end) mounted on top of the H-beam. Two similar beams are mounted opposite each other about 0.8 to 1m apart. A linoleum mat is spread over these two cylindrical metal tubes to make a U-shape between the two H-beams. The linoleum also goes half way the height of the H-beams on the outsides.

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

After fifteen strokes the sample was deemed to be thoroughly mixed (based on numerous tests with a fixed number of pea-size lateritic nodules) and everything is poured back into a big basin which is then split through a Jones riffle splitter to get the required 3 kg of sample for the laboratory for analysis.

Sample numbers were uniquely assigned to each sample and are written on the sample bags. Sequentially numbered sample books with two tear tags are used. The first tag is placed inside the sample bag. The second tag is clipped (stapled) onto the same bag and the last tag remains in the booklet with details of the hole number and metreage. (Lalande, 2010).

3.4 QUALITY ASSURANCE, QUALITY CONTROL (QAQC)

Quality assurance quality control procedures for the Kobada project have varied from one drill campaign to another prior to 2010 when a fixed sequence of blanks, field standards and duplicates was introduced. The purpose and importance of the quality control protocols have always been adhered to irrespective of the campaign season. However the more recent campaigns (2010, 2011 and 2012) have seen a well preserved quality control data and an improved level instituted by the Qualified Person, **Pierre Lalande.**

The following QAQC protocols have been used during the last three campaigns.

- A duplicate sample is taken every twentieth drilled sample with each sample analyzed by Leachwell on 2-kg aliquot.
- An external laboratory duplicate sample is taken every thirtieth drilled sample with both samples analysed by Leachwell (on 2-kg aliquot at the main laboratory and 1-kg aliquot at the external laboratory) .
- A method duplicate sample is taken every hundred drilled sample (50-g aliquot fire assay and 1.000-g screened metallic fire assay).
- One of two standard samples are included every twentieth drilled sample (one standard is twice the overall grade of the 2008 resource estimate and the second one is about the overall grade of that estimate.
- A blank sample is included every twentieth drilled sample.
- The Leachwell tails are tested every 25th sample for remaining gold and analytical recovery.

3.4.1 Blank Material

AGG uses certified blank materials from the ALS Laboratory of Bamako. It is composed of crushed sandstone from the Taoudeni basin deposits near Bamako. It is well known to record below detection gold values with two 50-g fire assay check for each 25 kg of blank received.

3.4.2 Certified Standards

Again different standards have been used with regards to different drill campaigns. Where assayed values were available these standards have been statistically treated to ascertain if the reported assays were within precision limits . However, with regards to the 2010, 2011 and 2012 drilling campaigns, two standards CRM G306-4 and CRM G901-8 were used. These standards were purchased from Geostat Laboratory (Australia) that has validated and certified their content. As per the quality control protocol that existed during the campaign, 100g bag of certified standard is added to 3kg of blank material. The contained grade is then calculated back after the analysis from the laboratory taking into account the dilution factor. In the case of fire assay, 50 g out of the 3100 g is assayed and for leachwell, 2000 g out of 3100 g is analyzed within precision limits.

3.5 MODEL TERRAIN FEATURES

3.5.1 Duricrust and Mottle Clay Zone

Includes all material logged as laterite, red lateritic and mottle clay usually iron rich forming notably some form of horizontal cap directly overlying the saprolite. This zone is hardly thought as being in-situ but rather seen as a remobilised gold zone. The highly discontinuous nature of the grade distribution tend to make the grade interpolation quite difficult as in some instances the zone follows the structural interpretation in some form of continuity down into the saprolite zone. From a geologic perspective, none of the lithologic units were considered to be hard boundaries to gold mineralisation.

3.5.2 Saprolite

Completely oxidised material logged as saprolite was modelled as a zone overlying the partially oxidized and primary rock. Most mineralisation in the study area and the Kobada Zone 1 occur in this corridor. The saprolite could be as deep as 120 meters vertically in some areas in the Kobada Zone 1.

3.5.3 Primary or Freshrock

This include all materials that fall below the saprolite. For the purposes of this study, the partially oxidized and the fresh rock zones were combined as a unit for metallurgical reasons. The mineralisation in the bedrock is characterised by intense veining/silicification and disseminated pyrites, arsenopyrites in intercalation of shales and greywackes.

3.6 ORE ZONE DEFINITION

The methodology employed in this estimation study utilised a 3-dimensional solid model of mineralisation interpreted with 0.2 g/t Au sample cut-off grade, defined on cross sections on the drill fences. All post 2007 RC holes have the same diameter hammer for sample support uniformity. For the westerly directed drilling there are six drill fences , the drill spacing is averagely 50 meters for the first four fences and 100 meters for the last two northerly fences. The southerly directed drilling however is a staggered grid fenced on averagely 40 meters having eight drill fences.

The process used in generating the 3-dimensional ore zone solids are as outlined in Figure 3.2 utilizing Gemcom's Surpac 3D software. The same approach was employed for both drilling directions:

Individual sections were digitized on the drill fences at 50 meters centres/slices for the westerly directed drilling and 40 meters fences on the southerly directed drilling. These grade shells were generated using the 0.2 g/t Au cut-off at a composite length of 5m or by intuition where the grade is appreciable. In addition a 0.2 g/t Au grade cut-off seemed to provide a consistent and clearly visible pattern of mineralization which was helpful in defining reasonably fairly continuous ore zones especially in the westerly directed.

This approach was achieved by joining the grades on individual drill holes by the various digitizing tools available in Surpac 3D environment such as points, polylines/segments etc.

The shells/polylines which by the software convention should be in clockwise direction were closed off after digitising each mineralised shell. Line validations approaches such as removing duplicate points/segments, crossover point, spikes, etc. were all checked. End of sections were closed off by either extension for half of the respective drill spacing where there are no drill holes or where there are drill holes, midpoint perpendicular to the drill hole. The digitizing was all done based on the structural information that has been presented on the respective stereonets presented in chapter 1.

Solid modelling was done by interactively connecting individual grade shells from section to section where applicable by using solid triangulation tools available in the Surpac software.

The end of the solid is closed by extruding the solid halfway with regards to the drill spacing used. Where grade continuity is visibly reliable, sections are joined to form grade solid models. In the case of the Kobada North where the study is carried out, the drilling is sparse and the grade appear to be more discrete, the model appear to be more individual than being continuous. Once constructed, the shells were checked for duplicate faces, openings and self intersecting triangles are all resolved by validating the solid.

Exploration data is captured in the WGS84 UTM grid with Mapinfo® being the preferred software used. Exploration Data Analysis (EDA) and the resource modeling work has been done in Surpac with support from MS Excel, SPSS and QCAssure softwares. Spatial data analysis (variography) is done in Surpac with some confirmations in Sage2001 software by Isaaks.

3.7 RESOURCE ESTIMATION PROCESS

In the following Figures 3.1 through 3.3 are illustrations that summarizes the various categorical flow sheet diagrams of the various stages used in the generation of the resource estimate for the Project Area.

3.7.1 Database Flow Chart

The resource estimation process starts with a good and a consistent database. This has been achieved quite easily due to the relatively small nature of the database. The database definition, mapping, review and auditing were all done in Surpac. After the database was connected, it was subjected to the software's audit capabilities and underwent a thorough scrutiny. Checks were done to verify the corresponding maximum depths as contained in the collar table with regards to the other attribute tables such as the surveys, the lithology, and the assays. Data consistency pertaining to other aspects such as sample overlaps, duplicate samples and missing samples have been thoroughly checked and dealt with appropriately prior to the statistical analysis on the database (as presented in the database flow chart, Figure 3.1).

Having been personally involved in the project especially data collection and the subsequent entry and management of the database it was relatively easier for me to nonetheless undertake these aspects of the database validation, and the issues that arose which I dealt with were very few. In case any of the above mentioned protocols had failed it would have meant going back to the original database to try and identify and fix all such errors before commencing statistical analysis, this is professionally acceptable.

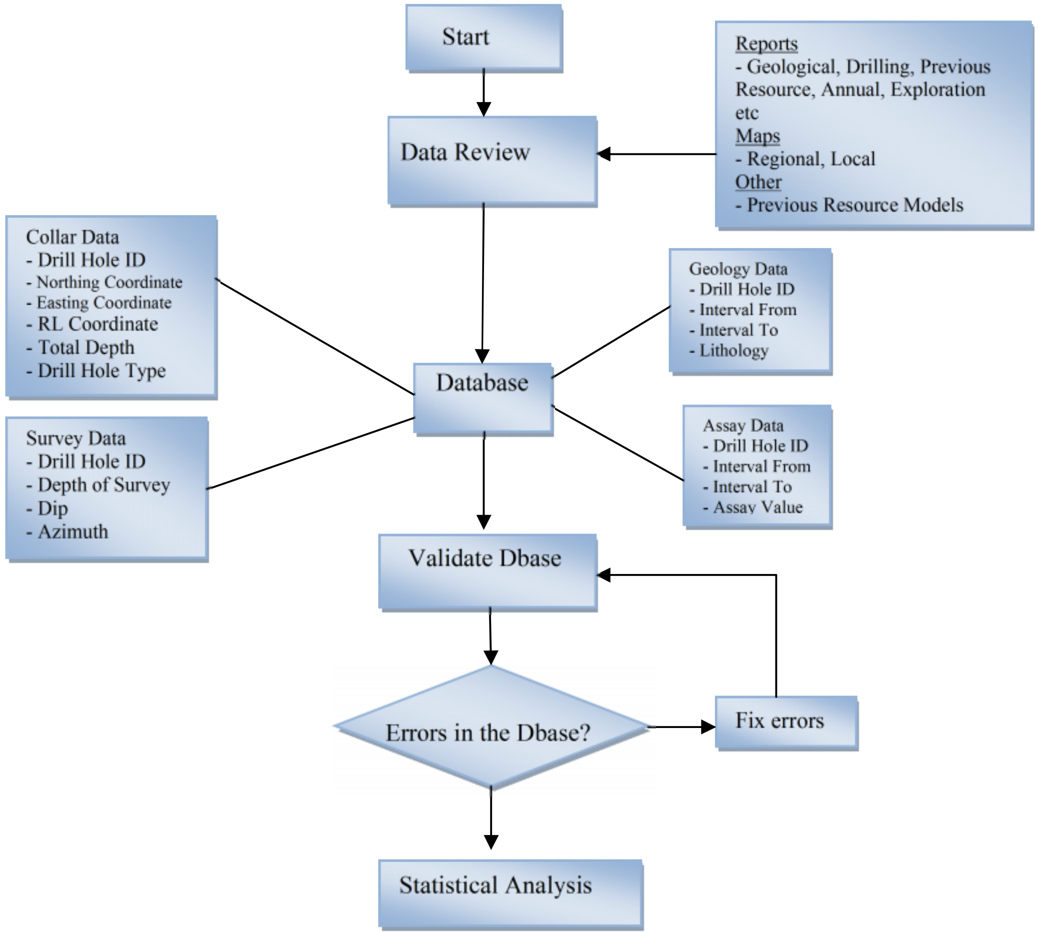


Figure 3.1: Database Flow Chart for ensuring data integrity (modified after Fieldgate (2009)

3.7.2 Wireframe Flow chart

The procedures constituting the flow chart of the wireframe/solid modeling (Figure 3.2) is as briefly illustrated below.

Prior to the commencement of the wireframe or solid modeling, a search distance of half the width of the major grid spacing of each section either side of the grid was decided upon to be used; for the westerly directed drilling a distance of 25 meters whereas the southerly directed drilling, a distance of 20 meters. The digitizing of the grade shells were done in 2-dimensional (2D) space before the eventual interpolation into a 3-dimensional space. A lower cut mineralised envelope of 0.2 g/t Au was used to digitize out the 2D grade sections.

Minimum envelope width of 2 meters which includes 1m @ <0.2 g/t Au, where required to bulk out envelope to 2 meters width.

With regards to this specific project (may not apply to the entire Kobada project), maximum internal dilution of 5 meters has been allowed, unless interval comprises a broad envelope capable of being correlated. Uncorrelated grade intervals that were not able to be tied in on and between sections were not digitized as grade polygons or in some cases where it made geological sense treated as isolated polygons.

The digitizing direction that I employed was the conventional clockwise direction with section facing northward for the east-west (westerly directed drilling) and facing west for the north-south (section southerly directed drilling).

All section envelopes have been extended midway between their respective drill holes. When the envelope ends, the median slope of the surrounding drill holes have been used to determine the end shape of the envelope. No mineralised grade envelope has been ended perpendicular to the grid, the halfway rule has been used to end all mineralised grade envelopes between drill holes. I have employed the same approach to end Sectional envelopes beyond drill holes. In cases where an envelope hasn't terminated between drill holes, I have extended the envelope half the section width of the last drill hole intercept at the apparent dip of the overall envelope. Unless the mineralised envelope is supported by intercepts on pervious sections. In which case the envelope has been extended down to the level supported by other sections. All envelope ends have been set to taper. No tapered end was <2 meters wide, which is the minimum envelope width. During tapering I have used the natural dip/plunge of the grade pod to determine the final taper width unless the assay interval from which the envelope is being extended is already 2 m wide. In which scenario the 2 meters width has been maintained across the width of the entire extension.

I thereby triangulated the individual sections by way of joining adjacent sections into desirable solids models using Surpac.

With the bifurcation as was the case in some of the solids, copies of the un-bifurcated envelopes were placed half way between sections, these were used as the joining point between the bifurcations and the main body of the envelope. This minimised the input of waste material into the model.

After adhering to all the above mentioned protocols I made sure the mineralised solid was consistent with the grade distribution by superimposing it with the raw assay display. After achieving this satisfaction with the model, I validated the individual grade polygons to be sure I have been able to generate solids without openings.

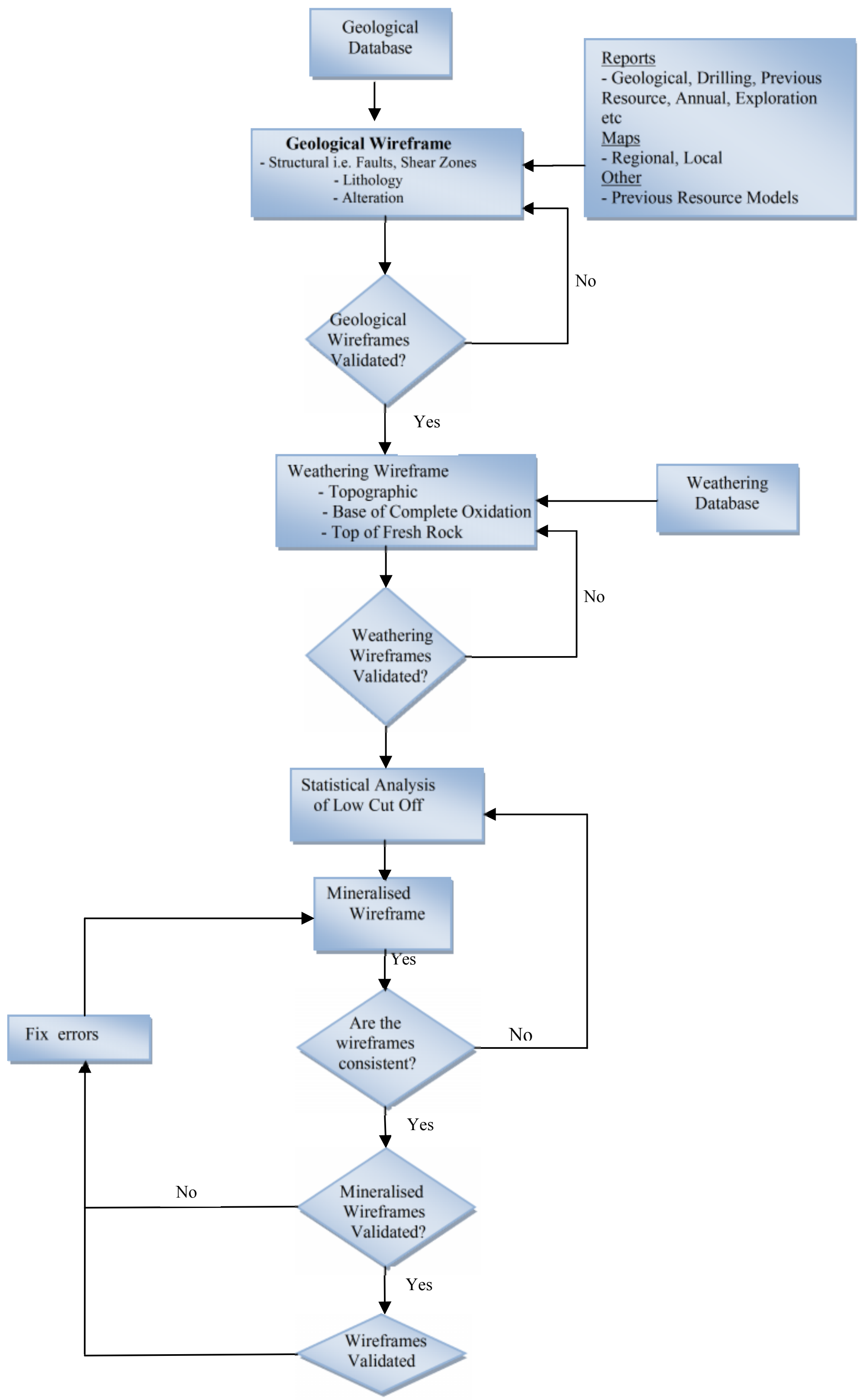


Figure 3.2: Wireframe flow chart (modified after Fieldgate, 2009)

3.7.3 Variography Flow chart

Spatial analysis is undertaken to establish the relationship between sample points at varying distances apart, and to establish a level of confidence in the estimation of grade at any point between sample locations. The variogram analyses was also performed to assist establish the major directions of continuity and to provide the variogram parameters required for geostatistical grade interpolations.

A search region or ellipse was defined based on observed directional grade continuity, and this was achieved having a crucial consideration for the geological observations.

Downhole and directional variograms were generated and modeled for each drilling direction for the two geostatistical domains.

After validating the wireframe the data was separated into two domains being the duricrust and the saprolite domains for both drilling directions. Data compositing was done within the validated wireframes to enable for statistics to be performed on the composited dataset. Histograms of the composited data were produced to visualise the data distribution both in the normal histogram and log transformed mode. The data was positively skewed in all the cases with regards to both drilling directions. However, the positive skewness was as a result of the presence of some few extreme values otherwise known as outliers and not by mixed populations. Outliers concentrate in the zone of erratic grade distribution on the histogram resulting in the positive skewness.

After further statistical analysis on the composited data I found the co-efficient of variation (COV) to be higher than the industry accepted COV of 1.2 (as stated by Coombes, 2008). The high COV was indicative of the need to apply a top cut to the dataset. This was achieved by using Surpac string maths by the following expression D1= iif(D1>V1,V1,D1) which implies that If D1>V1, it becomes V1, if not it retains the initial value, where D1 is the field where the composite gold values are stored and V1 is the value at which the top cut has been set.

After the treatment of the outlier values, the composite data was then set for the spatial/variogram modeling. The spherical model was selected due to its rigidity and the importance of the nugget variance to this estimation. The spatial modeling is a very important aspect of the estimation process in that it presents an understanding on how samples relate to each other spatially an then use this information to infer how samples may relate to a location that has not been sampled. This information has been used to build up an estimate of the block grades. The objective for the spatial analysis is to establish a model of grade variability in 3D, which in turn controls the model weights during Kriging estimation.

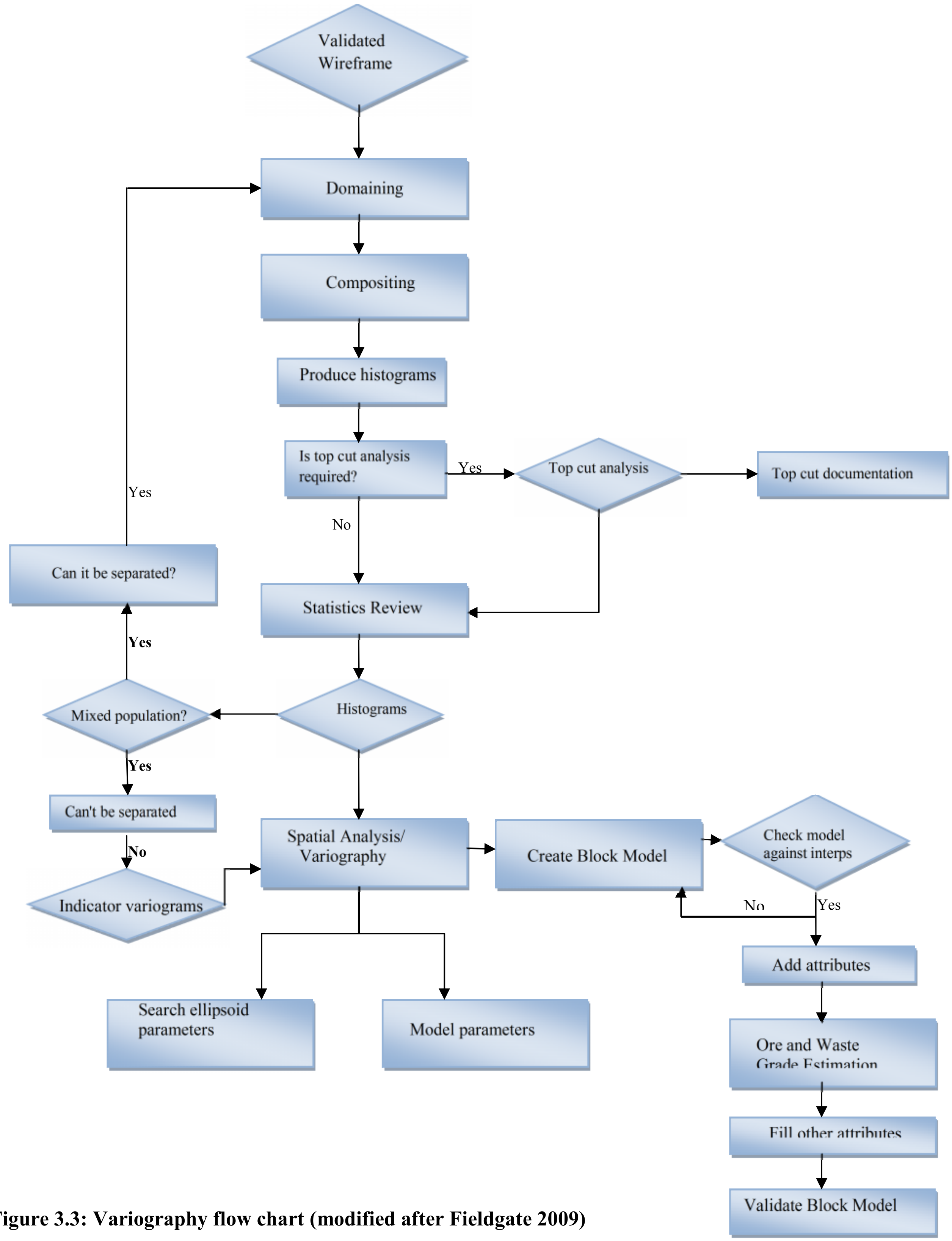


Figure 3.3: Variography flow chart (modified after Fieldgate 2009)

CHAPTER FOUR

RESULTS AND DISCUSSIONS

4.1 Exploratory Data Analysis (EDA)

Exploratory data analysis (EDA) incorporates descriptive statistics (Tables 4.1 and 4.2) and other boundary classification statistics in evaluating the grade distribution of domains that would be accessed during the estimation process. This is an important precursor to the block modeling process as it highlights the various problematic areas that would be encountered during interpolation and hence provide a means of mitigation prior to the interpolation process.

The available input data is subdivided into groups and then re-combined into a series of quasi-homogenous populations, called domains.

The project area consists of a total of 56 RC drill holes (Figure 4.1), 28 holes each respectively drilled to the west and the south. Southerly directed holes have been orientated in a staggered pattern approximately 50m x 40m grid at an azimuth of 200° (Figure 4.2), with sectional view of N70°W. To the west, the remaining 28 holes have been drilled at an azimuth of 290° (Figure 4.3) with an orientated sectional view of N20°E.

The first 8.956 samples of the 2012 campaign were analysed by 2 methods: Leachwell on 2-kg aliquot and fire assay on 50-g aliquot. Statistical analysis of the two sets of results shows that the partial Leachwell procedure gives 31% more gold than fire assay on arithmetic average of uncut high results (>10 g/t). After cutting back high results to 10 g/t, Leachwell is higher by 42%. All the samples contained in the southerly directed 2012 drilling were analyzed by Leachwell.

With the exceeption of KBRC068 and KBRC069 which were analyzed by 50 g Fire Assay (FA50), the rest of the KBRC9-xyz, KRC10-xyz and KBRC11-xyz drilled westerly series were analyzed by Leachwell.

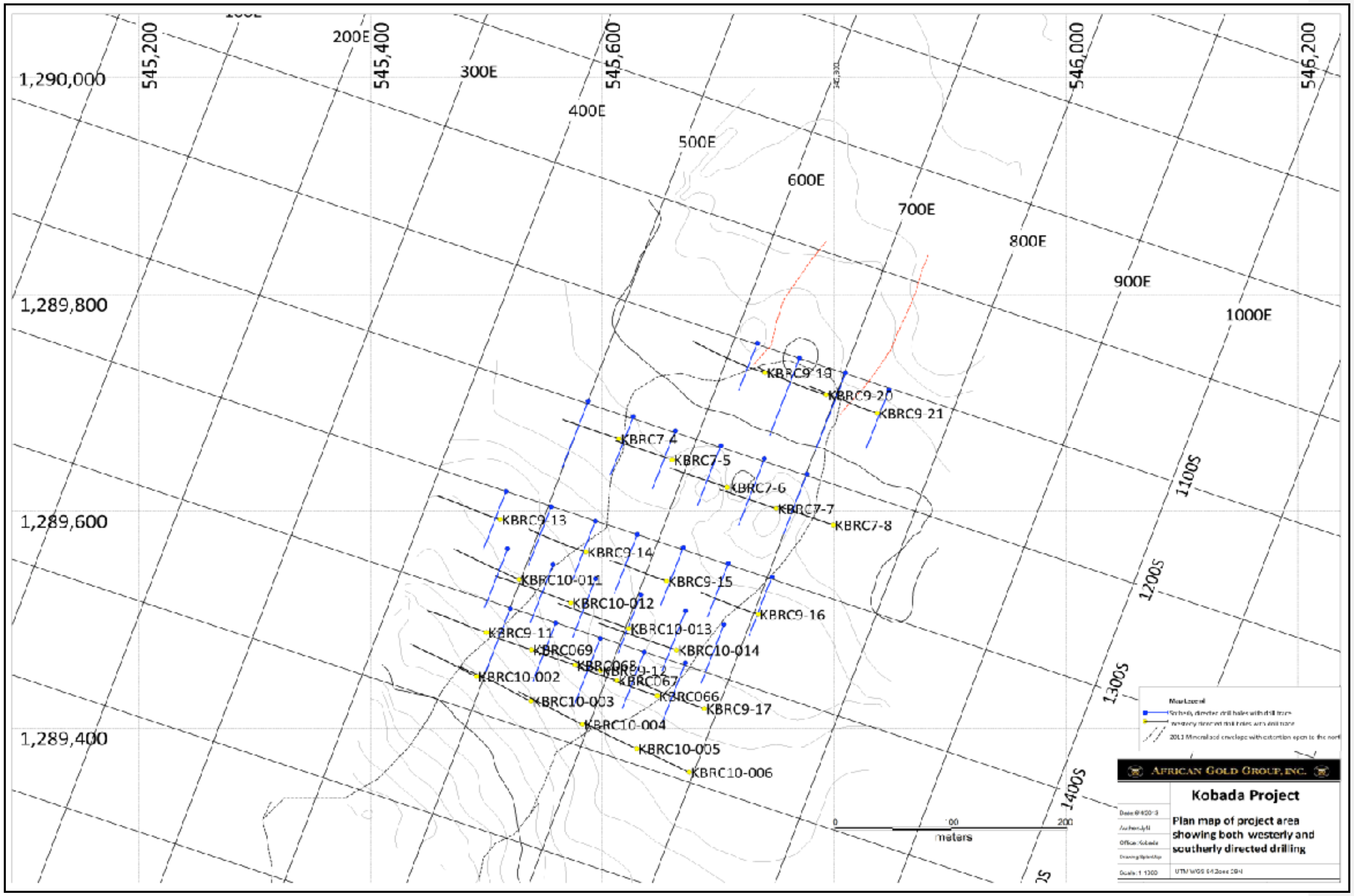


Figure 4.1: Drill holes of both orientations with regards to the local grid placement of the project area

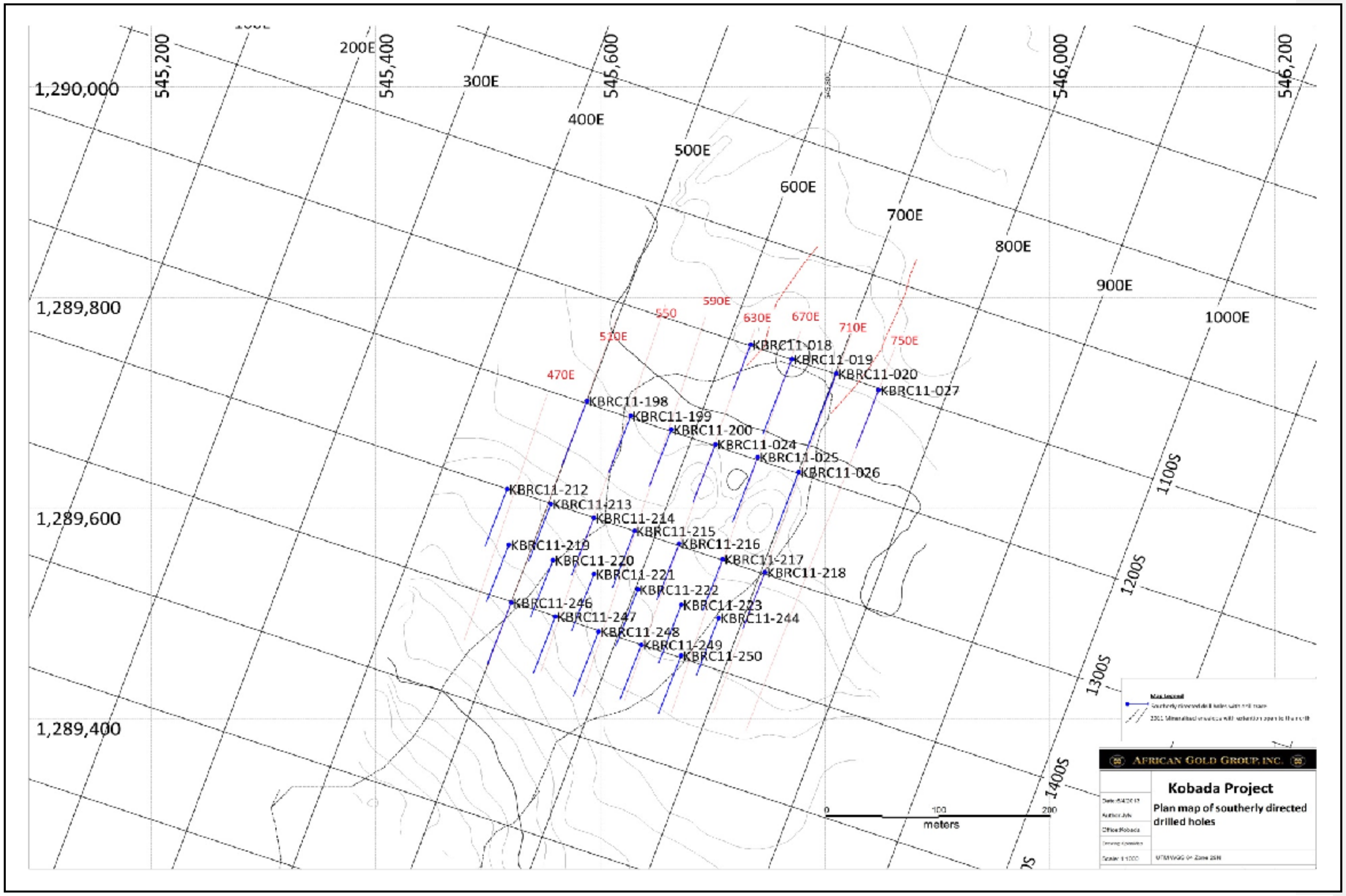


Figure 4.2: Drill collars and drill traces of southerly directed drilling

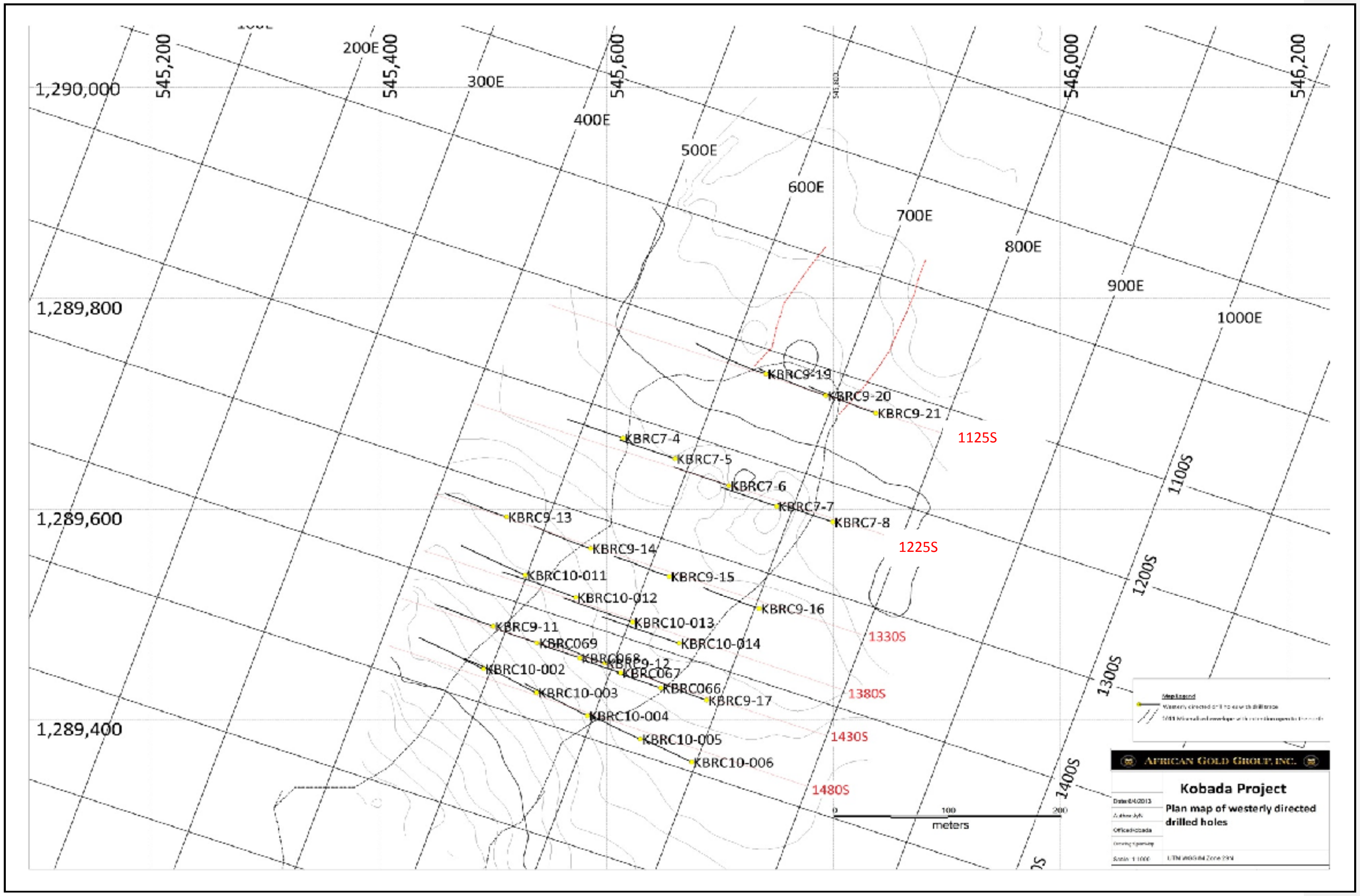


Figure 4.3: Drill collars and drill traces of westerly directed drilling

Table 4.1: Descriptive statistics of the raw data as obtained from the database

			DESCRIPTIVE STATISTICS						
Drilling Orientation	Drill holes	Assay Method	N	N Missing	Mean	Min	Max	CV	Median
Azimuth 200 degrees	28	Leachwell	2828	85	0.33	0.0005	48.31	5.71	0.04
Azimuth 290 degrees	28	Fire Assay (2) Leachwell (26)	2757	55	0.37	0.005	100.00	7.40	0.03

Drill holes have been logged and samples assayed and maintained in Microsoft access database. The study area represents a small portion of the Kobada deposit. Geologic model was not done due to limited geological data interpretation. The prime purpose of the project is to ascertain the best drilling orientation that could yield the most economically viable resource estimates. The main production strategy of Kobada project is to access the economic feasibility of oxide gold ore having in mind of the gravity ore treatment plant. The main geostatistical domain was a 0.2 g/t Au cutoff grade shell in both orientations.

Surface mapping of the project area reveals Au grades are proximally in place right over the bedrock mineralization (residual anomaly). However grade concentrations in primary fresh rock are rather low and might not be economical granting high operation cost.

Geostatistical domains are therefore grades within a 0.2 g/t grade shell and outside. This was done exclusively for each orientation drilling.

4.2 Data Validation

Data verification and validation of the whole Kobada project of which this project is part has been ably taken care of by Lalande (2010) and briefly presented below.

- **Field duplicates of 1 in 20 samples**: A total of 890 originals were duplicated in the field. There are no significant bias between the set of field originals and duplicates (Table 4.4).

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When excluding low grade pairs with an average below 0.1 g/t, this remains 349 pairs. The duplicates have a mean 9% lower than the originals (0.611 vs 0.678 g/t) while the median is 3% lower. For pairs (176) with an average above 0.3 g/t, the duplicates have a mean 11% lower than the originals (1.130 vs 1.269 g/t) while the median is 2% higher (Figure 4.4).

The variance from the pair average is very high at ± 44% for the 349 pairs higher than 0.1 g/t compared to the original. Such high variance for even 2-kg aliquot may not allow for the estimation of indicated resources, less so for measured.

In an effort to reduce the large variance, it was recommended for the 2011 drilling program that field samples be circulated twice through the 3-tier riffle splitter (or Jones riffler) to blend the samples before mass reduction to 1/8 for shipping to the laboratory.

Table 4.2: Descriptive statistics of the assayed pairs

Mean	0.678	0.611
Standard Error	0.084	0.063
Median	0.235	0.229
Mode	0.004	0.005
Standard Deviation	1.63	1.225
Sample Variance	2.658	1.5
Kurtosis	45.487	25.91
Skewness	6.02	4.582
Range	16.99	10.914
Minimum	0.0005	0.001
Maximum	16.99	10.915
Sum	254	229
Count	349	349

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Field Code Changed

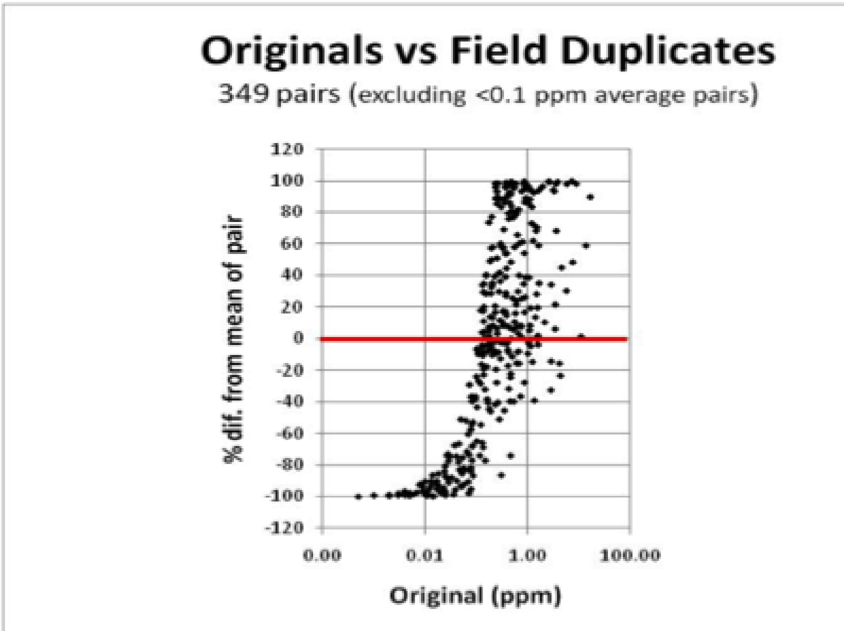

Figure 4.4: Assayed pairs of field original samples against field duplicates

- **Blanks**: Every 20th samples, a blank or one of two field prepared standards is inserted into shipments to the laboratory.

Blanks are prepared by the ALS laboratory in Bamako. They are from a quarry of flat lying sandstone near Bamako that has been crushed to minus 3 mm, with two 50-g fire assays (FA50) per 25 kg. They are delivered to the Kobada camp in 25-kg rice bags in 500-kg shipment. A total of 6.5 tonnes of blanks were delivered represented by a total of 650 FA50; of these 420 (65%) are below the detection limit (0.005 ppb); the remaining range from 5 to 9 ppb. The overall grade is 1.7 ppb and the median 1 ppb.

A total of 921 blanks (<0.005 by fire assay on 50g) were sent to lab and analyzed by Leachwell of these 904 (98%) were equal to or below 0.005 g/t; 5 were between 0.005 and 0.010 g/t. Eight (8) blanks from as many lab certificates were between 0.044 and

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0.231 g/t and the discrepancies cannot be explained. The remaining four (4) blanks could have been blanks to which a 100-g sachet of commercial standards were added (G306-4 and G901-8).

- **Field Standards**: Two grades of prepared standards (3.1 kg each) were used. Each standard is made up as follows: 1) put 1.5 kg of blank is put in a plastic bag; 2) add a 100-g sachet of commercial standards from Geostat Pty, Australia (G306-4 at 21.73 g/t or G901-8 at 47.24 g/t); and 3) covered with another 1.5 kg of blank. The bags are not blended on purpose so that a sample with a very strong "nugget effect" is delivered to the laboratory to test both sample preparation and analytical procedures, not just analytical procedure which is done extensively by the laboratory internally as part of their QA/QC procedure. The field standards are sent to the laboratory for Leachwell analysis on 2-kg aliquot. The expected grades of the two standards with a range of +/- 2 standard deviations are respectively 647 to 748 ppb and 1,426 to 1,626 ppb by Leachwell assuming a grade of 1.7 ppb for the blank portion.

A total of 465 low-grade field standards were analyzed by Leachwell. Of these, 235 samples (51%) fall with the expected 0.612-0.793 range; 35 samples (8%) are within +/-15% of the expected range; 35 samples (8%) fall better than +/-25% of the expected range. There are 55 samples (12%) which show minor discrepancies (better than +/-40%) and 12 show (3%) major discrepancies (from down to -40% to +150%) of which 8 could be due to a G908-4 sachet (instead of G306-4) mistakenly added to the blank.

A total of 444 higher grade field standards were analyzed by Leachwell. Of these, 143 samples (32%) fall with the expected 1,426 to 1,626 ppb range; 162 samples (36%) are within +/-15% of the expected range; 101 samples (23%) fall better than +/-25% of the

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expected range. There are 23 samples (5%) which show minor discrepancies (better than +/-40%). Sixteen (16) samples (4%) show major discrepancies (from down to -40% to +150%) of which 9 could be due to a G306-4 sachet (instead of 908-4) mistakenly added to the blank and one sample could simply be a blank. The major discrepancies (16) for both field standards were reported from different laboratory certificates. Overall, the two field standards are 10% lower than the expected value which can be explained by Leachwell being a partial analysis compared to fire assay (a total analysis).

4.3 Model Generation

Gold mineralisation has been constrained by the development of wireframes modeled at a 0.2 g/t Au lower cutoff grade at an average dip of 74^{o} respectively for both drilling directions. Grade pods were extrapolated approximately halfway beyond the limiting drill hole for both the down dip and along strike directions. The interpretation was completed based on visual geological review and has captured the vast majority of the mineralization within the zone. Two main geostatistical domains were generated for the deposit which basically encompasses the duricrust and saprolitic for both drilling orientations. All datasets within the transition and fresh zones were also categorized into saprolitic zone because of limited data for spatial analysis. Weathering or oxidation surface was interpreted based on drill hole and comprises of the Base of Duricrust and Base of Saprolite. The steeply dipping mineralised corridor occurring within the NE-SW Kobada shear zone is much more continuous in the westerly directed orientation drilling (WDD), compared to the mineralisation found in the southerly directed orientation drilling (SDD) which occur in pods and swamps. Figures 4.5 through to 4.8 show the respective vertical sections which were used to generate the wireframe respective models shown in Figures 4.9 and 4.10.

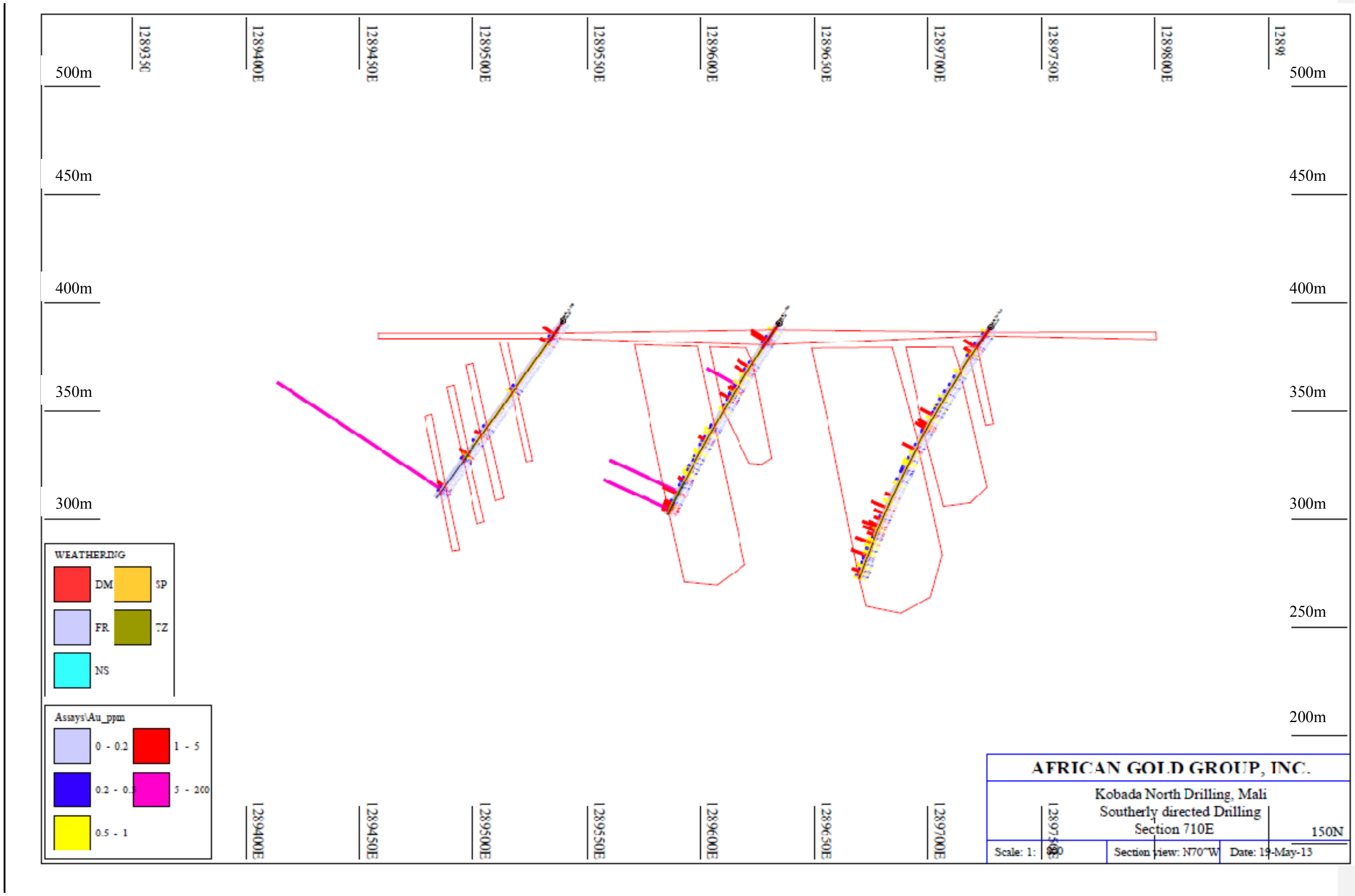


Figure 4.5: Cross section_710E, interpretation of ore zones with drill hole intercepts

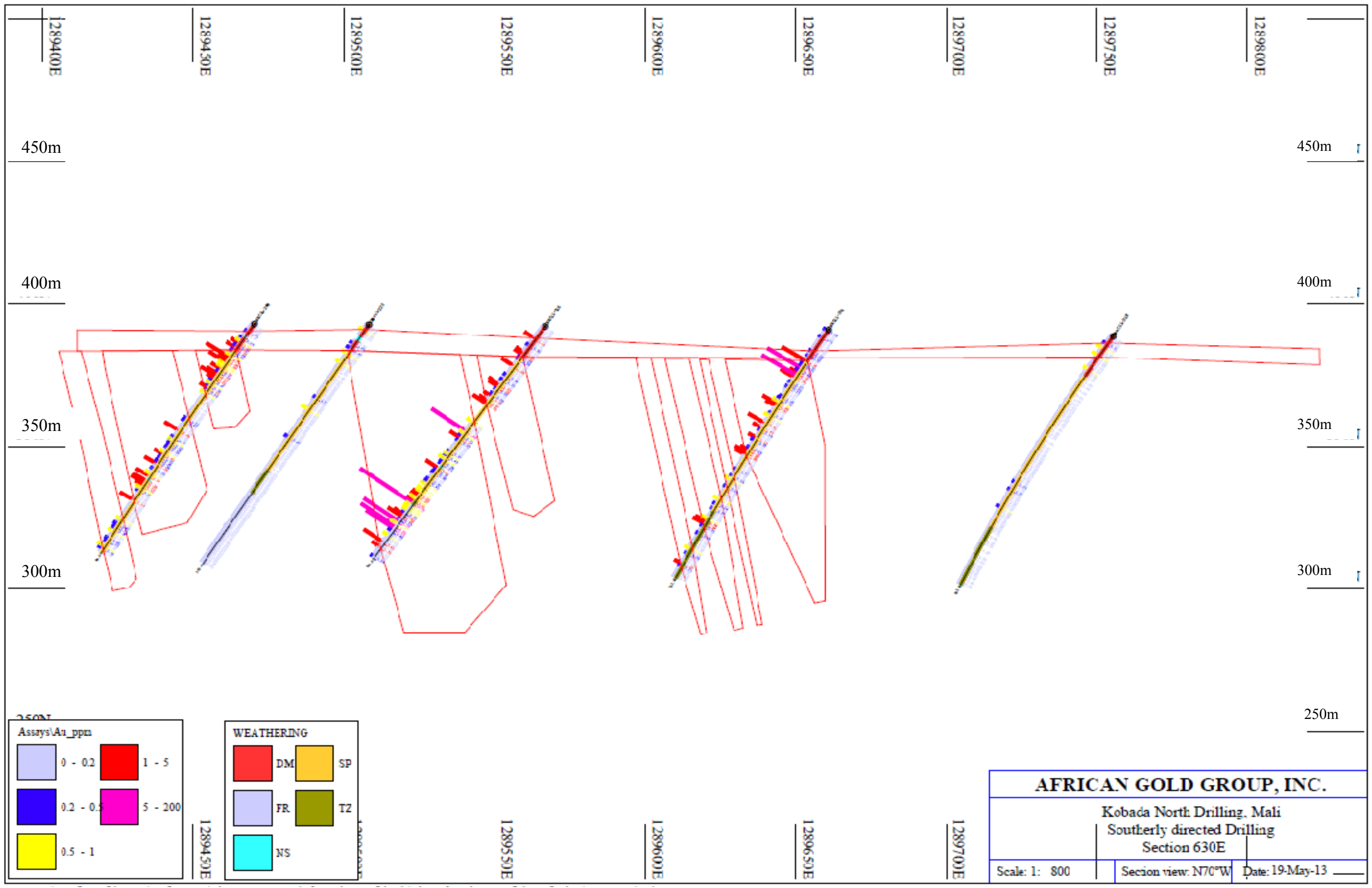
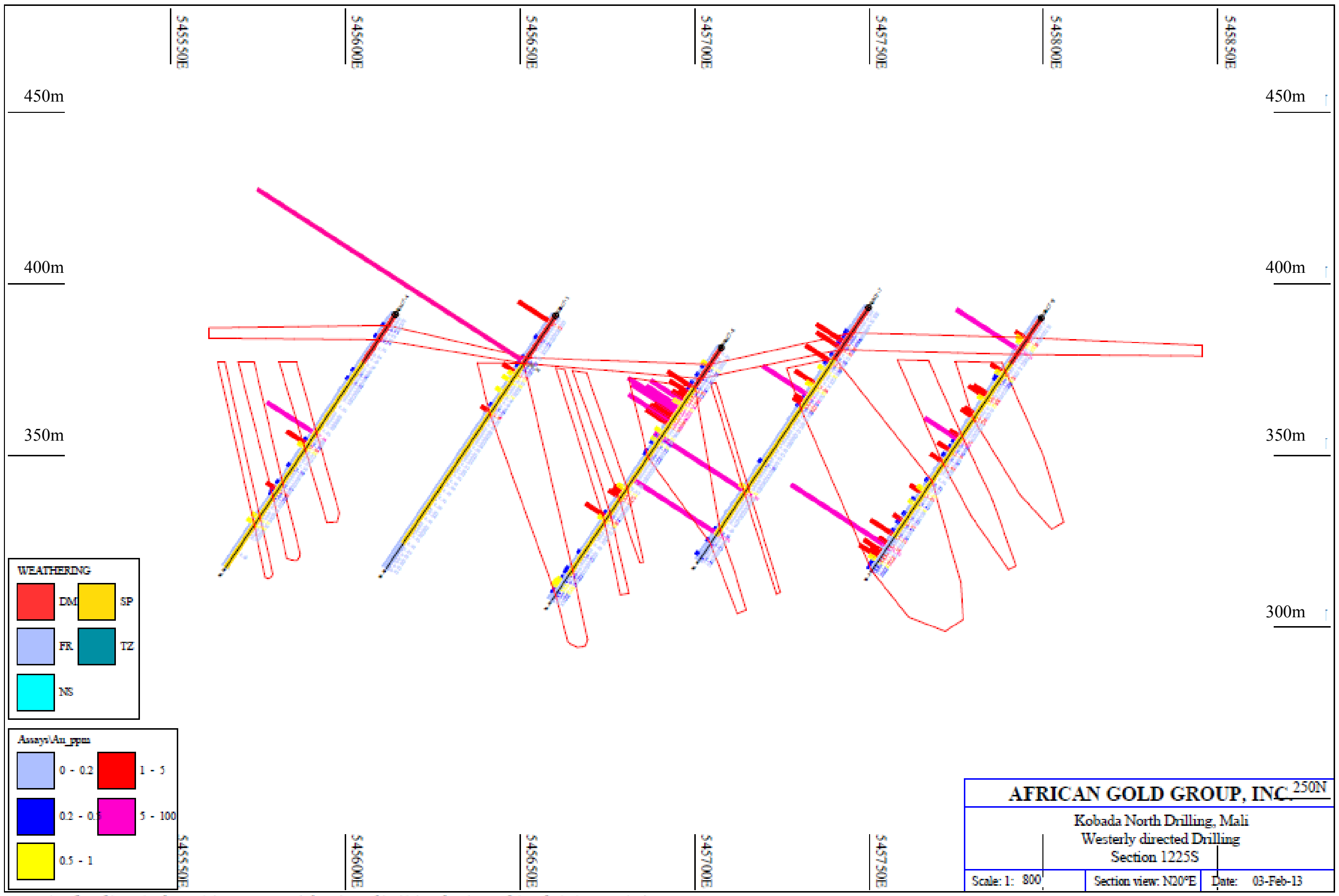


Figure 4.6: Cross section_630E, interpretation of ore zones with drill hole intercepts



Westerly directed section 1225S (showing digitized mineralised intercepts)

Figure 4.7: Cross section 1225S interpretation of ore zones with drill hole intercept

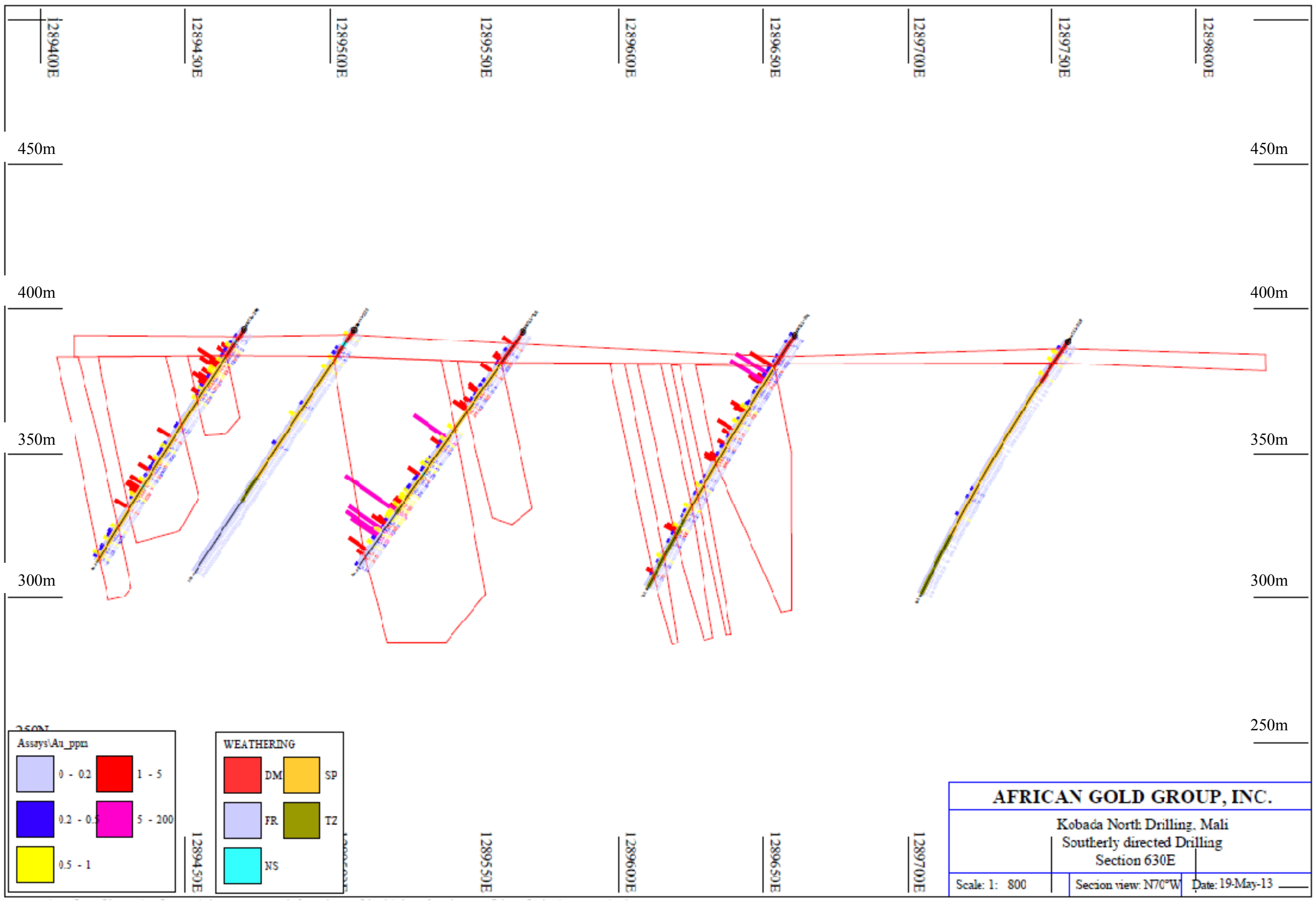
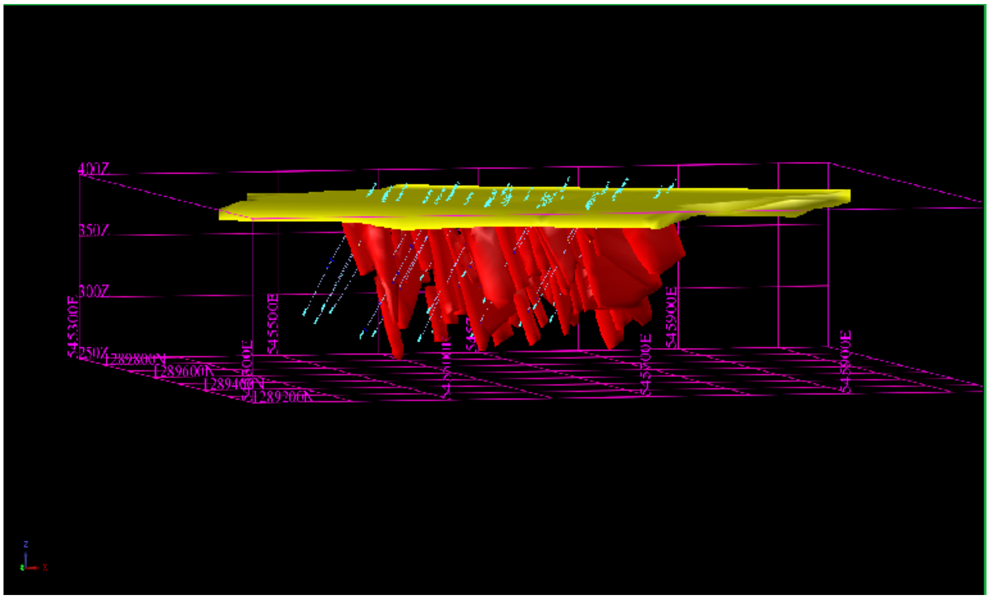


Figure 4.8: Cross section1380S interpretation of ore zones with drill hole intercepts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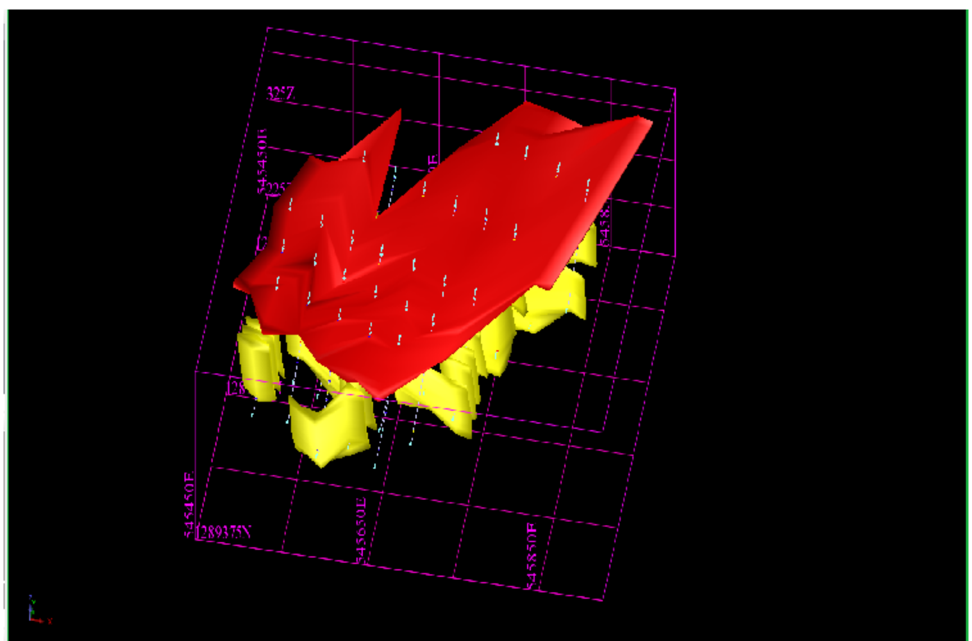
1289200N


325Z
545450E
1289375N
545650E
545850E

4.3.1 Data Distribution

The data distribution for the westerly directed drilling is nominally 50m X 50 m drilling pattern except for the extreme north which is sparsely drilled on 100 m X 50 m grid as presented in **Figure 4.3**. The data distribution for the southerly directed holes is 40 m X 50m in a staggered drilling pattern (**Figure 4.2**). All the reverse circulation (RC) have been drilled at 1 meter intervals and have been drilled specifically to target the saprolite but there are instances where either the transition or fresh rock zone has been encountered depending on the depth and location where a particular hole has been drilled to their respective planned depths on the prospect.

4.3.2 Sample Composite

A 2 m sample composite length was adopted after consideration of the sampled interval lengths in the drill hole database and mineralization geometry. For subsequent analysis and estimation, the raw samples were extracted from the drill hole database using the mineralised ore zone wireframes constrained with the lower cutoff of 0.2 g/t Au and then composited to 2m intervals within these zones, with a minimum composite length of 0.5 m. Residual composite lengths resulted were retained in the analysis of the model to ensure no loss of data. This declustering method is to ensure the samples have equivalent effect on the samples statistics.

4.3.3 Statistical Analysis

Statistical analysis was undertaken to establish the difference between sample populations and to establish the suitability of the grade interpolation methodology to employ. Univariate analysis of the composite data, from within the mineralised wireframes, consisted of summary statistics, log-normal probability plots in combination of histogram plots for the data.

Top-cuts were selected based on the statistical analysis from the assay data for each of the domains. Table 4.3 and 4.4 respectively shows the summary statistics of the individual domains of the WDD and SDD. The high coefficient of variation emanating from the raw and composite data of the two drilling directions strongly suggests high degree of variability in the datasets. This suggests there is the need to top cut the data to reduce the variability. This is one of the basis that informed the need to top cut the data to their respective domains. As per the tables 4.2 and 4.3, the WDD with coefficient of variation 3.12 and 3.0 respectively for duricrust and saprolite domains seem more variable suggesting the presence more extreme values than in the SDD with coefficient of variation of 2.8 and 2.5 respectively for its duricrust and saprolite domains.

Table 4.3: Summary statistics for domains 1 and 2 composite data for WDD

	Westerly Directed Drilling					
	Duricrust (Domain 1)			Saprolite (Domain 2)		
	Raw data	2m Composite No top-cut	2m Composite top-cut @ 9g/t	Raw data	2m Composite No top-cut	2m Composite top-cut @ 5g/t
Number of samples	126	71	71	538	485	485
Minimum value	0.066	0.080	0.080	0.007	0.002	0.002
Maximum value	100.00	50.110	9.000	77.990	40.737	5.000
Mean	2.030	1.970	1.370	1.214	0.787	0.624
Median	0.290	0.340	0.340	0.354	0.230	0.230
Geometric Mean	0.430	0.580	0.570	0.378	0.194	0.191
Variance	85.740	37.690	5.040	15.485	5.583	1.082
Standard Deviation	9.260	6.140	2.250	3.935	2.363	1.040
Coefficient of variation	4.570	3.120	1.640	3.242	3.002	1.667

Table 4.4: Summary statistics for domains 1 and 2 composite data for SDD

	Southerly Directed Drilling					
	Duricrust (Domain 1)			Saprolite (Domain 2)		
	Raw data	2m Composite No top-cut	2m Composite top-cut @ 7g/t	Raw data	2m Composite No top-cut	2m Composite top-cut @ 10g/t
Number of samples	145	82	82	723	377	377
Minimum value	0.001	0.001	0.001	0.001	0.007	0.007
Maximum value	45.780	22.934	7.000	48.310	30.825	10.000
Mean	1.181	1.098	0.777	0.909	0.908	0.816
Median	0.245	0.295	0.295	0.304	0.394	0.394
Geometric Mean	0.295	0.366	0.356	0.279	0.397	0.395
Variance	21.790	9.479	1.818	8.733	5.244	2.025
Standard Deviation	4.668	3.079	1.348	2.955	2.290	1.423
Coefficient of variation	3.951	2.805	1.734	3.252	2.521	1.744

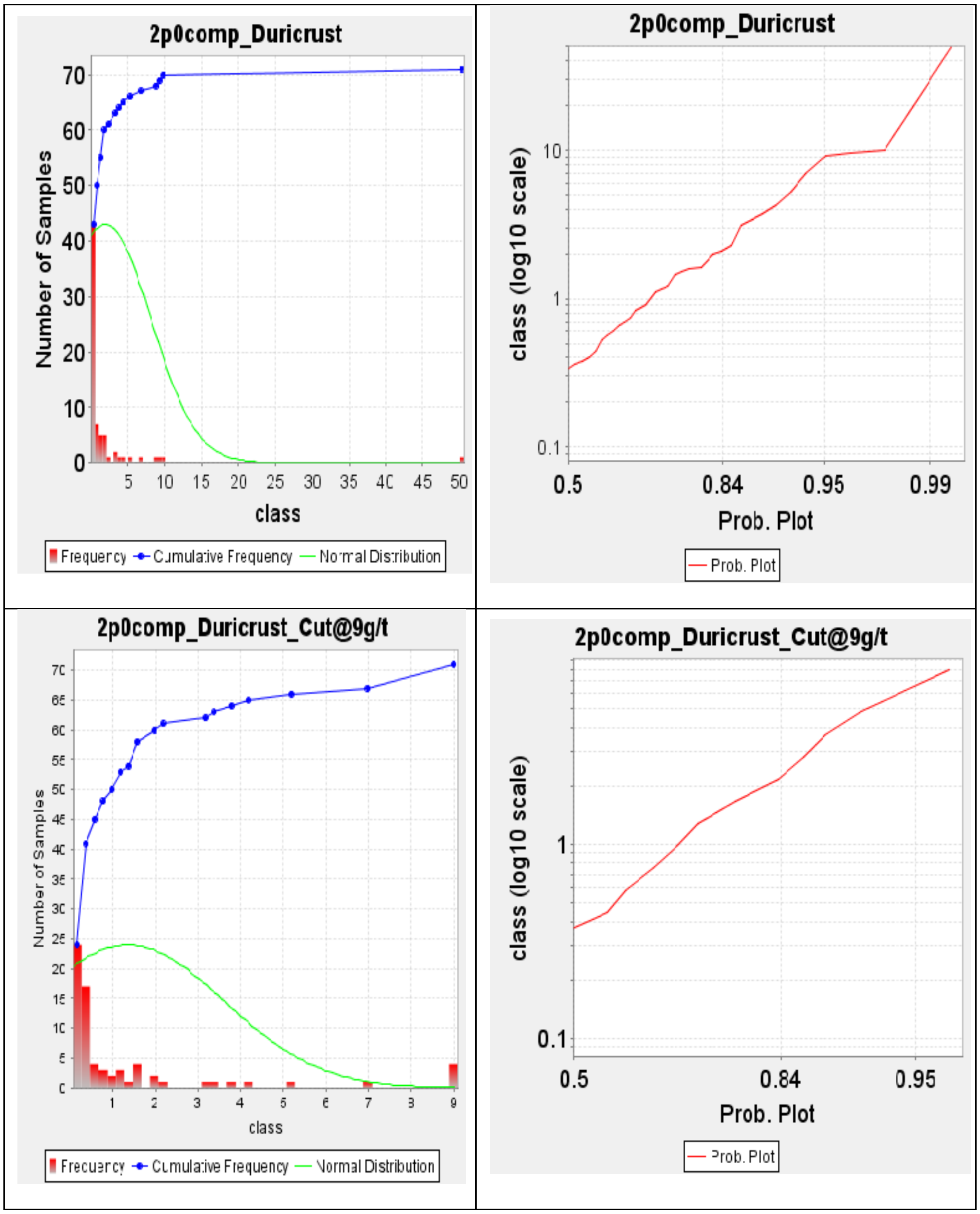


Figure 4.11a: Histogram and log-normal probability plots for the duricrust (WDD)

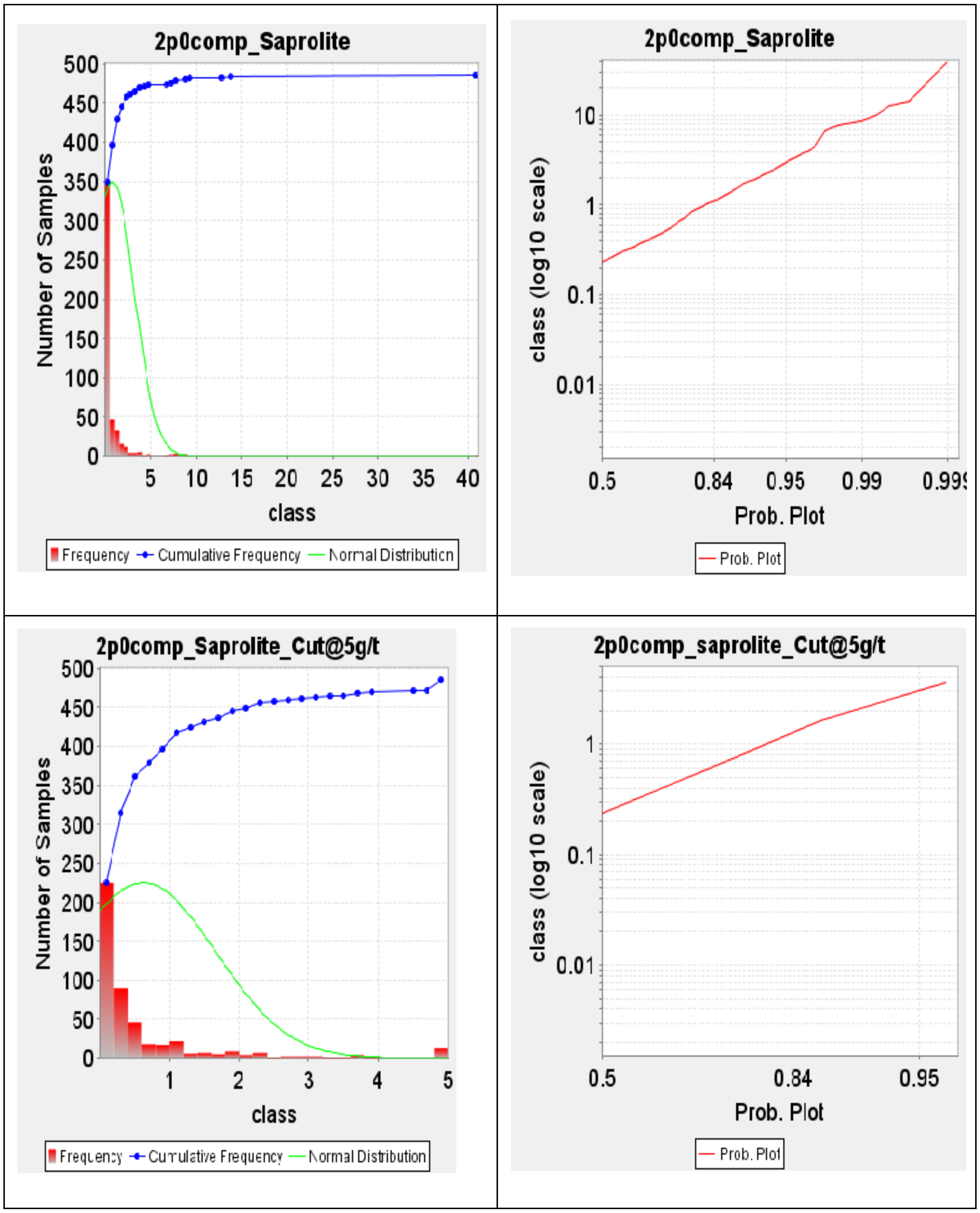


Figure 4.11b: Histogram and log-normal probability plots for the saprolite (WDD)

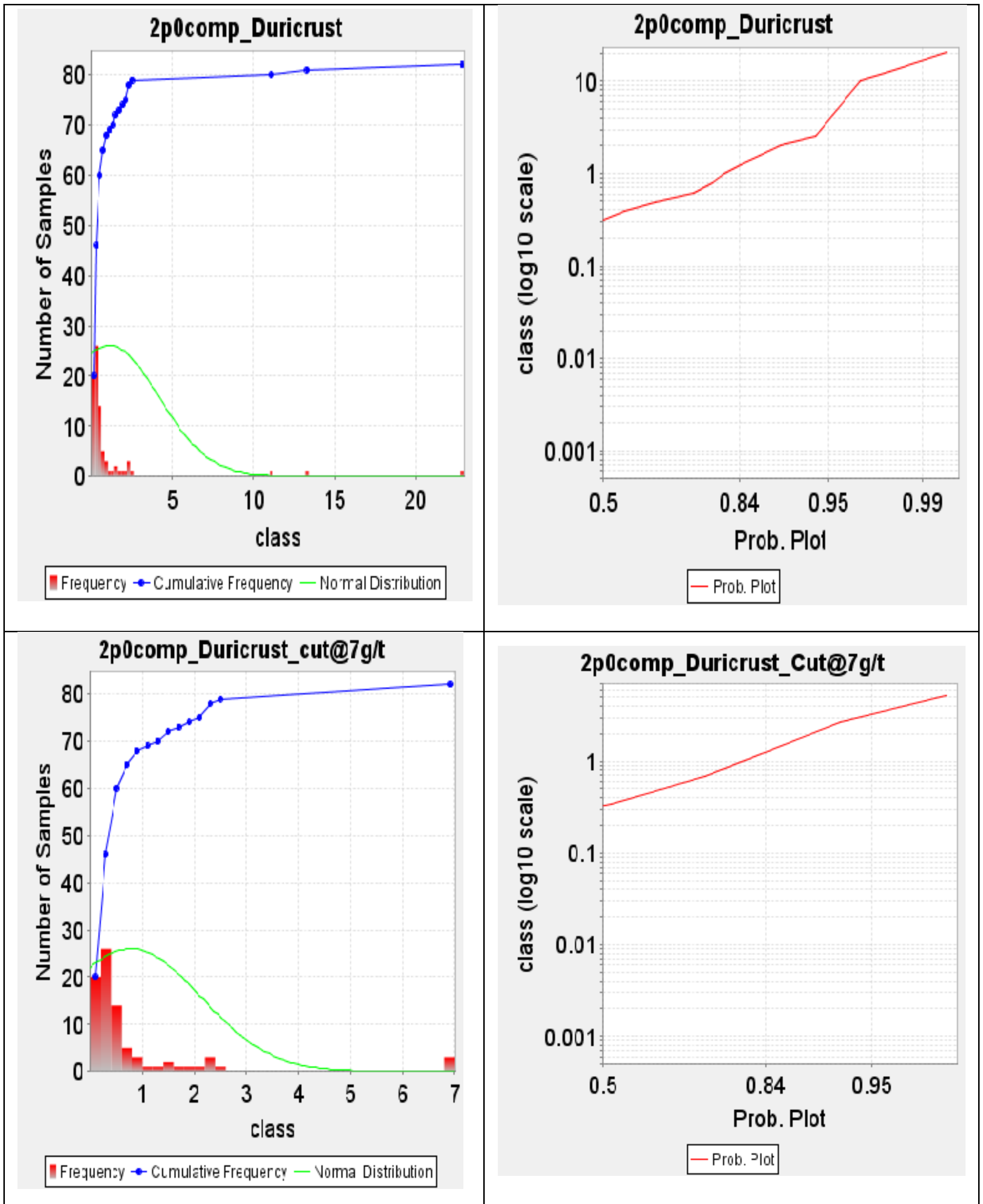


Figure 4.11c: Histogram and log-normal probability plots for Duricrust (SDD)

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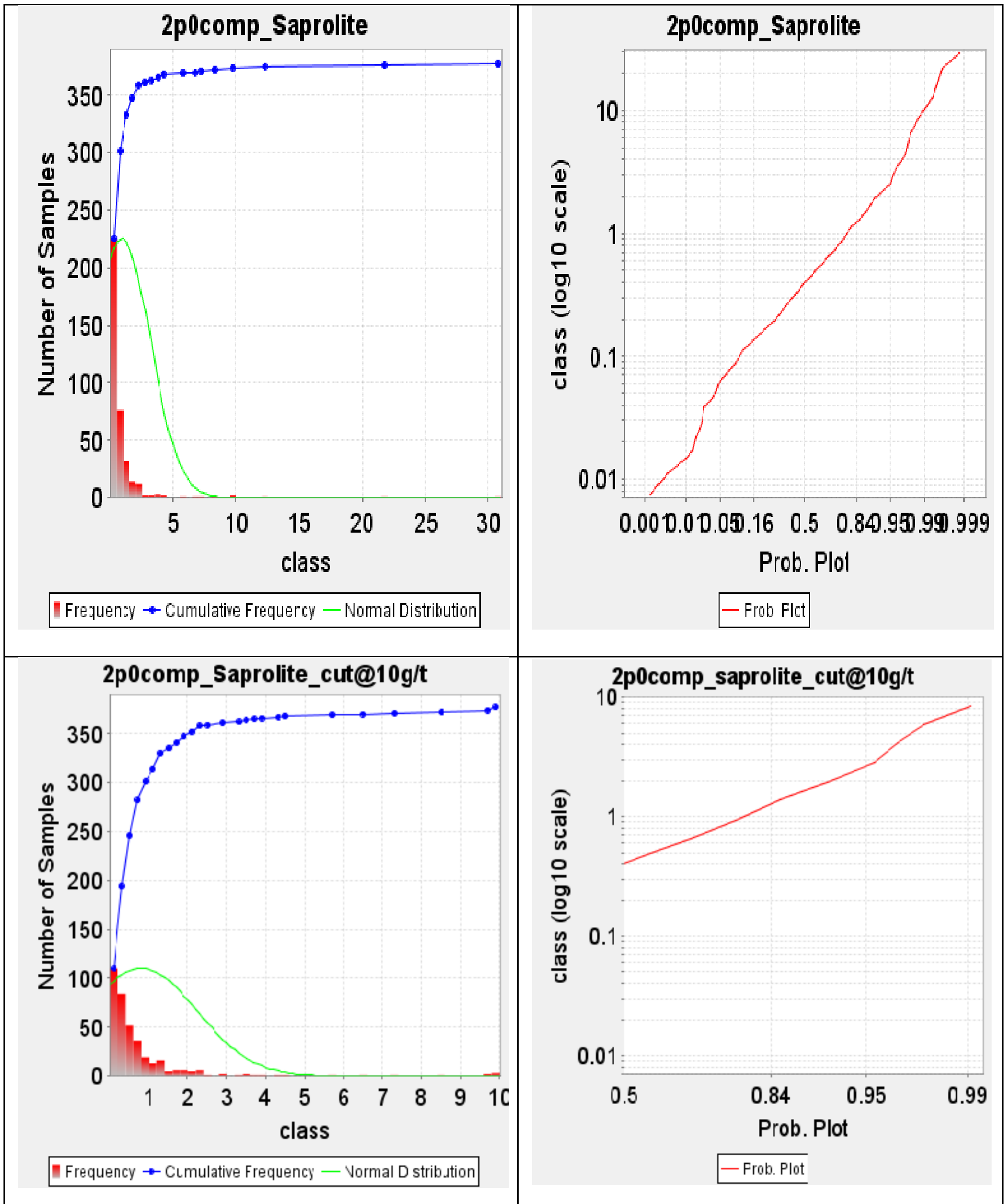


Figure 4.11d:Histogram and log-normal probability plots for Saprolite (SDD)

Histogram and log-normal probability plots for the duricrust and the Saprolite domains (Figures 4.11a through 4.11d) for the respective drilling directions are presented as examples of the population distributions and the respective top cuts that have been applied to the datasets.

With regards to data homogeneity and spatial modeling, variable top cuts have been applied to the different domains in the respective drilling directions based on the histogram plots. In the WDD, a top cut value of 9 g/t Au has been applied to the duricrust domain whereas 7 g/t Au has been assigned to the SDD. In the saprolite domain analysis, 5 g/t Au and 10 g/t Au are the top cuts that have been applied to the WDD and SDD datasets respectively. However, the estimation has both been presented using both the raw composite data and the cut data as a means of comparison.

4.3.4 Spatial Analysis

Variography was completed on the 2m composite for the duricrust and saprolite domains for each drilling direction. Separate variograms were modeled for the various domains in both downhole and directional manner. The downhole variograms generally have short primary ranges. The down dip directional data showed poor continuity in both drilling directions with most of the variance occurring for ranges <60m for the duricrust model.

In fitting the variograms, the spherical and exponential models were considered. The spherical model was selected due to its rigidity and the importance of the nugget variance to this estimation. In order to arrive this selection, various potential variogram models were constructed based on different parameters and after comparing the outcomes, the best-fit variogram which turned out to be the spherical model was selected for subsequent analysis.

The somewhat discontinuous nature of the data in terms of grade continuity made the modeling quite a difficult task. The westerly directed drilling model however is seen far more continuous compared to the southerly directed drilling. The southerly drilling look more discrete with some distinct high grade occurrences. Typical examples of variograms are shown in Figures 4.12 to 4.14 (appendix A1.1 to A1.4 shows more on variograms). Details of the parameters for the variogram models are presented in the Table 4.5 and 4.6.

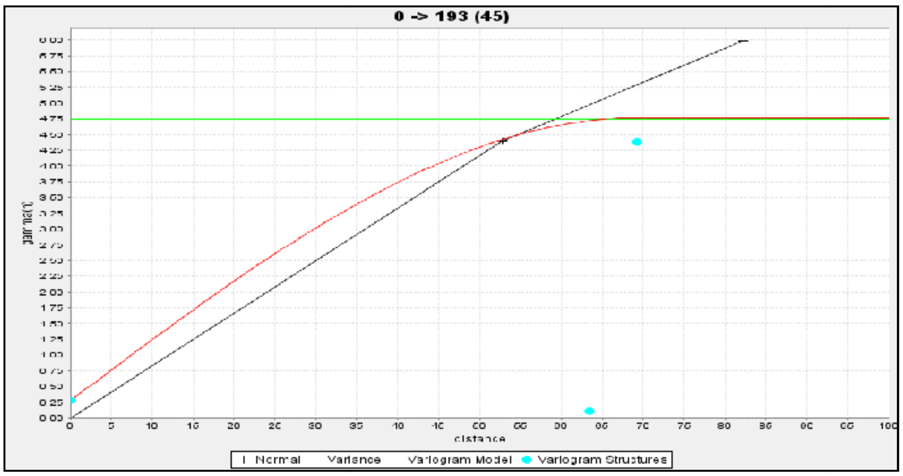


Figure 4.12: Directional Variograms for the WDD duricrust (domain_1)

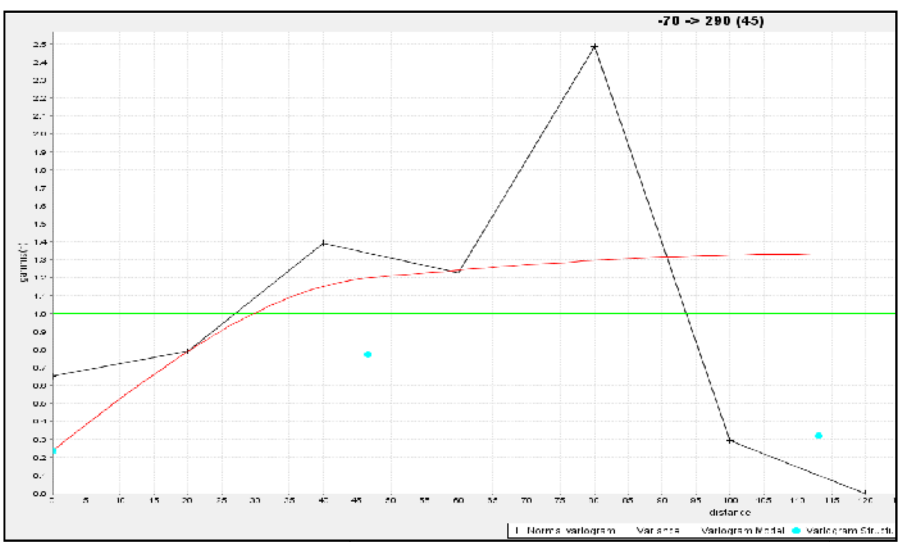
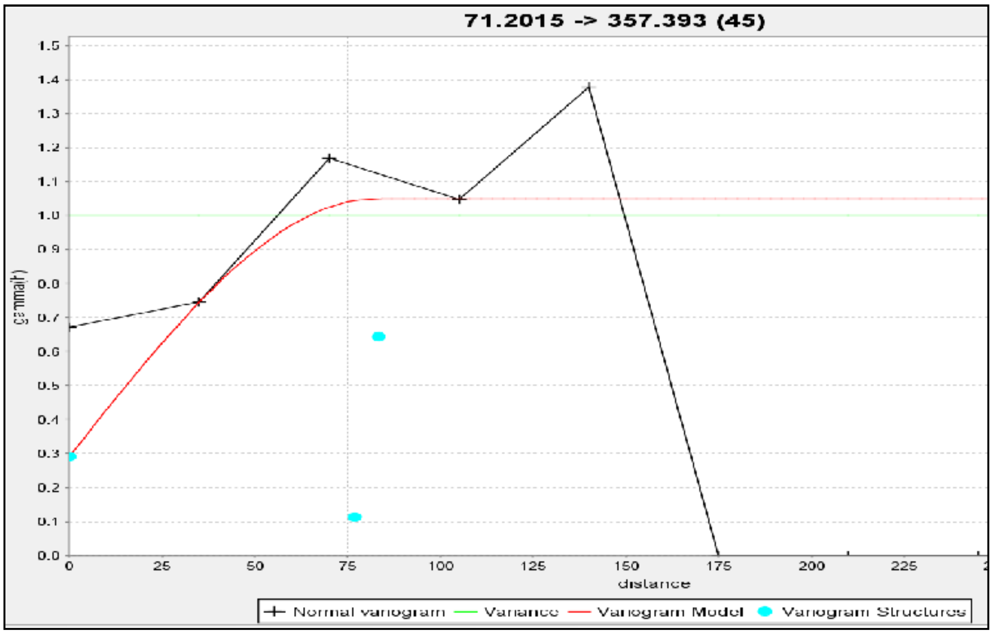


Figure 4.13: Directional Variograms for the WDD saprolite (domain_2)



Domains	Westerly Directed Drilling Nugget (C_O)	Rotation (dip/dip drx) Major	Semi	Minor	Structure 1 Sill1 (C_1)	Range (m) Major	Semi	Minor	Structure 2 Sill1 (C_2)	Range (m) Major	Semi	Minor
Grade Variography Au(g/t)												
Duricrust (Dom 1)	2.5	0/30	-10/120	-80/300	4.088	180	95	25	0	0	0	0
Saprolite (Dom 2)	0.274	12/24	-65/88	-22/299	0.857	105	48	23	0	0	0	0

Table 4.6: Variogram Model Parameters for SDD

Domains	Nugget (C_O)	Rotation (dip/dip drx)			Structure 1				Structure 2			
					Sill1	Range (m)			Sill1	Range (m)		
		Major	Semi	Minor	(C_1)	Major	Semi	Minor	(C_2)	Major	Semi	Minor
Grade Variography Au(g/t)												
Duricrust (Dom 1)	0.209	-4/150	-0/60	86/150	1.488	110	60	30	0	0	0	0
Saprolite (Dom 2)	0.1	-19/106	-68/315	9/20	4.719	160	60	55	0	0	0	0

4.3.5 Geological Model

No geological model was readily deduced considering the limited geological data and the logging variations encountered from one drilling campaign to another. However, the respective domains were coded from the grade shell solid (geostatistical domains) generated from the duricrust and saprolite zones for assay and composite data.

4.3.6 Grade Estimation

The parameters used in the generation of the block model for the respective datasets have been presented in Tables 4.7 and 4.8. Grade estimation was completed in multiple iterations using Ordinary Kriging (OK). Validation checks of the "OK" model were also conducted with other modeling algorithms such as Inverse Distance Weighting to the 2nd Power ("IDW") and Nearest Neighbour (NN) for each drilling direction. The model estimate was completed in three pass (Table 4.9) with expanding searches for each pass.

Table 4.7: Block Model Parameters for WDD

	East	North	Elevation
Minimum Coordinates	545425	1289350	50
Maximum Coordinates	545905	1289830	500
Parent Block Size(m)	10	20	10
Minimum Sub-Block Size(m)	1.25	2.5	1.25
Number of blocks (Parent)	48	24	45

Table 4.8: Block Model Parameters for SDD

	East	North	Elevation
Minimum Coordinates	545425	1289350	50
Maximum Coordinates	545925	1289850	500
Parent Block Size(m)	10	20	10
Minimum Sub-Block Size(m)	1.25	2.5	1.25
Number of blocks (Parent)	50	25	45

Table 4.9: Sample Search parameters for the WDD and SDD

	WDD		SDD	
	Duricrust (Dom 1)	Saprolite (Dom 2)	Duricrust (Dom 1)	Saprolite (Do
Pass 1 (along strike x down dip x across strike)	60m x 32m x 8m	60m x 25m x 15m	50m x 32m x 8m	50m x 25m 15m
Pass 2 (along strike x down dip x across strike)	75m x 48m x 12m	75m x 37.5m x 22.5m	60 x 48m x 12m	60m x 37.5m 22.5m
Pass 3 (along strike x down dip x across strike)	120m x 48m x 12m	120m x 50m x 30m	75m x 48m x 12m	75m x 50m 30m
Discretisation (x/y/z)	3/3/3	3/3/3	3/3/3	3/3/3
Min samples (search 1/2/3)	6/4/2	6/4/2	6/4/2	6/4/2
Max samples (search 1/2/3)	12/20/20	12/20/20	12/20/20	12/20/20

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4.3.7 Specific Gravity Measurement

No density data was readily available for the specific gravity determination. Specific gravities in Table 4.10 below were applied to all the resource estimation based on experience in similar terrains within the Birimian setup such as has been used by Sadiola Gold Mines in Mali, Robertson (2004) and Newmont Akyem mine in Ghana, AMEC (2004).

Table 4.10: Specific Gravity

Material Type	SG
Duricrust	1.70
Saprolite	1.73
Fresh	2.8

4.3.8 Model Validation

For model validation, Nearest Neighbor (NN) and Inverse Distance Weights (IDW) model which utilized the same search criteria as the OK estimate was completed. The NN and IDW estimate were used for comparison of summary statistics in the block model and in the swath plots for the OK. Model Validation consisted of visual inspection of cross-sections and plan-sections.

A model validation process included the visual examination of block model versus composites, and the building up of a model grade profile (swath plots), to compare average grades on vertical slices, as derived from the composites directly as well as from interpolated model grades.

Models are checked to see if they have been efficiently Kriged using model files created with efficiencies. Charts are prepared examining the portion of the model that has been efficiently Kriged. Where a large portion of the model fails to pass, the interpolation parameters will be

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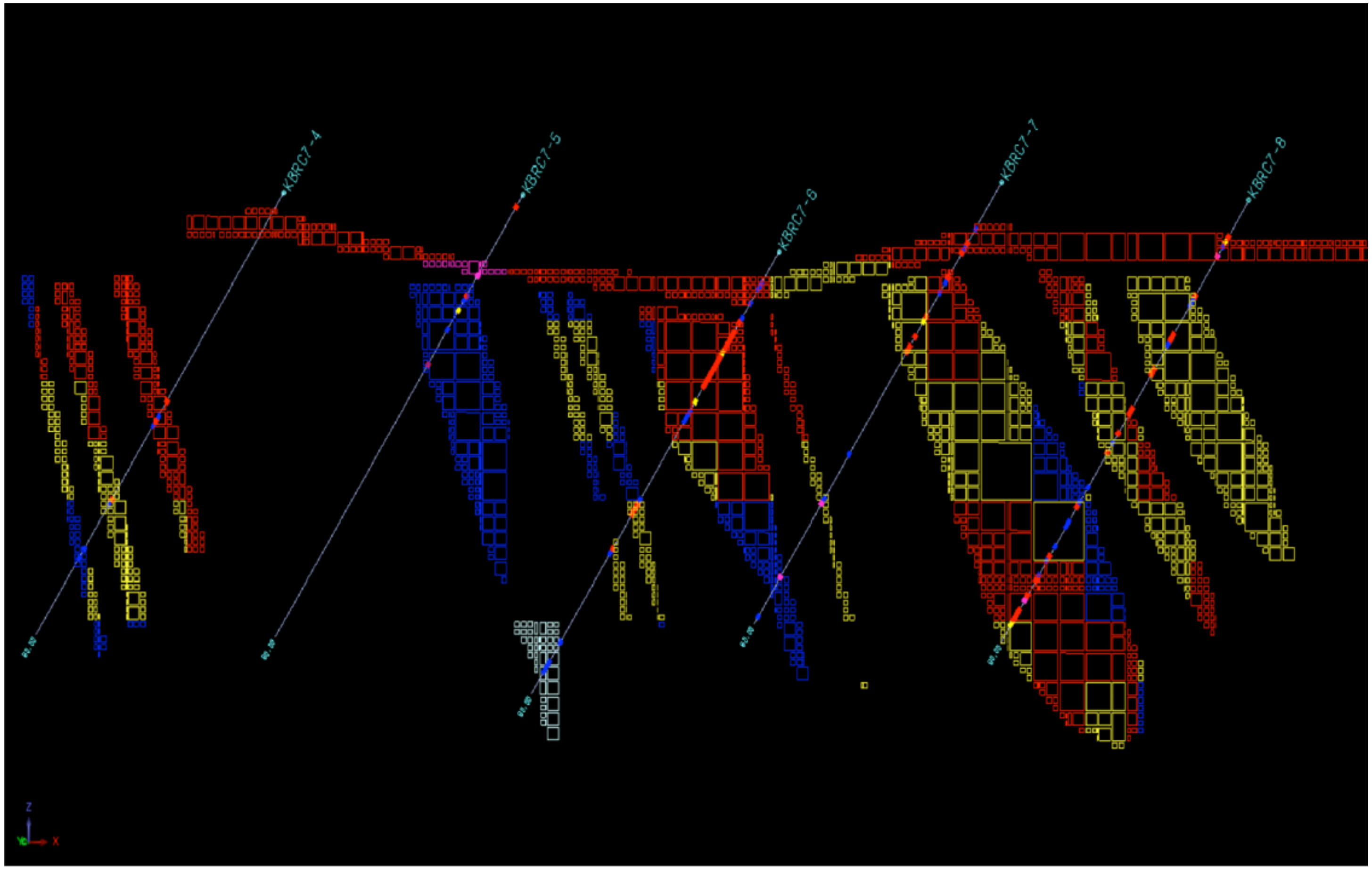
4.3.8.1 Visual Comparison

Visual examination of the estimated block model and gold grades in cross section and level plan by comparing interpolated grade with the supporting data has been used. This involves sectioning the estimated block model and comparing the vertical sections with the raw drill hole composite grades. On screen visual comparison of drill hole and block model grades reveal that the block model generally honours the drill hole composite grades both in the WDD and in the SDD. Figures 4.15 through to 4.18 are respective sections of estimated block model and composite gold grades in vertical cross sections of the respective drilling directions. The rest of the sections used in this model have been presented in the appendices.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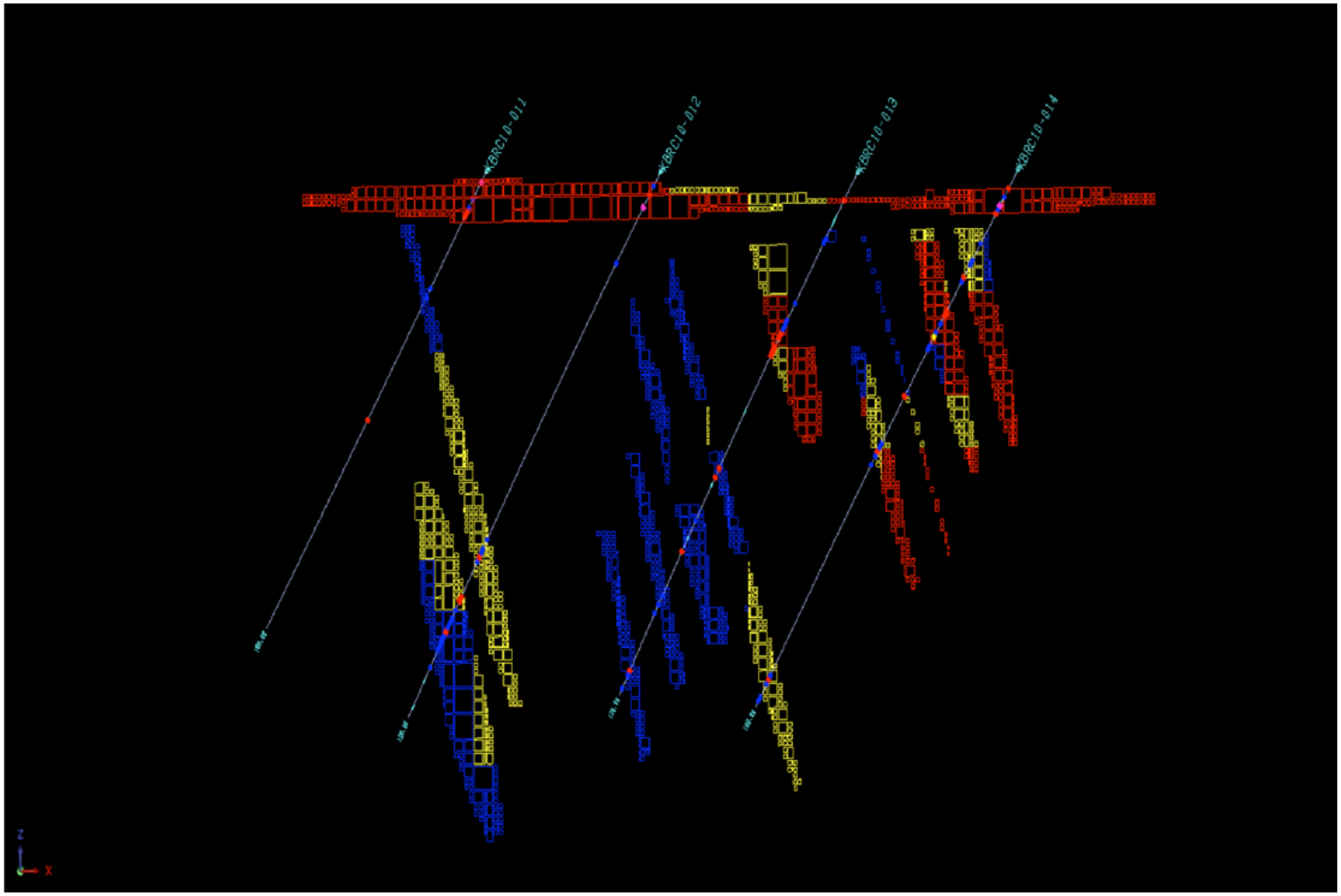

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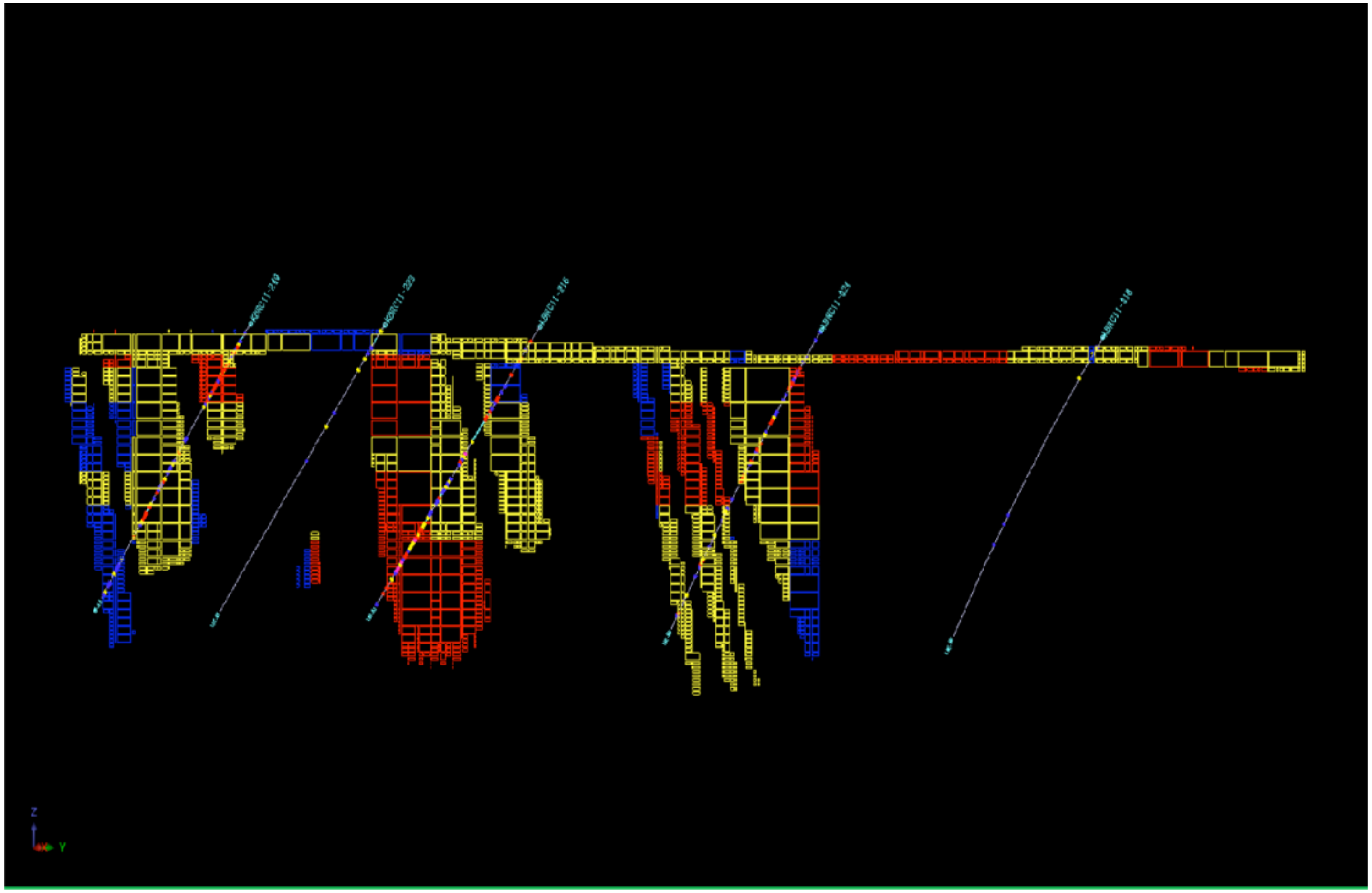

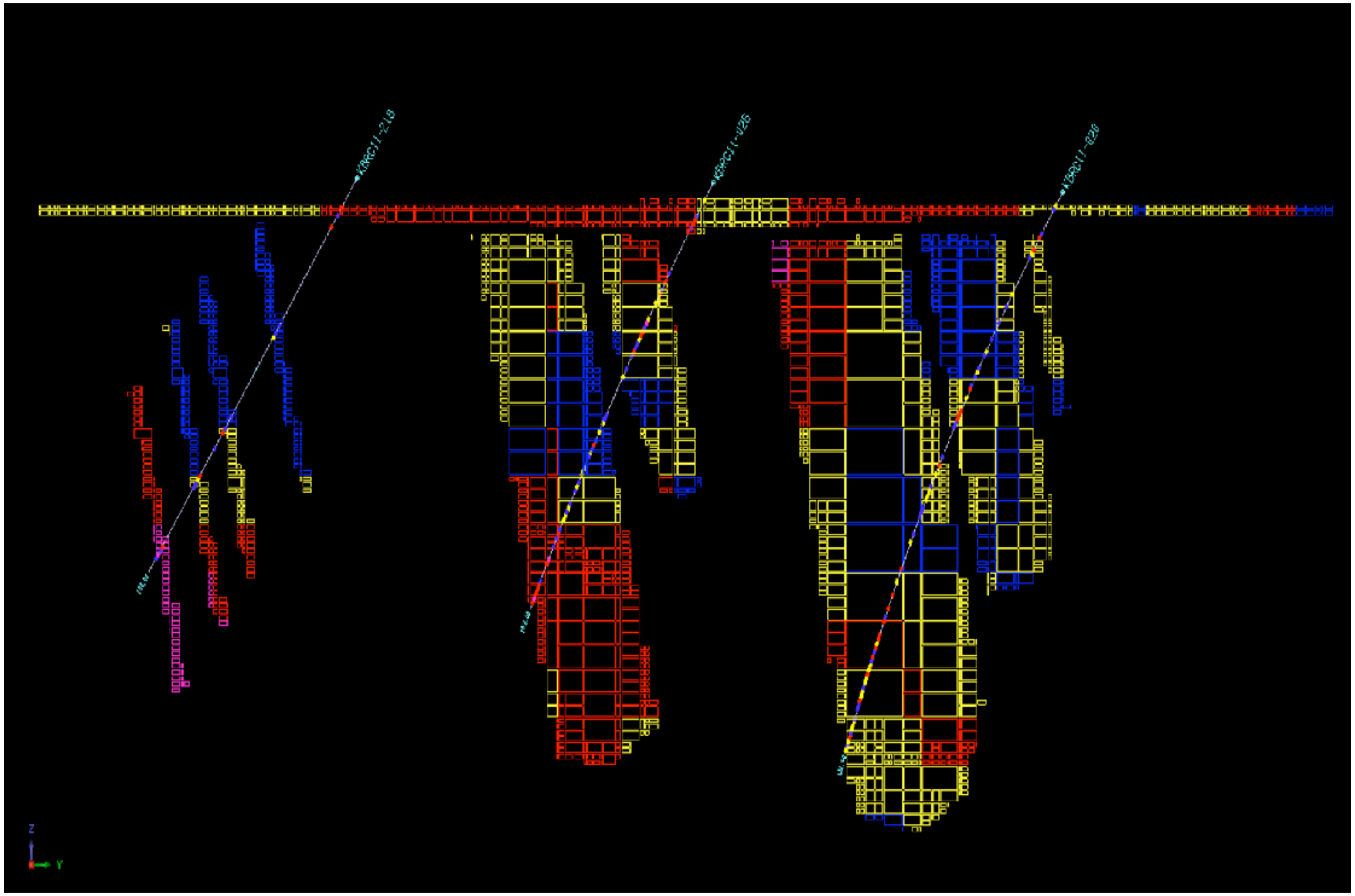
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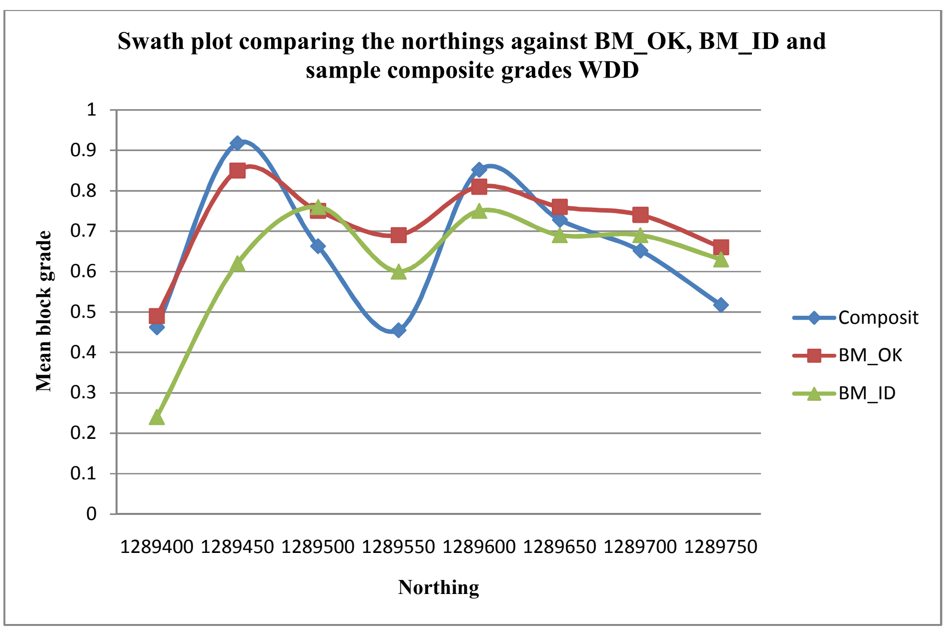
Swath plot comparing the northings against BM_OK, BM_ID and sample composite grades WDD
Mean block grade
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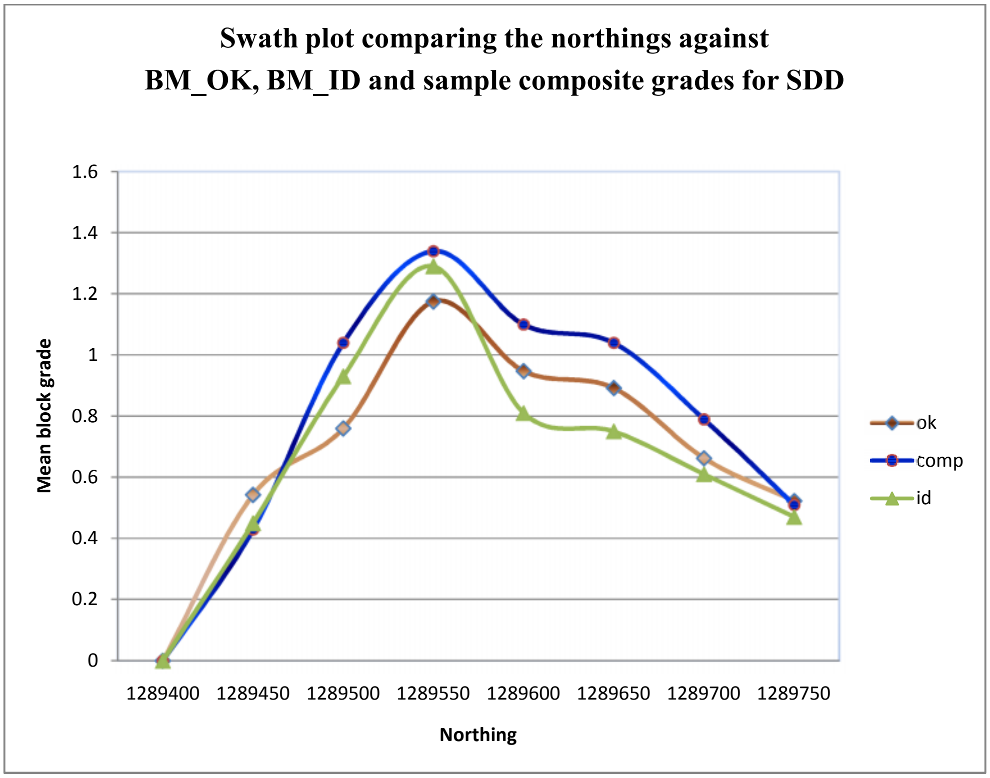
Swath plot comparing the northings against
BM_OK, BM_ID and sample composite grades for SDD
Mean block grade
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1.4
1.2
1
0.8
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0.4
0.2
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1289450
1289500
1289550
1289600
1289650
1289700
1289750
Northing
ok
comp
id

Table 4.11: Model estimates comparison for ordinary kriging between westerly and southerly directed drilling (topcut applied)

West						South					
Material Type	Dom_tc	COG	Tonne(t)	Grade(g/t)	Oz	Material Type	Dom_tc	COG	Tonne(t)	Grade(g/t)	Oz
Duricrust		0.2	523,105	1.66	**27,918**	Duricrust		0.2	748,584	0.86	**20,698**
	9	0.5	452,666	1.86	**27,069**		7	0.5	516,255	1.08	**17,926**
		1.0	362,121	2.14	**24,915**			1.0	229,711	1.58	**11,669**
Saprolite		0.2	1,777,737	0.93	**53,154**	Saprolite		0.2	1,489,172	1.01	**48,356**
	5	0.5	1,171,656	1.23	**46,333**		10	0.5	890,572	1.44	**41,230**
		1.0	505,829	1.92	**31,224**			1.0	278,030	3.13	**27,978**
Fresh_Rock		0.2	1,057,361	0.95	**32,295**	Fresh_Rock		0.2	529,134	1.59	**27,049**
	5	0.5	672,783	1.30	**28,119**		10	0.5	341,370	2.24	**24,584**
		1.0	346,405	1.84	**20,492**			1.0	200,517	3.34	**21,532**
TOTALS (All doms)		0.2	3,358,203	1.05	**113,367**	**TOTALS (All doms)**		0.2	2,766,890	1.08	**96,103**
		0.5	2,297,105	1.37	**101,522**			0.5	1,748,197	1.49	**83,740**
		1.0	1,214,355	1.96	**76,631**			1.0	708,258	2.69	**61,179**

Table 4.12: Model estimates comparison for ordinary kriging between westerly and southerly directed drilling (uncut raw data input)

Material Type	COG	Tonne(t)	Grade(g/t)	Oz	Material Type	COG	Tonne(t)	Grade(g/t)	Oz
Duricrust	0.2	523,139	3.28	**55,167**	Duricrust	0.2	746,904	1.16	**27,855**
	0.5	452,870	3.74	**54,454**		0.5	530,938	1.48	**25,263**
	1.0	362,190	4.48	**52,168**		1.0	297,141	2.08	**19,871**
Saprolite	0.2	1,888,173	0.99	**60,098**	Saprolite	0.2	1,489,172	1.01	**48,356**
	0.5	1,388,149	1.23	**54,894**		0.5	890,572	1.44	**41,230**
	1.0	556,560	1.98	**35,429**		1.0	278,030	3.13	**27,978**
Fresh_Rock	0.2	1,068,022	1.07	**36,741**	Fresh_Rock	0.2	529,134	1.59	**27,049**
	0.5	771,606	1.34	**33,242**		0.5	341,370	2.24	**24,584**
	1.0	363,561	2.03	**23,728**		1.0	200,517	3.34	**21,532**
TOTALS	0.2	3,479,334	1.36	**152,006**	**TOTALS**	0.2	2,765,210	1.16	**103,260**
	0.5	2,612,625	1.70	**142,590**		0.5	1,762,880	1.61	**91,078**
	1.0	1,282,311	2.70	**111,325**		1.0	775,688	2.78	**69,381**

Table 4.13: Model estimates comparison for ordinary inverse distance weight (idw) between westerly and southerly directed drilling (top cut applied)

West						South					
Material Type	Dom_tc	COG	Tonne(t)	Grade(g/t)	Oz	Material Type	Dom_tc	COG	Tonne(t)	Grade(g/t)	Oz
Duricrust		0.2	531,732	1.53	**26,156**	Duricrust		0.20	699,643	0.95	21,369
	9	0.5	433,067	1.8	**25,062**		7	0.50	508,818	1.17	19,140
		1.0	329,019	2.14	**22,637**			1.00	271,987	1.60	13,991
Saprolite		0.2	1,834,773	0.79	**46,601**	Saprolite		0.20	1,907,156	0.72	44,147
	5	0.5	2,421,857	1.16	**90,322**		10	0.50	1,233,044	0.89	35,282
		1.0	407,341	1.54	**20,168**			1.00	295,553	1.54	14,633
Fresh_Rock		0.20	1,037,351	0.84	**28,015**	Fresh_Rock		0.20	731,850	1.03	24,235
	5	0.50	723,234	1.05	**24,415**		10	0.50	492,942	1.34	21,237
		1.00	259,426	1.61	**13,428**			1.00	219,177	2.22	15,643
TOTALS		0.2	3,403,856	0.92	**100,772**	**TOTALS**		0.2	3,338,649	0.84	**89,752**
		0.5	3,578,158	1.22	**139,798**			0.5	2,234,804	1.05	**75,658**
		1.0	995,786	1.76	**56,234**			1.0	786,717	1.75	**44,268**

Dom_tc -Domain topcut

COG - Cut off grade

Oz - Ounces (1 oz Au = 31.10385 g/t)

In the duricrust/mottle zone domain, the WDD looked more impressive as opposed to the SDD with regards to both grade and tonnages at the various cut-offs within the domain at 9g/t Au and 7g/t Au top-cut respectively, producing more gold ounces. Specific grades top cuts have been selected with regards to the statistics that were run on the generated samples within the assigned domains. As mentioned earlier, histograms resulting from these statistical presentations amongst other factors were used in selecting desirable top cuts for the respective domains.

In the saprolite domain, the WDD actually reported more tonnage than contained grade with respect to the SDD at the respective cut offs of 5g/t Au and 10g/t respectively. However, the highly mineralised tons of ore made it possible for the WDD to report more contained ounces than present in the SDD (Table 4.11)

The same scenario is repeated in the transition/fresh rock domain where the grade favours the SDD but the high tonnage material within the WDD present more tons of ore and subsequently much more ounces than there is in the SDD. However, it could also be that the westerly directed drilling overestimate the tonnage and grade, core drilling in "Zone 1" shows many low angle quartz veins in fresh core. This may be more so in the sulphides where the absence of weathering does not redistribute the gold in the wall rock.

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CHAPTER FIVE

CONCLUSIONS AND RECOMMENDATIONS

5.1 CONCLUSIONS

The primary objective of this research was to use Geostatistical methods and tools to evaluate the resource potential of southerly directed drilling as opposed to westerly directed drilling of the Kobada Zone 1. It appears that the gold at Kobada was introduced during the hydrothermal events that introduced the quartz veins that are common throughout the area.

Grade shells were generated based on a 0.2 g/t Au cut off and variography for spatial interpretations were also done for the whole project area.

After the exercise, it has been found out that though there is the presence of E-W mineralized veins occurring within the N-S Kobada shear zone, mineralisation in its entirety is not exclusive to the E-W mineralised veins. It is therefore clear that in each drilling direction, there is the possibility of encountering sets of differently oriented mineralised veins; typical of areas of intense stockworks as a result of the hydrothermal fluid injection. It has also been established from the respective modelling that mineralization is more continuous in the westerly directed drilling than it is in the southerly directed drilling sense. Mineralization in the south direction as shown in the model presented seem to be more discrete in a manner appearing in many swamps and pods.

In the fresh sulphide core, there is a significant amount of low core angle quartz veins and veining. This could be that the westerly directed drilling may possibly be drilling down dip of the tension veins, or the continuity may be related to weathering, remobilising the gold into the wall rock away from the E-W tension gashes.

In determining the optimal drilling direction at the Kobada Project, grade continuity played a major role even though the small size of the data used in the modelling had a major constraint on the analysis, the outcome was never in doubt.

From the geostatistical resource estimation carried out in this project, the following conclusions have been duly deduced;

- Drilling to the west is more viable than drilling south

  The westerly directed drilling reported at least 15,000 ounces more gold than the southerly directed drilling, at the various respective grade cut offs.
- Mineralization is much more continuous in the westerly directed drilling than in the southerly directed drilling.

  Ground water flow may be parallel to the strike of the Kobada Zone and dissolved gold moved from the westerly tension gash to the wall rock as dissemination thus making the ENE-WSW direction more continuous.

5.2 RECOMMENDATIONS

Based on the conclusions the following recommendations are being proposed;

- The current drill spacing should be closed in to help define the continuity and structural controls of the deposit (this might also help bring the resource from the current inferred to possible indicated resources, with all factors and protocols observed).
- A geological model if developed would benefit the project in determining the mineralisation shape, style and the possible geological associations.

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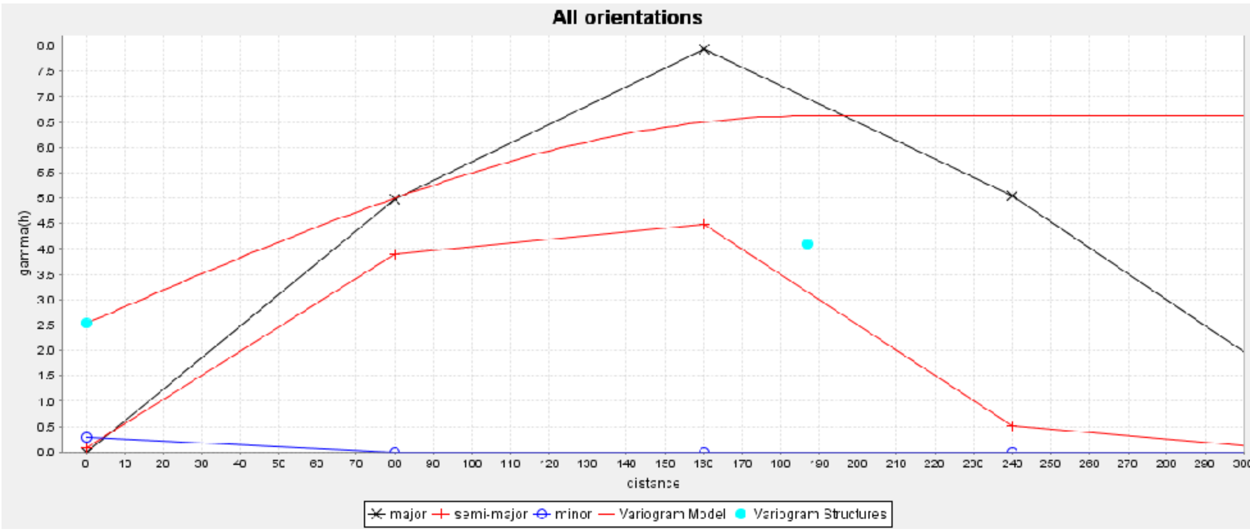
Important websites

http://www.isaaks.com
www.jorc.org
www.africangoldgroup.com
http://en.wikipedia.org/wiki/File:Mali_Map.jpg
www.gemcomsoftware.com

APPENDIX A

A1.1: WDD Variogram Model for Duricrust Domain (all directions)



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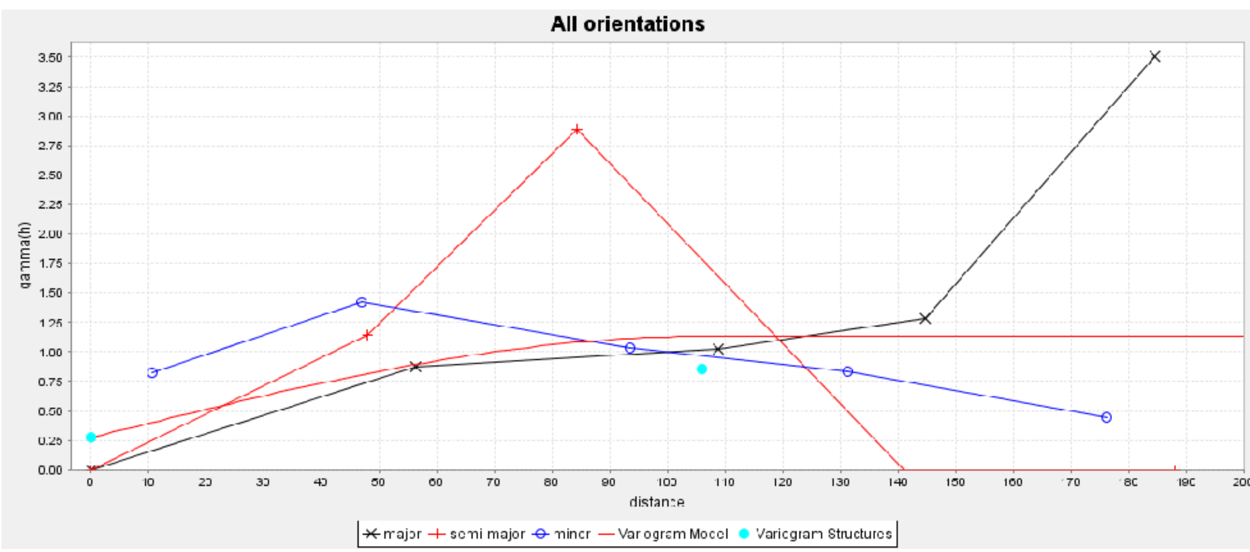

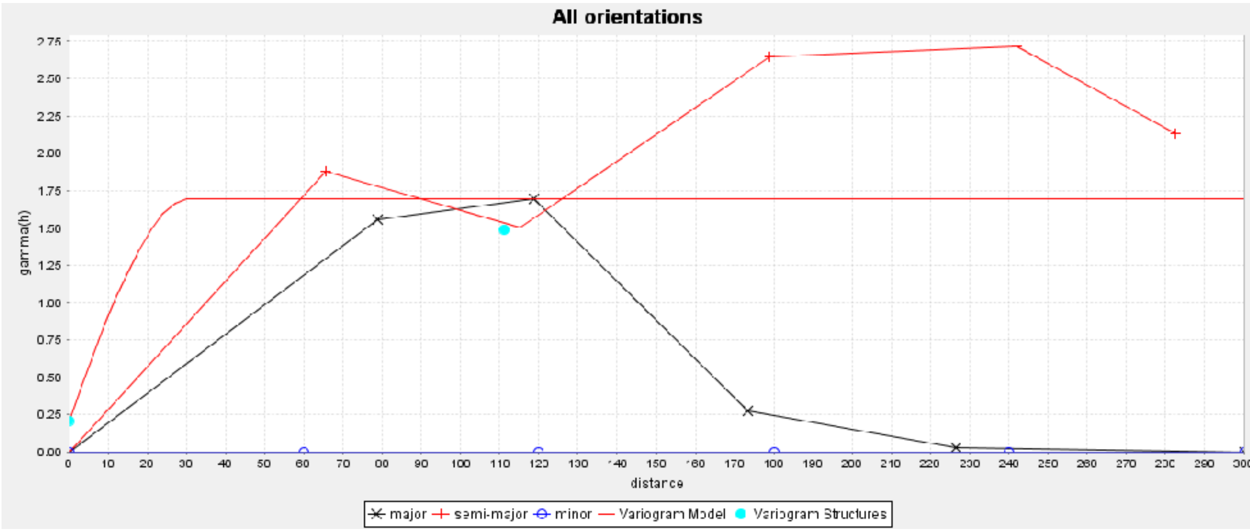
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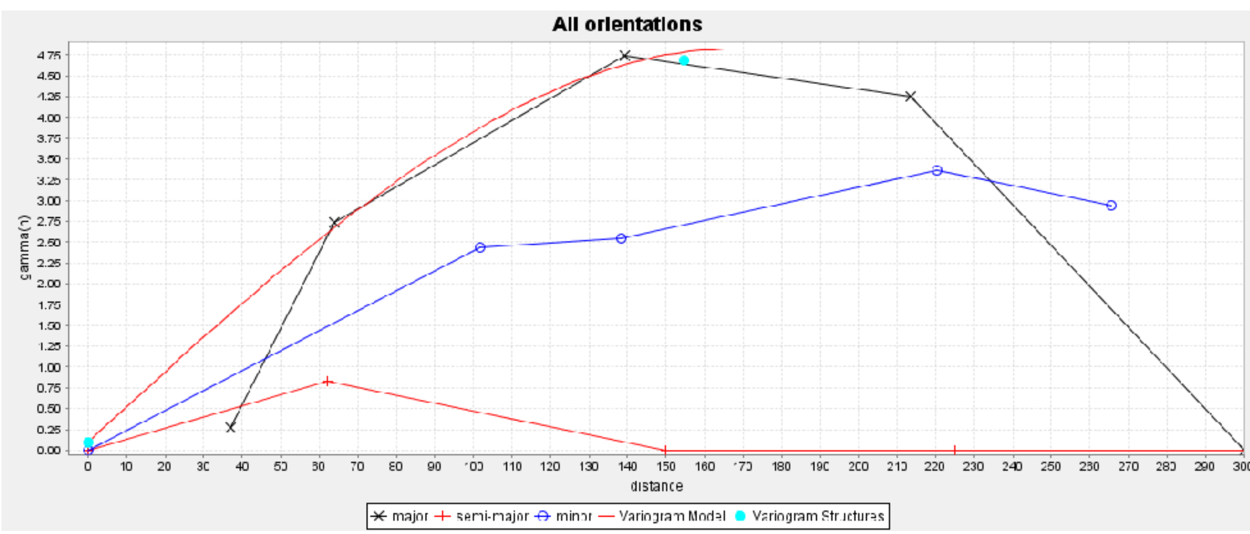
Variogram Structures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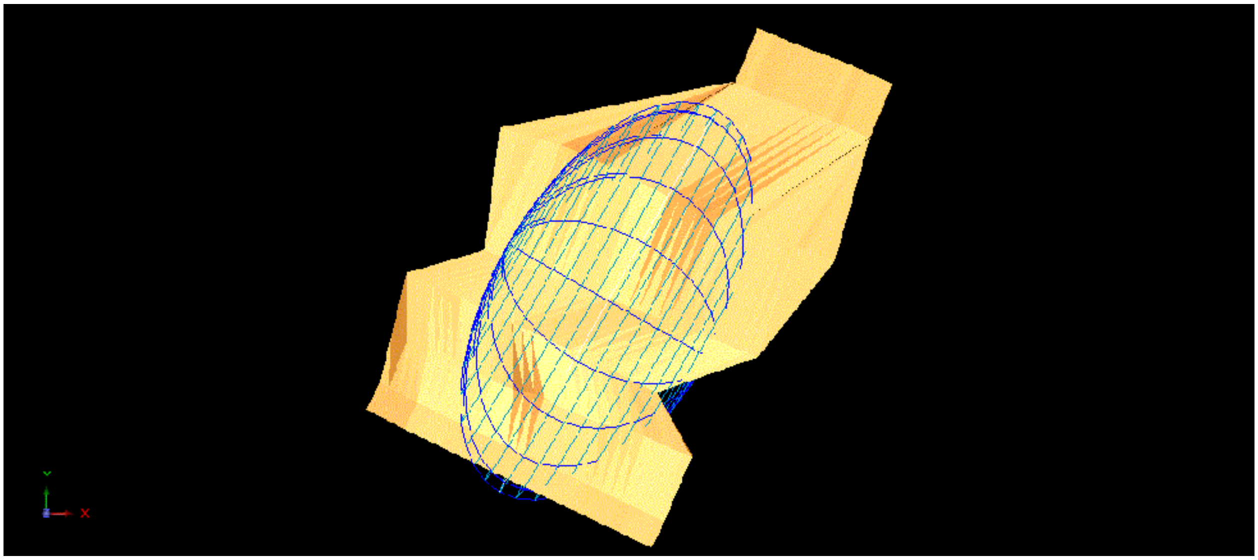

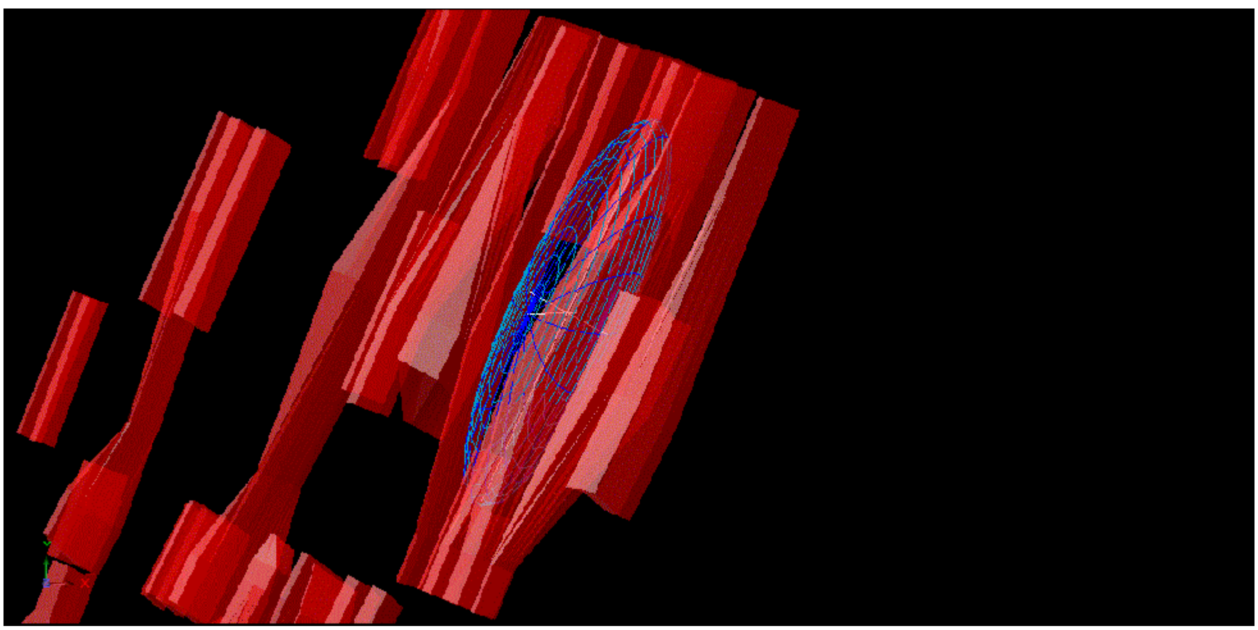

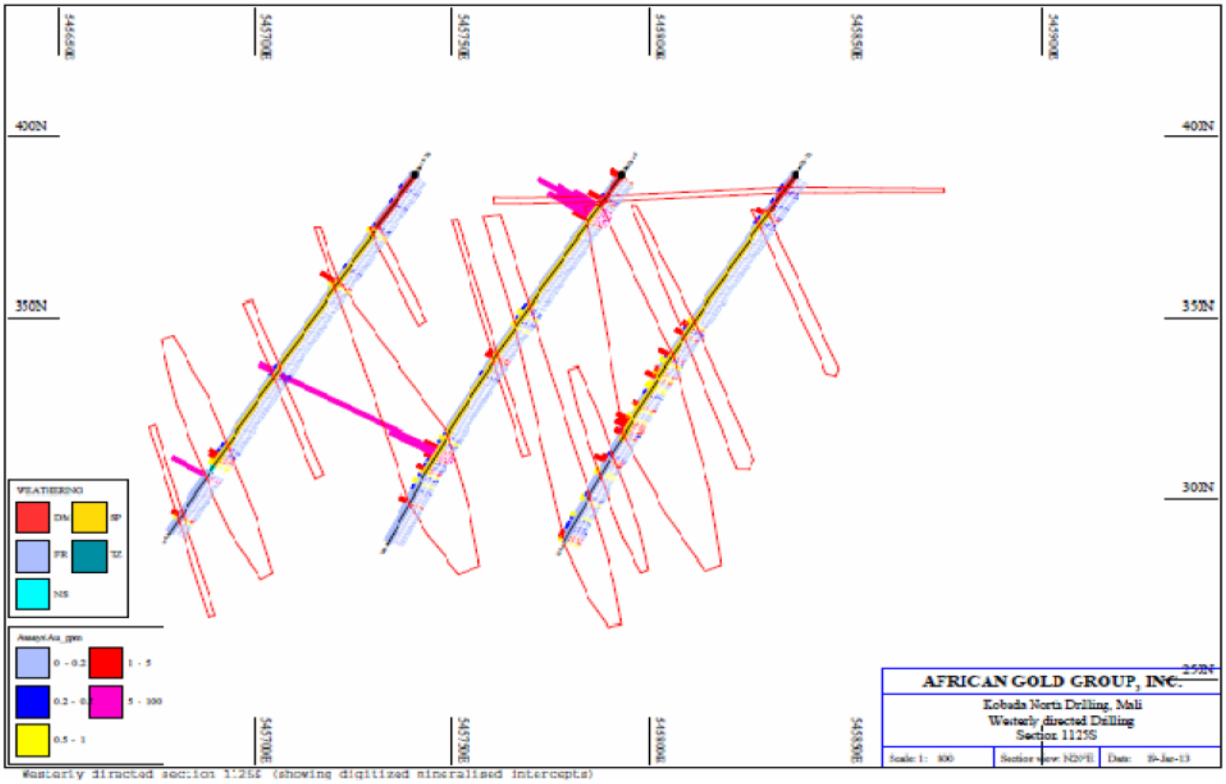

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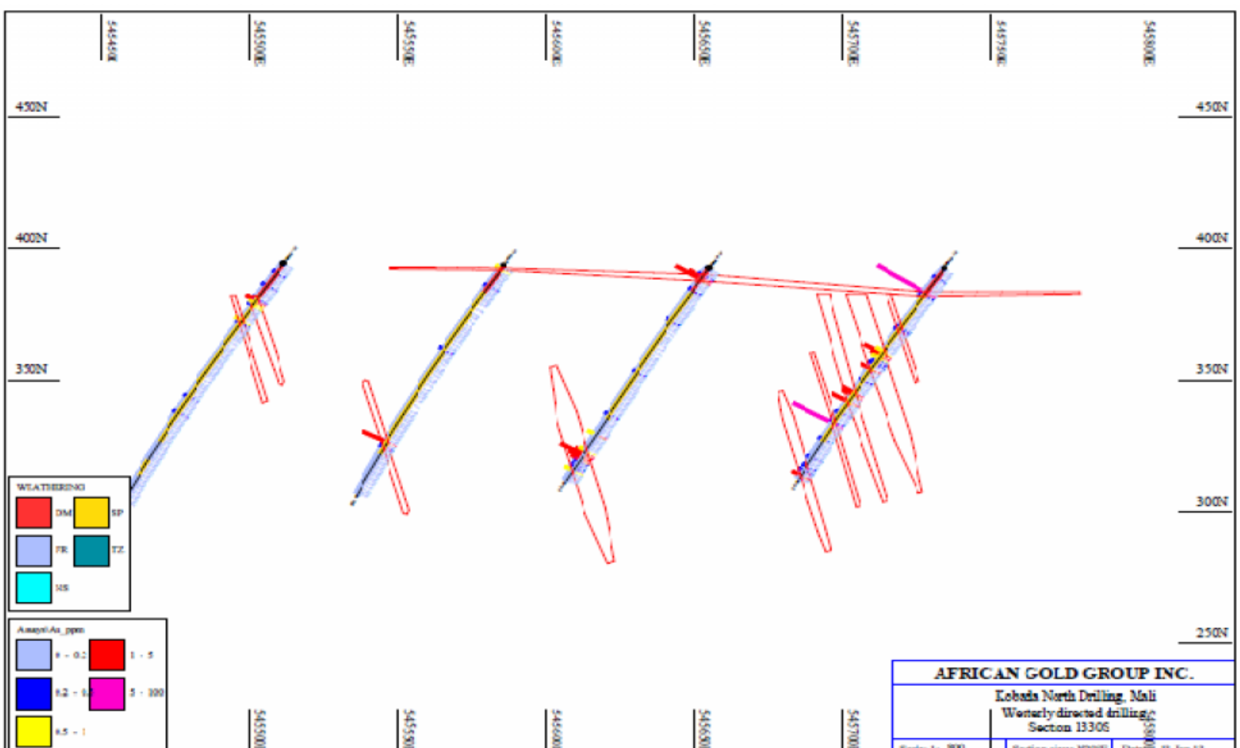

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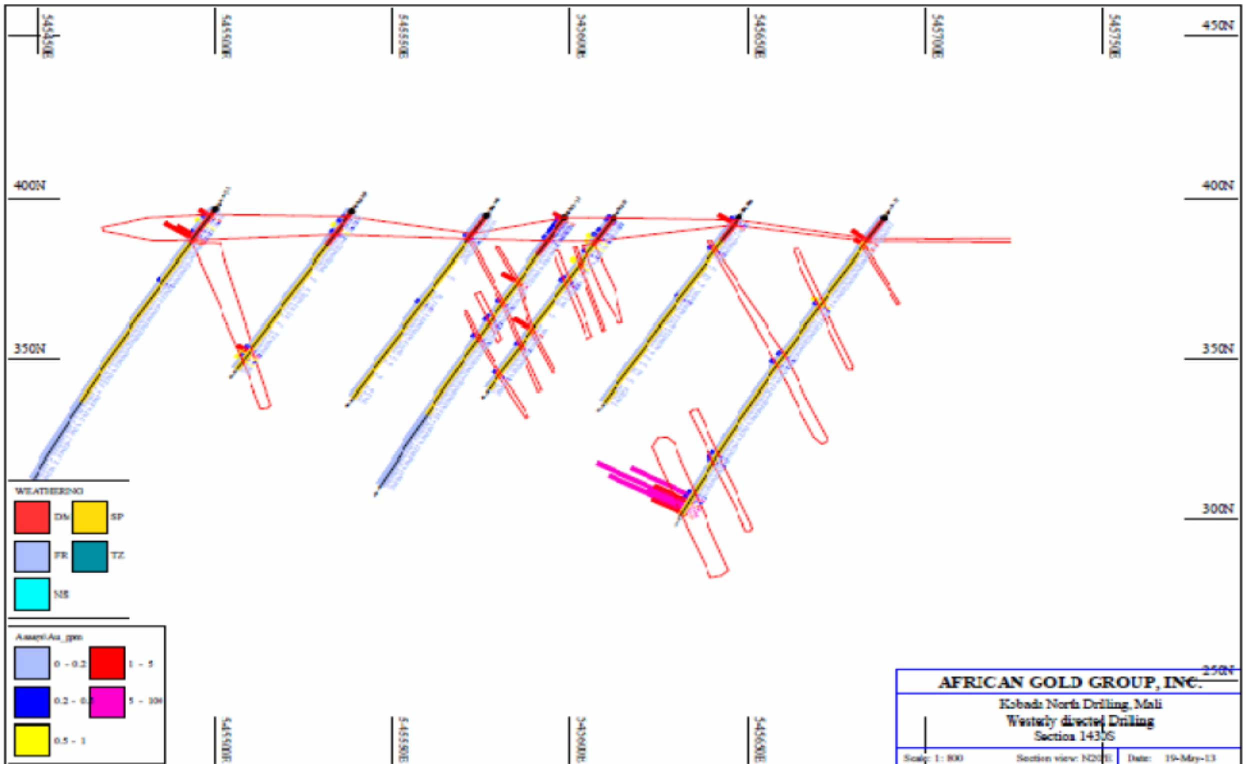

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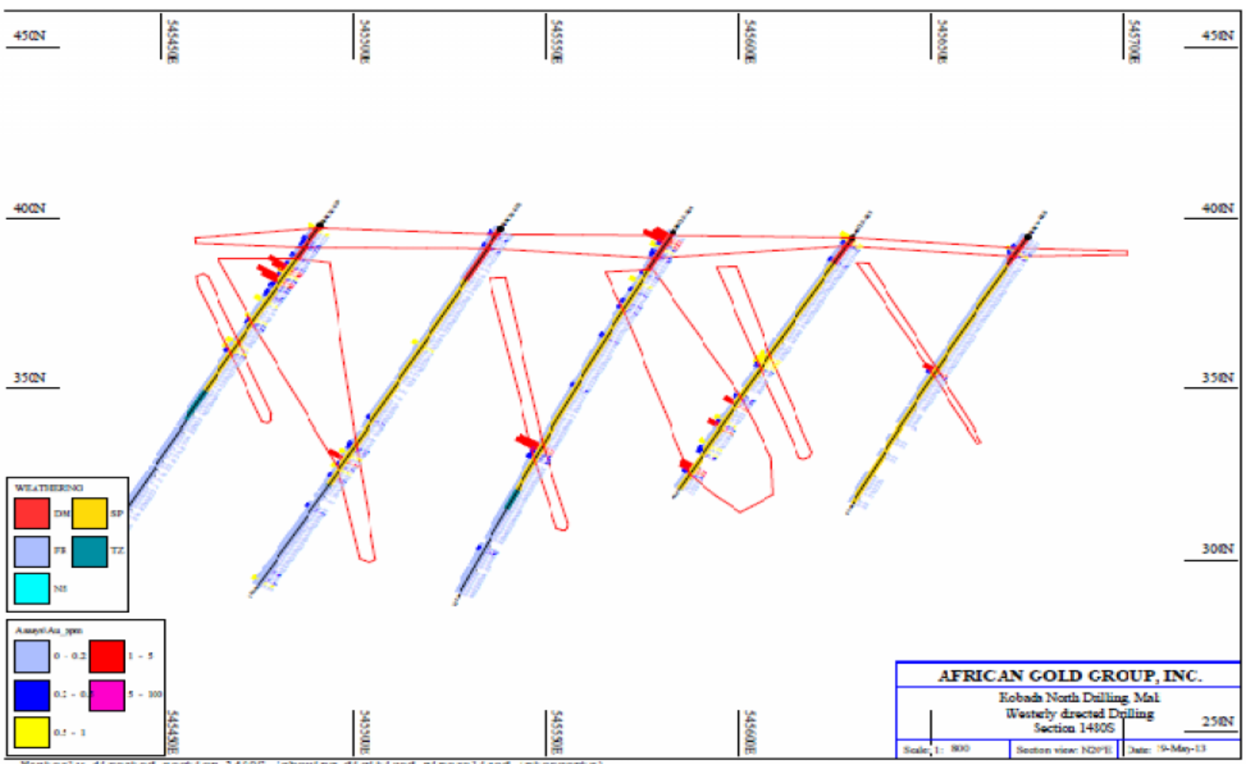

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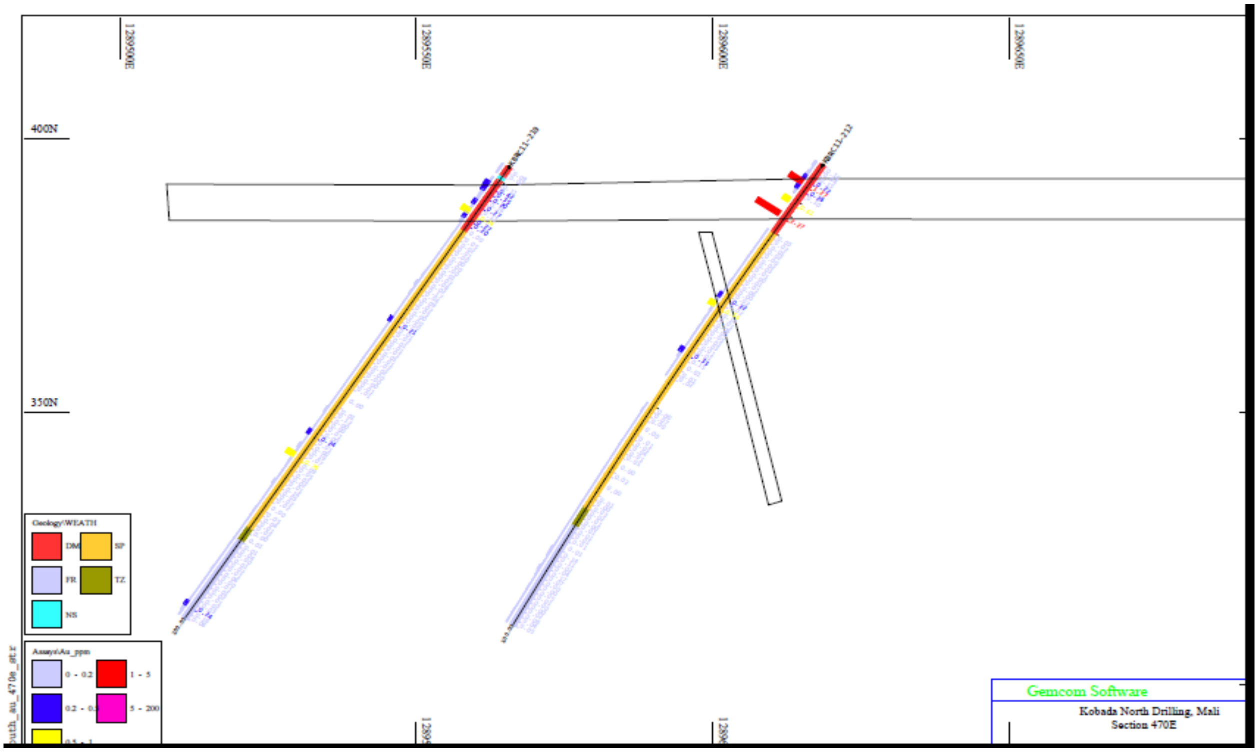

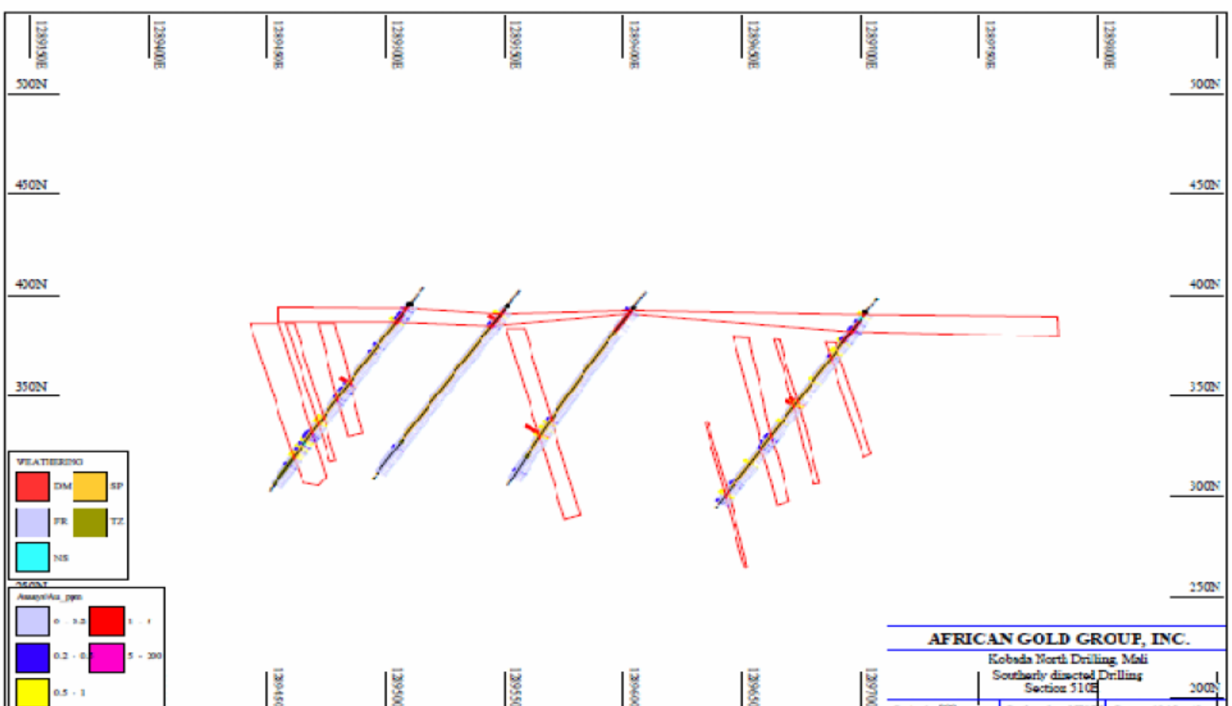

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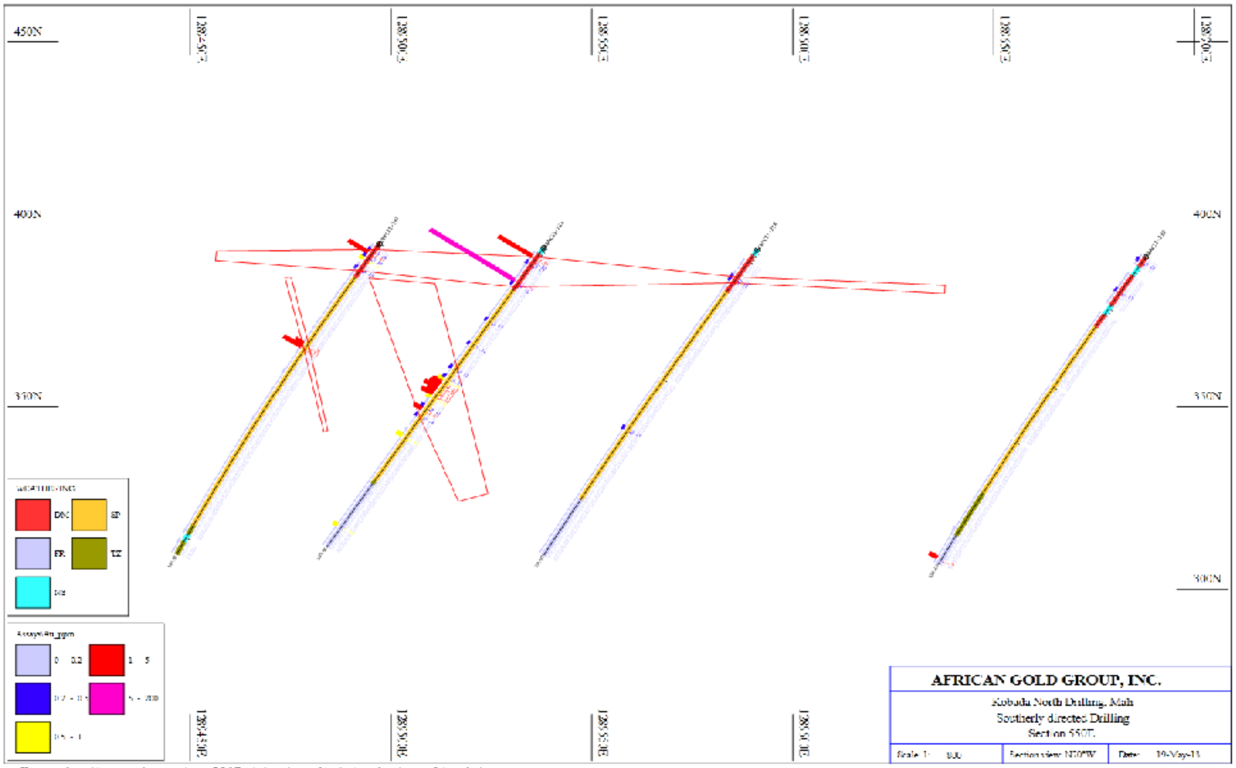

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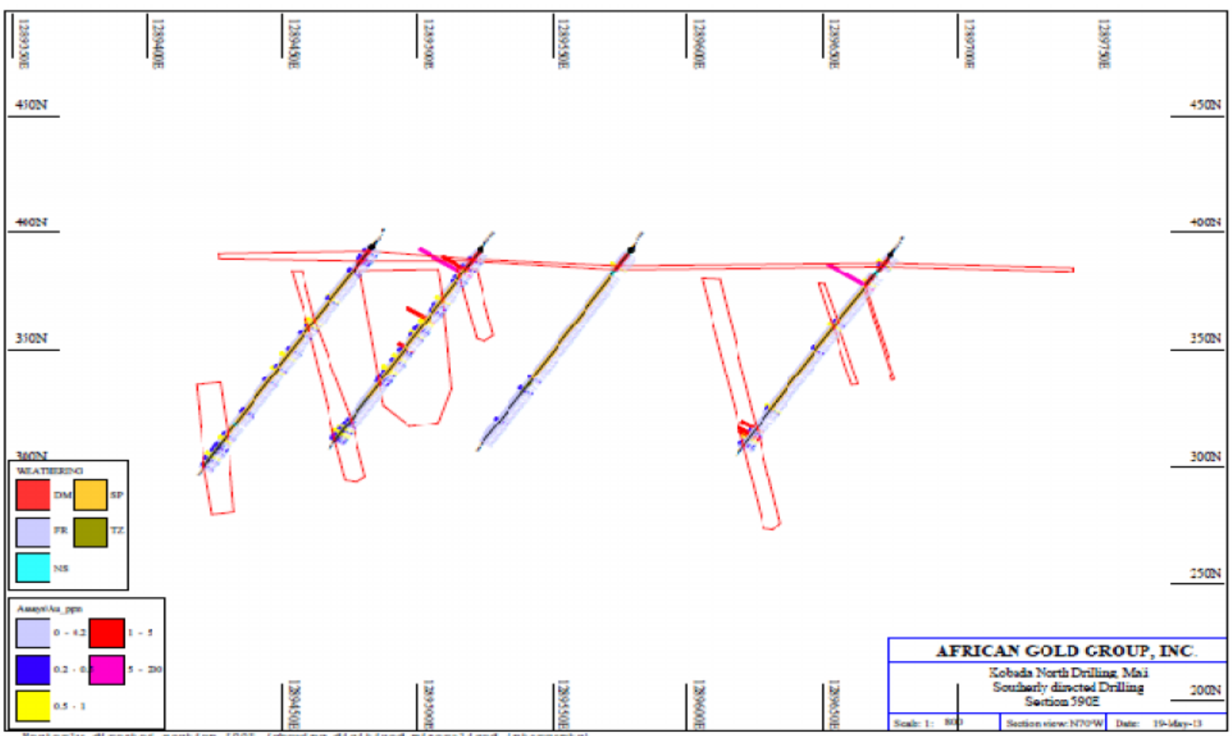

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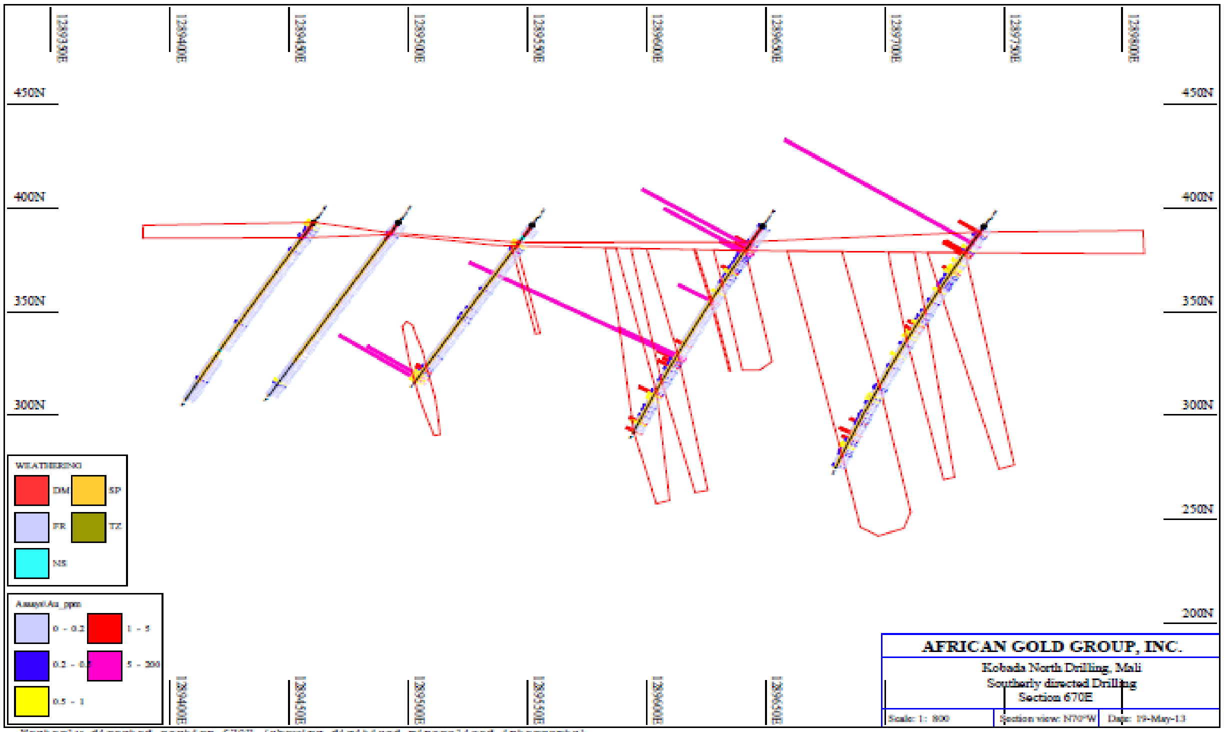
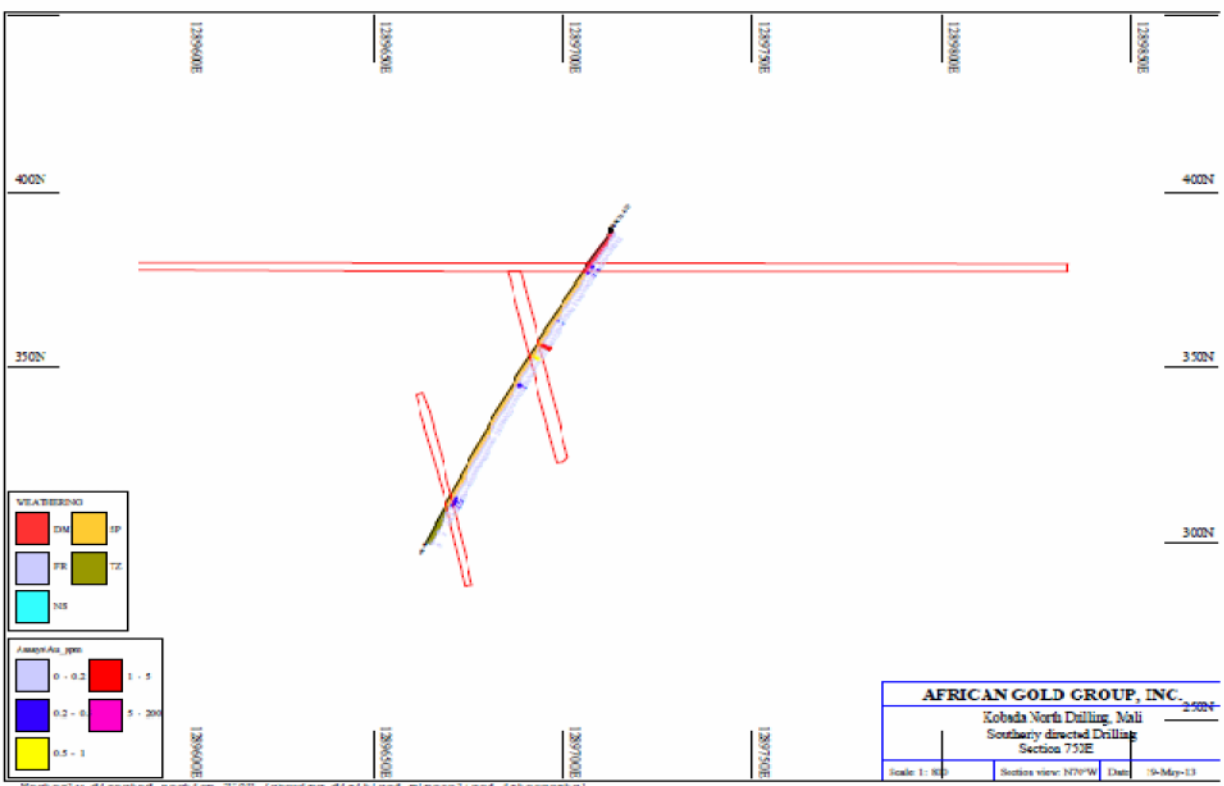
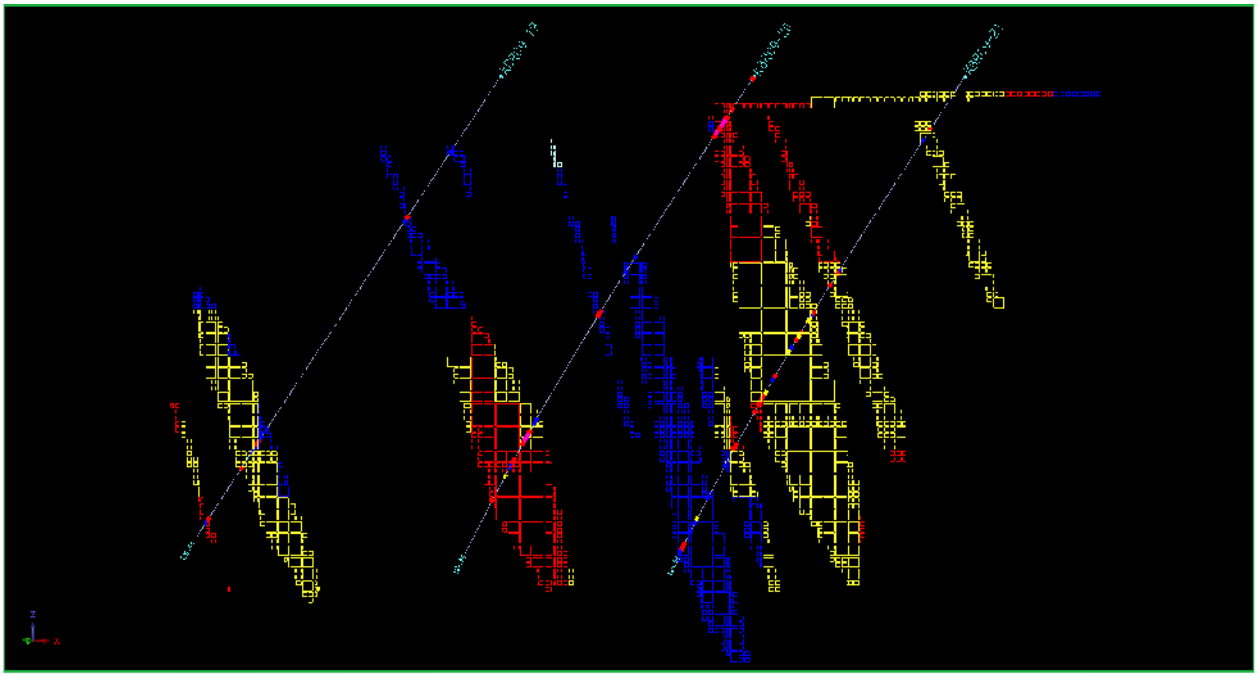
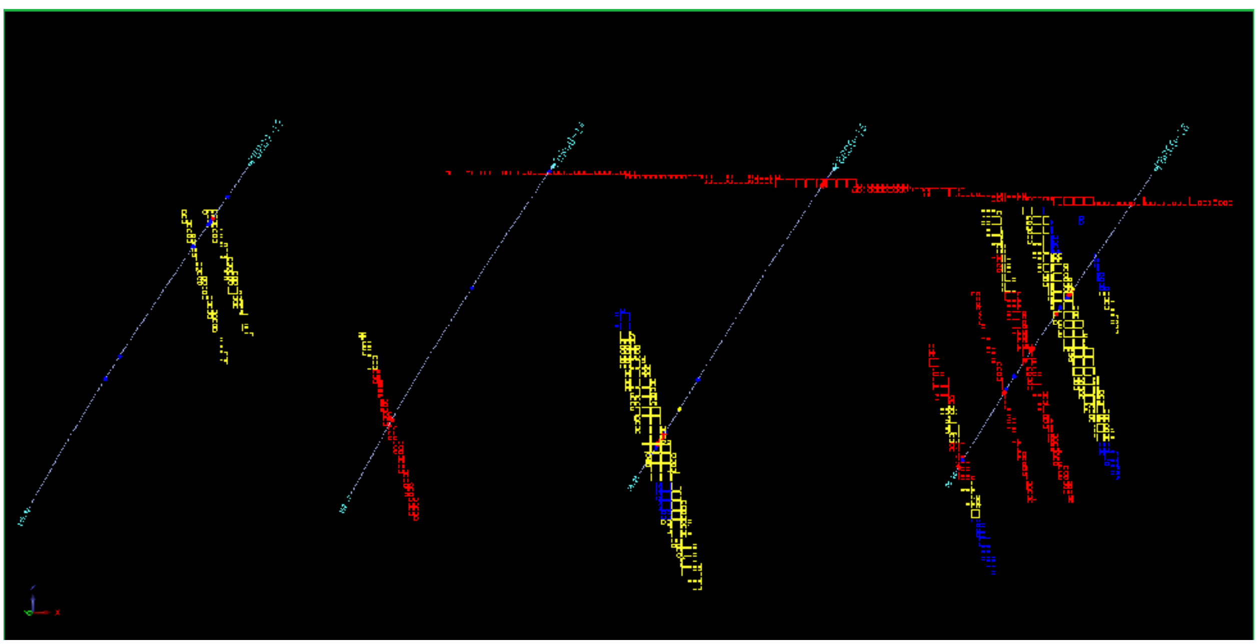


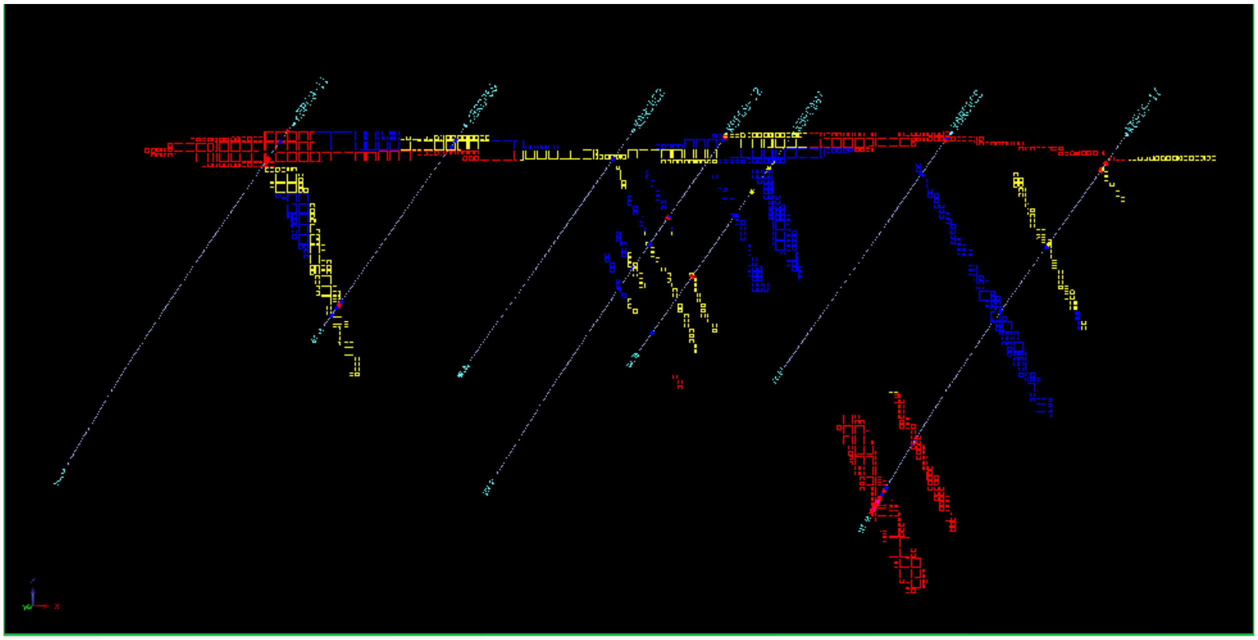
Westerly directed section 670E (showing digitized mineralised intercepts)

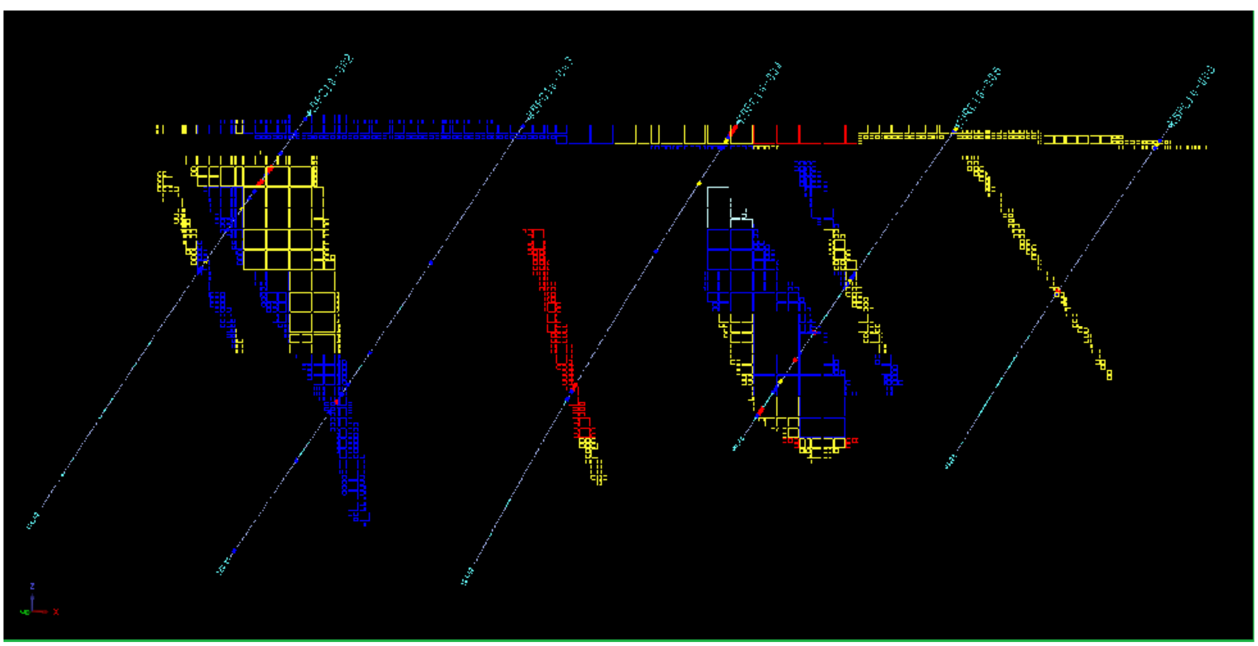


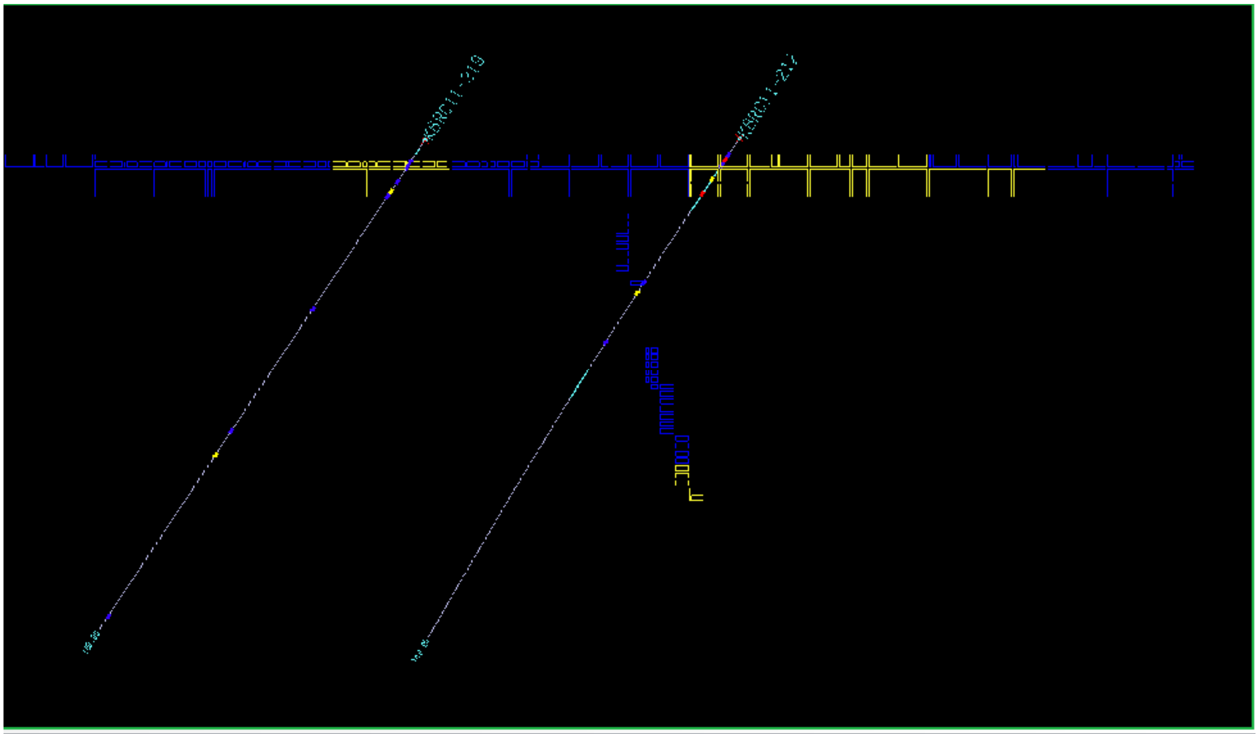
Westerly directed section 750E (showing digitized mineralised intercepts)

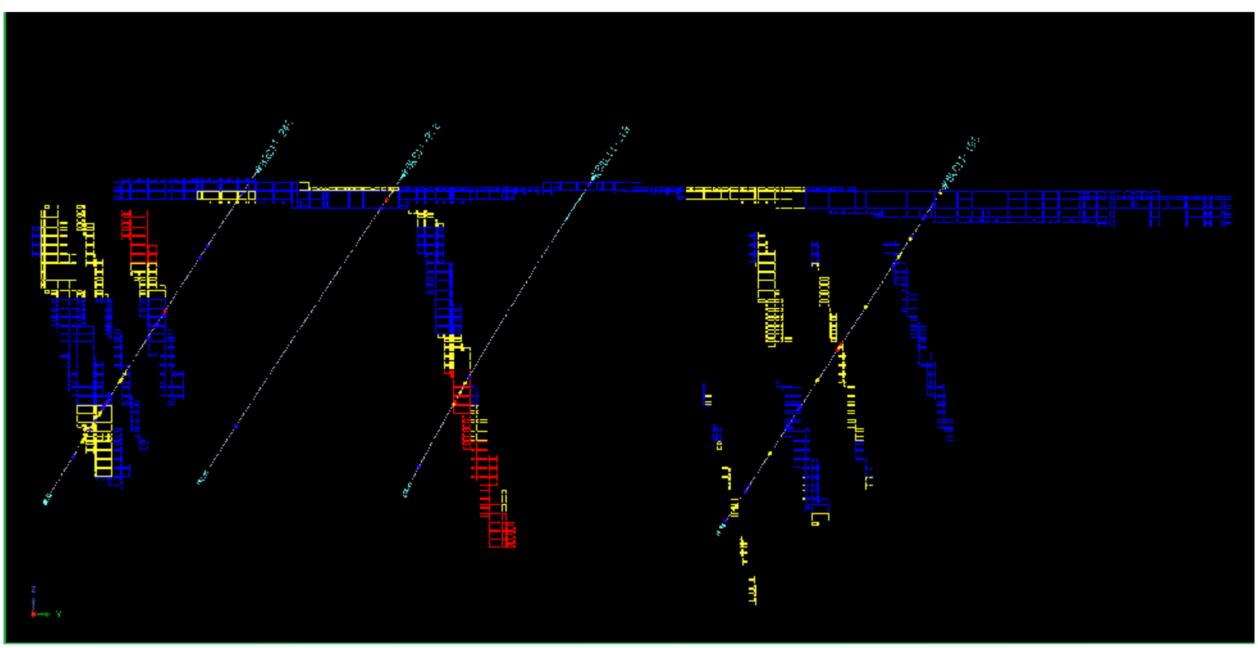


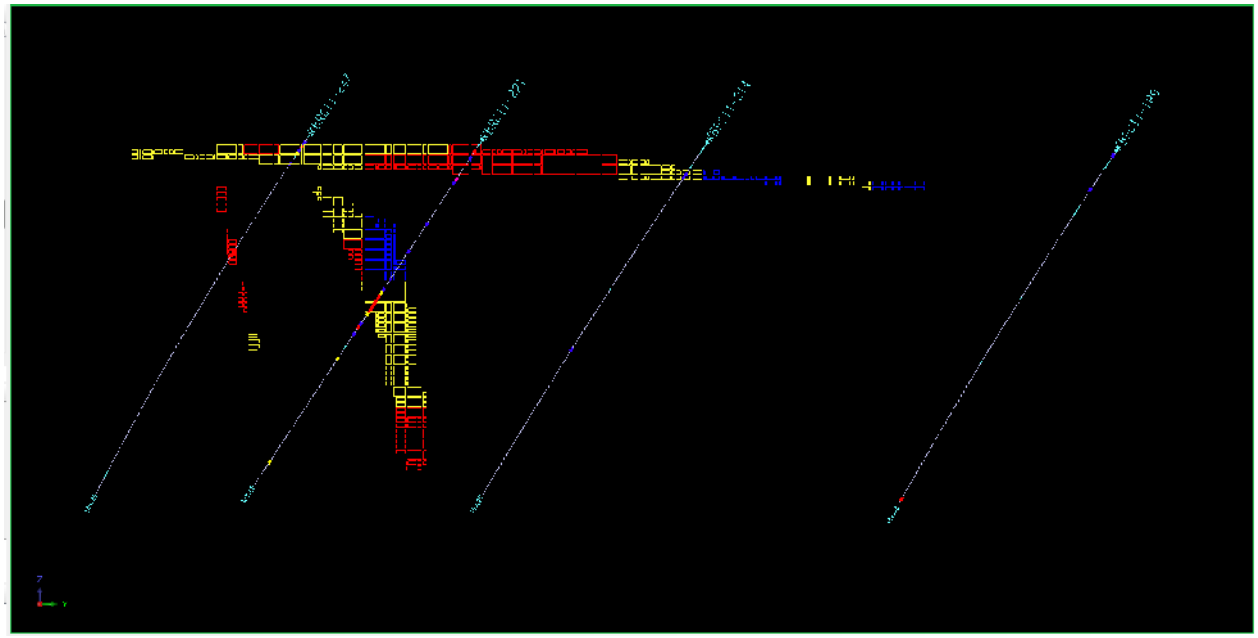


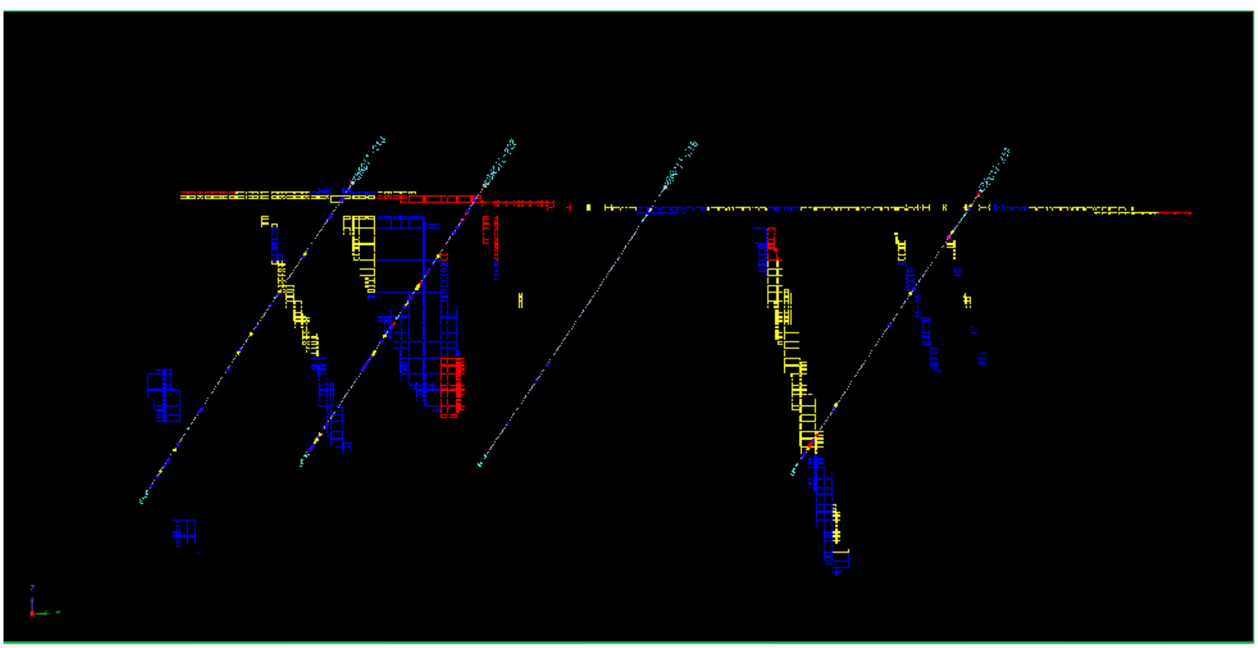


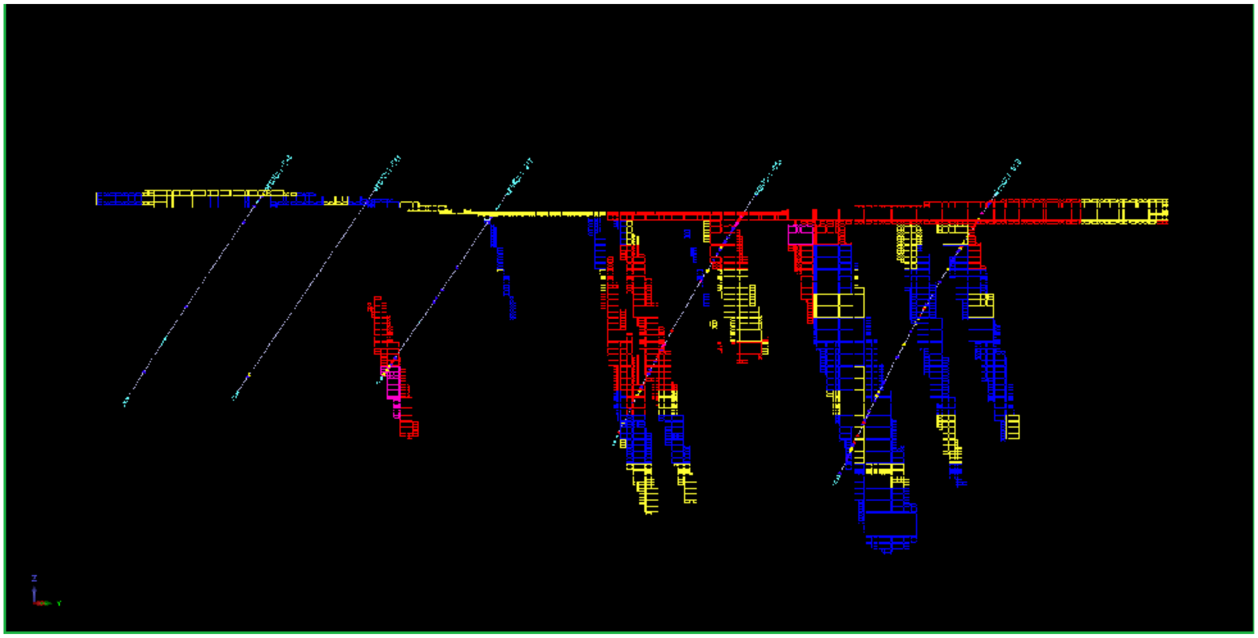


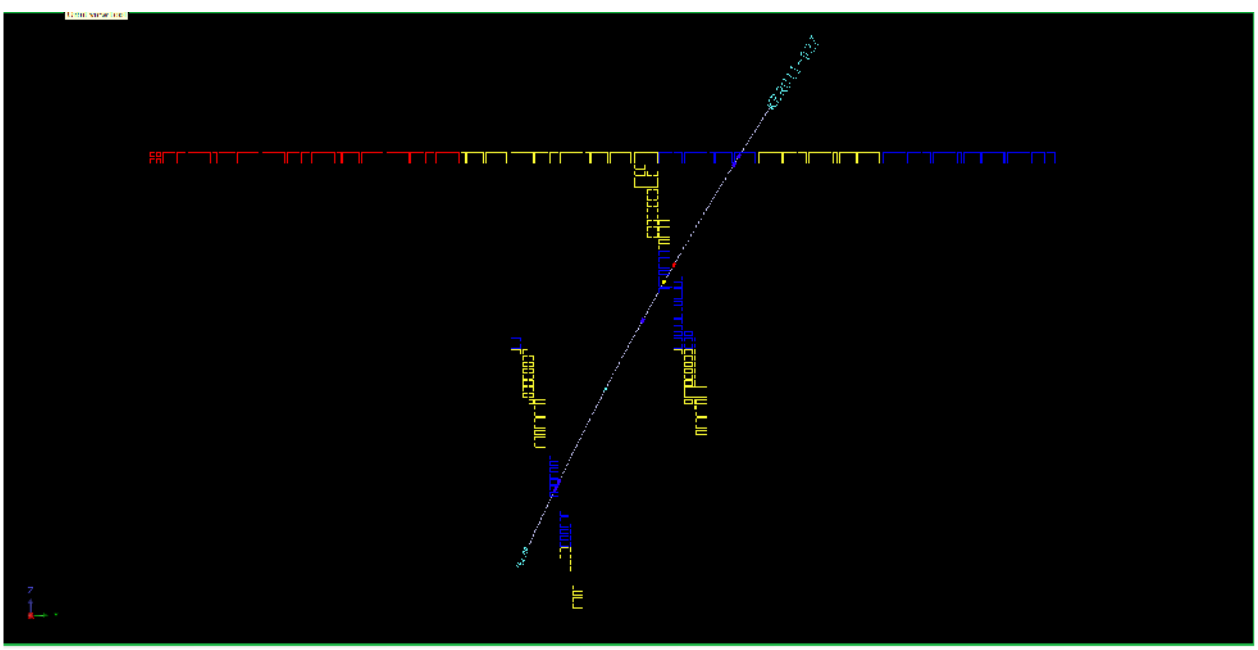


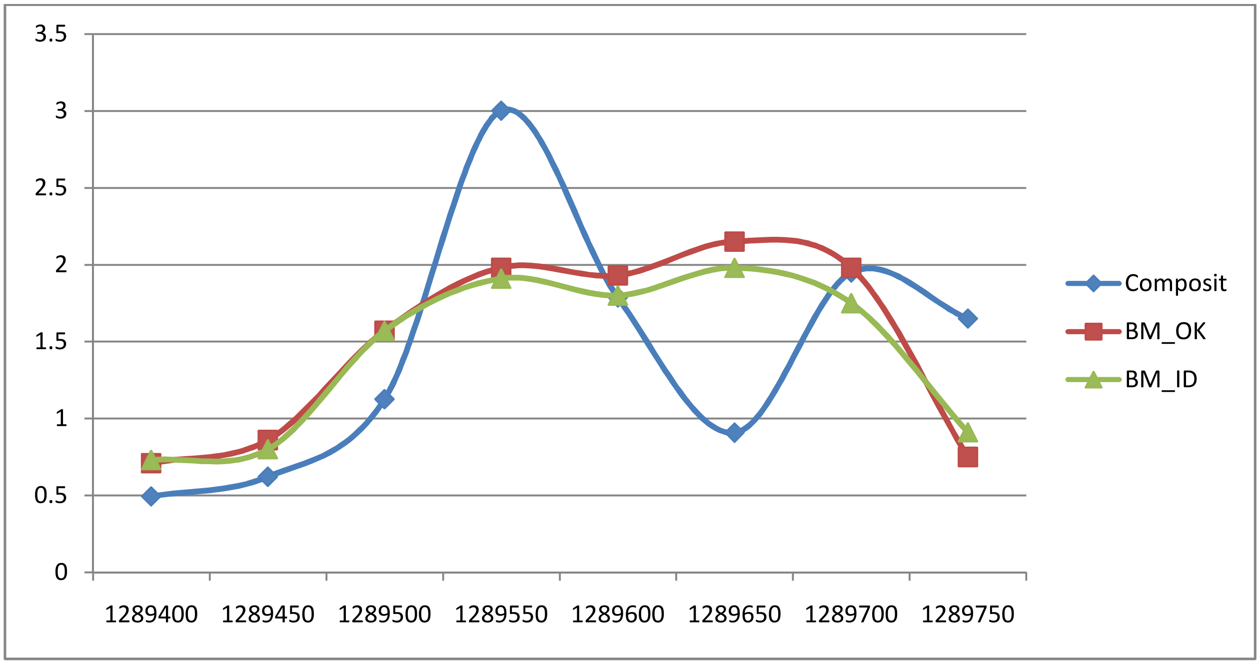








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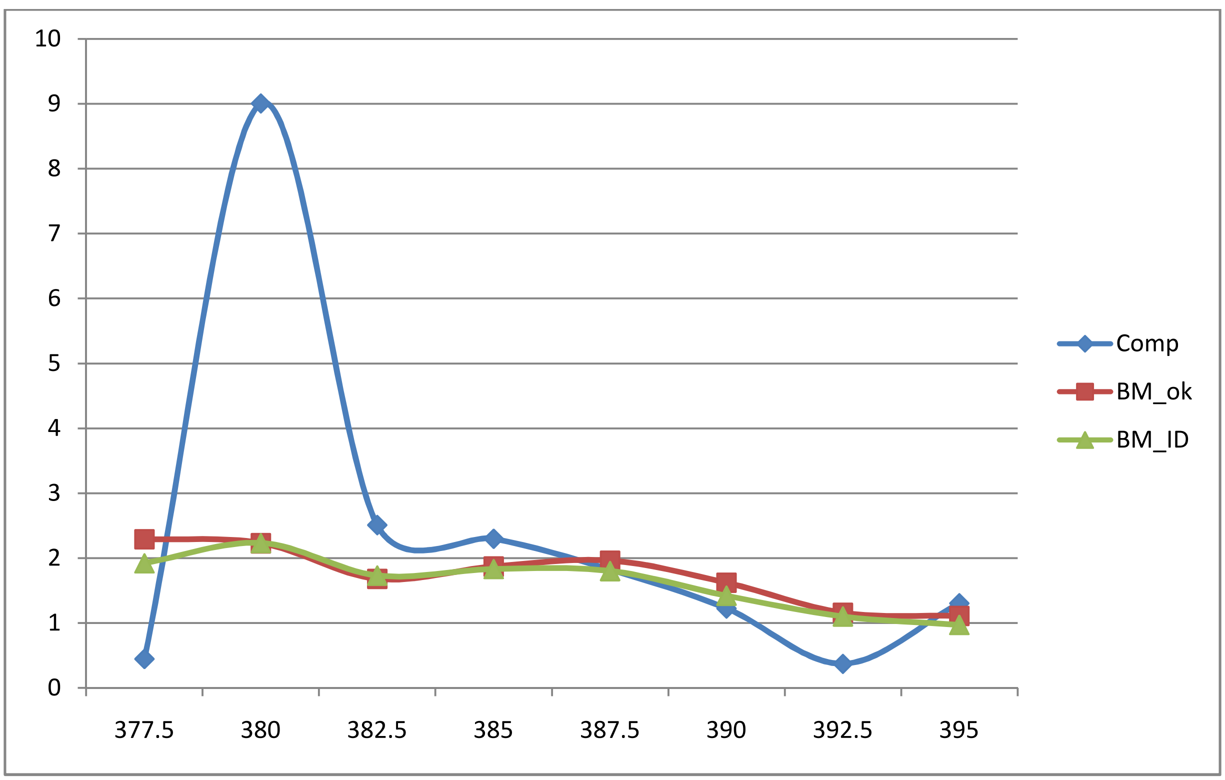
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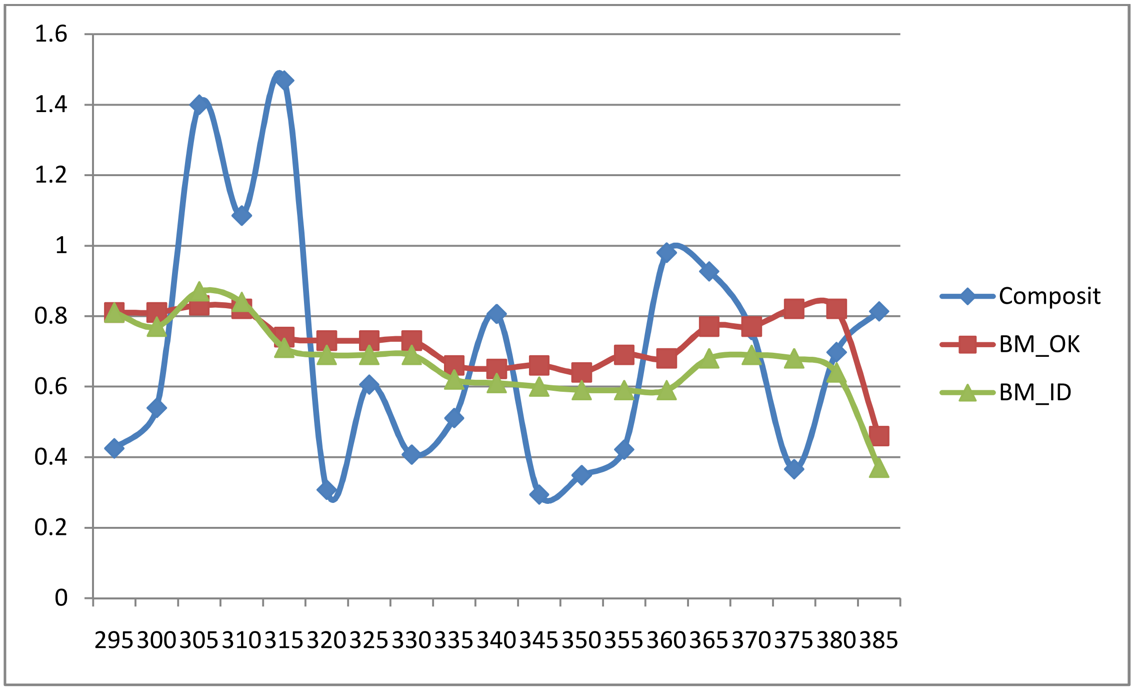
Comp
BM_ok
BM_ID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1.4
1.2
1
0.8
0.6
0.4
0.2
0
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Composit
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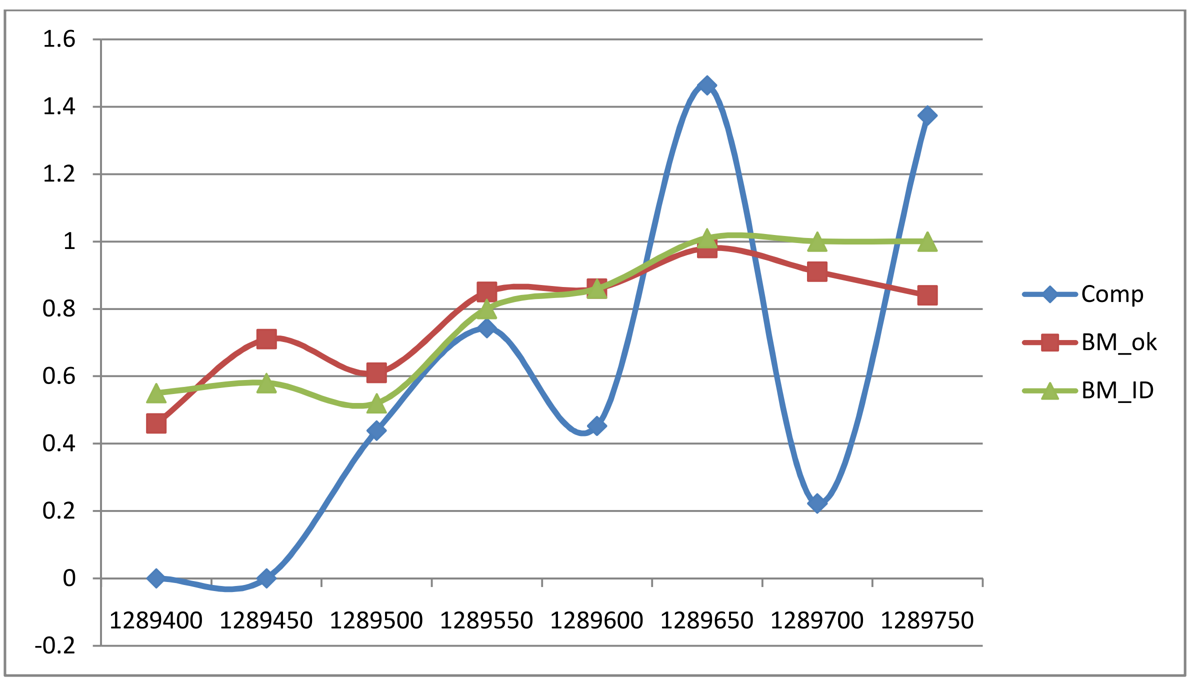
BM_ID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1.4
1.2
1
0.8
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Comp
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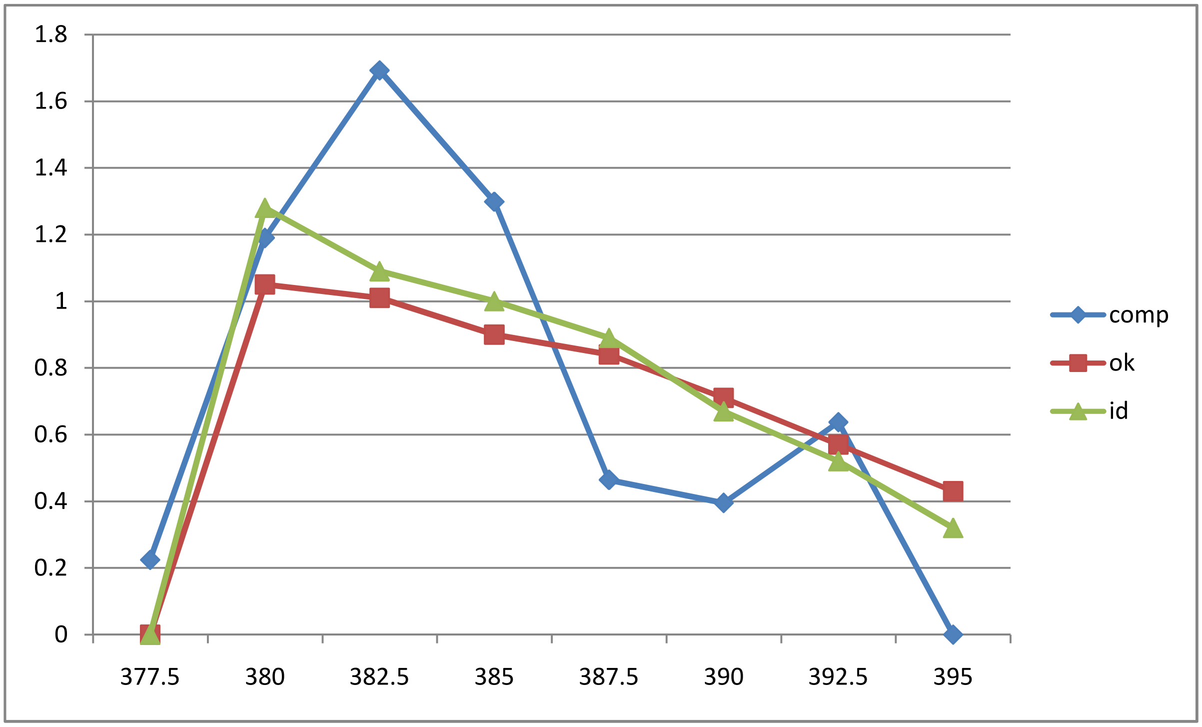
BM_ID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387.5
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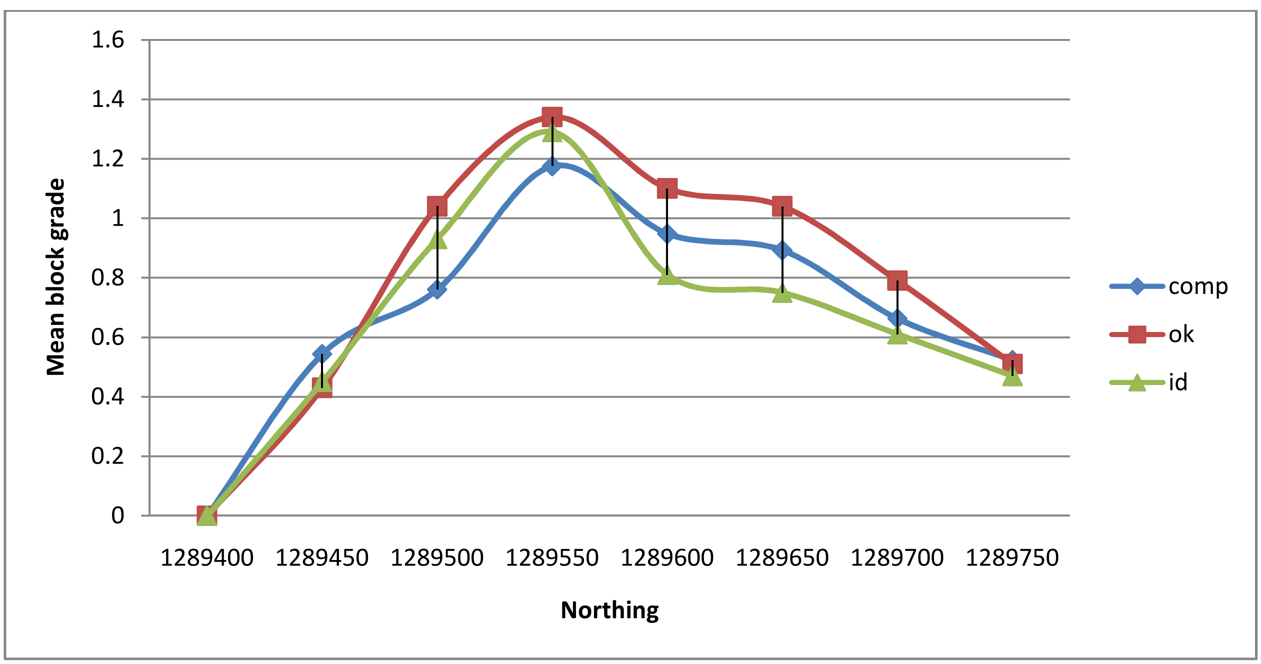
Mean block grade
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0.6
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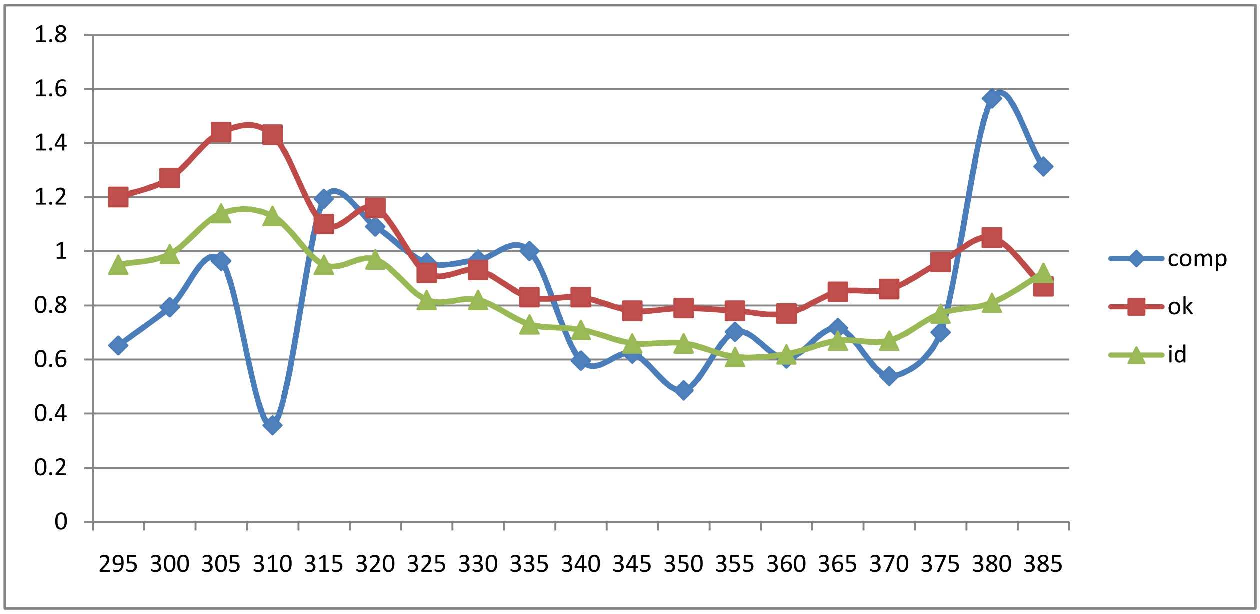
1289700
1289750
Northing
comp
ok
id



Attribute Name	Type	Decimals	Background	Description
ani_dist	Float	3	-99	Anisotropic distance
auok	Float	2	0	Au estimation by OK
av_anidist	Float	3	-99	Average anisotropic distance
bvar	Float	3	-99	Block variance
cb_slope	Float	3	-99	Conditional bias slope
domain	Character	-		Coded mineralization domains
krig_eff	Float	3	-99	Kriging efficiency
kvar	Float	3	-99	Kriging variance
lag_m	Float	3	-99	Lagrange multiplier
mat_type	Character	-		Material Type
nos	Integer	-	-99	Number of samples
nwt	Integer	-	-99	Number of weighted negative samples
oxidation	Character	-		Level of weathering
sg	Float	2	0	Specific Gravity

A1.36: South_Dom_1 & 2 general statistics

File	Rwsamp Alldom0.str	Rwsamp Dom2.str	Rwsamp Dom1.str
String range	All	All	All
Variable	rwsamp_alldom	rwsamp_dom_rock	rwsamp_dom_oxi
Number of samples	2828	723	145
Minimum value	0.0005	0.0005	0.0005
Maximum value	48.310001	48.310001	45.779999
	Ungrouped Data	Ungrouped Data	Ungrouped Data
Mean	0.333105	0.908774	1.181176
Median	0.0435	0.304	0.245
Geometric Mean	0.04092	0.279084	0.294842
Variance	3.55202	8.733322	21.790081
Standard Deviation	1.88468	2.955219	4.667985
Coefficient of variation	5.65791	3.251875	3.951981
Moment 1 About Arithmetic Mean	0	0	0
Moment 2 About Arithmetic Mean	3.55202	8.733322	21.790081
Moment 3 About Arithmetic Mean	120.142984	288.370265	770.630979
Moment 4 About Arithmetic Mean	4975.999133	12034.72627	30913.55099
Skewness	17.94673	11.173295	7.576318
Kurtosis	394.393293	157.789199	65.10753
Natural Log Mean	-3.196128	-1.276243	-1.221316
Log Variance	4.320131	2.489655	2.056362
10.0 Percentile	0.003	0.0325	0.094
20.0 Percentile	0.007	0.09	0.1295
30.0 Percentile	0.012	0.1535	0.166
40.0 Percentile	0.023	0.2385	0.2055
50.0 Percentile (median)	0.0435	0.304	0.245
60.0 Percentile	0.081	0.426	0.2885
70.0 Percentile	0.139	0.603	0.471
80.0 Percentile	0.2405	0.9105	0.7555
90.0 Percentile	0.5665	1.649	1.495
95.0 Percentile	1.1365	2.744	3.303
96.0 Percentile	1.3325	3.252	3.865
97.0 Percentile	1.668	4.2545	4.758
98.0 Percentile	2.4525	6.545	13.3555
99.0 Percentile	4.2705	12.86	35.615
100.0 Percentile	48.310001	48.310001	45.779999
Trimean	0.06775	0.363375	0.3135
Biweight	0.060435	0.339745	0.27688
MAD	0.055435	0.254	0.15988
Alpha	0.000399	0.036165	-0.000495
Sichel-t	0.353996	0.965353	0.812825

A1.37: South_Stats_Dom_2 general statistics

File	Rwsamp Dom2.str	2p0comp Dom2.str	2p0comp10 Dom2.str	
String range	All	All	All	
Variable	rwsamp_dom_rock	2p0comp_dom2	2p0comp10_dom2	Kriged Value
Number of samples	723	377	377	333
Minimum value	0.0005	0.007	0.007	0.151391
Maximum value	48.310001	30.825	10	6.650424
	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data
Mean	0.908774	0.908197	0.816051	0.778845
Median	0.304	0.3935	0.3935	0.508556
Geometric Mean	0.279084	0.396911	0.394712	0.580615
Variance	8.733322	5.243574	2.025029	0.61342
Standard Deviation	2.955219	2.289885	1.423035	0.783212
Coefficient of variation	3.251875	2.521352	1.743806	1.005607
Moment 1 About Arithmetic Mean	0	0	0	0
Moment 2 About Arithmetic Mean	8.733322	5.243574	2.025029	0.61342
Moment 3 About Arithmetic Mean	288.370265	105.64515	13.09954	1.591275
Moment 4 About Arithmetic Mean	12034.72627	2726.658677	109.076112	6.635842
Skewness	11.173295	8.798496	4.545787	3.312134
Kurtosis	157.789199	99.169004	26.599108	17.635172
Natural Log Mean	-1.276243	-0.924042	-0.929598	-0.543668
Log Variance	2.489655	1.45395	1.412987	0.50089
10.0 Percentile	0.0325	0.09525	0.09525	0.23714
20.0 Percentile	0.09	0.15575	0.15575	0.324633
30.0 Percentile	0.1535	0.21465	0.21465	0.390434
40.0 Percentile	0.2385	0.29525	0.29525	0.439479
50.0 Percentile (median)	0.304	0.3935	0.3935	0.508556
60.0 Percentile	0.426	0.511751	0.511751	0.637276
70.0 Percentile	0.603	0.711251	0.711251	0.801808
80.0 Percentile	0.9105	1.03875	1.03875	0.99679
90.0 Percentile	1.649	1.77075	1.77075	1.488344
95.0 Percentile	2.744	2.354	2.354	2.352429
96.0 Percentile	3.252	3.34945	3.34945	2.672962
97.0 Percentile	4.2545	4.0877	4.0877	2.927468
98.0 Percentile	6.545	6.12025	6.12025	3.398471
99.0 Percentile	12.86	9.67075	9.67075	4.429684
100.0 Percentile	48.310001	30.825	10	6.650424
Trimean	0.363375	0.446438	0.446438	0.560784
Biweight	0.339745	0.423023	0.423023	0.541705
MAD	0.254	0.270523	0.270523	0.229891
Alpha	0.036165	-0.00693	-0.00693	-0.133146
Sichel-t	0.965353	0.81842	0.79749	0.745156

A1.38: West_Dom_1 & 2 general statistics

File	Rwsamp Alldoms Kw.str	Rwsamp Dom Rock.str	Rwsamp Dom Oxi.str	Rwsamp Dom Sap.str	Rwsamp Dom Frsh.str
String range	All	All	All	All	All
Variable	rwsamp_alldoms_kw	rwsamp_dom_allrock	rwsamp_dom_oxi	rwsamp_dom_sap	rwsamp_dom_frsh
Number of samples	2757	538	126	2231	395
Minimum value	0.0005	0.007	0.066	0.0005	0.0005
Maximum value	100	77.989998	100	77.989998	15.85
	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data
Mean	0.367658	1.21376	2.026532	0.506398	0.222842
Median	0.039	0.354	0.2865	0.034	0.031
Geometric Mean	0.04428	0.378316	0.434509	0.047538	0.034565
Variance	7.295621	15.485343	85.742414	10.201617	0.806491
Standard Deviation	2.701041	3.935142	9.25972	3.193997	0.898049
Coefficient of variation	7.34662	3.242108	4.569245	6.30728	4.029983
Moment 1 About Arithme	0	0	0	0	0
Moment 2 About Arithme	7.295621	15.485343	85.742414	10.201617	0.806491
Moment 3 About Arithme	544.927011	880.296563	7562.92361	645.556023	9.988281
Moment 4 About Arithme	49142.06364	65054.35098	732885.4268	48708.13284	151.830782
Skewness	27.653169	14.446005	9.525684	19.812094	13.790856
Kurtosis	923.270205	271.290573	99.68841	468.018928	233.432025
Natural Log Mean	-3.117225	-0.972025	-0.833539	-3.04623	-3.364924
Log Variance	3.307316	2.204724	1.761254	3.734022	3.532223
10.0 Percentile	0.005	0.05	0.125	0.005	0.004
20.0 Percentile	0.01	0.11	0.162	0.01	0.006
30.0 Percentile	0.014501	0.2	0.209	0.014	0.01
40.0 Percentile	0.023	0.2745	0.237	0.021	0.0195
50.0 Percentile (median)	0.039	0.354	0.2865	0.034	0.031
60.0 Percentile	0.064	0.529	0.3915	0.06	0.0475
70.0 Percentile	0.1075	0.767	0.514	0.11	0.082
80.0 Percentile	0.183	1.27	1.241	0.236	0.1735
90.0 Percentile	0.479	2.5985	2.943	0.682	0.583
95.0 Percentile	1.2305	4.3445	8.1315	1.987	1.103
96.0 Percentile	1.619	5.283	9.607	2.557	1.1335
97.0 Percentile	2.266	7.276	12.0475	3.1145	1.1755
98.0 Percentile	3.06	9.4925	16.3375	4.977	1.7435
99.0 Percentile	5.9365	13.73	59.905	9.4765	2.7085
100.0 Percentile	100	77.989998	100	77.989998	15.85
Trimean	0.05625	0.4615	0.39825	0.058875	0.046
Biweight	0.050117	0.414952	0.326911	0.049533	0.036788
MAD	0.042117	0.313952	0.168411	0.040533	0.031788
Alpha	-0.000495	-0.002375	-0.06534	-0.000495	-0.000495
Sichel-t	0.231042	1.134344	1.034787	0.306784	0.199682

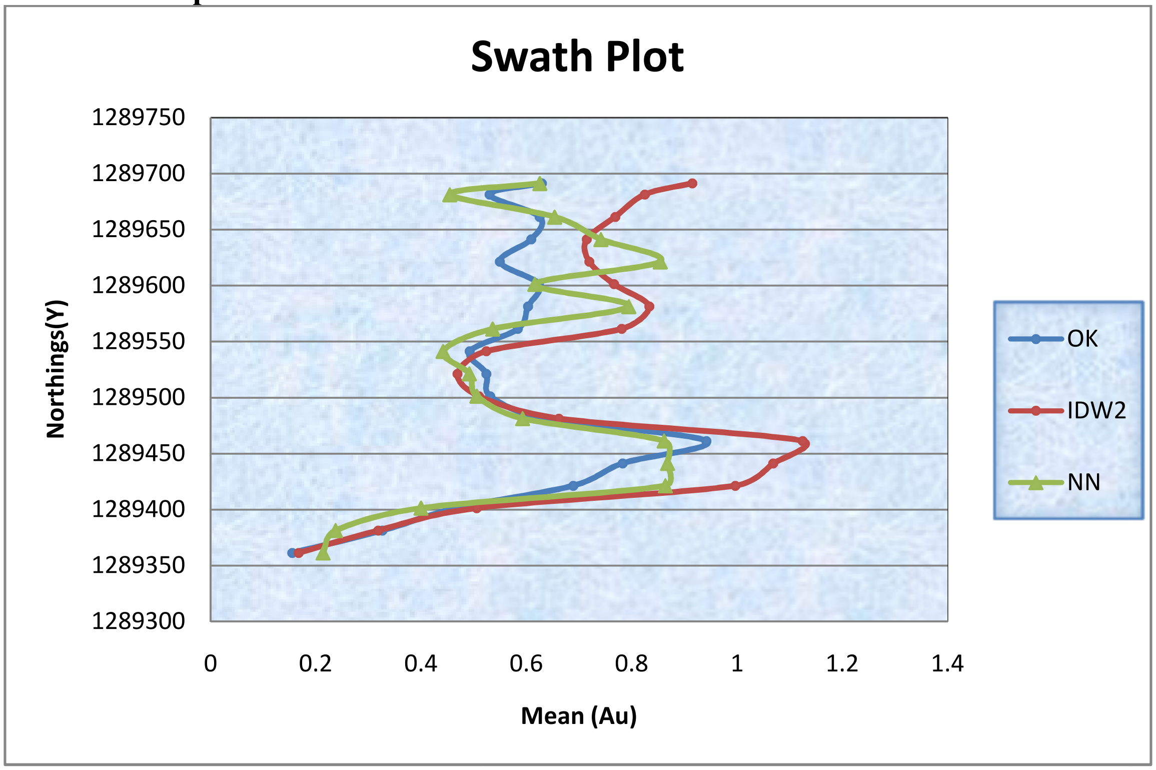
A1.39: West_Stats_Dom_1 general statistics

File	Rwsamp Dom Oxi.str	2p0comp Dom1.str	2p0comp09 Cdom1.str	
String range	All	All	All	
Variable	rwsamp_dom_oxi	2p0comp_dom1	2p0comp09_cdom1	Kriged Value
Number of samples	126	71	71	41
Minimum value	0.066	0.08	0.08	0.174
Maximum value	100	50.11	9	4.40927
				0
	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data
Mean	2.026532	1.968373	1.366359	0.786886
Median	0.2865	0.34	0.34	0.464536
Geometric Mean	0.434509	0.580804	0.565544	0.529855
Variance	85.742414	37.693295	5.041573	0.859668
Standard Deviation	9.25972	6.139487	2.245345	0.927183
Coefficient of variation	4.569245	3.119066	1.643305	1.178293
Moment 1 About Arithmetic Mean	0	0	0	0
Moment 2 About Arithmetic Mean	85.742414	37.693295	5.041573	0.859668
Moment 3 About Arithmetic Mean	7562.92361	1589.350243	28.260787	2.097603
Moment 4 About Arithmetic Mean	732885.4268	75811.20108	211.441635	6.878092
Skewness	9.525684	6.867892	2.49652	2.631643
Kurtosis	99.68841	53.358691	8.318755	9.306929
Natural Log Mean	-0.833539	-0.543343	-0.569967	-0.635151
Log Variance	1.761254	1.695944	1.507559	0.639125
10.0 Percentile	0.125	0.16875	0.16875	0.213777
20.0 Percentile	0.162	0.208	0.208	0.25706
30.0 Percentile	0.209	0.2515	0.2515	0.295884
40.0 Percentile	0.237	0.29625	0.29625	0.404748
50.0 Percentile (median)	0.2865	0.34	0.34	0.464536
60.0 Percentile	0.3915	0.583	0.583	0.574348
70.0 Percentile	0.514	1.011	1.011	0.657143
80.0 Percentile	1.241	1.6335	1.6335	0.980903
90.0 Percentile	2.943	3.964	3.964	1.670737
95.0 Percentile	8.1315	8.04675	8.01475	3.491141
96.0 Percentile	9.607	9.2795	9	3.491141
97.0 Percentile	12.0475	9.7845	9	3.964457
98.0 Percentile	16.3375	30.092	9	3.964457
99.0 Percentile	59.905	30.092	9	4.40927
100.0 Percentile	100	50.11	9	4.40927
Trimean	0.39825	0.557688	0.557688	0.508181
Biweight	0.326911	0.50152	0.50152	0.484985
MAD	0.168411	0.31027	0.31027	0.225131
Alpha	-0.06534	-0.0792	-0.0792	-0.147965
Sichel-t	1.034787	1.327449	1.180246	0.722054

A1.40: West_Stats_Dom_2 general statistics

File	Rwsamp Dom Rock.str	2p0comp Dom2.str	2p0comp05 Cdom2.str
String range	All	All	All
Variable	rwsamp_dom_allrock	2p0comp_dom2	2p0comp05_cdom2
Number of samples	538	485	485
Minimum value	0.007	0.0015	0.0015
Maximum value	77.989998	40.7372	5
	Ungrouped Data	Ungrouped Data	Ungrouped Data
Mean	1.21376	0.787211	0.624076
Median	0.354	0.23	0.23
Geometric Mean	0.378316	0.193778	0.19053
Variance	15.485343	5.583159	1.082499
Standard Deviation	3.935142	2.362871	1.040432
Coefficient of variation	3.242108	3.001574	1.667155
		2.0	
Moment 1 About Arithmetic Mean	0	0	0
Moment 2 About Arithmetic Mean	15.485343	5.583159	1.082499
Moment 3 About Arithmetic Mean	880.296563	147.628603	3.138323
Moment 4 About Arithmetic Mean	65054.35098	5413.140434	12.686242
Skewness	14.446005	11.190526	2.786483
Kurtosis	271.290573	173.655794	10.826248
Natural Log Mean	-0.972025	-1.641042	-1.657947
Log Variance	2.204724	3.078103	2.95074
10.0 Percentile	0.05	0.02	0.02
20.0 Percentile	0.11	0.0366	0.0366
30.0 Percentile	0.2	0.08	0.08
40.0 Percentile	0.2745	0.1385	0.1385
50.0 Percentile (median)	0.354	0.23	0.23
60.0 Percentile	0.529	0.32875	0.32875
70.0 Percentile	0.767	0.4645	0.4645
80.0 Percentile	1.27	0.89275	0.89275
90.0 Percentile	2.5985	1.815	1.815
95.0 Percentile	4.3445	3.03185	3.03185
96.0 Percentile	5.283	3.73225	3.73225
97.0 Percentile	7.276	4.16275	4.16275
98.0 Percentile	9.4925	7.4164	5
99.0 Percentile	13.73	8.7161	5
100.0 Percentile	77.989998	40.7372	5
Trimean	0.4615	0.282563	0.282563
Biweight	0.414952	0.250965	0.250965
MAD	0.313952	0.214615	0.214615
Alpha	-0.002375	0.012073	0.012073

A1.41: Swath plot for various estimation methods on WDD



A1.41: Swath plot for various estimation methods on SDD