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SCHOOL OF MINES, KUMASI

THESIS REPORT

TOPIC: *MINERAL RESOURCE ESTIMATION OF ALLUVIAL GOLD DEPOSITS - A COMPARATIVE STUDY*

SUBMITTED IN PARTIAL FULFILMENT OF THE REQUIREMENTS FOR THE MASTER OF SCIENCE (MSc) DEGREE IN MINERAL EXPLORATION

NAME: *ERIC WALTER ENGMANN* (BSc, Hons)

DEPARTMENT: *GEOLOGICAL ENGINEERING*

SUPERVISOR: ....

DR. D. MIREKU-GYIMAH (PhD,MSc,DIC,CEng,MIMM,MAIME,MGSG)

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ABSTRACT

Nkroful Mining Ltd has acquired a concession at Nkroful town which has both hard rock (vein type) and alluvial gold mineralization. The alluvial gold mineralization is erratic and complex so it demands a thorough investigation to establish the geological characteristics of the deposit and a good estimation method to estimate the gold resource potential of the deposit.

This thesis is aimed at, studying the geological characteristics of the deposit and selecting an appropriate mineral resource estimation method through a comparative study of various estimation methods. The outcome should assist Nkroful Mining Ltd in estimating the gold resource potential of the concession and also contribute towards the improvement of alluvial gold resource estimation in Ghana.

The geological characteristics of the deposit were studied through literature review and geological mapping in the field. Statistical and geostatistical analysis were used to check and quantify some of the parameters obtained from the geological studies. Based on the geological characteristics of the deposit and sample grade values, four blocks were demarcated for the resource estimation.

Triangular, sectional, inverse square distance, statistical and geostatistical methods were used to estimate the gold potential of the deposit. Grade estimates from the various estimation methods did not differ significantly. Comparatively, statistical estimation gave higher values. Triangular and sectional estimation gave values that fluctuated between high and low grades. Geostatistical and inverse distance estimation gave values that were average and compared favourably to each other.

Despite the constraint that mined figures were not available at the end of the research work to substantiate the method chosen, it is still proposed that inverse distance square method should be adopted by Nkroful Mining Ltd due to its simplicity and comparability with the geostatistical method which took into account the regionalization of the variables.

ACKNOWLEDGEMENTS

I express my heartfelt thanks and appreciation to my supervisor, Dr Dan Mireku-Gyimah for his fatherly supervision and immense contribution towards the success of this work. His commitment and devotion to this work are unparalleled.

I am equally thankful to the management of Nkroful Mining Ltd., for granting me access to their concession and making available any data used in this study. The immeasurable help of Dr. B. A Barko who took me through the field procedures at Nkroful and gave useful suggestions towards the completion of this work is worth mentioning.

My profound gratitude is also expressed to the authorities and staff of U.S.T School of Mines, Tarkwa, for putting their computing and library facilities at my disposal. The contribution made by Dr. S. K. Asiedu-Asante, whose personal software I used for the greater part of this work is very much appreciated.

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Finally, I give all the glory to God Almighty, whose divine assistance and guidance saw to the successful end of this work.

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CHAPTER ONE

INTRODUCTION

1.1 PROBLEM DEFINITION

Nkroful Mining Ltd has acquired a gold concession at Nkroful which has both hard rock (vein type) and alluvial gold mineralization. The alluvial gold mineralization is unusual and may be the first of its kind to be officially mined in Ghana (Barko & Thorton, 1993) in the sense that, apart from its normal alluvial free gold mineralization in the gravel layer, the angular quartz pebbles within the auriferous gravel layer contain substantial amount of gold. This peculiar characteristic of the Nkroful deposit compounds the usual problem of mineral resource estimation associated with alluvial deposits i.e the erratic grade distribution. Unless such a deposit is carefully studied and a proper resource estimation procedure adopted, any attempt to estimate the gold content can give erroneous values.

This thesis aims at studying the geological characteristics of the Nkroful alluvial gold deposit and selecting an appropriate mineral resource estimation method through a comparative study of various mineral resource estimation methods.

1.2 OBJECTIVES

The objectives of this thesis work are:

(i) To study the geological characteristics of the Nkroful alluvial deposit;

(ii) To estimate the gold resource of the deposit using different estimation methods; and

(iii) To compare the results and select the most appropriate method.

1.3 SCOPE OF WORK

The work is limited to parts of the concession that have already been estimated by Nkroful Mining Ltd. Other mineral resource estimation methods are used and the results compared. The geological characteristics of the deposit are studied through literature review and geological mapping in the field. Statistical and geostatistical analyses are also used to check and quantify some of the parameters obtained from the geological studies.

1.4 ORGANIZATION OF THE REPORT

The report has been divided into five chapters. Chapter One defines the problem, objectives and scope of work.

Chapter Two highlights on relevant information about the concession whilst Chapter Three reviews the literature on mineral resource estimation.

Chapter Four involves data analysis and comparison of the results and Chapter Five draws conclusions based on findings and makes recommendations for further work.

CHAPTER TWO

RELEVANT INFORMATION ABOUT NKROFUL MINING CONCESSION

2.1 INTRODUCTION

Information about the concession has been obtained from the company's feasibility report, discussions with the consulting geologist and observations in the field.

MINCONSULTS Ltd initially acquired the concession but entered into a partnership venture after reconnaissance with Union Mining Ltd of Monaco under the name "Nkroful Mining Ltd". The partnership venture was approved in 1992 by the Government of Ghana to further explore and develop the property.

At present, the alluvial deposit has been explored and evaluated but the hard rock prospecting is still in progress.

2.2 GEOGRAPHICAL ELEMENTS

2.2.1 LOCATION

The Nkroful Mining concession which covers an area of 31.46 km^2 (12.29 sq. miles) is located at Nkroful, a town in Eastern Nzema District in the Western Region of Ghana (Fig.2.1). The study area is just east of Nkroful town and can be reached by an untarred laterite road from Nkroful. Nkroful town is connected to the Ivory Coast-Ghana-Nigeria international highway at Esiama by a 4.5 km first-class road. Esiama is 55 km east of Takoradi.

FIG. 2·1 NKROFUL GOLD MINING CONCESSION (BARKO & THORTON, 1993)

2.2.2 TOPOGRAPHY & DRAINAGE

With the exception of the northern part of the concession, the terrain may be described as gentle sloping with heights reaching 20 m above sea level. The low lying areas are covered by swamps.

The northern part is hilly with narrow and dry valleys. Heights reach 30 m and above.

Streams that drain the concession include the Yiri, Bruma, Subri and Mbohari (see Figs. 2.1 and 2.3).

2.2.3 RAINFALL & VEGETATION

The concession area falls within the tropical rain forest belt. Heavy rains occur from March to July and from October to December. The rainfall pattern changes with time. However, the area is considered wet with a short dry season. The dry season periods are from January to March and from August to the middle of October.

Vegetation of tropical rain forest has been severely tampered with through farming activities leaving only shrubs and grass. The farms are mostly coconut plantations. A thin forest, made up of mainly raffia palm trees, covers the banks of the streams.

2.2.4 INFRASTRUCTURE

The Nkroful town has very good infrastructure. The town has good motorable roads and a microwave telephone that links it to the main towns and cities in the country. It is soon to benefit from the rural electrification project that is currently going on throughout the country.

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2.3 TECHNICAL ELEMENTS

2.3.1 GENERAL GEOLOGY OF THE CONCESSION AREA

The concession area falls within the Birimian System and can be considered as the southernmost extension of the Obuasi-Prestea gold belt (refer to Fig. 2.2).

The rocks in the area are steeply inclined Lower Birimian sediments which have been subjected to intense deformation. Overfolds and overthrust faults are common. The rocks generally dip to the west at high angles except where they are close to a granitic intrusion. The strike direction changes with variable dips.

An intrusion of the Cape Coast granite type covers the northern part of the concession and this has, to a considerable extent, masked the structural features of the rocks.

The Dixcove type of granite has been encountered in two prospecting pits in the concession. It has been intruded by quartz veins which give a stockwork structure of the outcrop. The host granite and quartz veins are heavily pyritized.

The country rocks are carbonaceous phyllites, sericite-schists, spotted phyllites and metavolcanics of the Birimian System.

2.3.1.1 Geology of the Deposits

Kesse (1985) describes three main types of auriferous deposits found in Ghana. They are:

i) Primary reef-vein type gold deposit;
ii) Auriferous quartz-pebble conglomerate; and
iii) Recent alluvial and elluvial deposit.

The primary reef-vein type auriferous deposit which is found in the Birimian System can occur in any of the following forms:

a) As auriferous quartz vein or reefs. This has been the most important source of gold in the country.
b) As veins and stockworks in granite porphyries which intrude the Birimian System.
c) As sulphide ores which have arisen through mineralization of the country rocks in the Birimian system.
d) As oxidized ores which have been concentrated by chemical and mechanical weathering of gold-bearing veins.
e) As pegmatite dykes associated with the granite rocks in the Birimian System.

The auriferous quartz-pebble conglomerate which is also an important source of gold in Ghana is found in the Banket conglomerate of the Tarkwaian System.

The alluvial gold deposit is by far the most common of the third type of auriferous deposits found in the country. It is believed to have been derived mainly from the primary reef-vein type gold deposit through several cycles of erosion and deposition.

Two of these three types of auriferous deposits are found at the Nkroful concession (Barko & Thorton, 1993). They are:

i) Primary reef-vein type gold deposit; and

ii) Alluvial gold type deposit.

A series of auriferous vein systems of the primary reef-vein type ore bodies have been identified in the concession. These are located near or at the contact between the Lower and the Upper Birimian successions. These auriferous vein systems include the Bakrobo vein system, Abuoso vein system, Plant site vein system and Atome vein system (Fig 2.3).

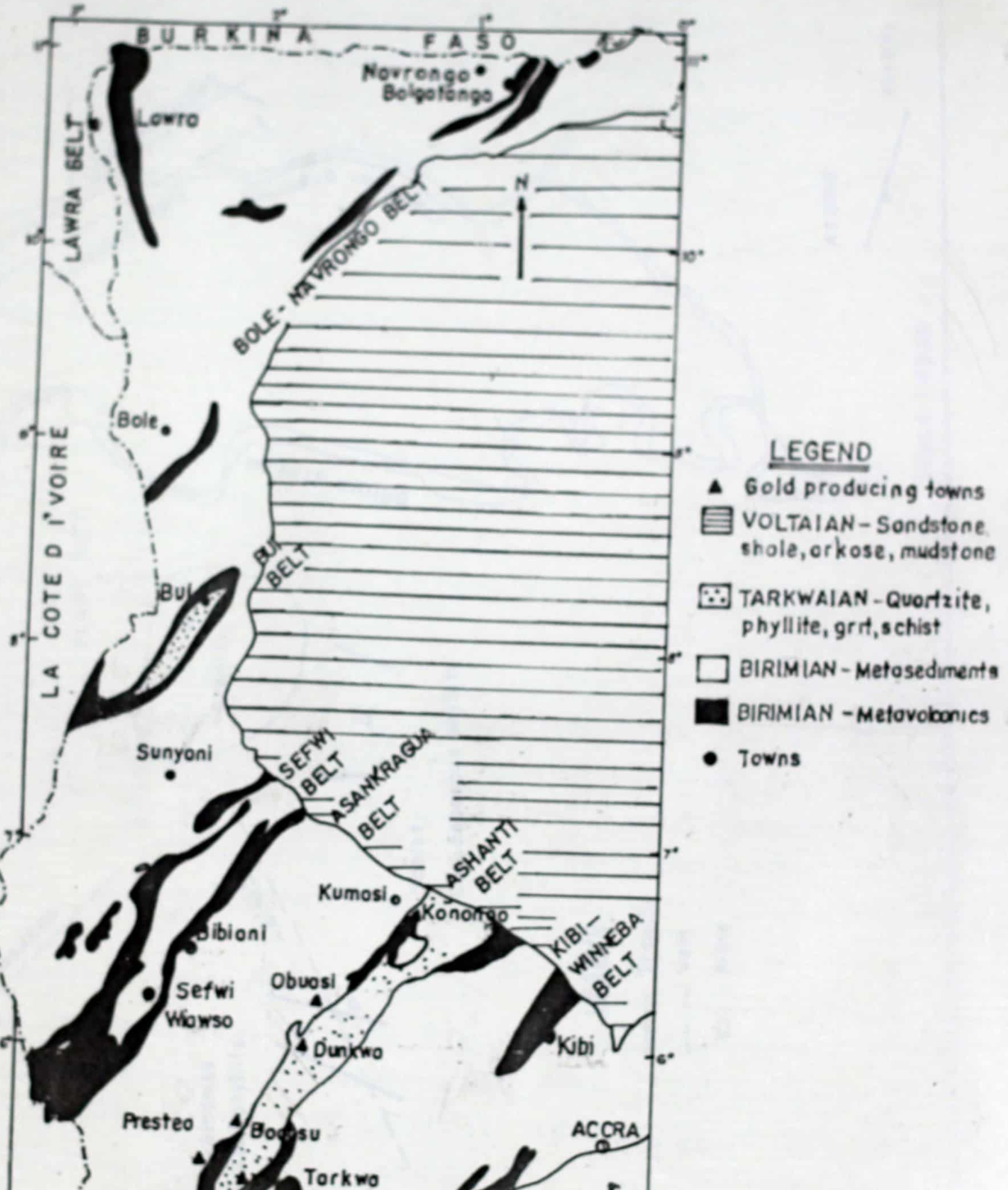


FIG. 2·2 THE GOLD BELTS OF GHANA
(Modified from Dzigbodi-Adjimah - 1993)

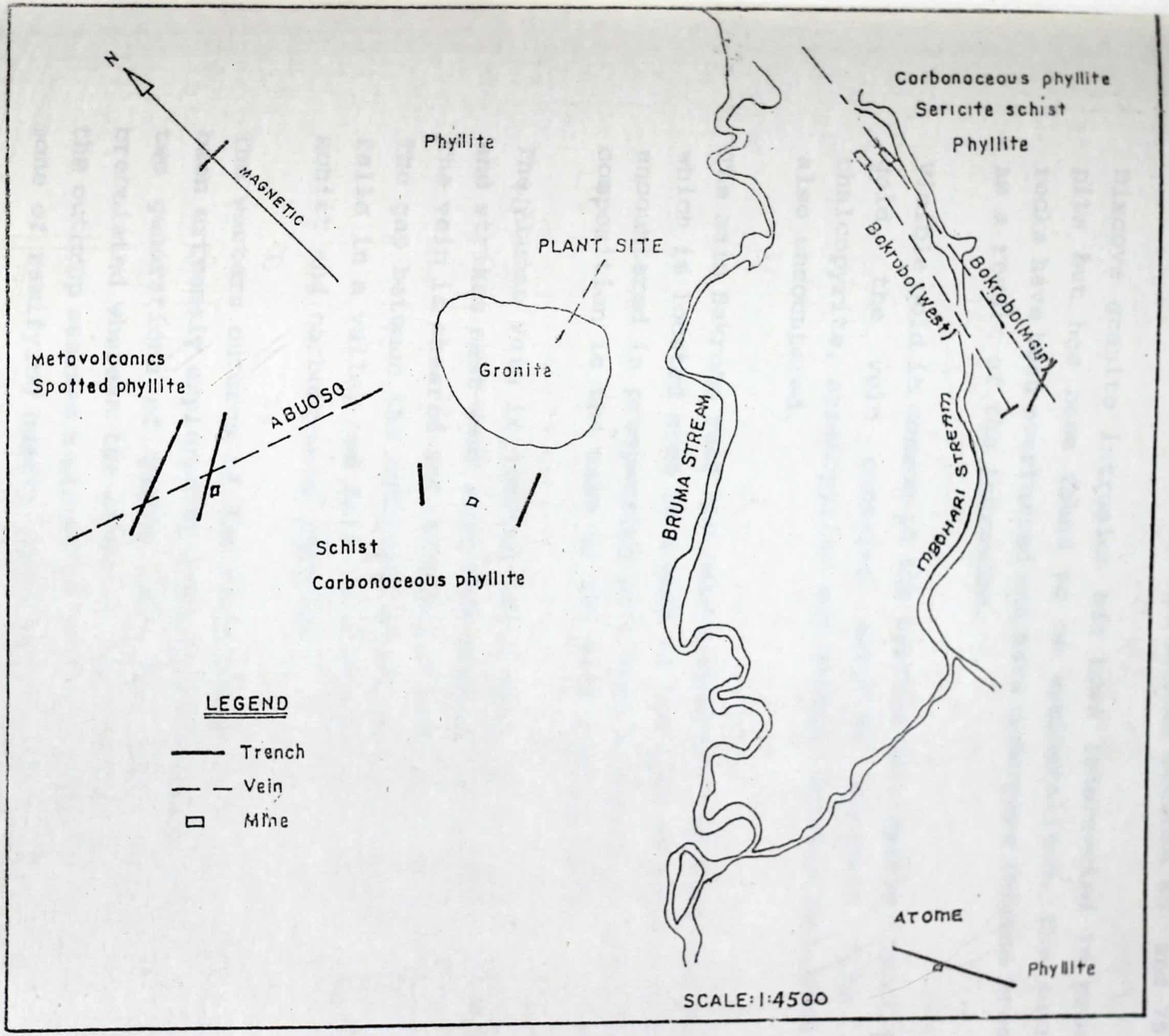


FIG. 2·3 HARDROCK GEOLOGY AND WORKINGS OF THE NKROFUL DEPOSIT
(NKROFUL MINING LTD, MODIFIED)

The Bakrobo vein system is hosted by carbonaceous phyllites, sericite-schists, phyllites and is found close to the Upper and Lower Birimian contact. It strikes north-south, about 8 m in width and dips to the west at angles between 65° and 75°.

Dixcove granite intrusion has been intersected in prospecting pits but has been found to be unmineralized. The surrounding rocks have been overturned and have undergone intense brecciation as a result of the intrusion.

Visible gold is common in the Bakrobo vein system. Apart from the gold, the vein contains sulphide minerals like pyrite, chalcopyrite, arsenopyrite and galena. Calcite and ankerite are also encountered.

The main Bakrobo reef has been traced for 300 m and the West reef which is located some 14 m west of the main Bakrobo reef has been encountered in prospecting pits over a length of 180 m. Mineral composition is the same as the main Bakrobo vein.

The Abuoso vein is located 440 m west of the main Bakrobo vein and strikes east-west with dips between 15° and 20° to the south. The vein is sheared and outcrops at two places from east to west. The gap between the outcrops is 280 m. A major part of the vein falls in a valley and this is unmineralized. The host rocks are schist and carbonaceous phyllite.

The western outcrop of the Abuoso vein which is brecciated has been extremely explored by trenches and bulk sampling. At least, two generations of quartz have been noted. One is intensely brecciated whereas the other is less brecciated. In most cases, the outcrop assumes a stockwork structure representing a confused zone of ramifying quartz veins and stringers in phyllites.

The Atome vein is exposed in a shaft. It strikes northwest-southeast with a dip of 45° to the northeast. The quartz is black-grey in colour and contains significant amounts of pyrite.

Phyllite is the host rock.

The Plant site vein is found in the immediate vicinity of the pilot plant location and it outcrops for a distance of 30 m. It is hosted by Dixcove granite which has been picked in a nearby pit. The vein is shattered and masked by boulders. The exploration of this vein is just at its initial stage.

The areas covered by alluvial deposits are shown in Fig. 2.4 The alluvial deposit discovered downstream along the Bakrobo area which is boarded by the Bruma and Mbohari streams is peculiar, for apart from the normal alluvial free gold contained in the gravel layer, the angular quartz pebbles within the auriferous gravel layer contain some substantial amount of gold.

Strangely enough, the pebbles within the auriferous gravel layer beyond the Bakrobo reef (upstream) contain no gold. Studies have however shown that the auriferous angular pebbles were derived from the Bakrobo vein system.

In general, the overburden of the alluvial deposit is composed of clay and sand, and at times layers of carbonized clay with remnants of fruit nuts. The bedrock is weathered phyllite which has turned into clay (kaolin). The formation, at times, reaches depths of about 4 m with little or no overburden. The thickness of the gravel bed is between 0.5 and 4 m. A typical section of a pit showing the various layers is shown in Fig. 2.5.

Gold is erratically distributed throughout the deposit, both in horizontal and vertical directions. The only exception is along stream channels where the gold is fairly and evenly distributed. Accessory heavy minerals are zircon, monazite and rare grains of staurolite, feldspar and tourmaline.

Apart from the stream deposits which are found in the stream beds, channels and in gravel flats adjacent to the stream beds, there are also auriferous terrace gravels formed on higher

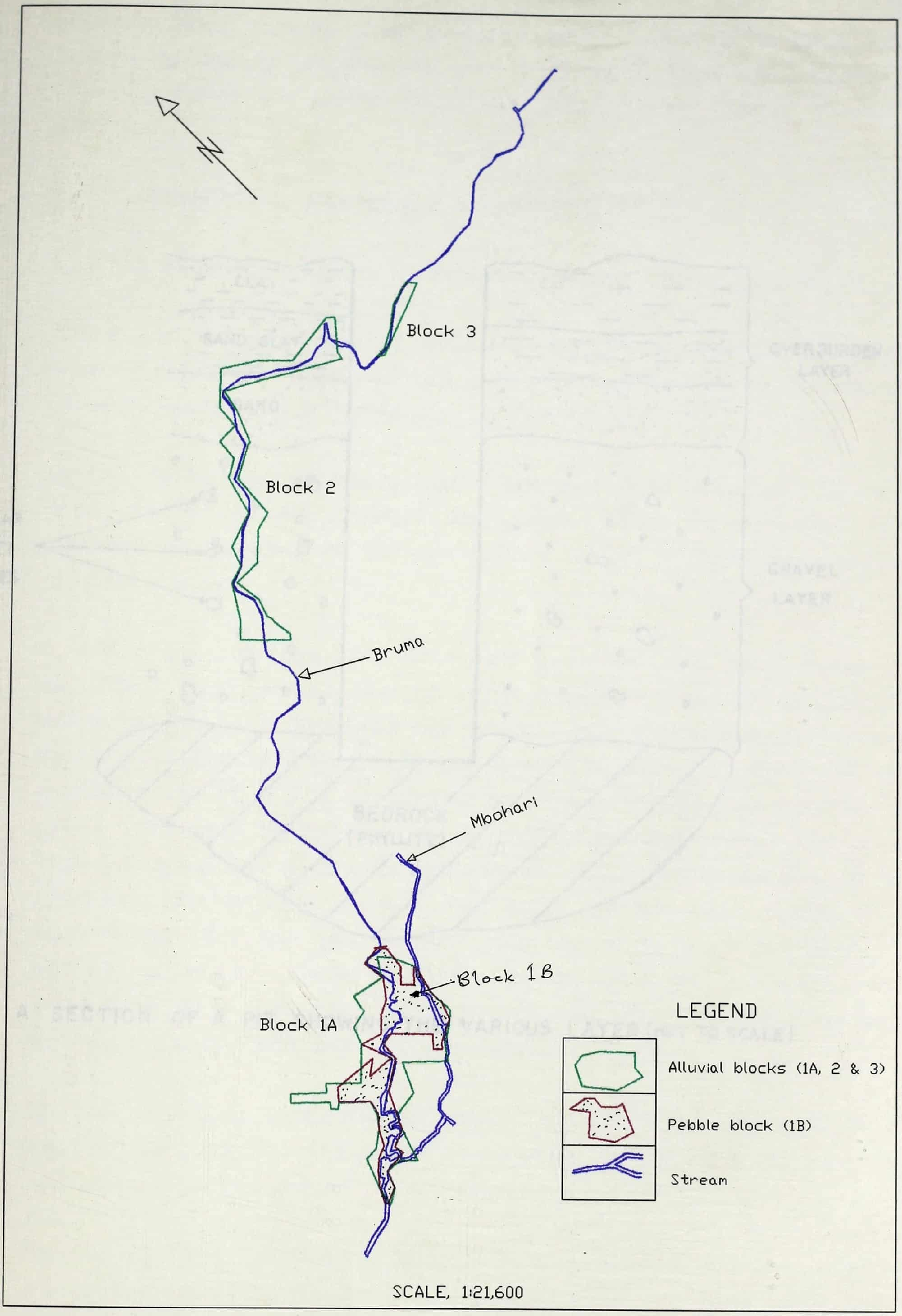
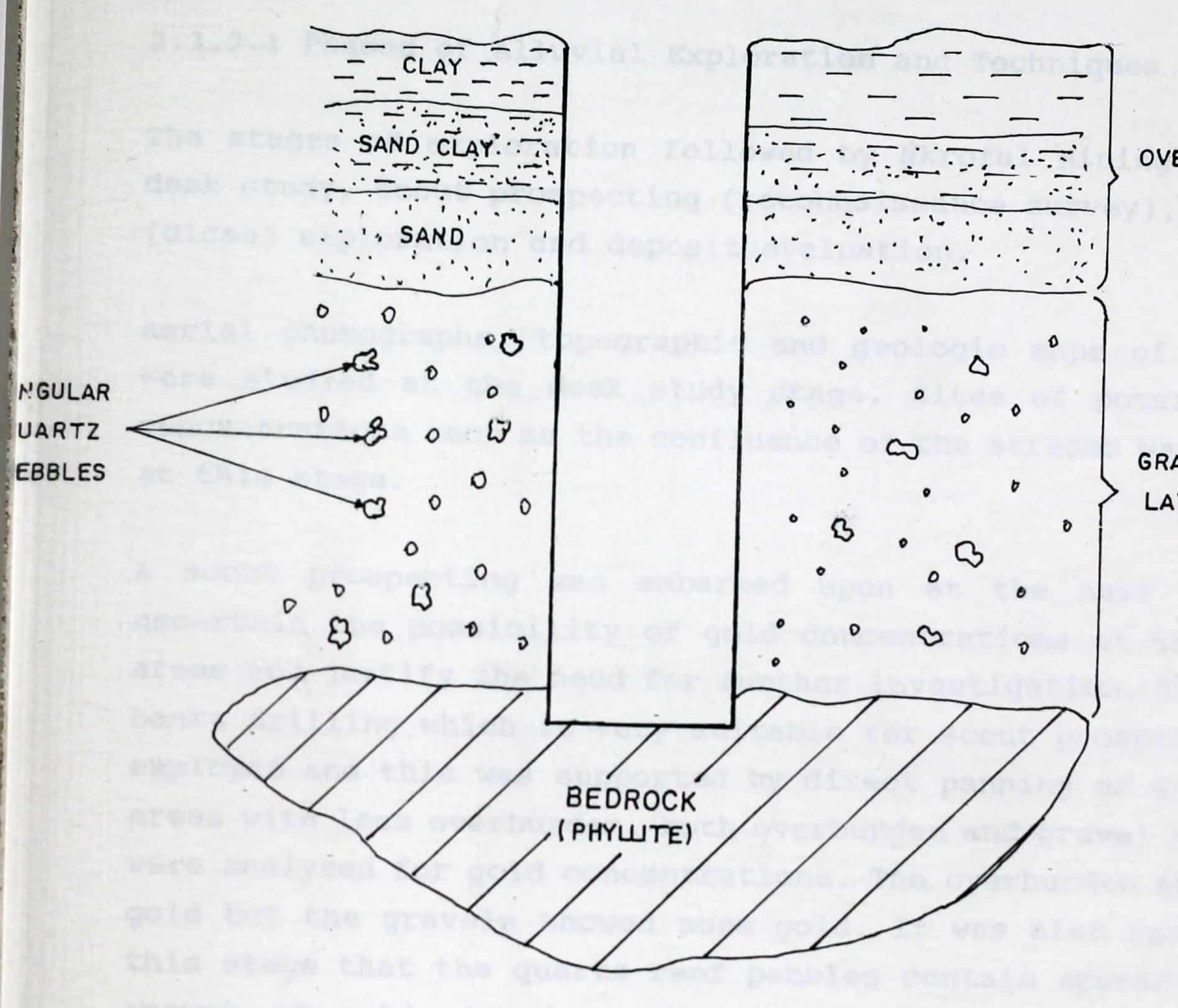


Fig 2.4 Nkroful gold concession showing the various auriferous blocks



2·5 A SECTION OF A PIT SHOWING THE VARIOUS LAYER (NOT TO SCALE) grounds even though they are residual in form. About 90% by volume of the gold bearing gravel in this type of deposit is located above the water table and will present no problem in mining.

2.3.2 EXPLORATION PROCEDURES AT NKROFUL

2.3.2.1 Phases of Alluvial Exploration and Techniques Employed.

The stages of exploration followed by Nkroful Mining Ltd were desk study, scout prospecting (reconnaissance survey), detailed (close) exploration and deposit evaluation.

Aerial photographs, topographic and geologic maps of the area were studied at the desk study stage. Sites of possible gold concentrations such as the confluence of the streams were mapped at this stage.

A scout prospecting was embarked upon at the next stage to ascertain the possibility of gold concentrations at the target areas and justify the need for further investigation. The use of banka drilling which is very suitable for scout prospecting was employed and this was supported by direct panning of gravels at areas with less overburden. Both overburden and gravel materials were analyzed for gold concentrations. The overburden showed no gold but the gravels showed some gold. It was also realized at this stage that the quartz reef pebbles contain appreciable amount of gold. Results of the scout prospecting were very promising so a detailed exploration was planned for the next stage, skipping the initial exploration stage.

At the detailed exploration stage, a grid system was laid to facilitate pitting. The cross lines were spaced at 60 m interval and pits were spaced at 30 m intervals.

During pitting, square pits of 1 m x 1 m were dug through the overburden and gravel layers into the bedrock using pick axes, shovels and mattocks. Buckets were used to hoist the material from the pits manually. Utmost care was taken to ensure uniform pit dimensions.

The problem faced during pitting was the in-rush of ground water into the pit, which eventually leads to the collapse of the walls. Pits were thus cribbed using boards to control the in-rush of water and also to stabilize the walls of the pits from caving in or collapsing.

2.3.2.2 Sampling and Logging of Pits

The overburden material was not sampled because it contained no gold but the gravel layer was sampled and the volume noted. As a result of the cribbing, actual volume of pits was reduced to 0.85 m x 0.85 m x thickness. Oversize materials (quartz pebbles) were also sampled as they were mineralized.

Pitting was allowed to continue down to 0.5 m below the gravel layer into the bedrock and samples collected from the bedrock were assayed for any gold value.

Bulk sampling was also carried out on the alluvial material to test the grade of the ore and also assist in developing volume beneficiation flow sheet.

The pits were carefully logged and the contacts of different layers identified and recorded for interpretation of geological sections, structures and grade computations.

2.3.2.3 Assaying Procedures

Materials excavated from pits were sluiced and the end product (heavy mineral concentrate) collected. Tailings obtained were panned several times to ensure that there was no speck of gold in them. Concentrates obtained were washed with solution of soda ash and detergent to facilitate amalgamation. Concentrates were amalgamated and the returned gold obtained weighed in grams. This method recovered only the free gold.

Part of the samples were sent to S.G.S Laboratory Services (Ghana Ltd) and UST School of Mines Laboratories at Tarkwa for analysis. Some of the samples were also sent to Cambourne School of Mines in U.K. Sample analysis in these cases were for the recovery of both free and unliberated gold.

2.3.2.4 Mineral Resource Estimation

Exhaustive studies were conducted to determine the densities for the alluvial material and quartz pebble material. The alluvial material has a density of 2.0 g/m^3 and the quartz pebble material has a density of 2.2 g/m^3.

With the known weight of gold in grams (g) and volume of gold from each material in cubic meters (m^3), the grades were calculated in grams per cubic meter (g/m^3) for each pit. When multiplied by the density, grades were obtained in terms of grams per tonne (g/t).

The triangular method was used for the mineral resource estimation by Nkroful Mining Ltd. The reason attributed to this was that even though it is the longest in computations as compared to other geometrical methods it is the most accurate. The value obtained is from the average of the three pits and the effect of these pits is localised within the triangles and thus provide better interpolated estimates. This method of estimation has been discussed fully in the next chapter.

CHAPTER THREE

LITERATURE REVIEW OF MINERAL RESOURCE ESTIMATION

3.1 MINERAL RESOURCE ESTIMATION

Estimation of ore reserves involves not only evaluation of the tonnage and grade of a deposit but also consideration of the technical and legal aspects of mining the deposit, of beneficiating the ores, and of selling the product (Noble, 1992). This thesis addresses only the aspects of reserve estimation that include determination of the tonnage, size, shape and location of minerals so the term " Mineral Resource" is used (refer to Annels, 1991 and Annon, 1992 for further reading on this subject). Mineral resource as defined by the Australasian Institute of Mining and Metallurgy (AIMM) and the Australian Mining Industry Council (AMIC) (Annon, 1992) is an identified in-situ mineral occurrence from which valuable or useful minerals may be recovered.

The basic parameters of sample points needed for mineral resource estimation are:

i) Grade;

ii) Thickness and area; and

iii) Tonnage factor.

Grade is a qualitative indicator of values and their distribution in the deposit. Grade on its own can only be used where the support is constant i.e. where the thickness and areas to which the grade is assigned is constant. Where the support is variable then there must be some form of weighting to determine the average grade.

Thickness and area are quantity indicators of form, size and volume of a mineral body. The area together with the thickness gives the volume of the deposit.

Tonnage factor converts volume to tonnage. A common formula used in computing tonnage is:

$$T = \frac{V}{F}$$

or

$$T = Vf$$

where T is tonnage, V is volume, F is volume-tonnage factor expressed in cubic meter per ton and f is tonnage-volume factor expressed in weight-units per cubic meter (also known as specific gravity).

There are various estimation methods and these can be broadly put under the following:

(1) Conventional or traditional methods;

(2) Classical statistical methods; and

(3) Geostatistical methods.

3.2 CONVENTIONAL OR TRADITIONAL METHODS

These methods are based on geometry and distance relationship among samples. They can be grouped into:

1. Geometrical methods;
2. Inverse distance weighting methods; and
3. Contouring methods.

According to Popoff (1966) there are three basic principles employed by the conventional methods in computing mineral resources. They are:

i) The rule of gradual changes;

ii) The rule of nearest points or equal influence; and

iii) The rule of generalization.

The rule of gradual changes assumes that all sample elements of a mineral body change gradually and continuously as a linear function along a straight line connecting two adjacent sample points. Apart from grade and weight factors, this rule can be applied to other parameters of a mineral body like volumes and tonnages. Triangular method is based on this principle.

According to the rule of nearest point or equal influence the value of any point between two samples is constant and equal to the value of the nearest sample. Put in another way, the rule assumes that the value of a sample extends halfway to any sample. The polygonal and the rectangular block methods are based on this principle.

The rule of generalization is known as the empirical method and, in its extreme, as the rule of thumb. Usually, this rule is arbitrarily applied as a matter of judgement reflecting past experience and opinion. An example is adapting a definite weighting factor for resource computations from other similar mineral deposits or projecting continuity of mineralization beyond the outermost workings.

The problems associated with the rule of generalization are related to subjectivity and the difficulty of duplicating results by another examiner.

3.2.1 GEOMETRICAL METHODS

This method of estimating mineral resources includes polygonal, triangular, rectangular and sectional methods. Polygonal, rectangular, and triangular methods are suited for tabular and low dipping ore bodies whereas the sectional method is particularly applicable to non tabular ore bodies or those with somewhat irregular outline which have been evaluated by drilling on mine sections (Annels, 1991).

3.2.1.1 Polygonal & Rectangular Methods

The polygonal method is used where drill-holes are randomly distributed (i.e. not on a regular grid). Polygons are constructed around each drill-hole with influence of each hole extending half way to the adjacent hole.

Grade and thickness of each polygon are assigned to the hole at its center. The area of each polygon is determined and then multiplied by its thickness to determine the volume. The average grade is calculated by using a method known as the metre-% method. This method is given by the formula:

$$G = \frac{\sum_{i=1}^{n} G_i x_i}{\sum_{i=1}^{n} x_i}$$

where,

G_i = variable grade
x_i = variable support
G = weighted average grade

According to Reedman (1979), where there is a strong positive correlation between G_i and x_i variables, there is an overestimation of grade and where there is a negative correlation, there is an underestimation of grade by the metre-% method. This is a disadvantage using this method.

Where the sample data is on a regular grid or regular offset grid, rectangular blocks can be fitted to the drill-holes as shown in Fig. 3.1 (a). Grade calculations follow that of the polygonal method since they are very similar.

3.2.1.2 Triangular Method

This method divides the area into triangles by drawing lines between the holes. The thickness and grade for each triangle is determined as a weighted average of the value in the holes at the corners of the triangle. The overall average grade can be determined by using the meter-% method.

The average thickness can be determined as a single mean of the three thicknesses or as a weighted thickness according to the size of the included angle at each corner.

Even though the triangular method overcomes the under and over evaluation, it does not allow extrapolation of the mineralization

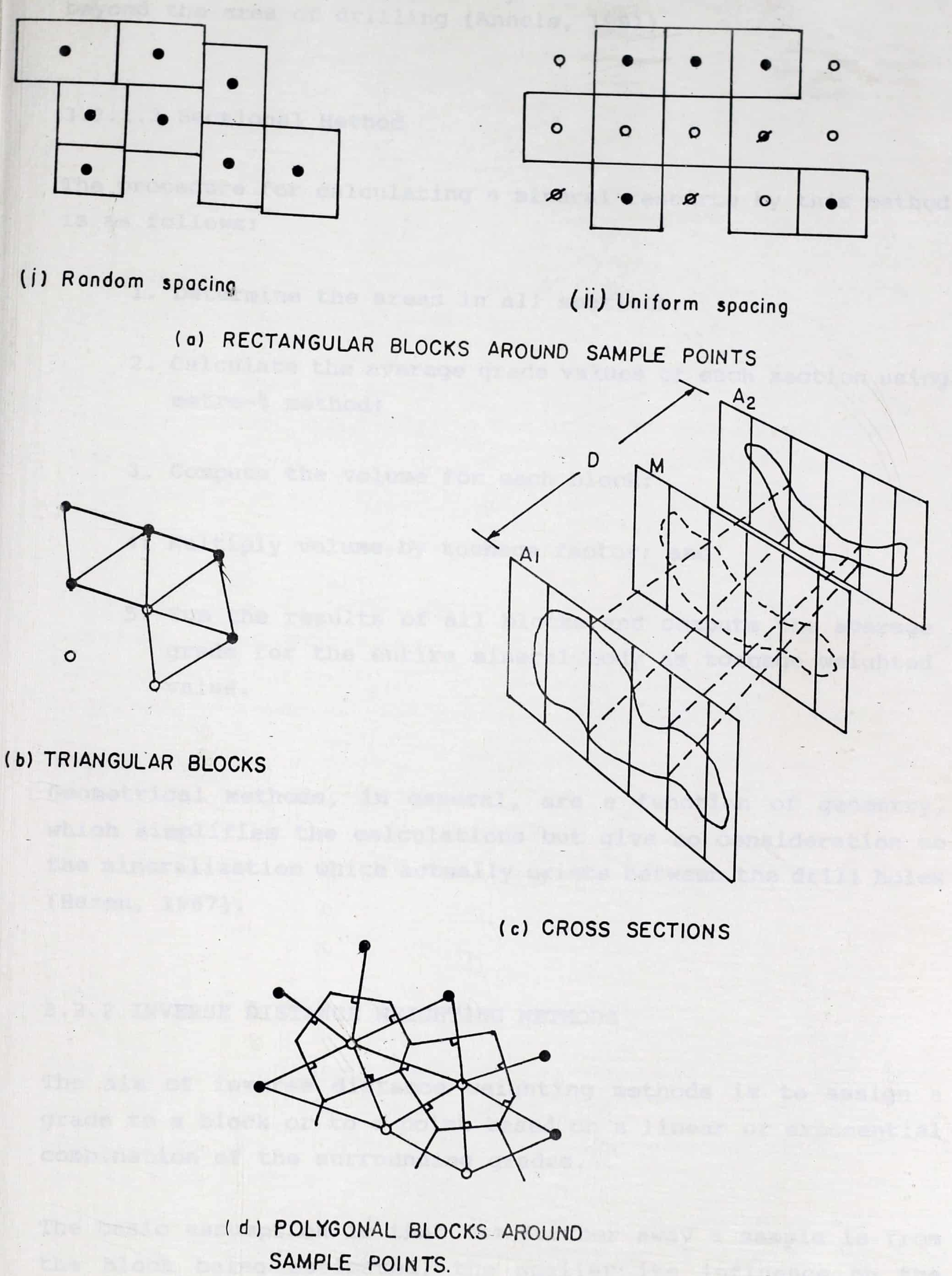


FIG. 3.1 GEOMETRIC PATTERNS USED IN ASSIGNING AREAS OF INFLUENCE TO SAMPLE POINTS. (After Hazen, 1967)

beyond the area of drilling (Annels, 1991).

3.2.1.3 Sectional Method

The procedure for calculating a mineral resource by this method is as follows:

1. Determine the areas in all sections;
2. Calculate the average grade values of each section using metre-% method;
3. Compute the volume for each block;
4. Multiply volume by tonnage factor; and
5. Sum the results of all blocks and compute the average grade for the entire mineral body as tonnage weighted value.

Geometrical methods, in general, are a function of geometry, which simplifies the calculations but give no consideration to the mineralization which actually exists between the drill holes (Hazen, 1967).

3.2.2 INVERSE DISTANCE WEIGHTING METHODS

The aim of inverse distance weighting methods is to assign a grade to a block or to a point based on a linear or exponential combination of the surrounding grades.

The basic assumption is that the further away a sample is from the block being estimated, the smaller its influence on the block. Grade thus becomes a function of inverse distance as shown in equation below:

$$G = \frac{G_i}{f(d)} \qquad for \quad i = 1,2,3,...n$$

where G is an unknown grade influenced by G_1, a known grade. G and G_1 are separated by a distance d.

The problem that is faced most often is the inverse power of distance relationship between the grade to be estimated and the surrounding grades in a particular deposit. A method for determining the inverse power of a distance function is discussed below.

This method uses nearby samples to compute the grade at the location of another sample. The computed grade is then compared to the sample grade at that location. Various powers for inverse distances are used to compute this grade from the equation:

$$G = \frac{\sum_{i}^{n} G_i/d_i^m}{\sum 1/d^m}$$

where m is the power being considered.

The variance or error between the computed grade and grade assigned to the location is accumulated for all holes for each value of m investigated. The lowest error accumulation should be the best weighted average model for the deposit. It has been found that the inverse square distance is generally the most suitable distance interpolation function for mineral deposits (Barnes, 1980).

Normally, samples falling within a specified search area or volume are weighted in this way. The search area could be circular or ellipsoidal depending on the trend of mineralization in the deposit.

3.2.3 CONTOUR METHODS

This method involves contouring samples to give areas and volumes of differing grades. The method assumes the values are continuous between drill-holes of similar grade. Annels (1991) has outlined four main methods used in calculating mineral resources by contouring. They are:

1. The grid superimposition method;

2. Moving window method;

3. Graticular method; and

4. Metal accumulation method.

For the description of the various methods, refer to Annels (1991).

3.3 CLASSICAL STATISTICAL METHODS

Statistical methods can be used to determine the overall grade of a deposit (i.e. mean, median and geometric mean) depending on the frequency distribution. They, however, require that the sample values be far apart enough to be independent variables (Annels, 1991) which is rarely the case in gold mineralization. It can still be very useful especially at the early stages of exploration where sample spacing are far apart.

Statistics, is essentially, the study of variability which is an important study in gold mineralization for resource estimation.

According to Noble (1992), basic statistical studies help in:

1. Detecting high grade or low grade outliers;
2. Differentiating of complex (eg multimodal) grade distribution into simple (eg unimodal) distributions for resource modelling; and
3. Identifying of highly skewed and / or highly variable distributions that will be difficult to estimate.

In addition, it forms the basis for geostatistical analysis (Annels & Boakye, 1990). It must therefore be incorporated in any mineral resource estimation analysis.

The nature of frequency distribution of gold mineralization most often take the form of a two parameter log-normal distribution. Normal and three parameter log-normal distributions are also encountered.

3.3.1 NORMAL DISTRIBUTION

For a normal distribution, the following conditions must be satisfied (Annels, 1991):

Mean ≈ Median

Skewness ≈ 0

Kurtosis ≈ 3

Chi-square test and Coefficient of Variation (COV) can also be used to test for normality of a distribution. For normality of a population, chi-square value must be less than the book value.

Coefficient of Variation is often used to describe the variability of assays in a deposit. According to Annels (1991), Koch and Link (1970) suggest that COV should be less than 0.5 for a normal distribution. Larger values of COV indicate either log-normality or an erratically distributed data set. However, this value is very much a function of length, for variance of assays decreases as sample length increases due to an averaging out, or smoothing, of localized high grade patches by increased sample volume.

If the assay frequency of the distribution is unimodal, indicating a set of assay data from a single population, the statistics (mean, variance and standard deviation) of the distribution are computed. If the assay frequency distribution is multimodal, indicating a more than one population from a set of assay data, screening is required to break the data down into unimodal distribution before calculating the statistics (Hazen, 1967).

Screening must also detect high grade and low grade outliers and find out whether they are true reflections of the mineralization within the deposit.

3.3.2 TWO PARAMETER LOG-NORMAL DISTRIBUTION

Where a population is positively skewed, a log-transformed plot of the data is required to check if the population becomes normalized. If it becomes normalized, then the distribution is described as a two parameter log-normal population (the parameters being log mean and log variance). Log-normality can be tested by plotting a log-probability diagram which should produce a straight line graph (Annels, 1991).

The log-normal statistics is related to the normal statistics as follows (Noble, 1992):

$$\text{mean} = e^{(\alpha+\beta)}$$

$$COV = (e^{\beta^2} - 1)^{0.5}$$

$$\text{standard deviation} = (e^{(\alpha+\beta)})(e^{\beta^2} - 1)^{0.5}$$

where α = mean of the logarithms of raw data.

β = variance of the logarithms of raw data.

3.3.3 THREE PARAMETER LOG-NORMAL DISTRIBUTION

If a log-probability plot produces a gentle curve, then a three-parameter log-normal population is suspected. The third parameter which is called the additive constant λ can be calculated as shown from the equation below (Rendu, 1981):

$$\lambda = \frac{(x_{50}{}^2 - x_{90}.x_{10})}{(x_{10} + x_{90} - 2x_{50})}$$

where x_{10}, x_{50}, x_{90} are 10, 50, and 90 percentile values respectively.

The additive constant λ is added to the raw data and the results log transformed giving a new population $\ln(x_i + \lambda)$ which, when plotted on a log-probability graph should produce a straight line.

The procedure explained for the two-parameter population is then repeated for this population with λ being subtracted from the final results.

Where an assay population is small and log-normal, and the raw data has a high coefficient of variation, Sichel's t estimator can be used to estimate the mean. It is obtained from the formula:

$$\text{mean} = m \times f(\beta)_n$$

where $m = e^a$ the geometric mean of the log-transformed data.
β = variance of the log-transformed data
n = number of the samples

$f(\beta)_n$ can be obtained from tables. This is achieved by interpolating between values of β and n.

3.4 GEOSTATISTICAL METHODS

Geostatistics is the study of the spatial relation between sample values, which could be grade, ore width or other geological parameter. It is based on Matheron's theory of regionalized variables.

Barnes (1980) defines a regionalized variable as a variable whose magnitude depends on neighbouring values which are distributed in two or three-dimensional space. This concept takes into consideration the position as well as the magnitude of a sample in the deposit.

Some deposits show excellent continuity of grade over considerable distances while others are highly erratic. Sample value comprises two elements. They are:

i) The regionalized element which is a function of its position within the ore zone; and

ii) The random element of a sample which is independent of the location of nearby samples. Such samples can be treated statistically as such.

The principle of geostatistics is a major advancement in the mineral evaluation process because it provides a sound theoretical and practical basis for quantifying the geological concepts of:

(i) area of influence of a sample;

(ii) the continuity of or lack of continuity of mineralization within the ore body; and

(iii) anisotropic or isotropic conditions within the deposit.

The objective of geostatistics is to estimate the most likely values of blocks of ore, or the values of the whole deposit and to estimate the errors of such estimates.

The basic tool of geostatistical analysis is the semi-variogram.

3.4.1 THE SEMI-VARIOGRAM

The semi-variogram permits the quantification of the geological concepts mentioned previously. Blais and Carlier (1968) defines it as a mathematical function expressed by a curve which gives the degree of dispersion of assay values. This mathematical function should account for the spatial structure of the given variable as well as the random fluctuations which are represented in a mineral deposit.

A semi-variogram is computed by averaging the square difference between pairs of samples at specified distances or lags apart. It is represented by the function:

$$\gamma(h) = \frac{\sum (f(x+h) - f(x))^2}{2n}$$

where f(x) is the grade at sample point x, f(x+h) is the grade at sample point x+h, h is lag distance, and n is the number of data pairs counted at each directional lag. The semi-variogram function γ(h) computed for different lag distances provides what is known as an experimental semi-variogram. A typical experimental semi-variogram is shown in Fig 3.2.

The nugget variance is identified as the y-intercept of the semi-variogram curve and represents random and short distance variability factors such as sampling error, assaying error, and erratic mineralization.

Small nugget variance reflects good sampling and analytical techniques and locally continuous mineralization whereas high nugget variance is the reverse.

The range is the distance at which the semi-variogram levels off to a constant value and corresponds to the area of influence of sample. Beyond the range, samples no longer correlate with other samples. The sill is usually equal to statistical sample variance.

When the slope and range of a semi-variogram vary in different directions, anisotropic conditions exist.

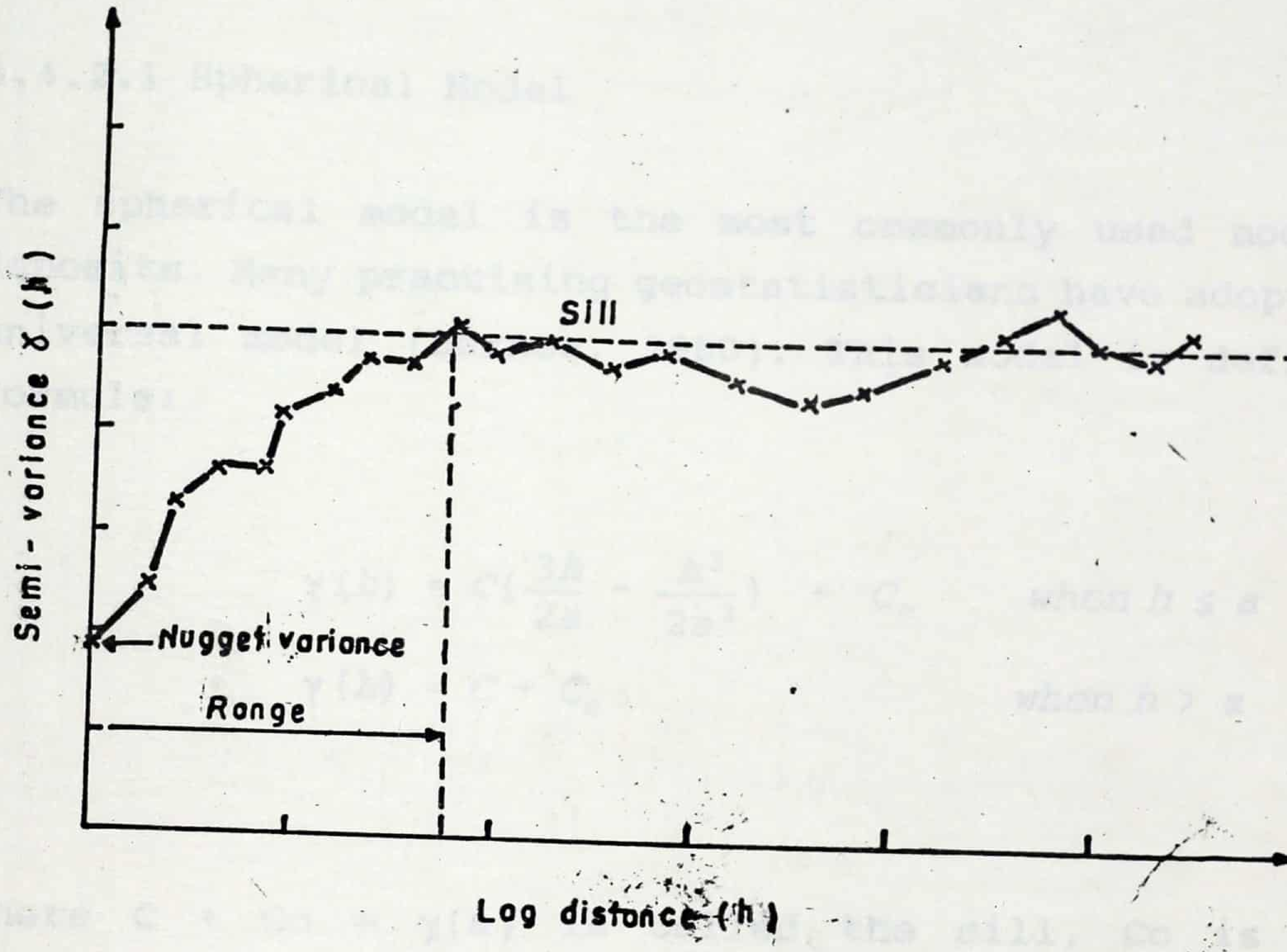


FIG. 3.2 TYPICAL EXPERIMENTAL SEMI-VARIOGRAM

3.4.2 SEMI-VARIOGRAM MODELS

Several mathematical models have been proposed to represent a probability function that describes the behaviour of various experimental semi-variograms. These mathematical models describe the spatial behaviour of the regionalized variables in an ore body. The spherical model is by far the most widely used. The other models, even though not often used, still have their applications in some deposits. Several models have been summarised in Table 3.1.

3.4.2.1 Spherical Model

The spherical model is the most commonly used model for ore deposits. Many practising geostatisticians have adopted it as a universal model (Barnes, 1980). This model is defined by the formula:

$$\gamma(h) = C\left(\frac{3h}{2a} - \frac{h^3}{2a^3}\right) + C_o \qquad \text{when } h \leq a$$

$$\gamma(h) = C + C_o \qquad \text{when } h > a$$

where C + Co = $\gamma(\alpha)$ is called the sill, Co is the nugget variance, and "a" is the range or maximum zone of influence.

A practical technique for determining the model is to draw a tangent through the first two or three points (this defines Co and C on the γ(h) axis) to intersect the sill level at a point 2a/3 . Range "a" in this case represents the point at which the curve reaches the sill (Annels, 1991).

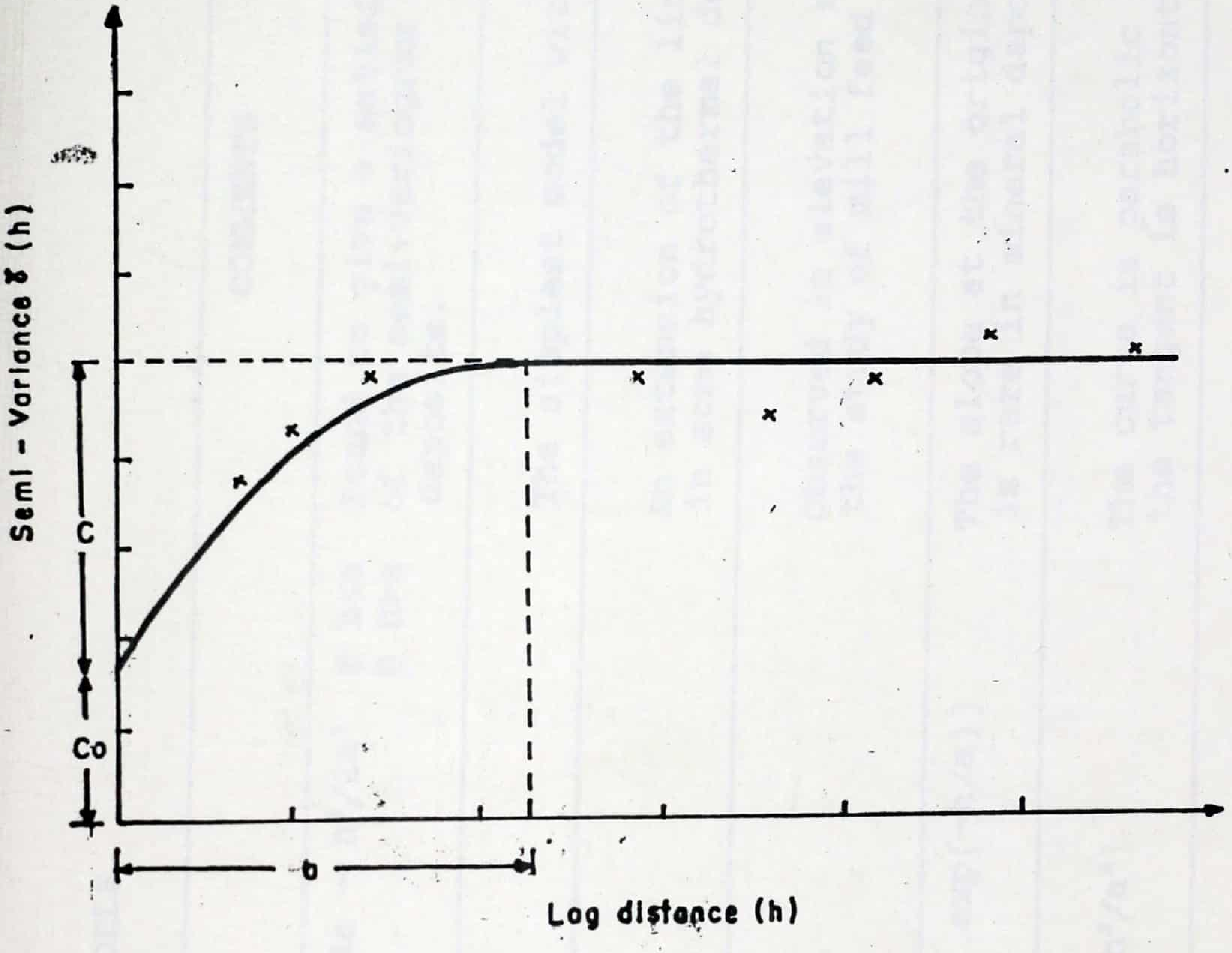


FIG. 3.3 SPHERICAL SEMI - VARIOGRAM MODEL

TABLE 3.1 : SEMI VARIOGRAM MODELS

MODEL TYPE	EQUATION	COMMENTS
SPHERICAL	$(h) = Co + C(3h/2a - h^3/2a^3$ @ $h \leq a$ $= Co + C$ @ $h > a$	Found to give a satisfactory representation of the semi-variogram of many different deposits.
LINEAR	$(h) = Ah + B$	The simplest model without a range.
DE WIJSIAN	$(h) = A\ln(h) + B$	An extension of the linear model and found in some hydrothermal deposits.
ah MODEL	$(h) = ah$	Observed in elevation semi-variogram or in the study of mill feed variability.
EXPONENTIAL	$(h) = Co + C\ [1 - \exp(-h/a)]$	The slope at the origin is C/a. This model is rare in mineral deposits.
GAUSSIAN	$(h) = C[1 - \exp(-h^2/a^2]$	The curve is parabolic near the origin and the tangent is horizontal at the origin.

TABLE 3.1 CONT'D

MODEL TYPE	EQUATION	COMMENTS
PARABOLIC	$(h) = 1/2\ (a^2h^2)$	Observed when there is a linear drift.
RANDOM	$(h) = S^2$	This model has no continuity, indicating that there is a high degree of randomness in the variable distribution. (h) is then equal to the statistical variance.
HOLE EFFECT	$(h) = C[1 - (\sin(ah)/ah)]$	This model has a periodic behaviour and it is observed when there is a succession of rich and poor zones.

The symbols stands for the following:

Co = nugget variance; C = transition variance; h = distance between sample pairs;

a = range; A,B = constants; = slope; and S^2 = statistical variance of the same population.

Source: Ashong (1990)

3.4.3 PRACTICAL CONSIDERATIONS IN COMPUTING SEMI-VARIOGRAMS

Different sample support normally produces different nugget variances (Barnes, 1980). It is therefore necessary to have a uniform sample support for semi-variogram computation and this demands compositing the samples to provide equal sized data. Where the deposit has a varying thickness like in vein and stratigraphic deposits, metal accumulation can be calculated. This is obtained by multiplying the grade by the length. The grade of an ore block estimated by the accumulation variable can be obtained by dividing the estimated accumulation for the block by its estimated average thickness.

In practice, semi-variograms are constructed in 3 or 4 principal directions to check for anisotropic conditions. Tolerance distance and angle of regularization are also given in semi-variogram construction so that samples that are unevenly spaced and deviate a little from the principal direction of construction can be captured.

Other considerations that can aid in computing semi-variograms are (Noble, 1992):

a) At least 30 pairs of samples are required to compute a valid semi-variogram. More pairs produce a more stable semi-variogram.

b) Semi-variograms must be computed within continuous zones of mineralization and not across contacts between different geologic domains.

c) The maximum lag distance used should be less than one-half the length of the mineralized zone in the direction of the semi-variogram.

d) The distance increment should be approximately equal to the average spacing between samples in the direction of the semi-variogram.

f) Samples must be obtained by the same or similar methods.

3.4.4 CROSS VALIDATION

In practice, the preliminary selected semi-variogram model is fitted by inspection. The cross validation process is used to check the validity of the parameters derived from the fitted semi-variogram.

Essentially, each sample in the data set is removed in turn and its value estimated from neighbouring sample values. The kriging technique is employed in this estimation using the fitted semi-variogram parameters.

Requirements of a good model should satisfy the following conditions (Mireku-Gyimah, 1990):

i) Equality of the mean of kriged estimates and the mean of actual values; and

ii) A zero mean difference between actual and estimated values.

The actual and kriging estimated values must meet or approximate the above requirements else the parameters of the initially selected model may have to be varied with the aim of meeting the set out conditions.

3.4.5 KRIGING

The evaluation of individual blocks (or panels) of an ore body can be accomplished by using the technique of kriging. It calculates the best estimates for these blocks by developing a set of linear weighting coefficients that are applied to sample values in the vicinity of the blocks. The kriging technique takes the position of the sample with respect to the block and the continuity of mineralization into account.

The kriging estimator is defined by the equation:

$$Z^*_v = \sum_{i=1}^{n} a_i x_i \ldots\ldots\ldots\ldots (1)$$

where

Z^*v = the best estimated value of the true but unknown value Zv of the block of volume V,

x_i = the value of a sample in the vicinity of the block,

a_i = the weighting coefficient assigned to x_i, and

n = the selected number of samples in the vicinity of the block to be used in the estimation.

For Z^*v to be the Best Linear Unbiased Estimator (BLUE), the following conditions must be satisfied:

i) The estimation variance must be a minimum, i.e. $\sigma_E^2 = E(Z^*v - Zv)$ must be minimum; and

ii) The estimation must be unbiased, i.e. $\sum a_i = 1$ (for i = 1,2,...n).

The estimation variance as expressed by Rendu (1981) in terms of semi-variance is given by the equation:

$$\sigma_E^2 = 2\sum_{i=1}^{n} a_i\gamma(x_i, v) - \sum_{i=1}^{n}\sum_{j=1}^{n} a_i a_j \gamma(x_i, x_j) - \gamma(v, v) \ldots\ldots\ldots (2)$$

where

$\gamma(x_i,v)$ = the average semi-variance between sample x_i and every point in the block,

$\gamma(x_i,x_j)$ = the average semi-variance between all pairs of points in the sample set, and

$\gamma(v,v)$ = the average semi-variance between all pairs of points in the block.

To minimize the error of estimation, the estimation function (Equation (2)) is differentiated with respect to the weighting coefficients and the derivative set to zero. However, when there is a constraint ($\Sigma a_i - 1 = 0$), a Langrange multiplier λ is introduced and the function defined as follows:

$$G(a_i \ldots a_n, \lambda) = \sigma_E^2 + 2\lambda(\sum a_i - 1) \ldots\ldots\ldots\ldots\ldots (3)$$

or

$$G(a_1, a_2, \ldots a_n, \lambda) = 2\sum_{i=1}^{i=n} a_j\gamma(x_i, v) - \sum_{i=1}^{n}\sum_{j=1}^{n} a_i a_j \gamma(x_i, x_j) - \gamma(v, v) \quad \ldots (4)$$

The minimizing conditions become:

$$\frac{\delta G}{\delta a_i} = \sum_{i=1}^{n} a_j \gamma(x_i, x_j) + \lambda - \gamma(x_i, v) = 0 \quad \text{for } i = 1,2,\ldots.n \text{ -(5)}$$

$$\frac{\delta G}{\delta \lambda} = \sum_{i=1}^{n} a_i - 1 = 0$$

This is known as the kriging system of equation. When the first n equations are multiplied by a_i ($i = 1,2,3,\ldots.n$ respectively) and the results summed up, the following equation is obtained:

$$\sum_{i=1}^{n}\sum_{j=1}^{n} a_i a_j (x_i, x_j) + \lambda = \sum_{i=1}^{n} a_i \gamma(x_i, v) \ldots\ldots\ldots\ldots\ldots\ldots (6)$$

Substituting Equation (6) into Equation (2), the kriging estimation variance σ_k^2 can be expressed as:

$$\sigma_k^2 = \sigma_E^2 = \sum_{i=1}^{n} a_i \gamma(x_i, v) + \lambda - \gamma(v, v) \ldots\ldots\ldots\ldots\ldots\ldots (7)$$

The kriging system of equation and kriging estimation variance can be expressed in a matrix form as:

$$[S]\,[A] = [D] \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (8)$$

with the solution

$$[A] = [S]^{-1}[D] \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (9)$$

where

$$S = \begin{bmatrix} \gamma(x_1,x_2) & \gamma(x_1,x_2)\ldots & \gamma(x_1,x_n) & 1 \\ \gamma(x_2,x_1) & \gamma(x_2,x_2)\ldots & \gamma(x_2,x_n) & 1 \\ \cdot & \cdot & \cdot & \cdot \\ \cdot & \cdot & \cdot & \cdot \\ \cdot & \cdot & \cdot & \cdot \\ \gamma(x_n,x_1) & \gamma(x_n,x_2)\ldots & \gamma(x_n,x_n) & 1 \\ 1 & 1\ \ldots & 1 & 0 \end{bmatrix}$$

$$A = \begin{bmatrix} a \\ a \\ \cdot \\ \cdot \\ a \\ \lambda \end{bmatrix} \qquad D = \begin{bmatrix} \gamma(x,v) \\ \gamma(x,v) \\ \cdot \\ \cdot \\ \gamma(x,v) \\ 0 \end{bmatrix}$$

All γ terms can be derived from the semi-variogram.

CHAPTER FOUR

DATA ANALYSIS AND DISCUSSION OF RESULTS

4.1 INTRODUCTION

The objective of this thesis is to study the geological characteristics of the Nkroful alluvial deposit and to select an appropriate mineral resource estimation method through a comparative study of various estimation methods to assist Nkroful Mining Ltd in estimating its gold resource potential.

The concession area, which falls within the Birimian System and lies at the southernmost extension of the Obuasi-Prestea gold belt, has both hard rock (vein type) and alluvial gold mineralization. The study was focused on the alluvial deposit.

Based on the geological characteristics of the deposit and sample grade values, four blocks were demarcated for the resource estimation. The boundaries of these blocks were digitized using AUTOCARD software. The essential information (i.e sample position, overburden, gravel thickness and assay values) needed for the resource estimation were entered into a Personal Computer (PC) using LOTUS 123 software.

The data format was structured to suit GEOEAS software which was used for statistical and geostatistical analysis. The data format was also structured to suit SURPAC software which was used for triangular estimation analysis. The rest of the analysis for the other methods was carried out manually but with the aid of a scientific calculator.

The data in the computer were validated by cross checking them with the original data to ensure accuracy.

4.2 GEOLOGICAL ANALYSIS

The geological characteristics of the deposit were studied through literature survey from the mine and was followed up with basic geological mapping, trench mapping and pit logging in the field. Geological sections were also studied.

As described in Chapter Three, the overburden is made up of clay and sand, and at times a layer of carbonized clay with remnants of fruit nuts. The bedrock is weathered phyllite which has turned into clay (kaolin). Sandwiched between the bedrock and overburden is the auriferous gravel layer which ranges between 0.5 m and 4 m in thickness (see Fig. 2.5).

Angular quartz pebbles, some of which are iron stained, were found within the gravel layer near the Bakrobo reef (Fig. 2.3). These rocks are suspected to have been derived from the Bakrobo reef as they are similar in appearance.

Microscopic and chemical studies later confirmed that the angular quartz pebbles are derived from the rocks of the Bakrobo reef and that they are mineralized. This type of mineralization adds to the normal alluvial free gold mineralization found in the gravel layer. Thus two different kinds of mineralization are found within the gravel layer in Bakrobo area. For the purpose of mineral resource estimation, they had to be separated into different geological blocks. This gave rise to Block 1A and Block 1B (see Fig. 2.6). The overburden contained no gold.

The mineralogical studies also revealed that over 95 % of the material in the gravel layer contains quartz. Oxides, sulphides, silicates and organic material make up the rest of the material.

4.3 STATISTICAL ANALYSIS & ESTIMATION

Statistical analysis was carried out on Blocks 1A, 1B and 2. Block 3 was left out because of its limited number of sample data.

Histogram constructed for the various blocks showed that Blocks 1A and 2 follow a two parameter log-normal distribution whereas Block 1B follow a three parameter log-normal distribution.

The mean, median, standard deviation, skewness, kurtosis and coefficient of variation were calculated, first for the raw data and then for the log-transformed data. In the case of Block 1B, an additive constant of 0.15477 was added to the raw data before the log transformation. These results have been summarized in Tables 4.3.1 to 4.3.4

The log-normal distribution of Block 2 showed a single population. That of Blocks 1A and 1B were considered as single populations even though separate low grade outliers were evident. This confirms the geological basis for demarcating the blocks.

The statistics showed that Block 1B has a high variability as compared to Blocks 1A and 2. A 768.94 % error of estimation of the mean for Block 1B is highly risky as far as the expectation of the population mean is concerned. 18.35 % and 12.35 % error of estimation for Blocks 1A and 2 respectively though high are very reasonable for the population means to fall within when compared with Block 1B. This also confirms grade variability indicated by the geology.

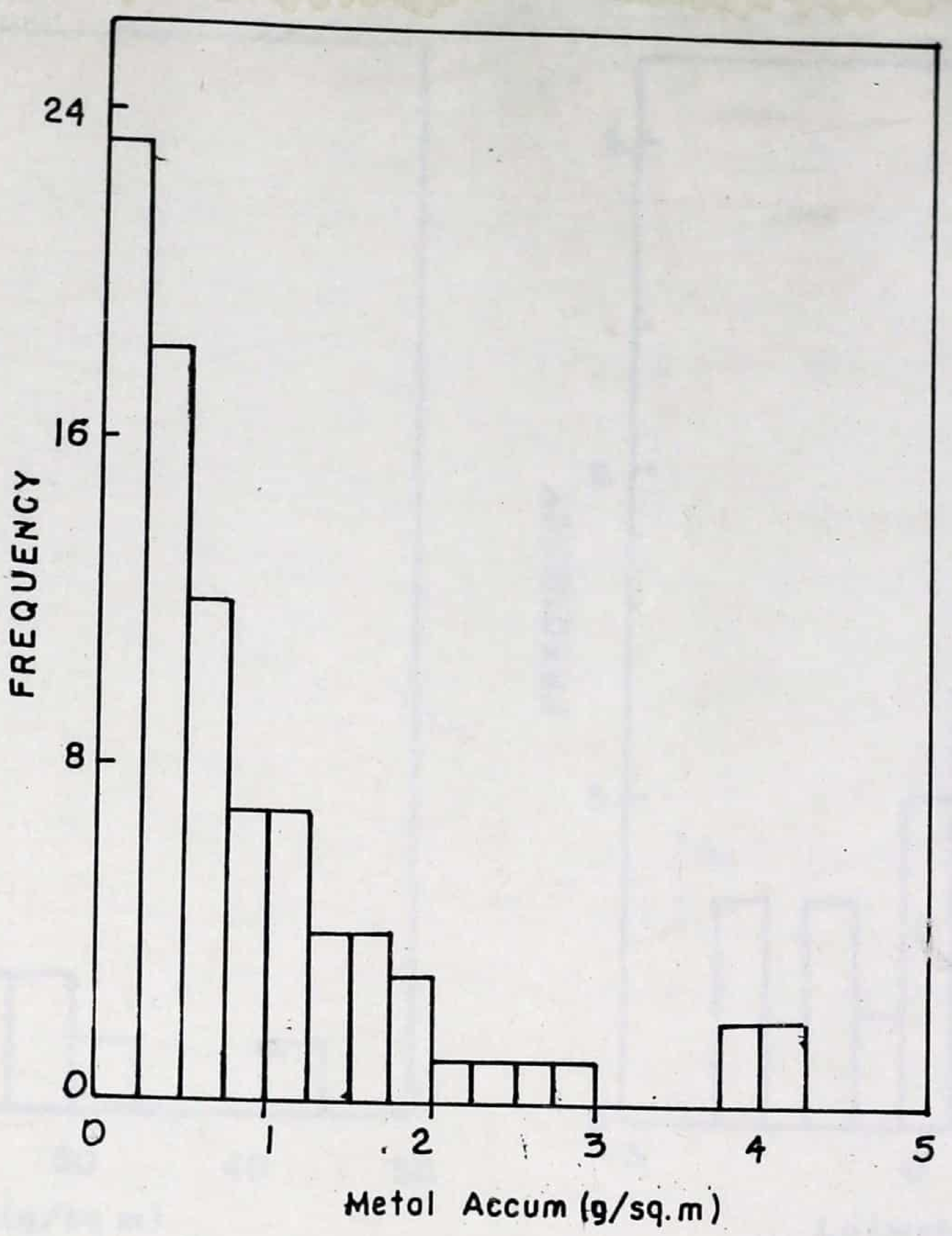


FIG. 4·3·1 (a) HISTOGRAM (METAL ACCUMULATION) OF BLOCK 1A

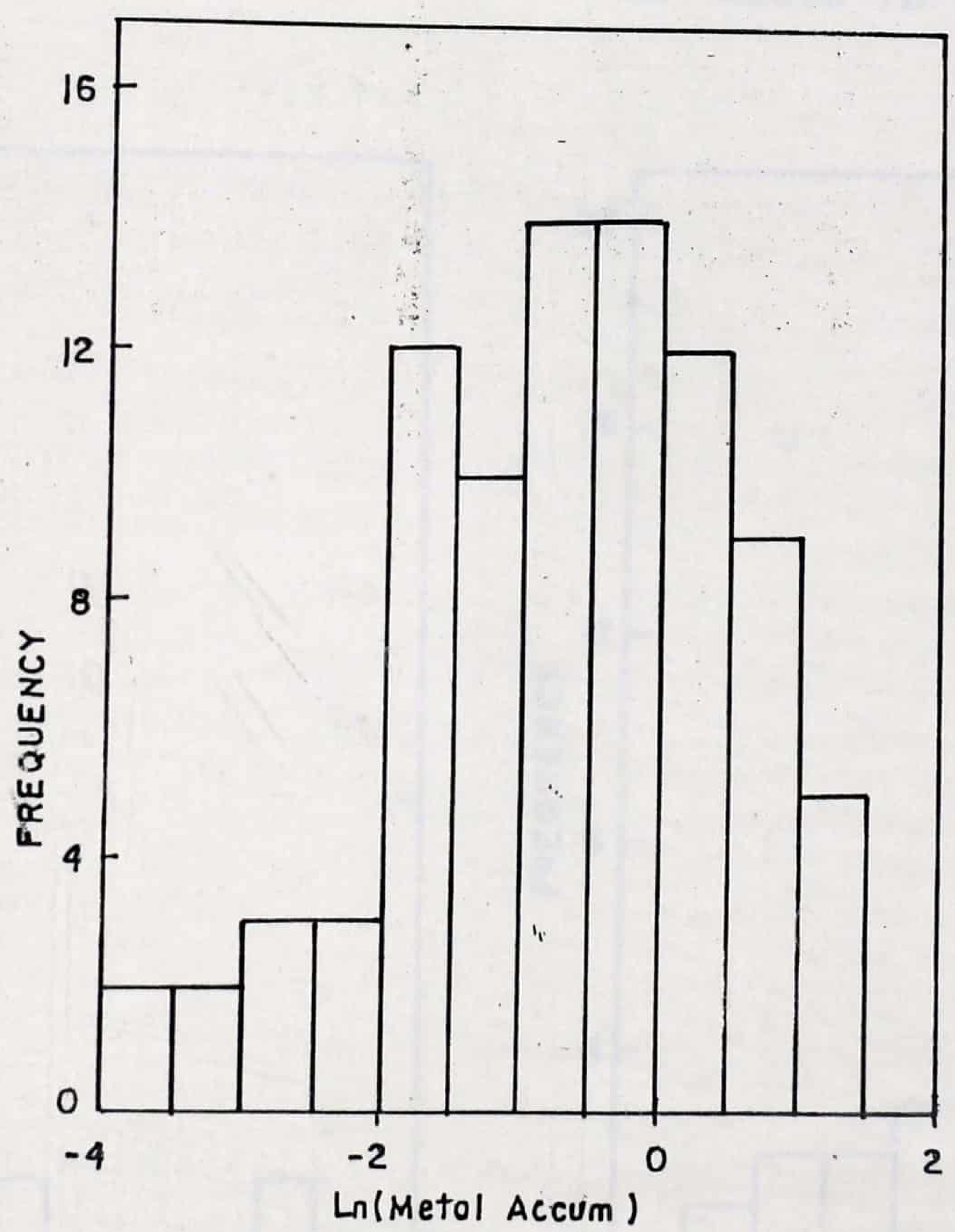


FIG. 4·3· 1(b) HISTOGRAM (NATURAL LOGARITHMS OF METAL ACCUMULATION) OF BLOCK 1A

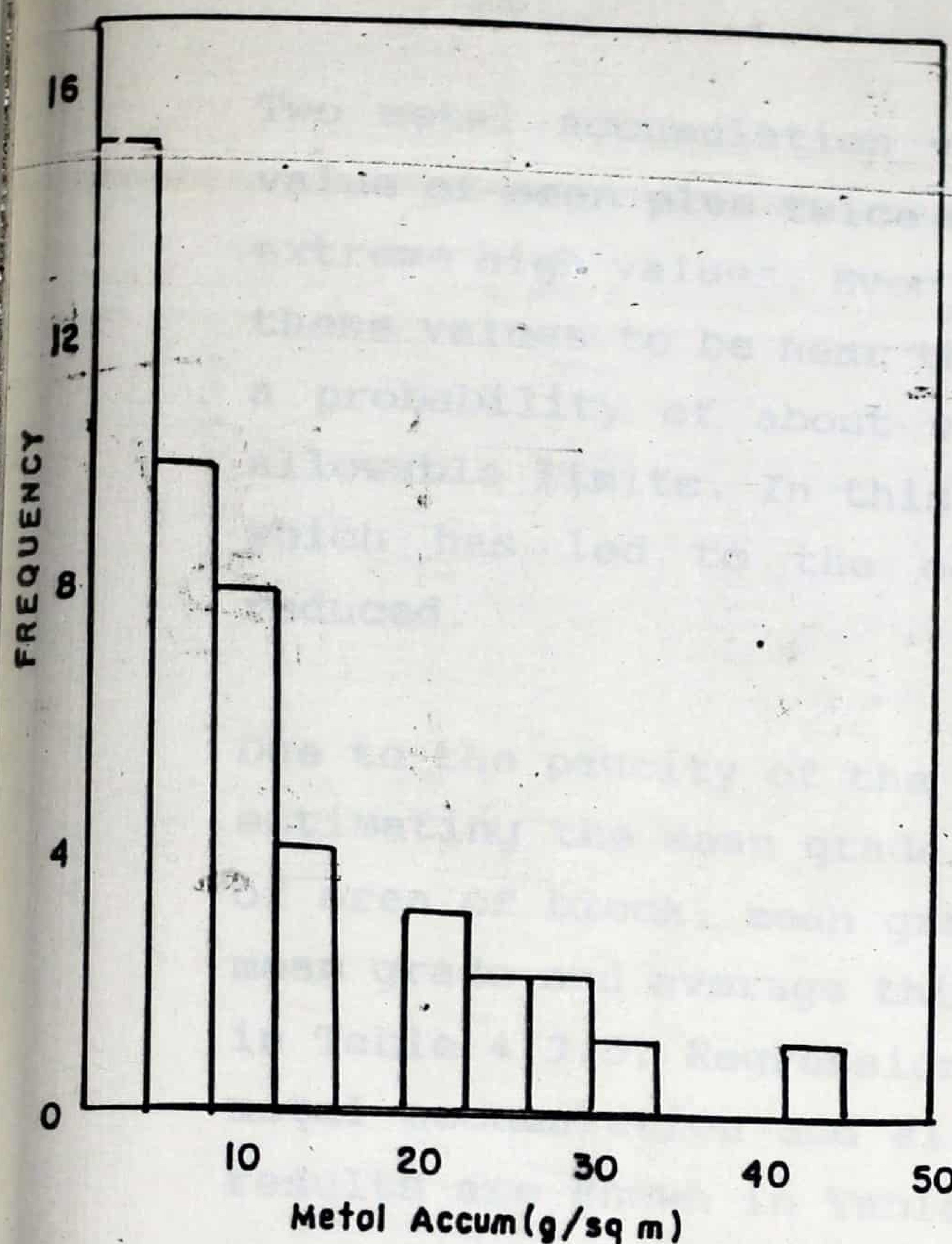


FIG. 4·3·2 (a) HISTOGRAM (METAL ACCUMULATION) OF BLOCK 1B

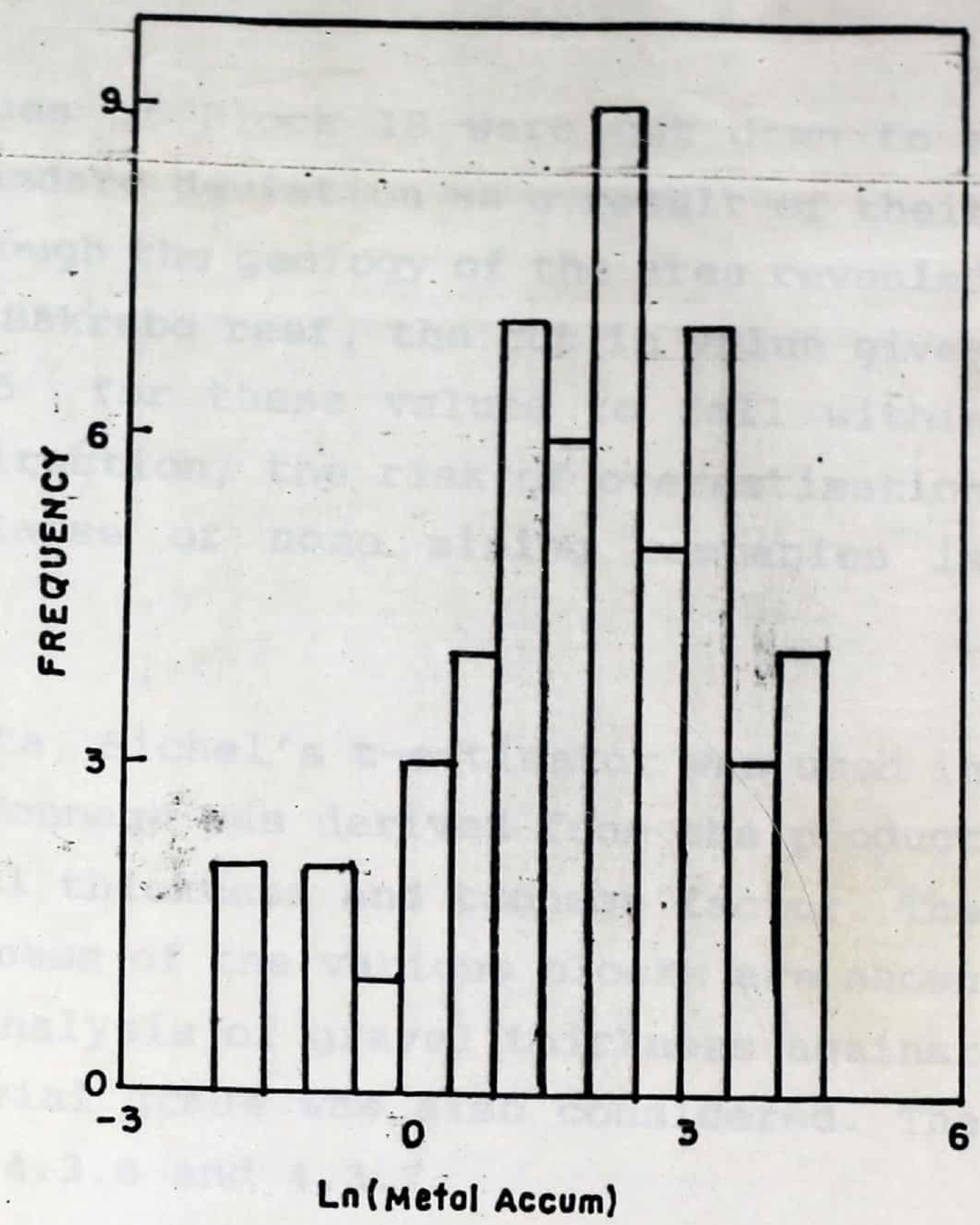


FIG. 4·3·2 (b) HISTROGRAM (NATURAL LOGARITHMS OF METAL ACCUMULATION) OF BLOCK 1B

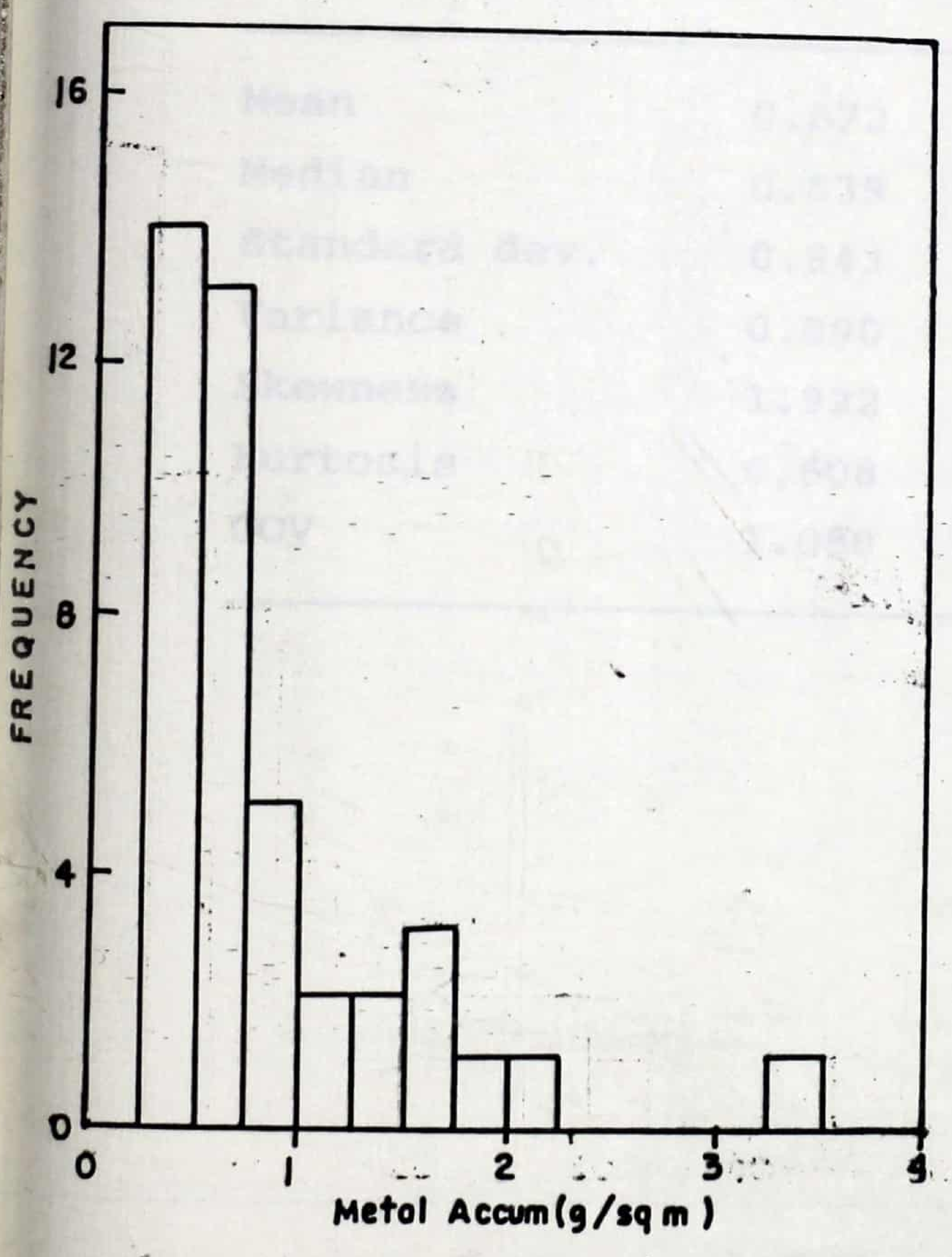


FIG. 4·3·3 (a) HISTROGRAM OF (METAL ACCUMULATION) OF BLOCK 2

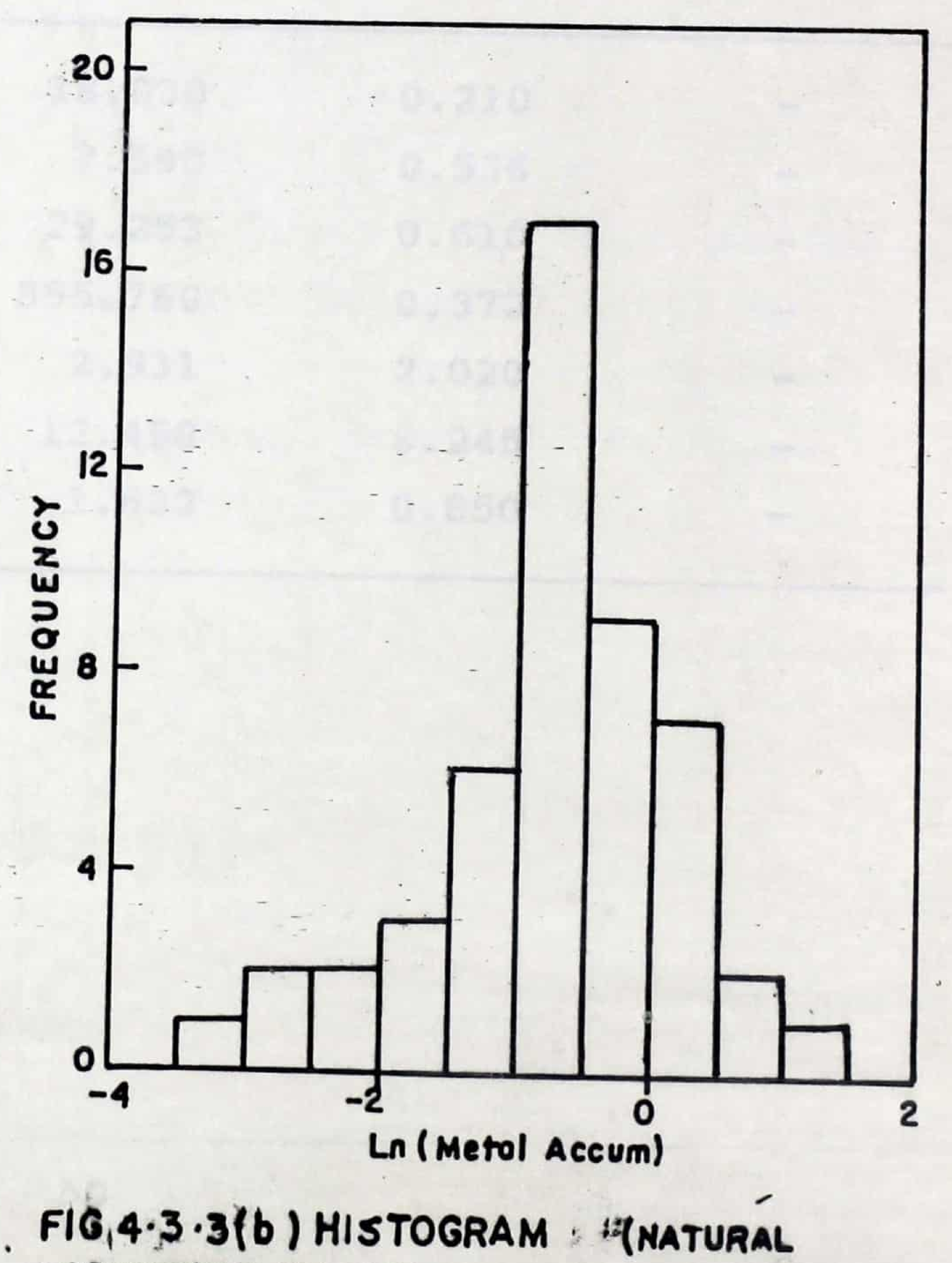


FIG. 4·3·3 (b) HISTOGRAM (NATURAL LOGARITHMS OF METAL ACCUMULATION) OF BLOCK 2

Two metal accumulation values of Block 1B were cut down to a value of mean plus twice standard deviation as a result of their extreme high values. Even though the geology of the area revealed these values to be near the Bakrobo reef, the cut in value gives a probability of about 0.95 for these values to fall within allowable limits. In this direction, the risk of overestimation which has led to the collapse of some mining companies is reduced.

Due to the paucity of the data, Sichel's t-estimator was used in estimating the mean grade. Tonnage was derived from the product of area of block, mean gravel thickness and tonnage factor. The mean grade and average thickness of the various blocks are shown in Table 4.3.5. Regression analysis of gravel thickness against metal accumulation and alluvial grade was also considered. The results are shown in Tables 4.3.6 and 4.3.7.

Table 4.3.1 Statistics of metal accumulation of the raw data distribution

BLOCK	1A	1B	2	3
Mean	0.872	18.030	0.210	-
Median	0.539	7.590	0.536	-
Standard dev.	0.943	29.253	0.610	-
Variance	0.890	855.760	0.372	-
Skewness	1.922	2.931	2.020	-
Kurtosis	6.608	12.458	8.245	-
COV	1.080	1.622	0.850	-

Table 4.3.2 Statistics of metal accumulation of two parameter log-transformed data distribution

BLOCK	1A	1B	2	3
Mean	-0.709	1.662	-0.689	-
Median	-0.619	2.027	-0.623	-
Standard dev.	1.177	2.157	0.917	-
Variance	1.385	4.653	0.841	-
Skewness	-0.435	-1.696	-0.702	-
Kurtosis	2.914	7.225	3.800	-
COV	1.660	1.297	1.330	-

Table 4.3.3 Statistics of metal accumulation of three parameter raw data distribution of Block 1B

Mean	Median	Standard Dev.	Variance	Skewness	Kurtosis	COV
15.669	7.835	19.767	390.323	1.761	5.106	1.26

Table 4.3.4 Statistics of metal accumulation of three parameter log-transformed data distribution of Block 1B

Mean	Median	Standard Dev.	Variance	Skewness	Kurtosis	COV
1.924	2.059	1.468	2.156	-0.472	2.937	0.76

Table 4.3.5 Statistical mean grade and average thickness of the various blocks

BLOCK	1A	1B	2	3
Metal Accum (g/m^2)	0.978	19.558	0.762	-
Standard Error of Estimation %	18.350	763.940	12.420	-
Mean Gravel Thickness (m)	1.675	2.227	1.803	-
Grade (g/m^3)	0.583	8.782	0.423	-
Grade (g/t)	0.292	3.992	0.212	-
Tonnage (t)	234860.2	377686.0	311046.3	-

Table 4.3.6 Regression analysis of gravel thickness and metal accumulation of the various blocks

BLOCK	1A	1B	2	3
No. of pairs	86	53	50	-
Slope	0.998	0.230	1.322	-
Intercept	0.805	1.936	0.864	-
Correlation coeff.	0.782	0.364	0.786	-

Table 4.3.7 Regression analysis of gravel thickness and alluvial grade

BLOCK	1A	1B	2	3
No. of pairs	86	53	50	-
Slope	0.689	-0.008	-0.060	-
Intercept	1.344	2.373	1.829	-
Correlation coeff.	0.170	1.000	1.000	-

Correlation coefficient of gravel thickness and metal accumulation values of 0.782 and 0.786 for Blocks 1A and 2 respectively, show good correlation. The value of 0.364 of Block 1B is quite low and shows poor correlation.

The correlation between gravel thickness and alluvial grade is very poor for Block 1A. The value of 1.00 for Blocks 1B and 2 with negative slopes show an inverse relationship between gravel thickness and alluvial grade. High alluvial grade corresponds to low gravel thickness and vice versa.

4.4 GEOSTATISTICAL ANALYSIS & ESTIMATION

4.4.1 SEMI-VARIOGRAM ANALYSIS

Experimental semi-variograms were constructed for metal accumulation for Blocks 1A, 1B and 2 at bearings of 000, 045 and 090 with an angle of tolerance of 22.5°. No geostatistical trend was realized from the experimental semi-variograms of metal accumulation. However, an omnidirectional experimental semi-variogram constructed showed geostatistical trend and was therefore adopted.

In fitting a mathematical model to the experimental semi-variograms, the number of sample pairs that contribute to the position of a sample data in addition to the general trend of data position were considered. Spherical models were fitted to the experimental semi-variograms for all the blocks (see Figs 4.4.(a) to 4.4.(c)). The initial values obtained from the experimental semi-variograms that were used in defining the spherical models have been summarized in Table 4.4.1

Table 4.4.1 Initial semi-variogram parameters used in defining the spherical models

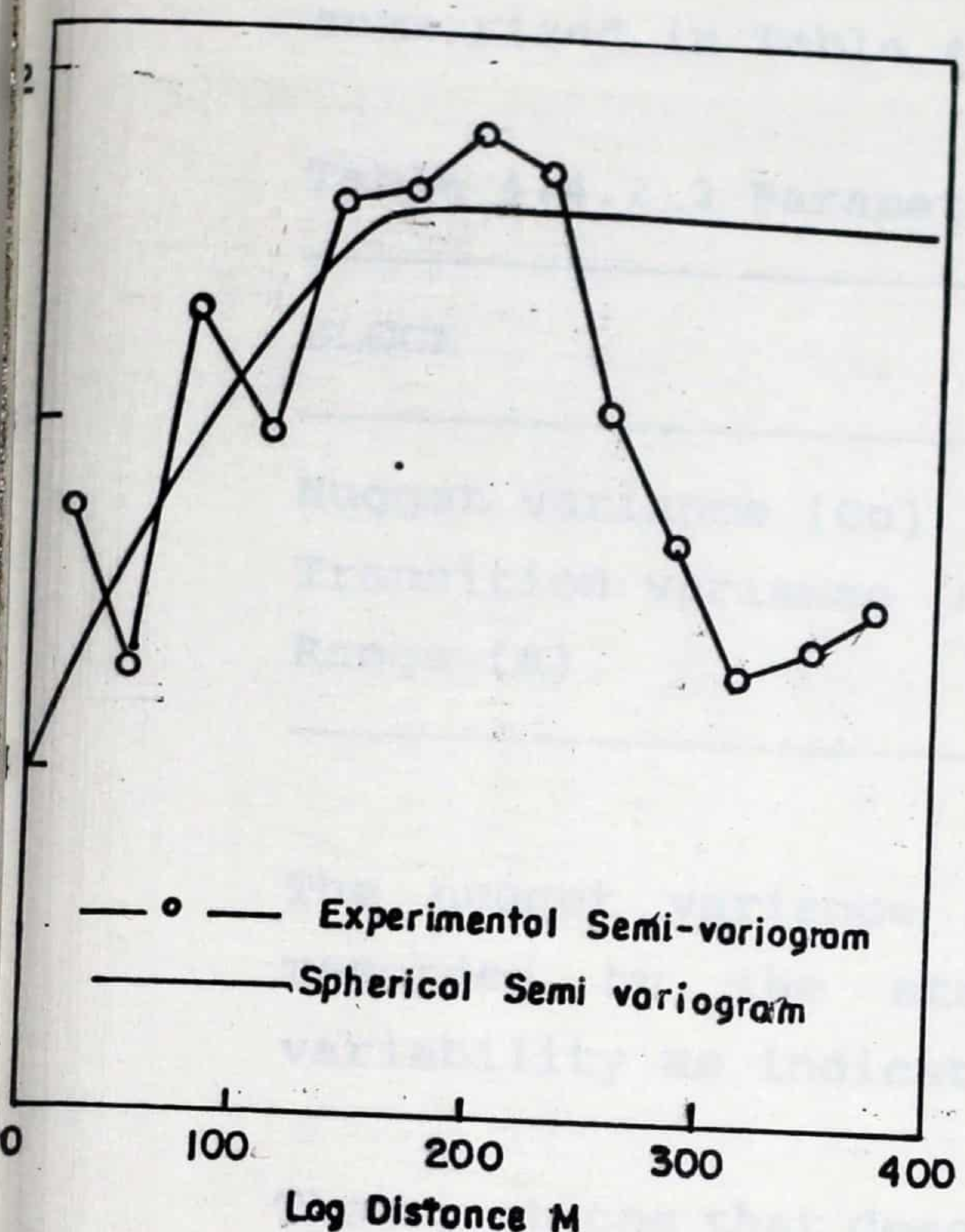
BLOCK	1A	1B	2	3
Nugget variance (Co)	0.425	250	0.130	-
Transition variance (C)	0.600	230	0.260	-
Range (a)	180	200	180	-

4.4.2 CROSS VALIDATION

The initially selected models were cross validated by point kriging. The best-fit models were obtained by varying the initially selected values of nugget variance, transition variance and range, with the objective of obtaining the same mean for the actual and the estimated values and also a zero mean error of estimation.

Table 4.4.2.1 Cross validation indices for semi-variogram models

BLOCK	1A	1B	2
No. of points available for kriging	86	53	50
No. of points kriged	86	53	50
No. of points missing	0	0	0
Mean kriged estimates	0.875	15.546	0.710
Standard deviation of kriged estimates	0.593	11.983	0.352
Mean of actual values	0.872	15.514	0.710
Standard deviation of actual values	0.943	19.757	0.610
Mean of difference (i.e. $Z - Z^*$)	0.003	0.031	-0.001
Standard deviation of difference	0.873	19.471	0.583
Mean of kriging standard error (σ/n)	1.075	25.402	0.583
Standard dev. of kriging standard error	0.118	2.922	0.082



G. 4·4 (a) OMNIDIRECTIONAL EXPERIMENTAL
MI-VARIOGRAM AND SPHERICAL SEMI-
RIOGRAM MODEL OF BLOCK IA

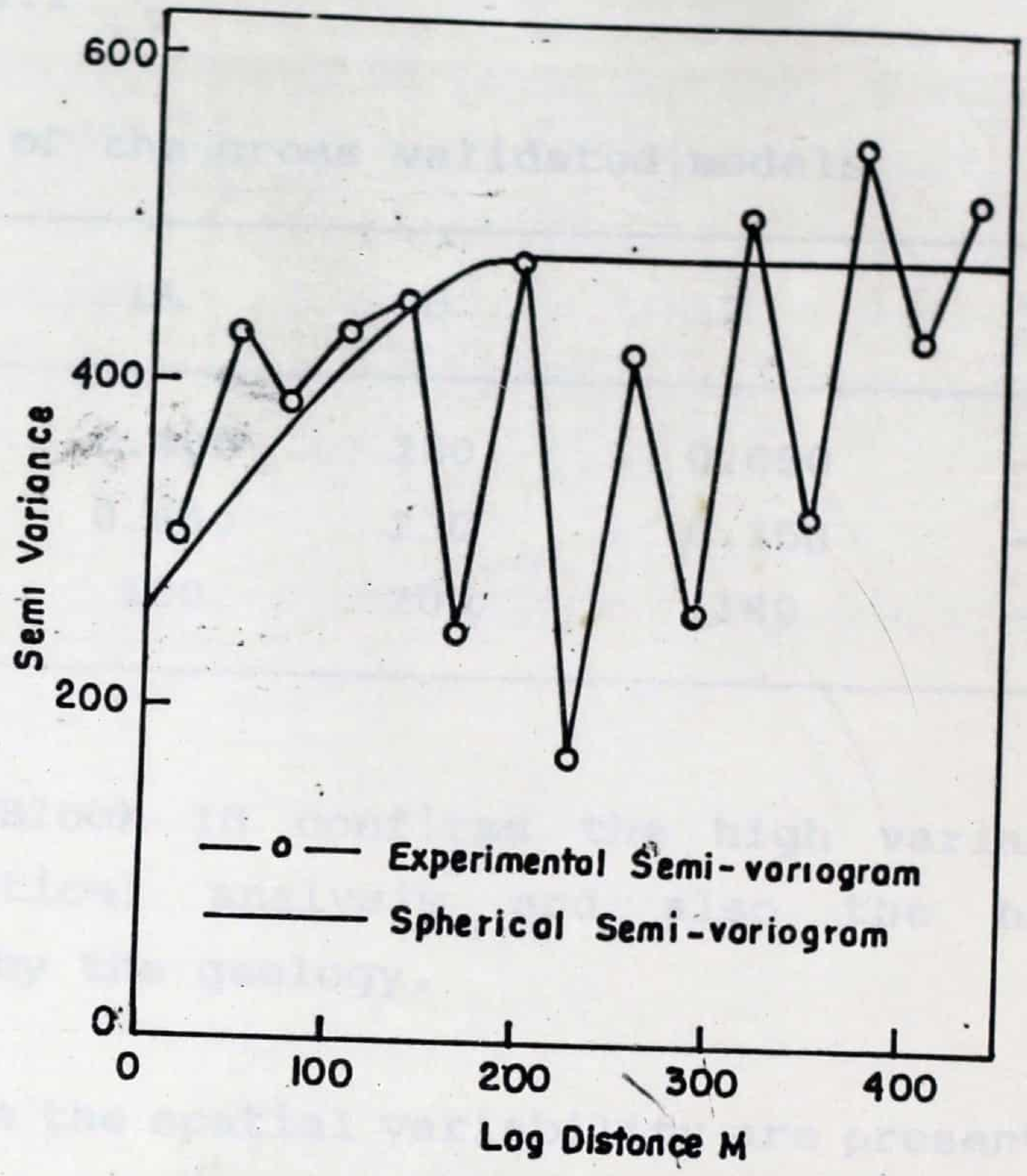


FIG. 4·4 (b) OMNIDIRECTIONAL EXPERIMENTAL
SEMI-VARIOGRAM AND SPHERICAL SEMI-
VARIOGRAM MODEL OF BLOCK IB

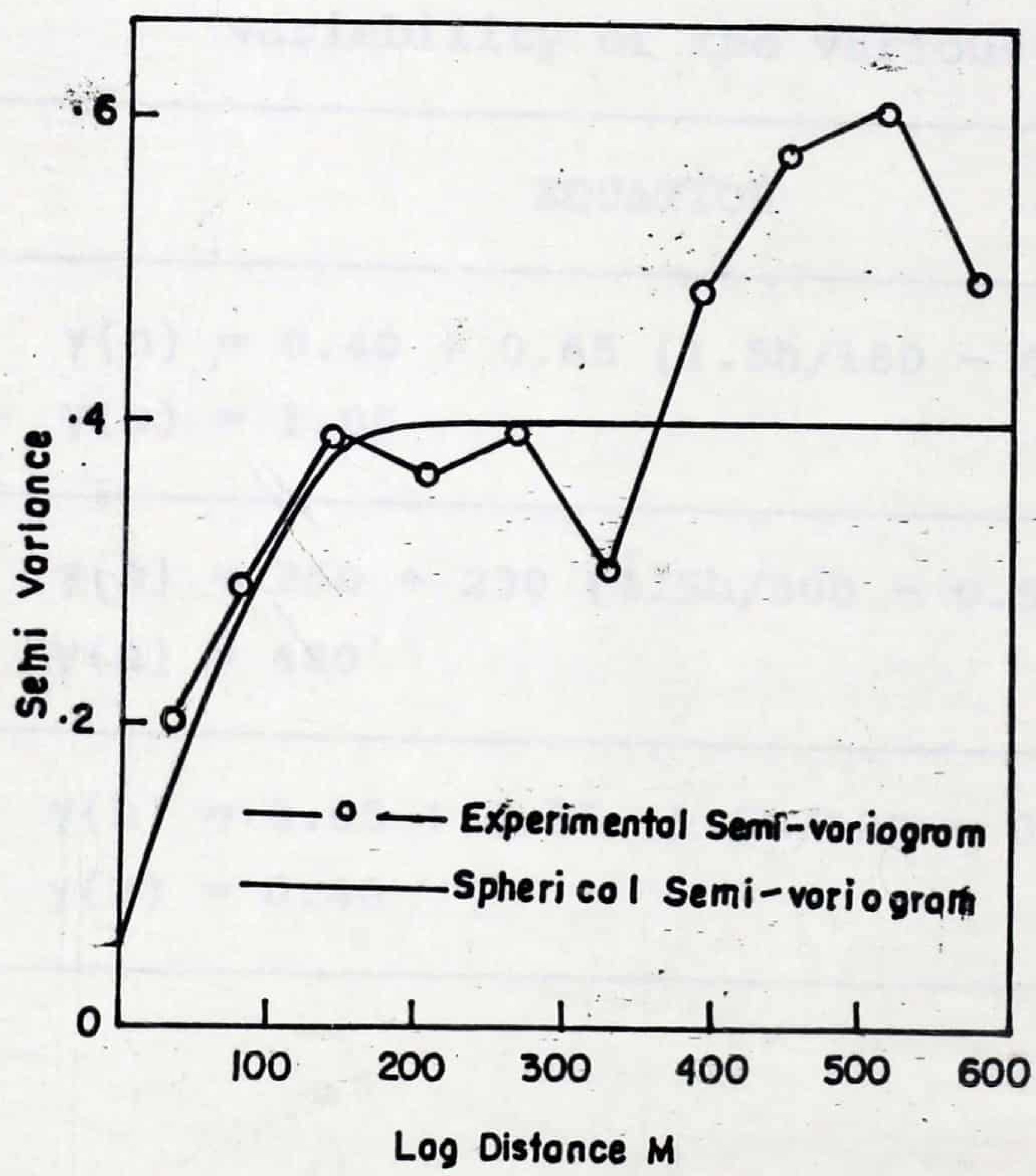


FIG. 4·4 (c) OMNIDIRECTIONAL EXPERIMENTAL
SEMI-VARIOGRAM AND SPHERICAL SEMI-
VARIOGRAM MODEL OF BLOCK 2

The final models obtained are shown in Fig 4.4.(a), Fig 4.4.(b) and Fig 4.4.(c). The parameters of these models have been summarized in Table 4.4.2.2

Table 4.4.2.2 Parameters of the cross validated models

BLOCK	1A	1B	2	3
Nugget variance (Co)	0.400	250	0.050	-
Transition variance (C)	0.650	230	0.350	-
Range (a)	180	200	180	-

The nugget variance of Block 1B confirms the high variance recorded by the statistical analysis and also the high variability as indicated by the geology.

The equations that describe the spatial variability are presented in Table 4.4.2.3

Table 4.4.2.3 Spherical model equations describing spatial variability of the various blocks

BLOCK	EQUATION	
1A	$\gamma(h) = 0.40 + 0.65\ (1.5h/180 - 0.5h^3/180^3)$	@ $h<180$
	$\gamma(h) = 1.05$	@ $h\geq180$
1B	$\gamma(h) = 250 + 230\ (1.5h/200 - 0.5h^3/200^3)$	@ $h<200$
	$\gamma(h) = 480$	@ $h\geq200$
2	$\gamma(h) = 0.05 + 0.35\ (1.5h/180 - 0.5h^3/180^3)$	@ $h<180$
	$\gamma(h) = 0.40$	@ $h\geq180$

4.4.3 KRIGING ESTIMATION

Block kriging was employed for the estimation procedure. Each block was divided into smaller units. A circular search area of radius 90 m for Block 1A and Block 2, and 100 m for Block 1B was centered on each unit in turn. The search area was selected on the basis of range of influence of samples from the variogram analysis. The number of samples captured for each unit estimation ranged from 4 to the maximum which is 8 in this case.

The mean grade and tonnage for an entire block are presented in Table 4.4.3.

Table 4.4.3. A summary of grade and tonnage estimates for the various blocks

BLOCK	1A	1B	2	3
Grade (g/t)	0.280	3.000	0.205	-
Tonnage (t)	245,810.6	356,147.6	300,830.3	-

4.5 INVERSE DISTANCE WEIGHTING METHOD

Inverse square distance was used for this estimation. Each block was divided into smaller units of dimension 60 m x 30 m. A circular search area of radius 90 m for Blocks 1A and 2, and 100 m for Block 1B was centered on each unit in turn. The unit dimension was based on the sampling grid at the detailed exploration stage and the search area was selected on the basis of the geostatistical range of influence of samples.

The number of samples captured for each unit estimation ranged from 4 to 8. The mean grade for an entire block was derived as a volume weighted value. The total volume for an entire block was derived from just the summation of block unit volumes. The results are summarized in Table 4.5.1

Table 4.5 Inverse distance grade and tonnage estimates of the various blocks

BLOCK	1A	1B	2	3
Grade (g/t)	0.280	3.432	0.199	0.231
Tonnage (t)	298432.2	361,172.7	295,904.8	8,760.79

4.6 SECTIONAL METHOD OF ESTIMATION

The method of estimation as described in Chapter Three was used and results have been presented in the table below.

Table 4.6 Sectional grade and tonnage estimates for the various blocks

BLOCK	1A	1B	2	3
Grade (g/t)	0.241	3.103	0.231	0.246
Tonnage (t)	304,799.0	352,081.9	307,991.5	8,760.8

4.7 TRIANGULAR METHOD OF ESTIMATION

This method of grade and tonnage estimation was used by Nkroful Mining Ltd. A computer package of SURPAC software which has a Digital Terrain Model (DTM) option was used for this exercise. The DTM is particularly useful for stratified deposits because it enables surfaces between successive strata to be modeled taking into consideration the undulating nature of the surfaces involved. Volume estimation is therefore very accurate.

Volume and grade estimation of the various blocks are presented in Table 4.7

Table 4.7 Triangular grade and tonnage estimates for the various blocks

BLOCK	1A	1B	2	3
Grade (g/t)	0.241	7.500	0.195	0.245
Tonnage (t)	322,108.4	269,040.6	315,383.2	10,362.4

4.8 COMPARISON OF ESTIMATION RESULTS

Tonnage variations were generally high in Block 1B as shown in Table 4.8. Apart from the triangular method of estimation which was purely computer base, the other methods were semi or purely manual which make them more prone to human error. Also, unlike the triangular method which takes the undulating nature of the surfaces into consideration, the manual methods assume a straight surface between sample locations. These reasons should account for the variation in tonnages.

The results of grade estimates from the various estimation method did not differ so much from one another. An exception is that of Block 1B (see Table 4.8) whose triangular estimation method did not cut down any high values.

Table 4.8 Summary of estimation results

BLOCK		1A	1B	2	3
TRIANGULAR METHOD	Grade (g/t)	0.250	7.500	0.195	0.245
	Tonnage (t)	322108.4	269040.6	315383.2	10362.4
SECTIONAL METHOD	Grade (g/t)	0.241	3.103	0.231	0.246
	Tonnage (t)	304799.0	352081.9	307991.5	8760.8
INVERSE DISTANCE METHOD	Grade (g/t)	0.281	3.432	0.199	0.231
	Tonnage (t)	298438.2	361172.7	295904.8	8760.8
STATISTICAL METHOD	Grade (g/t)	0.292	3.992	0.212	-
	Tonnage (t)	234860.2	377686.0	311046.3	-
GEOSTATIST-ICAL METHOD	Grade (g/t)	0.280	3.000	0.205	-
	Tonnage (t)	245810.6	356147.6	300830.3	-

Comparatively, results of the statistical grade estimates were found to be generally high. Sectional and triangular grades fluctuated between low and high values. Geostatistical and inverse distance grades were average, and compared favourably. The inverse distance parameters were however selected on the basis of geostatistical range of influence.

Though some of the estimation methods have been acclaimed as better than others due to their sound theoretical basis, the most logical and practical way of arriving at such a conclusion is to compare estimated results with mined figures. Unfortunately, Nkroful Mining Ltd had not been granted a mining lease yet.

CHAPTER FIVE

CONCLUSIONS & RECOMMENDATIONS

The Nkroful alluvial gold mineralization is erratic and complex in nature so it demands a thorough investigation to establish the geological characteristics of the deposit and a good estimation method to estimate the gold resource potential of the deposit. The objective of this thesis was to study the geological characteristics of the deposit, and select an appropriate mineral resource estimation method through a comparative study of the various existing estimation methods. The outcome should assist Nkroful Mining Ltd in estimating its gold potential resource.

The geological characteristics of the deposit were studied through literature survey and geological mapping in the field to ascertain facts gathered from the literature. Statistical and geostatistical analysis were used to check and quantify the parameters obtained from the geological studies.

Geological studies revealed two different kinds of mineralization in the Bakrobo area: normal alluvial free gold mineralization and quartz pebble mineralization within the gravel layer. The area was therefore separated into two different geological blocks (Fig. 2.6). Based on the geological characteristics of the deposit and sample grade values, the entire alluvial deposit was demarcated into four different geological blocks for the gold resource estimation.

Triangular, sectional, inverse square distance, statistical and geostatistical methods of estimation were used to estimate the mineral resources of the deposit. Grade estimates from the various estimation methods did not differ significantly. Comparatively, statistical estimation grades were higher. Triangular and sectional estimation values fluctuated between high and low grades. Geostatistical and inverse distance estimation grades were average and compared favourably to each other.

The study has shown that geostatistics, though laborious and requires high expertise, was able to take into account the regionalization of the variables which should allow for the satisfactory estimation of the various blocks.

The study has also shown that the inverse distance square estimation method, if selected on the basis of a thorough geostatistical study, approximates grade results produced by kriging. Furthermore, it is much simpler to use.

The author is of the view that the most appropriate way of choosing an estimation method is to substantiate the method chosen with available mined figures. Unfortunately, as at the time of completion of this research work, Nkroful Mining Ltd had not been granted a mining lease yet. Despite this constraint, it is still proposed that inverse distance square method should be adopted by Nkroful Mining Ltd for resource estimation due to its simplicity and comparability with the geostatistical method. However, the continued use of this method should be justified through a comparison with mined figures when they become available.

A thorough geological studies proved to be a good basis for mineral resource estimation. It is further recommended that for any fruitful mineral resource estimation of an alluvial deposit, a thorough geological study should be undertaken to establish the geological characteristics before any attempt is made to estimate the grade and tonnage of the deposit.

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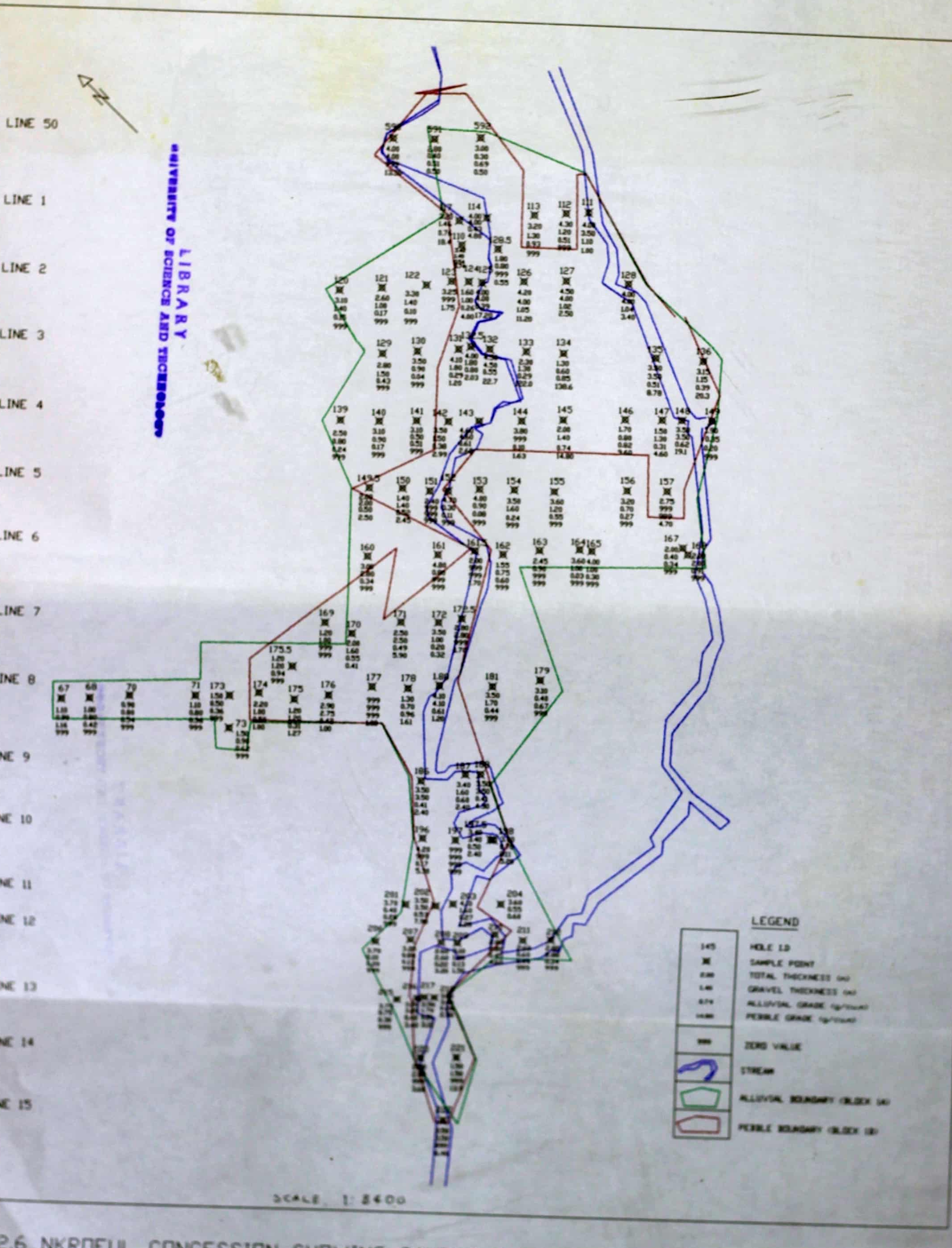

2.6 NKROFUL CONCESSION SHOWING SAMPLE POINTS AND VALUES OF BLOCKS 1A & 1B

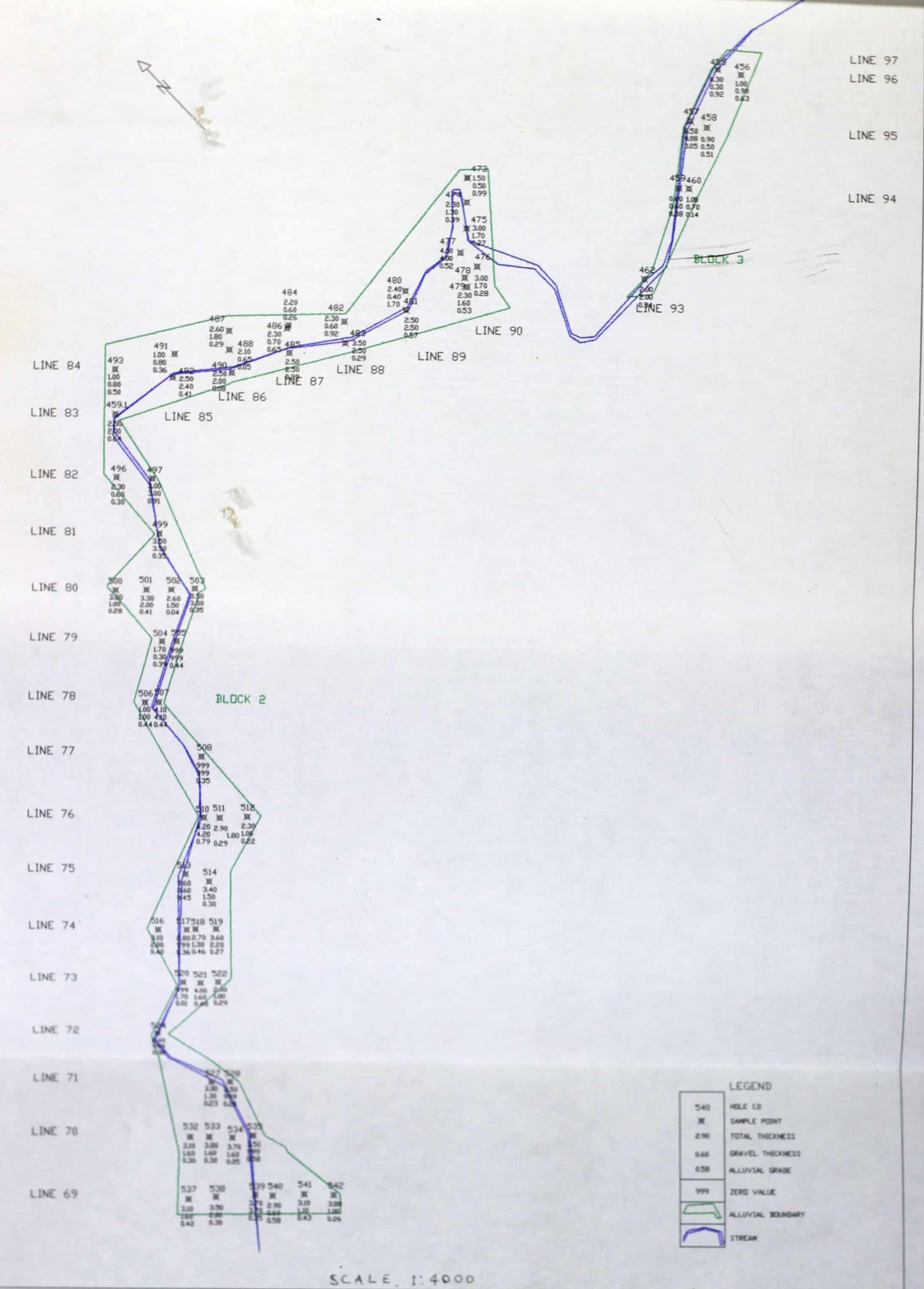

FIG 2.7 NKROFUL CONCESSION SHOWING SAMPLE POINTS AND VALUES OF BLOCKS 2 & 3