

GEOSTATISTICAL INVESTIGATION OF IRON
ORE GRADE CONTROL AT MARAMPA MINE,
SIERRA LEONE.

by

MYLES HARLE MCINTYRE

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Department of Geology,
Royal School of Mines,
Imperial College of Science and Technology
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ABSTRACT

The Marampa Mine in Sierra Leone produces a hematite concentrate (mean grade 64.5 per cent Fe) by Humphreys spiral beneficiation of a tropically weathered hematite-quartz-muscovite schist (mean grade 45.9 per cent Fe). The structure of the orebody has been interpreted as consisting of a series of overturned to recumbent isoclinal folds with axial planes dipping to the east. The ore is contained in four major synclines separated by anticlinal bodies of quartz-muscovite schist waste.

Grade control problems exist in the operations insofar as the mean grade of the concentrate has shown a significant decrease over a period of several years. Furthermore, it was evident that the concentrate grade could not be predicted from a knowledge of the mill feed grade alone.

These problems were investigated during the research, using methods of statistical analysis of assay data in conjunction with an evaluation of sampling methods and the geology and mineralogy of the hematite ores. The following pertinent factors emerged.

(1) A positive and statistically significant correlation exists between mill feed grade and concentrate grade. A significant decrease in mean ore grade has occurred with increasing depth of mining and this decrease is confidently expected to continue as mining extends downwards into less weathered material but was not detected over the thirty feet height of a single mining bench.

(2) The concentrate midsize fractions (-60 +170 B.S. mesh) are invariably of substandard grade due to excessive amounts of liberated quartz. This inclusion of quartz is apparently inherent in spiral beneficiation but may be counteracted to a certain extent by careful control of milling procedures.

(3) The concentrate grade cannot be predicted from a knowledge of mill feed grade alone despite the significant correlation between the two. Important factors influencing

the concentrate quality, besides variations in ore grade, are textural features of the ore and the grainsize and grainshape of the hematite which occurs in two forms;

(a) coarse, micaceous to tabular and brecciated hematite.

(b) fine, equidimensional and annealed hematite.

The two types of hematite respond differently to spiral beneficiation and do not occur naturally intermixed, but in separate ore synclines.

(4) Comparative results from different methods of sampling indicate that channel sampling is the most cost-effective method of sampling the Marampa ore. Ore grades are spatially anisotropic and a rectangular sampling grid with twenty feet channel samples taken across the strike along lines one hundred feet apart may improve grade control.

(5) Special care is necessary with the XRF method of analysis of ore samples in order to avoid analytical bias which can result in overvaluation of the content of iron in ore relative to iron in concentrates.

(6) Errors are common in the conversion of XRF spectrometer digital output data to actual percentage assay data and a self-checking assay procedure has been designed to eliminate the errors before the final conversion is made.

(7) Mining to an assay plan based on contoured moving means may reduce feed grade variations and improve concentrate quality by enabling the mill to be tuned to respond to smaller ranges of ore grade and hematite grainsize and grainshape.

ABBREVIATIONS AND SYMBOLS

CHEMICAL

Fe	- iron
SiO ₂	- silica
Mn	- manganese
Al ₂ O ₃	- alumina
Ti	- titanium
P	- phosphorous
Fe _c	- concentrate grade (per cent Fe)
SiO _{2 c}	- concentrate grade (per cent SiO ₂)
Fe _f	- feed grade (per cent Fe)
SiO _{2 f}	- feed grade (per cent SiO ₂)

STATISTICAL

x, y	- sample values
$\bar{X}, \bar{Y}$	- mean values
n, N	- number of samples
v	- coefficient of variation or number of degrees of freedom
s, s ²	- standard deviation, variance
s _p	- pooled standard deviation
$\sum X$	- sum of values of X
r	- total correlation coefficient
r ²	- coefficient of determination
r _{ij:k}	- first order partial correlation coefficient between variables i and j with the effect of variable k held constant.
r _{ij:kl}	- second order partial correlation coefficient between variables i and j with the effects of variables k and l held constant
x ²	- Chi-square
F	- Snedecors Variance Ratio
γ(h)	- variogram function

PHYSICAL AND BENEFICIATION

S.G.	- specific gravity
F ₁	- per cent weight of total silica in -44 +120 B.S. fraction

PHYSICAL AND BENEFICIATION (continued)

F ₂	-	per	cent	weight	of	total	silica	in	+44	B.S.	fraction
F ₃	-	"	"	"	"	"	iron	"	+44	B.S.	fraction
F ₄	-	"	"	"	"	"	silica	"	-120	B.S.	fraction
F ₅	-	"	"	"	"	"	iron	"	-44	+120	B.S. fraction
F ₆	-	"	"	"	"	"	"	"	-120	B.S.	fraction
X ₅	-	per	cent	weight	of	ore	in	+	16	B.S.	mesh fraction
X ₆	-	"	"	"	"	"	"	-	16	+44	B.S. mesh fraction
X ₇	-	"	"	"	"	"	"	-	44	+85	B.S. mesh fraction
X ₈	-	"	"	"	"	"	"	-	85	+120	B.S. mesh fraction
X ₉	-	"	"	"	"	"	"	-	120	+200	B.S. mesh fraction
X ₁₀	-	"	"	"	"	"	"	-	200	B.S.	mesh fraction

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CHAPTER I

INTRODUCTION AND THE MARAMPA GRADE CONTROL PROBLEM

A. THE MINE

Location: The Marampa orebody is situated in Sierra Leone, West Africa, at a latitude and longitude of $8^{\circ} 41'$ North and $12^{\circ} 30'$ West respectively (Fig. 1.1). It lies some 50 miles ENE of Freetown, the nations capital and is connected with Pepel, the shipping port for the iron ore concentrate, by a 52 mile railway. The mine is operated by the Sierra Leone Development Company.

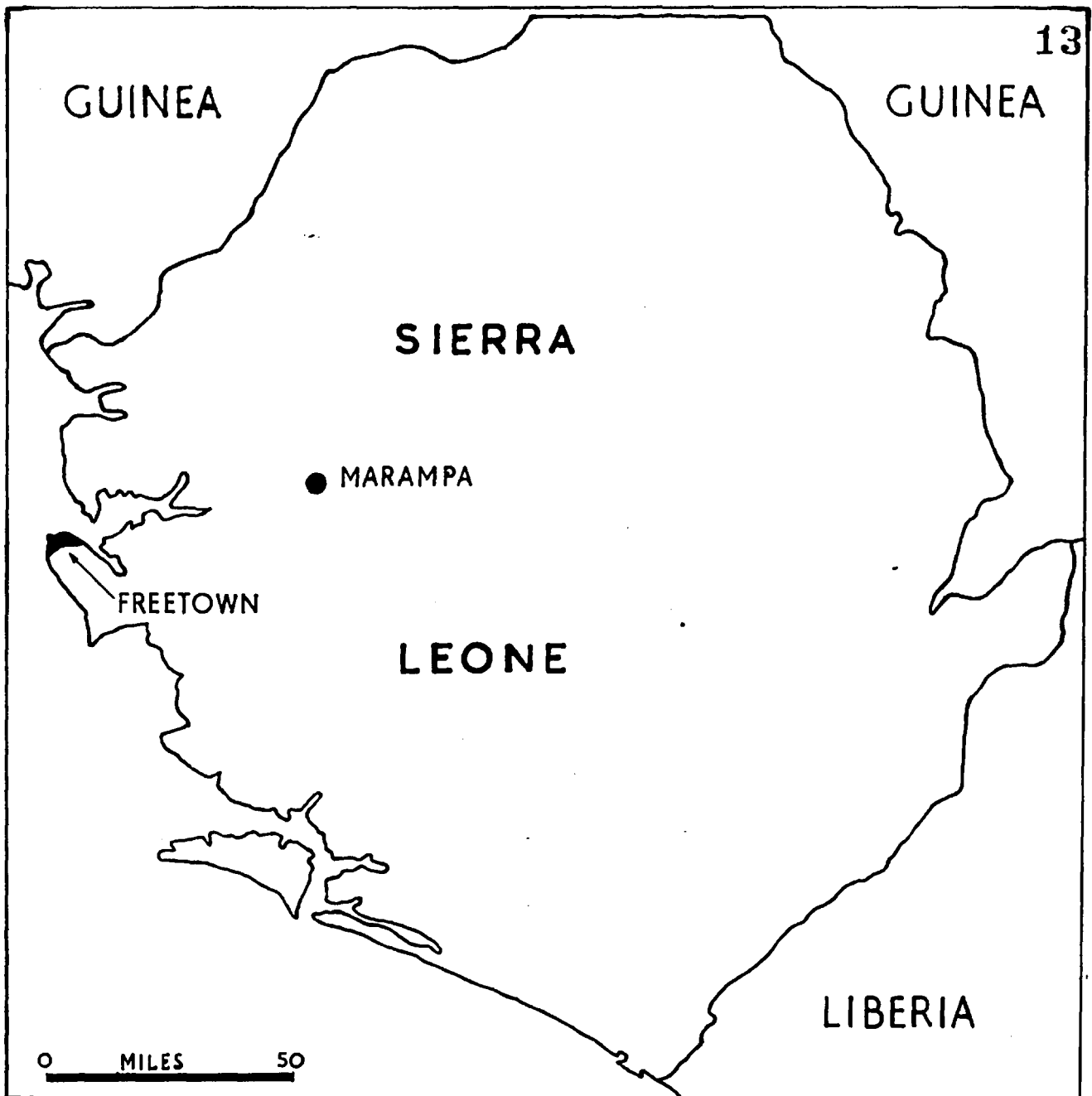
Climate: The climate is tropical with two seasons: the wet season, extending from July to October, is warm and humid with common rainstorms and mean daily temperatures exceeding 75°F . The dry season, from November to June, is warmer and much dryer. Between 100 and 125 inches of rainfall per annum falls at Marampa, the bulk of which falls between June and October inclusive.

Technical: Large scale iron ore mining at Marampa commenced about 1933 but there is evidence that primitive iron ore smelting was carried out by the local Africans much earlier.

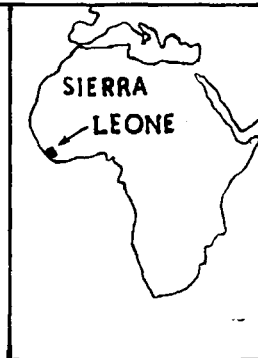
Production of concentrates during 1968 was about 2.504 million tons (Mining Annual Review, 1969) and production capacity at Marampa is being expanded from its present annual level of about 2.75 million tons to 3.1 million tons. Most of the concentrates are sold in Europe but a long term contract has been drawn up with Japanese interests for the sale of at least 12.6 million tons of concentrates over a $10\frac{1}{2}$ year period.

Distribution of concentrates during 1968 was (Mining Annual Review, 1969);

Germany	952 419 tons
Holland	789 656 "
United Kingdom	282 885 "
Italy	151 620 "



0 MILES 50



LOCATION MAP OF
MARAMPA AND SIERRA LEONE

Belgium	124 680 tons
Japan	110 312 "
Total	2 411 572 "

In addition to the production of iron ore concentrates, about 1000 tons of Ferromax is produced per annum. Ferromax is an extremely high grade (> 68.5 per cent Fe) hematite concentrate used in the manufacture of welding rods and is produced by exhaustive spiral treatment.

Several factors aided the development of mining at Marampa, including;

(a) The demand for iron ore during the 1930's initiated the development of Marampa and the Second World War stimulated production for the war effort. The present iron ore market with excess of supply over demand may act against long term mining at Marampa although the Marampa product may retain its usefulness for blending purposes in sintering and pelletising operations.

(b) The physical and chemical characteristics of the ore are such that beneficiation has been relatively inexpensive and deleterious elements such as phosphorous and sulphur are negligible. The ore has been remarkably friable although it is now becoming increasingly harder and modifications to mining methods and milling practice will eventually become necessary.

(c) African labour has been available at low cost.

B. MINING OPERATIONS

Mining operations are at present confined to Masaboin Hill which is known to contain the bulk of the ore reserves, although it is planned to mine small amounts of ore from other minor deposits, notably Ghafal Hill.

Masaboin Hill originally stood several hundred feet above the surrounding countryside but has been reduced some 250 feet by mining operations. The ore is contained within four major synclines, designated A, B, C and D, separated by

anticlinal quartz-muscovite schist waste. Although differential erosion produced the upstanding character of Masaboin and Ghafal Hills both the ore and waste have undergone prolonged tropical weathering and are almost completely friable and are readily mined without systematic blasting although harder ore is becoming more common particularly towards the centre of the hill. The hardness of the ore is a function of depth brought about by ground water leaching effects and it is recognized that ore hardness increases with depth. At present only a modest amount of blasting is necessary but is steadily increasing as mining progresses. The quartz-muscovite schist waste is even more friable than the ore and only a very minor amount of blasting has been necessary in the past for the removal of waste although this too is expected to increase as mining progresses.

Mining operations are carried out during two eight-hour shifts per day, seven days per week. The shifts are 11 a.m. to 7 p.m. and 11 p.m. to 7 a.m., the intervening four hour periods being used for preventive maintainance, repair of plant and the extension and transfer of conveyor belts or other ancillary equipment. A certain amount of mining is undertaken between shifts for ore-blending purposes or when ore bunkers become depleted.

The ore is mined by Ruston-Bucyrus RB 110 electric shovels equipped with $4\frac{1}{2}$ cubic yard buckets working on a 30 feet face, and is transported to the mill by conveyor belts fed directly by the shovels or by hoppers loaded from Euclid R 45 rear dump trucks. The use of the rear dump trucks is increasing at the expense of the conveyor belts and direct feeding of the conveyor belts will eventually be phased out. The conveyor system is rather inflexible and is not adaptable to a sensitive grade control scheme which may require the day to day mining of small blocks of ore from various localities scattered over the orebody.

In the past the mining policy has been to mine only one level at any one time and the mining has entailed the removal of horizontal sections of the hilltop. Changes in ore hardness, ore grade and the grade control problem have produced a

situation in which mining operations have been altered in such a way as to initiate multi-level mining in an attempt to combat these problems.

C. ORE BENEFICIATION

Marampa ore has an average in situ grade of about 46 per cent Fe and is beneficiated to give a concentrate with an average grade of about 64.5 per cent Fe. Beneficiation is carried out by washing minus $1/8$ inch ore on Humphrey spirals (Fig. 1.2).

The ore is dry screened before beneficiation commences in order to separate the lump ore from the fines. The oversize from the screening plant is crushed in a 17 feet diameter Aerofall mill which has a capacity of about 180 tons per hour. The size of the Aerofall mill feed varies considerably. During the wet season the screens are readily blinded due to the adhesive properties of the saturated ore and screen sizes are increased to combat the effects of blinding. This results in greater amounts of fines passing through the screens to the Aerofall mill. When ore from the mine is friable and without much lump ore, fine material must be fed to the Aerofall mill to keep the crushing charge at an efficient volume and this may result in overcrushing of the fine ore with consequent loss of some of the fine hematite on the spirals. The largest size ore fed to the Aerofall mill has a maximum dimension of about three feet. The Aerofall mill is usually operated with a variable, but considerable, amount of ore which could otherwise be directly beneficiated and it is necessary to operate the mill continuously on an inefficient feed rather than operate it periodically with a coarse lump feed alone.

The Aerofall mill product is separated in three stages. The first stage is a coarse classifier which produces two fractions, coarse and fine. The coarse fraction is screened with the oversize being recycled to the mill for further crushing and the undersize passing to the concentration plant for spiral beneficiation. The fine fraction from the first

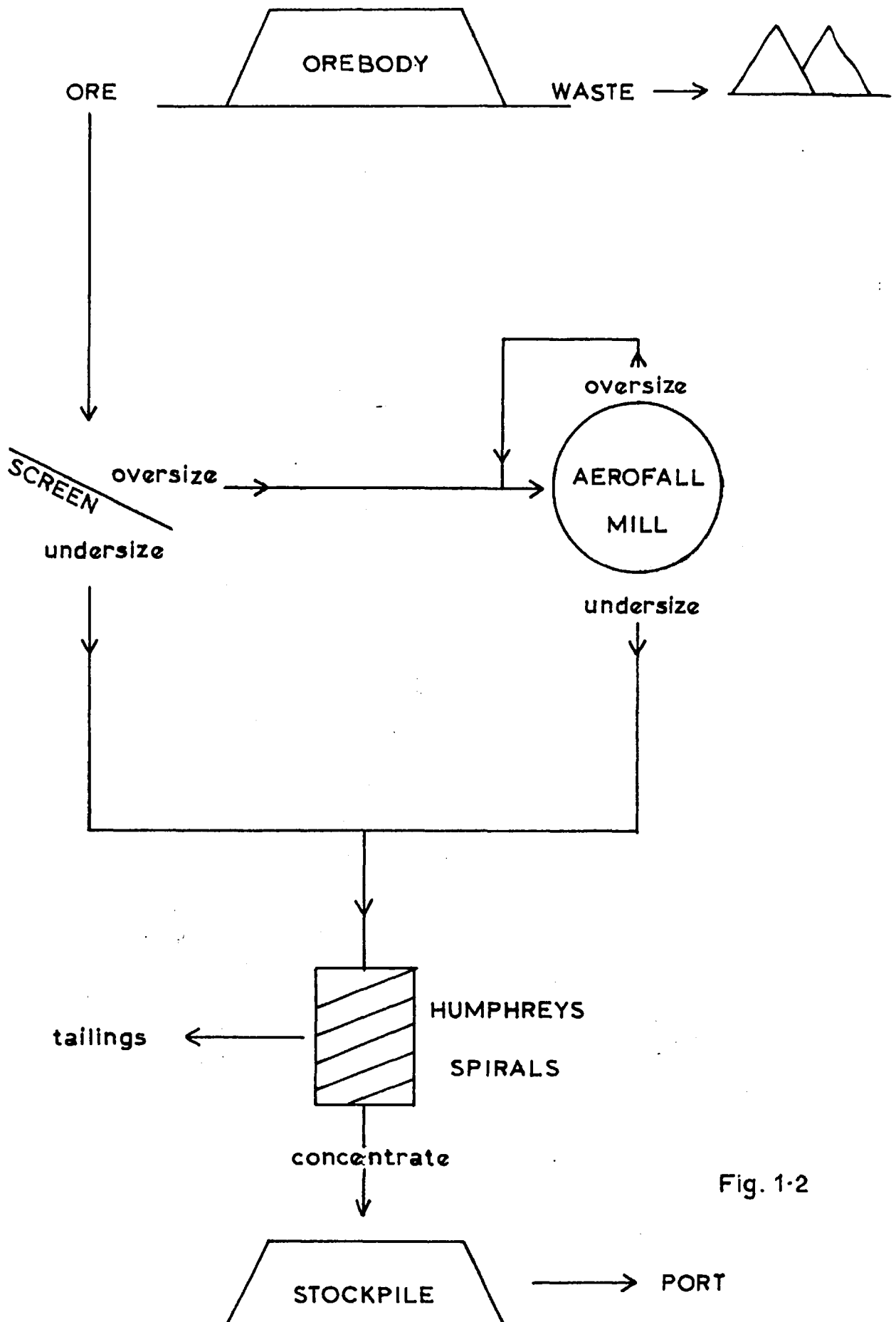


Fig. 1-2

SIMPLIFIED MARAMPA FLOWSHEET

stage coarse classifier passes to a second stage cyclone classifier, the coarse fraction from which passes to the concentration plant for beneficiation. The fine fraction from the cyclone classifier passes to a dust collector, the fine fraction from which is exhausted to the atmosphere. The coarse fraction from the dust collector is gravity transported to the tailings swamp. Magnetic and electrostatic beneficiation of part of the coarse fraction from the cyclone classifier is also carried out.

The main beneficiation of the ore takes place on Humphrey spirals, a simple separating device based on the specific gravity differentials of the valuable mineral hematite (specific gravity 5.2) and the gangue minerals, mainly quartz and mica which have a specific gravity of about 2.6 (Chapter IV). The ore is diluted with water at a water to ore ratio of about 7:1 (by weight) and then introduced at the top of the bank of spirals. The ore/water slurry passes down the spiral with the higher density hematite tending to remain at the inside of the spiral and the less dense gangue tending to be washed to the outside of the spiral. The hematite passes down ports on the inside of the spirals and is dewatered in rake classifiers and then stockpiled. The gangue passes through the spiral system to thickeners and eventually to a tailings swamp. Ancillary beneficiation equipment includes cones, hydrocyclones and thickeners.

The spirals are arranged vertically in various combinations. The usual arrangement is either a 5-turn spiral followed by a 3-turn spiral or three 5-turn spirals in series and it has been found that concentrate grade usually increases with the number of spirals over which the ore passes, and that a three 5-turn spiral system usually produces a higher grade concentrate than a 3 + 5-turn spiral system.

The concentrate is reclaimed from the stockpile by a bucket-wheel reclaimer and loaded into railway wagons each containing about 47 tons of concentrate. Each trainload contains about 1600 tons of concentrate.

Daily concentrate production at Marampa is about 8,000 tons,

the monthly total usually being in the range 210,000 to 230,000 tons.

The beneficiation of the Marampa ore and the operation of the Humphreys spirals are further described and discussed in Chapters III and IV.

D. NATURE OF THE MARAMPA GRADE CONTROL PROBLEM

Grade or quality control in the context of mining operations is an attempt to produce a consistently acceptable product at minimum cost. This present investigation of the Marampa grade control problem is concerned with the production of the consistently acceptable product and no attempt has been made to evaluate the costing side of the problem.

At present the grade of the Marampa iron ore concentrate shipments is acceptable and averages about 64.5 per cent Fe but it is apparent that the mean grade of the concentrate has decreased in recent years and if this trend towards lower concentrate grades is unchecked, and projected into the future, the mean grade of the concentrate may eventually fall below that grade accepted as a minimum. It will be demonstrated (Chapter III) that a statistically significant decrease in concentrate grade has occurred over the period January 1967 - June 1968 and it is also apparent that the mean grade of ore supplied to the mill has decreased in recent years. The main problem concerning grade control at Marampa is that the grade of the concentrate cannot be predicted from a knowledge of feed grade alone, i.e. two mill feeds of similar or even identical grade may yield concentrates with significantly different grades and it is now recognized that physical characteristics of the ore and technical factors associated with milling operations may have more influence on the grade of the concentrate than does the grade of the mill feed. This lack of a prediction model for concentrate grade is probably the single most important aspect of the Marampa grade control problem.

It has become apparent during the course of this present investigation that the hematite grainshape, number and

orientation of the Humphreys spiral withdrawal ports, the variable hardness of the ore and other factors affect the concentrate grade and at the same time it has also become apparent that any analysis of the effects of these variables is most difficult to quantify. Qualitative and semi-quantitative analyses may be made but the design of mathematical prediction models using such variables does not yet appear to be practicable.

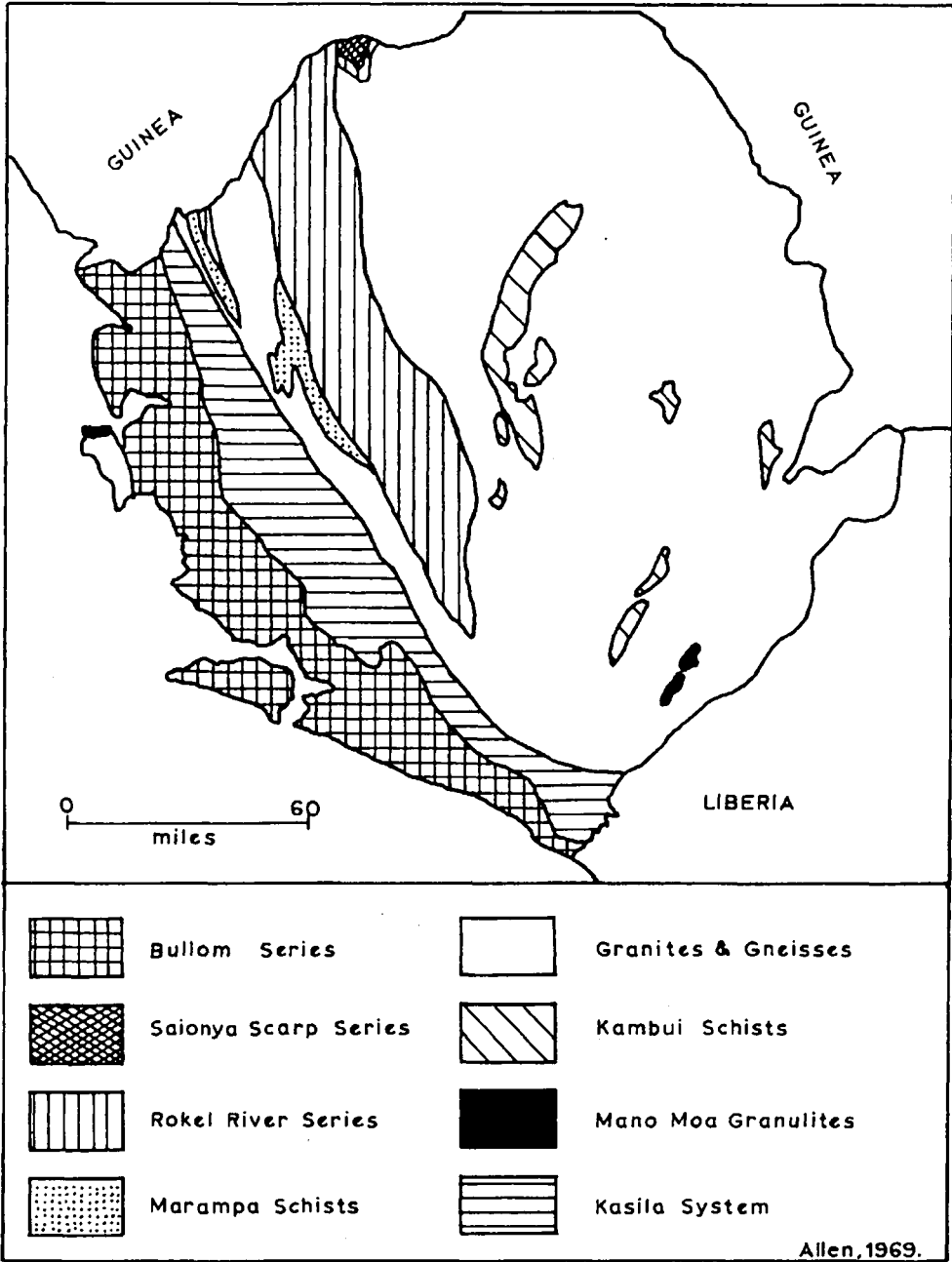
The present world iron ore scene favours the buyer and will probably do so for several decades; supply of high grade iron ore exceeds demand and the buyer is able to dictate the price of iron ore as perhaps never before. In such a competitive market those producers which offer a beneficiated product have the additional cost of ore-dressing to consider and, other factors being equal, the potential profits are less. If Marampa concentrate grades cannot be predicted, mean grades continue to decrease and sub-standard concentrate shipments are produced, the profitable sale of Marampa iron ore concentrates could conceivably be jeopardised.

This present investigation of the Marampa grade control problem embodies the results of investigations into various aspects of mining, milling, sampling, assaying and mineralogy of the Marampa deposit and is largely of a statistical nature since it is considered that analyses of assay data and other numerical information must be objectively studied and assessed and statistical analysis with its foundation of probability theory lends itself admirably to such an investigation.

E. THE GEOLOGY OF SIERRA LEONE

The bulk of this description of the geology of Sierra Leone is based on the work of Allen, (1969).

Seven main rock groups of Precambrian or Palaeozoic age occur in Sierra Leone (Fig. 1.3), in addition to various basic intrusions and the Pleistocene to Recent sediments of the Bullom Series. These seven rock groups are, from oldest to youngest; Kasila System, Mano-Moa Granulites, Kambui Schists, Granite and Acid Gneiss, Marampa Schists, Rokel River Series and Saionya Scarp Series.



GEOLOGICAL MAP OF SIERRA LEONE

Fig.1-3

Sierra Leone is structurally and morphologically divisible into two main parts; a western coastal plain about 70 miles wide and the central highlands. The eastern structural and morphological unit (central highlands) contains the Kambui Schists, Granites and Acid Gneisses and the Mano-Moa Granulites. A K/Ar age of 2195 ± 85 million years has been obtained on mica from a schist in the Kambui Schists (Allen, 1969). The western structural and morphological unit contains the Kasila System, Marampa Schists, Rokel River System and some rocks of the Granites and Acid Gneisses. The four rock groups of the western unit occur in belts parallel to the coast and are structurally similar and were probably deformed in a common tectonic event. K/Ar dating of mica from the Marampa Schists and the Kasila System gave ages of 550 and 480 million years respectively. Allen (1969) in his study of the western unit concluded that the Marampa Schists and the Rokel River Series constitute the filling of a geosyncline sited in a basement of Granites and Acid Gneisses and the Kasila System.

The Kasila System

The rocks of the Kasila System occur in a belt, up to 20 miles wide, extending from Guinea to Liberia. The rocks are mainly acid gneisses, charnockites, garnet-hornblende gneiss and garnet-plagioclase gneiss.

The Mano-Moa Granulites

The Mano-Moa Granulites consist of a physically small group of enderbites, migmatites and other metamorphic rocks occurring in southern Sierra Leone.

The Kambui Schists

The Kambui Schists consist of metamorphosed sedimentary, volcanic and ultrabasic igneous rocks in the Kambui Hills and of banded ironstones, quartzites, mica schists, amphibolites, conglomerates and metamorphosed ultrabasic rocks in the Sula Mountains.

Granites and Acid Gneisses

The Granites and Acid Gneisses are composed mainly of granodioritic synkinematic gneisses.

The Marampa Schists

The Marampa Schists consist of folded and metamorphosed sedimentary and volcanic rocks including sericitic quartz schists, quartzites, mica schists, garnetiferous biotite schists and the hematitic schists mined at Marampa.

Pelitic schist is the main rock type of the Marampa Schists although in the area around the Marampa orebody quartz-mica-epidote schist, quartzite and quartz-feldspathic schist are locally dominant. Also in this area bands of amphibolite, black quartzite and garnet-mica-amphibole schist occur within the pelites and are considered by Allen (op cit) to be metamorphosed lavas and pyroclastic rocks.

Metamorphic ultrabasic and basic rocks are present within the schists in the area around Marampa and include serpentinite, olivine-serpentinite, actinolite schist and garnet amphibolite.

The lithology of the Marampa deposit consists essentially of hematite schist ore and quartz-muscovite schist waste.

Most rocks of the Marampa Schists have been metamorphosed in the greenschist facies.

Four phases of folding have been identified from the minor structures of the Marampa Schists (Allen, op cit).

(a) The first phase of folding probably took place during the metamorphism and gave rise to schistosity parallel to the compositional banding. The only fold of this phase is an isoclinal fold with axial plane parallel to the external schistosity and it is suggested by Allen that this fold was formed during a phase of syntectonic isoclinal folding.

(b) The second phase of folding is widely represented and took place after the metamorphism. The folds formed are tight with flat or inclined axial planes and a crenulation cleavage

parallel to the axial plane in the hinge area but absent in the limbs.

(c) The third phase of folding resulted in open and symmetrical folds with vertical or steeply dipping axial planes with axes trending between 280 and 350 degrees and plunging gently NW or SE.

(d) The early structures, particularly in the vicinity of the Marampa deposit, have been re-oriented by large open folds with vertical NE-striking axial planes with gently plunging axes.

Rokel River Series

The rocks of the Rokel River Series consist of marls, quartzites, sandstones and volcanic rocks which include quartz andesite, basalt, dolerite, pillow lavas and acid tuffs.

Saionya Scarp Series

This series consists of unmetamorphosed, flat-lying arenaceous sedimentary rocks and form a small outlier on the northern Sierra Leone/Guinea border.

F. THE GEOLOGY OF THE MARAMPA DEPOSIT

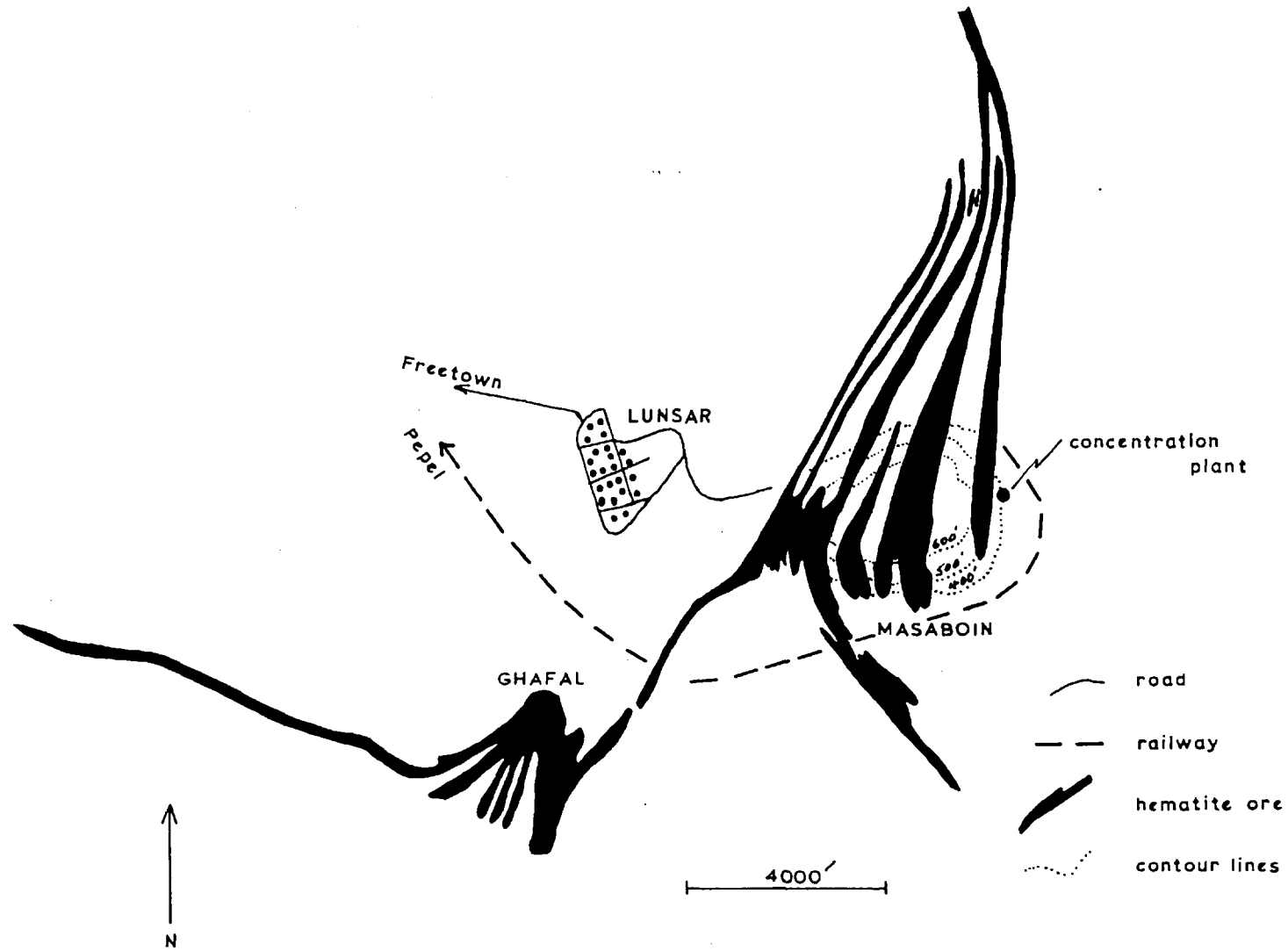
From the grade control and ore reserve estimation point of view the geology of the Marampa deposit is fairly simple, the structure of the Masaboin Hill area being understood sufficiently well to enable ore reserve estimations to be made, and the bodies of ore and waste being sufficiently well defined so as to obviate the need for detailed mapping or selective mining.

(a) Previous investigations

The hematite deposits at Marampa were discovered in 1926 by W.R. Junner, then Director of the Geological Survey of Sierra Leone, and production of iron ore commenced in 1933.

The main accumulations of hematite in the deposit form two upstanding hills, Masaboitanki (now known as Masaboin

Fig. 1-4



HEMATITE BODIES AROUND MARAMPA

Hill) and Dafila (now known as Ghafal Hill), (Fig. 1.4). Before mining commenced Masaboin and Ghafal Hills were about 550 and 250 feet above the level of the surrounding country respectively, but the height of Masaboin Hill, the main orebody and the only one presently being worked, has been reduced some 250 feet by subsequent mining operations.

A considerable amount of development work, by means of adits and shafts, was carried out by the African and Eastern Trade Corporation in 1928 and resulted in the following general description, (Junner, 1929).

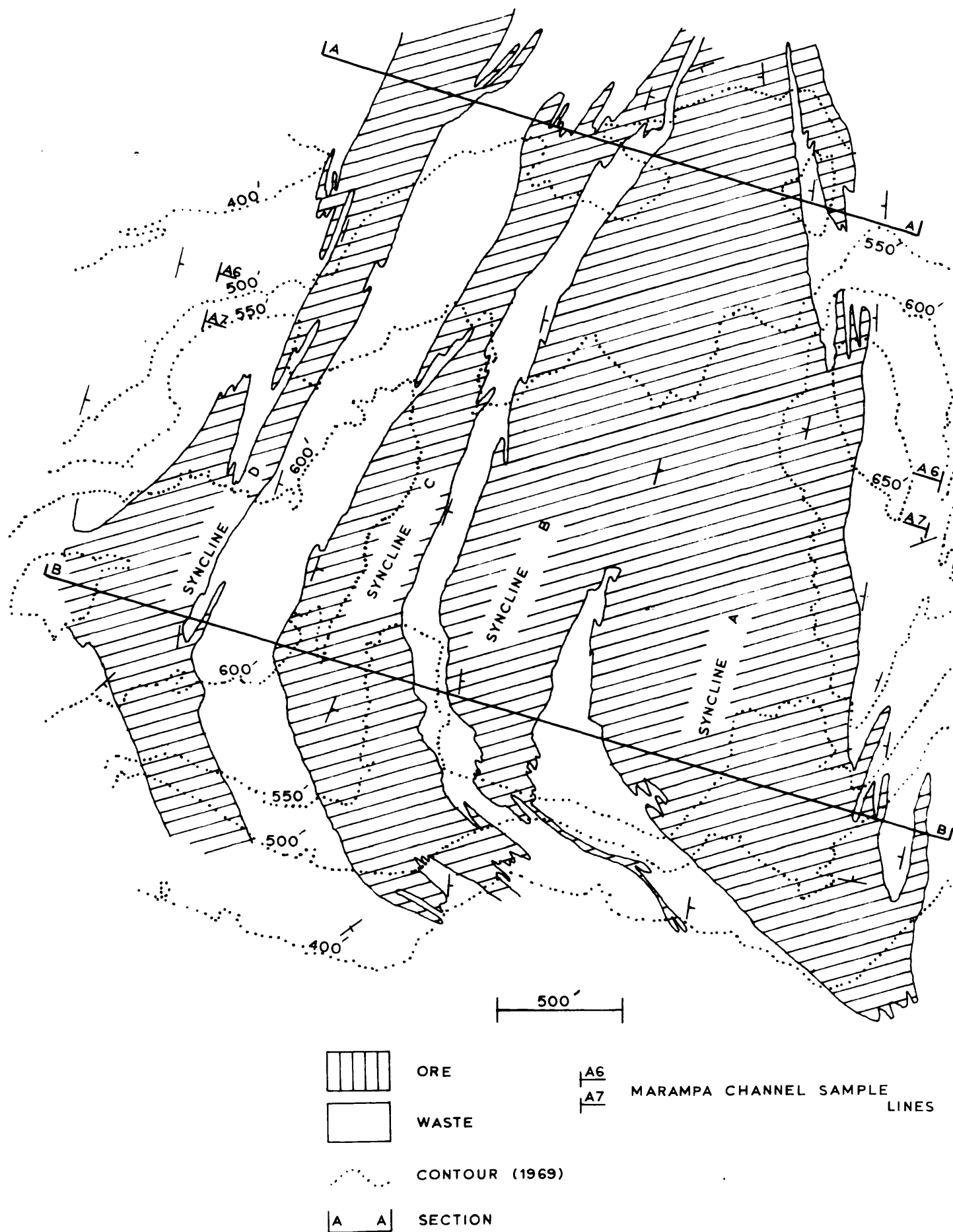
- (a) Thin capping of loose boulders of hard ore.
- (b) A layer of hard red, brown and black ore conforming roughly to the contours of the ground surface. This layer varied in thickness from a few feet to at least 35 feet and averaged between 15 and 20 feet. At the base of this layer the ore became softer and darker and gradually passed downwards into,
- (c) high grade specular hematite containing parallel partings and thin lenticular beds of muscovite schist.

Analyses of the ore made at this time included, (Junner, 1930);

	A	B
Fe	63.6 per cent	63.4 per cent
SiO ₂	1.9 " "	1.6 " "
Al ₂ O ₃	3.2 " "	4.2 " "
Mn	0.14 " "	0.18 " "
TiO ₂	0.23 " "	0.23 " "
P	0.04 " "	0.04 " "
S	0.09 " "	0.10 " "

- (A) Bulk sample from top of Masaboin Hill.
- (B) Bulk sample from top of Ghafal Hill.

No indications of iron-bearing carbonates or sulphides were noted during the initial investigation of the deposit and it was concluded (Junner, 1930) that the specular hematite represented primary ore of Precambrian age which had been subsequently metamorphosed, a view which is still held at present.



GEOLOGICAL MAP, MASABOIN HILL

(Kennedy and Thomson, 1962)

These early investigations were mainly concerned with determining the extent and grade of the hematitic ore and little, if any, detailed mapping of the orebody appears to have been carried out.

The relative ease of mining and milling of the Marampa ore and the high grade of the concentrates has, in the past, resulted in a lack of attention being paid to the mineralogical and structural features of the deposit and it was not until 1962 that the structure of the deposit was determined.

This investigation (Kennedy and Thomson, 1962) was based on the results of a combined mapping, diamond-drilling and auger-drilling programme and was mainly concerned with elucidating the structure of the Masaboin Hill area for the purpose of estimating the total ore reserves.

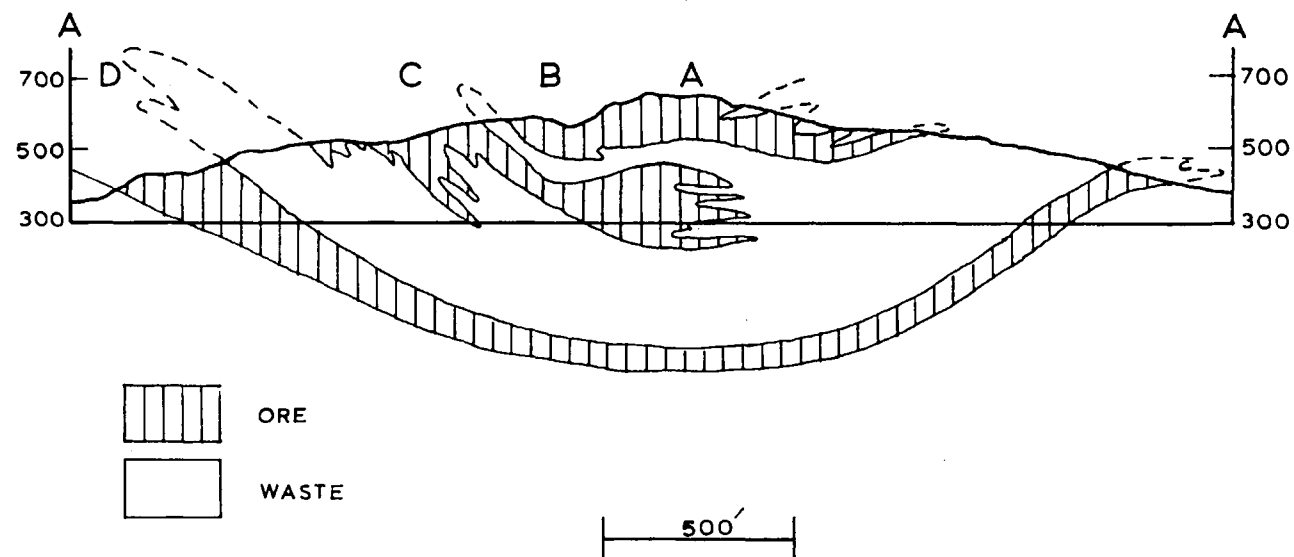
(b) Structure

The structure of the Masaboin Hill area of the Marampa deposit was interpreted by Kennedy and Thomson (1962) to be, "a series of overturned to recumbent isoclinal folds, the axial planes of which dip to the east, except along the extreme eastern side where the axial planes of recumbent folds dip to the west", (Figs. 1.5, 1.6, and 1.7). The validity of this interpretation appears to have been confirmed by the results of the subsequent mining operations.

The ore is contained in four major ore synclines, designated, from east to west as A, B, C and D (Fig. 1.5). These ore synclines are separated by anticlinal bodies of quartz-muscovite schist waste.

The structural interpretation of Kennedy and Thomson implies that the present ore distribution is due to the deformation of a single ore zone but mineralogical evidence and grainsize analyses (Chapter V) suggest that, in fact, two ore zones were originally present, one now being represented by synclines A and D, the other by synclines B and C.

The textural and grainsize differences which exist between the ores of synclines A and D and those of synclines B and C



GEOLOGICAL SECTION AA, MASABOIN HILL

are certainly significant but may have been caused by differential response to deformation, the ore now forming synclines A and D suffering more severe deformation than that now forming synclines B and C.

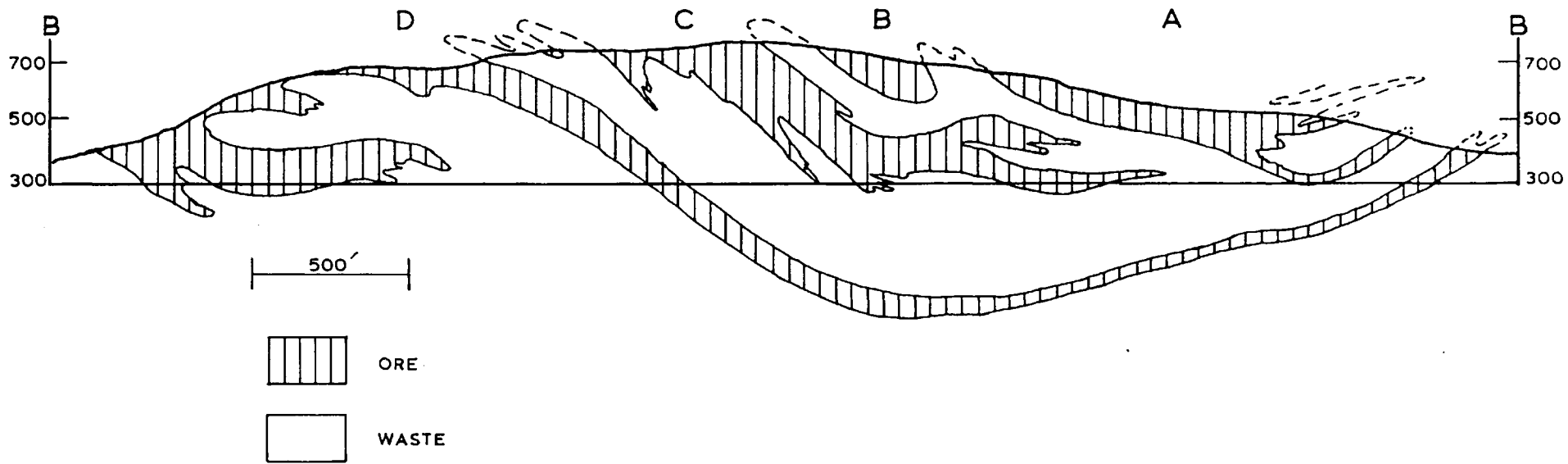
Very little faulting appears to have occurred at Marampa and only minor transverse displacements have been observed. Shearing has taken place parallel to the compositional banding and cleavage crenulations can be observed to have been sheared out. The shearing has had the effect of masking or destroying the continuity of the compositional banding to the extent that individual bands can rarely be traced along strike for distances greater than about 100 feet, particularly in synclines A and D, in which the continuity of the banding is usually much less. An analysis-of-variance (ANOVA) of along and across-strike assay data (Chapter VII) has however, shown that grade distributions in the orebody are anisotropic and that the greatest grade variations occur across strike, thus confirming that little across strike faulting has occurred.

(c) Lithology and mineralogy.

The ore consists essentially of hematite-quartz-muscovite schist with lesser amounts of manganese oxides which appear to be concentrated along the margins of synclines B and C. The ore synclines are separated by anticlinal bodies of quartz-muscovite schist waste.

The originally low grade ores of Masaboin Hill have been upgraded by leaching which has abstracted silica and made the ore friable to such an extent that blasting of the ore has generally been unnecessary. With increasing depth of mining below the original topographic surface of the hill however, the effects of the leaching are diminishing and in many areas of the orebody the boundary between leached and unleached ore has been reached and blasting of the ore (and waste) is becoming common. Together with the increase in ore hardness there has been a steady decrease of ore grade.

The Marampa grade control problem is intimately involved with the decrease in ore grade and the increase in ore hardness



GEOLOGICAL SECTION BB, MASABOIN HILL

and with the textural characteristics of the two types of ore, i.e. the ore of synclines A and D and the ores of synclines B and C. These textural differences are considered to have important economic implications in the beneficiation of the Marampa ore on Humphreys spirals.

The hematite of syncline A and D is coarse grained (44 per cent coarser than 60 B.S. mesh), micaceous to tabular and has undergone in situ brecciation. The hematite of synclines B and C is fine grained (23 per cent coarser than 60 B.S. mesh, compare with . synclines A and D), equidimensional, commonly exhibits an annealed texture and has not undergone in situ brecciation. The different types of hematite respond in different ways to spiral beneficiation and, as the mill feed at any one time consists of a mixture of ore from the various synclines, it is postulated that the optimum milling conditions are rarely, if ever, in operation.

The mineralogy and textural features of the Marampa ore and the effects of these features on spiral beneficiation are described in detail in Chapter V. The implications of the increase in ore hardness and the decrease in mean ore grade are discussed in Chapter IV.

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CHAPTER II

DESCRIPTION AND APPLICATION OF SOME STATISTICAL TECHNIQUES IN GEOLOGY

A. INTRODUCTION

This chapter briefly describes some of the statistical techniques employed in the analysis of the Marampa data and for a more detailed description of statistical methods and their application to geology the following texts are recommended; Moroney (1951), Dixon and Massey (1957), Miller and Kahn (1962), Krumbein and Graybill (1965), Rickmers and Todd (1967) and Blais and Carlier(1968).

Geology, as with the other sciences, has its foundation in measurement and comparison but until recently, analysis of numerical geological information depended upon subjective interpretations, which are prone to personal bias, rather than objective interpretation. In the past the application of statistics to geological problems has lagged behind similar applications in the other sciences but statistical analysis of numerical data is now common in virtually all branches of geology.

Application of statistical analysis to the earth sciences is readily divided into three periods (Miller and Kahn, 1962);

(a) 1890 to early 1930's. Mainly the statistical analysis of paleontological data.

(b) The middle 1930's to 1941 saw a considerable expansion in the statistical analysis of sediment size data but still only little application of statistics to geological fields other than sedimentology and paleontology.

(c) Post World War II saw considerable expansion of statistical analysis into virtually all fields of geology.

The combination of geology and statistics has given rise to the inter-disciplinary science of geostatistics which finds its greatest practical application in grade control and valuation or ore reserve estimation in the mining industry. The

statistical analysis of sampling data is still in its infancy and has only come into relatively common use in the last 20 years. At present three main geostatistical schools exist;

(a) The South African school of Krige (1951, 1960, 1961, 1962), Sichel (1946-47, 1952) and others who have been mainly concerned with the valuation of the South African gold deposits.

(b) The North American school of Krumbein (1954, 1955), Hazen (1958, 1961, 1962, 1963, 1967) and Agterberg (1961, 1964, 1967b) who have undertaken statistical analysis of sampling data from various types of ore deposits and other geological features for ore reserve estimation and optimization of sampling techniques and as an aid in geological interpretation.

(c) The French school of Matheron (1963) and Serra (1966, 1967) has been mainly concerned with developing statistical methods to deal with non-random and non-independant sampling data or regionalized variables such as ore grade. The geostatistical methods of the French school do not appear to have yet been used to any great extent in the mining industry, but are expected to be used to an increasing extent once the methods become more widely known.

B. STATISTICAL METHODS IN GEOLOGY

Two spheres of statistical activity exist. On the one hand is descriptive statistics which concerns the summary and presentation of large amounts of information in forms which are readily appreciated and able to be used as a basis for geological interpretation and generalization, i.e. tables, graphs or histograms of, say, heavy mineral content of beach sands or annual production of iron ore in the U.K. On the other hand is analytical or inferential statistics in which, "the samples are taken as part of a larger class of geological objects or events (the geological population) about which generalizations and predictions are to be made by statistical inference combined with geological judgement" (Krumbein and Graybill, 1965).

Classical statistics is based mainly on probability theory which is itself developed on the assumption of random data or random sampling methods in which every individual in the population has an equal and independent chance of being chosen for a sample (Dixon and Massey, 1957) but in practice, particularly in the collection of geological information, this prerequisite is rarely, if ever, met because of prohibitive cost or inaccessibility of data and due allowance must be made for the fact that geological samples are almost never collected according to random sampling methods. Drilling of orebodies for grade information is almost invariably carried out on a systematic pattern and while the location of the first drill-hole may be randomly decided the location of the remainder of the drillholes are non-random.

Although mine sampling data is obtained from sample locations which are rarely non-random the assay results may be considered to be random provided that the mineral particles being sampled are randomly distributed within the host rock or are associated with small-scale structural features that are randomly distributed (Hazen, 1962).

The techniques of statistical analysis employed in the analysis of the Marampa mine sampling data may be roughly divided into two groups;

- (a) Techniques concerned with tests of comparison and determination of significant differences.
- (b) Techniques concerned with determination of degrees of association and formulation of prediction models.

Techniques described in (a) above are used to determine whether statistically significant differences exist between, say, two (or more) sampling methods or between the mean assay values of two (or more) ore types. Statistically significant differences are those differences which cannot be explained by or attributed to chance alone. Techniques included in this group are;

- (i) Students "t" distribution test
- (ii) χ^2 test (Chi-square test)
- (iii) Analysis of variance (ANOVA)

(iv) Wilcoxon's Signed Rank test

The application of these tests in the field of mine planning has important economic implications in that they may be used to determine the "best" or most cost-effective sampling method or whether or not significant improvements have been effected in mining or milling practice by introduction of different techniques. Similar quality control based on statistical analysis of sampling data has long been used in other industries.

The techniques described in (b) above can be used to determine whether or not statistically significant associations or correlations exist between two (or more) variables and to make prediction models concerning the variables. These techniques are possibly the most important in the analysis of mine sampling data and are a prerequisite to the full interpretation and understanding of numerical geological data. Techniques included in this group are;

(i) Correlation and regression analysis.

(ii) The variogram and the geostatistical techniques of Matheron.

The main techniques used in the analysis of the Marampa sampling data are as follows.

(a) Students "t" Distribution Test

Applications of this test, devised in 1908 by W.S. Gosset under the pseudonym "Student", are used in two main forms;

(a) To determine whether or not a statistically significant difference exists between the mean values of two groups of samples. A value of t is calculated from the sample data and compared with standard tables listing the critical values of t . If the calculated value exceeds the critical value at the required confidence level then a statistically significant difference is taken to exist. The value of t is calculated from the equation;

$$t_{v_1+v_2} = \frac{\bar{X}_1 - \bar{X}_2}{s_p \sqrt{1/n_1 + 1/n_2}} \quad \dots \dots \dots \text{Equat. 2.1}$$

where, $\bar{x}_1, \bar{x}_2$ = mean values of sample groups 1 and 2.

n_1, n_2 = number of samples in sample groups 1 and 2.

v_1, v_2 = number of degrees of freedom of sample groups 1 and 2.

s_p = pooled standard deviation of the two sample groups.

$$= \sqrt{\frac{v_1 s_1^2 + v_2 s_2^2}{v_1 + v_2}} \quad \dots \dots \text{Equat. 2.2}$$

where, s_1^2, s_2^2 = variances of the two sample groups

(b) To determine whether or not statistically significant differences exist between paired data such as assay results of samples taken by two different methods from the same location. In this case the calculated value of t is given by the equation;

$$t_v = \frac{\bar{d} \sqrt{n}}{s_d} \quad \dots \dots \text{Equat. 2.3}$$

where, $\bar{d}$ = mean value of difference between the two values making up each pair.

n = number of pairs of samples

s_d = standard deviation of the difference between the two values making up each pair.

(b) Analysis of Variance (ANOVA)

Analysis of variance (ANOVA) is a statistical technique (developed by R.A. Fisher) with which the total variation in a set of data may be reduced to components associated with possible sources of variation. ANOVA techniques are based on the fact that the variance of a sum of random variables is equal to the sum of the variances of these random variables if the variables concerned are uncorrelated. In such circumstances, if two (or more) uncorrelated factors introduce variability into a set of measurements, the total variability can be separated into individual components adding up to the total.

In ANOVA procedures the sum of squares and the number of degrees of freedom are calculated for each source of variation.

Now, the sum of squares for the source of variation divided by the relevant number of degrees of freedom is simply the usual estimate of the variance and for each source of variation there is an appropriate variance estimate. If a Null Hypothesis is proposed of no significant difference between the variance estimates given by the different possible sources of variation, the different variance estimates are therefore estimates of the same parent variance. It would not be expected that the variance estimates given by the different sources of variation would be exactly the same since each would be a reflection of the variance of the common parent population, but it would be expected, according to the Null Hypothesis, that the differences would not be statistically significant. Snedecors Variance Ratio Test (F test) is a simple test for determining whether or not statistically significant differences occur between variance estimates. The test requires division of the numerically larger variance estimate by the smaller variance estimate and comparing the result with standard tables with the relevant number of degrees of freedom at the required confidence level. If the calculated value of F is greater than the critical (standard table) value then a statistically significant difference is shown to exist between the two estimates of the parent variance.

A useful relationship for comparing the relative variability of two distributions, e.g. two groups of assay values, is the coefficient of variation (v) in which the standard deviation is expressed as a percentage of the mean value.

$$\text{i.e.} \quad v = \frac{100s}{\bar{X}} \quad \dots \quad \text{Equat 2.4}$$

where $\bar{X}$ = mean value

s = standard deviation

(c) Chi-Square Test

The reality of apparent associations may be tested by application of the Chi-square (χ^2) distribution which allows quantitative comparisons of observed results with expected results, i.e. the χ^2 test determines whether the observed frequencies in a distribution are significantly different from the frequencies which may be expected according to some hypothesis which has been formulated. The test may be used, for example, to determine whether an observed set of data is normally distributed.

The Chi-square test statistic is;

$$X^2 = \sum \frac{(O - E)^2}{E} \quad \dots \quad \dots \quad \dots \quad \text{Equat 2.5}$$

where,

X^2 = calculated value of test statistic

O = observed frequency of variable

E = expected frequency of variable

The calculated value of X^2 is compared with standard tables at the required confidence level and with the relevant number of degrees of freedom. If the calculated value of X^2 is greater than the critical (standard table) value then a significant difference is shown to exist and a Null Hypothesis would be confounded.

(d) The Variogram and Geostatistics

Matherons variogram (Matheron, (1963), Blais and Carlier (1968)) embodies a mathematical method of studying the dispersal or spatial association of regionalised variables which are variables holding a definite value at each point in space and which numerically describe natural phenomenon such as ore grade, thickness of a lithological unit or other geological features having intrinsic dispersion. The variogram is the basic tool of the French school of geostatistics and, it is claimed, accounts for the non-random and non-independant relationships which are common occurrences in the field of geology.

It may be intuitively recognized that the grade of ore samples separated by increasing distances would tend to be increasingly different, i.e. samples close together would tend to have similar grades whereas samples separated by large distances would tend to have dissimilar grades. This is more easily visualised if two extremes of sample separation are considered; copper assays of two samples taken from within the same chalcopyrite grain would be expected to be almost identical whereas copper assays of two samples separated by a distance of miles would be expected to be completely different, i.e. they would be independant. This concept is related to the classical concept of the area of influence of a sample. In ore **reserve estimation and valuation each sample assay value is** assigned an area of influence related to the geometry of the sampling grid and it is tacitly assumed that the grade assigned to a sample can be extended halfway to the adjacent sample

location. This is the usual method of ore reserve estimation but it fails to account for the continuity and anisotropy of grade which exists in most ore bodies, e.g. in a sedimentary deposit (iron, bauxite or manganese deposits) the ore grade variations along strike are usually less than those across-strike.

Classical statistics based on probability theory considers random sampling data and does not effectively account for geometrical features of geological data such as sample location and sample size which are usually systematic or non-random features. Geostatistics, on the other hand, takes into account the fact that geological data (sample assay values, thickness of a lithological unit etc.) are often non-random and non-independent and that such data usually has a certain degree of spatial continuity. It should be noted however that the geostatistical methods of Matheron and others of the French school do not yet appear to have been accepted by the mining industry to any great extent. This is due to several factors including;

(a) The published work of Matheron and others of the French school have been, with very few exceptions, published in French. The main application of Matheron's work published in English include the papers of Matheron (1963), Blais and Carlier (1968), Budenicek and Hass (1969) and David (1969) but it is expected that much further application of such work will be made in the future after the theories of Matheron have become more widely known.

(b) The mathematical foundation of Matheron's theories appear, at first sight, to be rather ponderous and it has been reported (Bubenicek and Hass, 1969) that the methods are cumbersome and do not necessarily afford appreciable gains in accuracy.

(c) The older, accepted methods of ore reserve estimation based on probability theory or even rule of thumb have been satisfactory and a change has not been warranted, especially when dealing with normally (Gaussian) distributed data.

The variogram, which forms the basis of the geostatistical

methods of Matheron, is a graphical expression of a mathematical technique which considers the variance of the difference between regionalised variables separated by increasing distances. The calculated value of the function is plotted on the ordinate and the lag or distance between the sample locations is plotted on the abscissae, the resultant curve giving the degree of natural or intrinsic dispersion of the particular regionalized variable. The procedure for calculation is as follows:

Consider Fig. 2.1 which represents a number of sample assay values located along a linear sample traverse.

The variogram function $\gamma(h)$ is calculated for increasing separations of the samples and is given as the mean of the sum of the squares of the differences between sample values separated by constant intervals. There is therefore an ordinate value of the variogram for each sample separation value, i.e. the variogram for a sample traverse with N sample locations would have N - 1 ordinate values. The variogram function is given by;

$$\gamma(h) = \frac{\sum_{i=1}^{L-h} (x_{i+h} - x_i)^2}{2(L - h)} \quad \dots \quad \text{Equat. 2.6}$$

where,

$$\begin{aligned} \gamma(h) &= \text{variogram function.} \\ x &= \text{sample value (grade, thickness etc.)} \\ h &= \text{separation of adjacent samples.} \\ L &= \text{length of linear sample series.} \end{aligned}$$

The variogram function is calculated for each value of h and plotted against h to give the graphical expression of the variogram.

In the geostatistical treatment of sample assay values the variogram possesses the following attributes (Blais and Carlier(1968));

(a) It is an expression of the correlation and area of influence of the samples. Its rate of increase is a measure of the rate of decreasing influence of assays separated

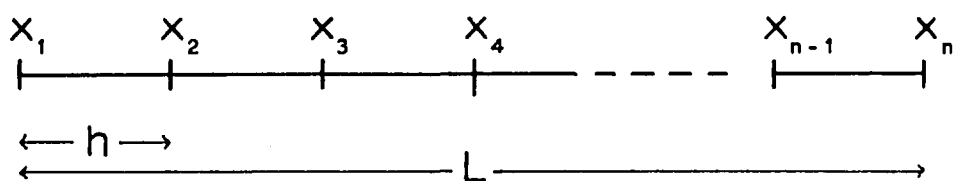
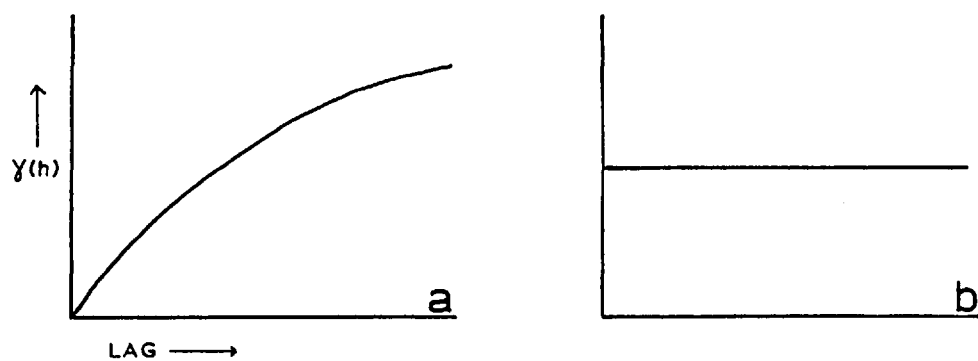


Fig. 2.1



VARIOGRAM

MODELS

Fig. 2.2

(a) De Wijsian

(b) Random

by increasing distances.

(b) Mineralisation anisotropies are revealed by differential behaviour of the variogram for different directions in the orebody.

(c) Transitional phenomena may be revealed by variogram features. The variogram function has a reasonably constant value when calculated with random data which indicates that when the variogram function has reached a maximum and levelled off to a plateau, then no correlation exists between sample values, i.e. the "area" of influence has been exceeded and no continuity of mineralisation exists between samples separated by the distance indicated on the variogram.

(d) The continuity and regularity of mineralisation are reflected by the behaviour of the variogram near the point of origin. The variogram function of bulk high grade deposits, e.g. iron ore, tends to zero towards the origin whereas that of low grade deposits, e.g. gold deposits, reveals a discontinuity at the origin indicating irregular grade (high variance) distribution and very low continuity of grade between samples. This discontinuity at the origin of the variogram and of the grade distribution is a numerical expression of what is known as the nugget effect which occurs when the ore type under consideration is more or less randomly distributed as discrete grains or clusters of grains in a large mass of gangue and is a feature of most low grade ore deposits.

Ore deposits of different types yield variograms with characteristic curves and three basic variogram models are associated with different types of mineralization. The three models are;

(a) The De Wijsian model which appears to be typical of precious and base metal deposits particularly of the vein type (Fig. 2.2a).

(b) The Random model, which appears to be typical of trace element and other low grade deposits, is characterised by a large nugget effect and a horizontal variogram (Fig. 2.2b).

(c) The Transitive model. This model appears to be typical of bulk high grade sedimentary deposits such as iron ore, bauxite and manganese deposits and, because of its application to the Marampa deposit, will be described in greater detail.

(e) The Transitive Variogram

Several types of ore deposits, particularly those characterised by high grades and large volumes (Fe, Al, Mn) also appear to be characterised by a transitive variogram in which the regionalised variable conforms to the following law of intrinsic dispersion (Blais and Carlier, 1968).

$$\gamma(h) = C_0 + CT(h,a)$$

where (Fig. 2.3),

h = Distance

a = Range of correlation ("area" of influence).

The range is the distance over which the regionalised variables, in this case the assay values, are spatially correlated.

C_0 = Nugget effect. This is a measure of dispersion of assay values at small values of h , i.e. when samples are very close together.

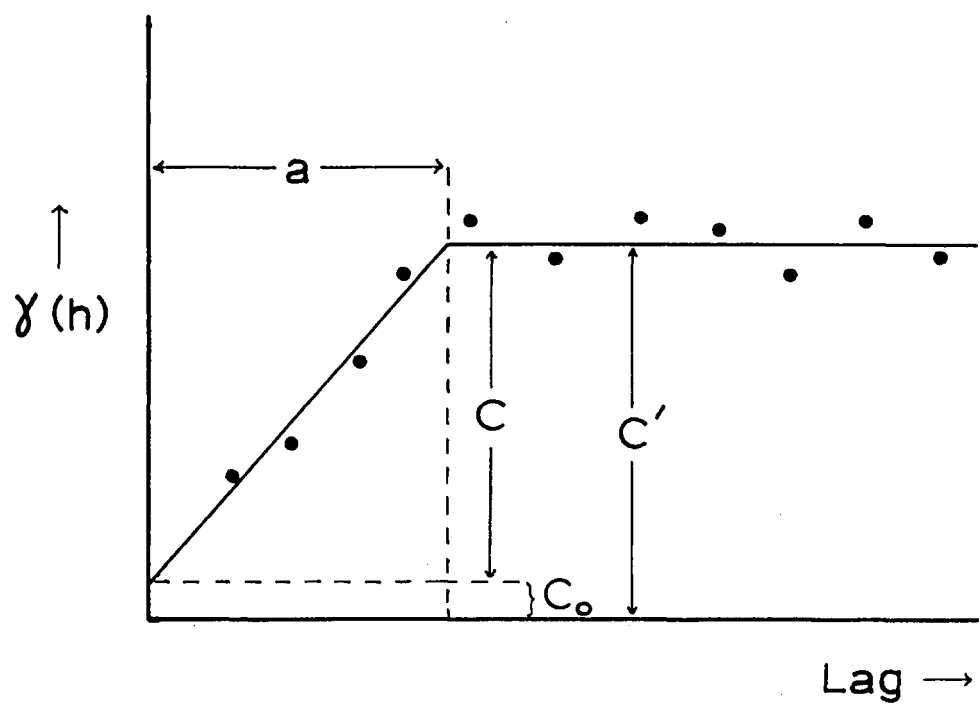
C = Pitch. This is a constant given by the difference between assay variance and the nugget effect.

$T(h,a)$ = Transition function equal to h/a for $h < a$ and equal to 1 for $h > a$

C' = Sill where $C' = C + C_0$. The sill is the limiting value of $\gamma(h)$ at distances $h > a$ and is equivalent to the assay variance.

Anisotropy of mineralization or grade distribution may be demonstrated by construction of a two (or three) dimensional variogram, that is, a variogram is constructed from assay values of samples taken along mutually perpendicular sample traverses. An isotropic grade distribution would yield two (or three) coincident variogram functions whereas an anisotropic grade distribution would yield non-coincident variogram functions.

The distance over which the regionalised variable, assay values for example, is spatially correlated is readily



TRANSITIVE VARIOGRAM

a = range

C_0 = nugget effect

C = pitch

C' = sill

Fig. 2.3

found by construction of a variogram and it is assumed that for distances greater than a (the range) the regionalised variables are independent. Serra (1966) considers that the minimum sample separation in an orebody characterised by a transitive variogram should be at least equal to 95 per cent of the full development of the variogram, i.e. approximately equal to the variogram range.

An isotropic grade distribution implies that the vertical and horizontal sample separations should be equal i.e. a square sampling grid, whereas an anisotropic grade distribution requires that the sampling grid should be rectangular to give the optimum sampling density.

(f) Wilcoxon's Signed Ranks Test

This test determines whether or not a statistically significant difference exists between pairs of data and can thus be used, for example, to compare two sampling methods, the pairs of data being made up of the assay values of samples from the two methods.

The test depends upon the fact that if there is no significant difference between the paired data, any chance differences should consist of about an equal number of positive and negative differences. This rank test for differences in paired data is often preferable to a Student's t test because it is "distribution-free", i.e. it is independent of the nature of the distribution of the parent population be it normal, lognormal or any other. It is also a sensitive test in that it takes into account not only the direction (positive or negative) but also the magnitude of the differences.

The difference between the two pieces of data making up each pair are arranged in increasing absolute order and assigned rank values (1 to n where n is number of pairs of data) for each difference value, tied ranks being assigned average rank values. Each rank value is then assigned a plus or minus sign according to whether the original difference between the two pieces of data making up each pair was positive or negative, and the positive and negative ranks are

separately summed.

The smaller the value of the smaller rank total (R), the greater the difference between the two sets of data. Critical values of R for six to 25 pairs of data are found by recourse to standard tables (Rickmers and Todd, 1967, pp 402). For greater than 25 pairs the significance of R is found by calculating Z;

$$Z = \frac{\frac{1}{2} n(n + 1) - 2R}{\sqrt{\frac{n(n + 1)(2n + 1)}{6}}} \quad \dots \quad \dots \quad \text{Equat. 2.7}$$

where,

R = smaller rank sum

n = number of pairs of data

The calculated value of Z is then compared with critical values of Z as follows;

Probability	10%	5%	1%	0.2%
Z	1.64	1.96	2.58	3.09

If the calculated value of Z is greater than the critical value at the desired confidence level then a significant difference is shown to exist between the two sets of data.

(g) Correlation Analysis

Correlation analysis is a statistical technique to provide measures of the degree of association existing between variables and is closely related to the concept of regression. The correlation coefficient is related to the standard error of estimate (S_E) which measures the dispersion of the points about the regression curve, i.e. $r = f(S_E)$ where r is the correlation coefficient.

If the correlation coefficient, r, is to be a satisfactory measure of association or correlation it should have four main features;

(a) It should be large when the degree of association is high and small when the degree of association is low.

(b) It should be independent of the units in which the variables are measured.

(c) It should indicate whether the association between the variables is positive (direct) or negative (inverse).

(d) It should have theoretical upper and lower numerical limits in order for comparisons to be made between correlation coefficients.

The correlation coefficient is defined by the equation

$$r_{xy} = \frac{n \sum x_i y_i - \sum x_i \sum y_i}{\sqrt{(n \sum x_i^2 - (\sum x_i)^2)(n \sum y_i^2 - (\sum y_i)^2)}} \quad \text{Equat 2.8}$$

and $-1 < r < 1$, where $r = -1$ or $+1$ represents perfect negative (inverse) or positive (direct) linear association respectively between the two variables X and Y. A value of zero indicates that no linear association exists between the variables.

The value of the square of the correlation coefficient is known as the coefficient of determination and is a measure of the amount of variation in one variable which may be attributed to simultaneous variation in the second. It is usually expressed as a percentage, e.g. $r_{xy} = 0.707$, $\therefore 100r_{xy}^2 = 50.0$ per cent, indicating that 50 per cent of the variation occurring in X may be attributed to variation in Y.

Values of the coefficient of determination may be used arbitrarily to group correlation coefficients into orders of significance after statistical significance has been shown by recourse to standard tables.

- (a) $r = \pm 0.71$, $100r^2 = 50$ per cent — strongly significant
- (b) $r = \pm 0.50$, $100r^2 = 25$ " " — moderately significant
- (c) $r = \pm 0.32$, $100r^2 = 10$ " " — weakly significant

The significance of a particular correlation coefficient increases with increasing sample size, that is, the confidence

limits about the value of r tend towards r with increasing n , and a value of 0.5 for r for sample size $n = 10$ may not be as significant as a value of $r = 0.2$ for sample size $n = 100$, and care must be exercised in determining the significance of r in relationship to the sample size.

Although the correlation coefficient quantifies the degree of association existing between two variables it is, in the first instance, a statistical association and not necessarily a cause and effect relationship and should be used as a predictor only with caution. A high degree of association may be purely fortuitous and in a statistical analysis of geological data, geological criteria should be used to determine the cause of any significant relationship.

Partial correlation methods determine the degree of association between two variables when other variables acting simultaneously are "eliminated" or regarded as being held constant. They are utilised when the magnitude of the total or simple correlation coefficient between two variables may be masked by the influence of other variables which were ignored for calculation purposes. The partial correlation coefficient is therefore used to find the correlation coefficient between two variables when the effect of other variables is accounted for but held constant.

The standard formula for first order partial correlation coefficients is:

$$r_{ij:k} = \frac{r_{ij} - r_{ik}r_{jk}}{\sqrt{(1 - r_{ik}^2)(1 - r_{jk}^2)}} \quad \dots \quad \text{Equation 2.9}$$

where $r_{ij:k}$ denotes the partial correlation coefficient between variables i and j when the effects of variable k is held constant and where r_{ij} etc. are the total correlation coefficients between the variables i , j , and k . Similarly, a second order partial correlation coefficient is given by the equation:

$$r_{ij:kl} = \frac{r_{ij:k} - r_{il:k}r_{jl:k}}{\sqrt{(1-r_{il:k}^2)(1-r_{jl:k}^2)}} \dots \text{Equat 2.10}$$

which measures the degree of association between variables i and j while the effects of variables k and l are held constant.

Standard tables are used to determine significance levels and confidence limits for the coefficients of correlation.

(h) Regression Analysis

Regression analysis is a statistical technique for describing the relationship between two (or more) variables, and is presented in equation form giving values of the response variable (Y) in terms of the predictor variable or variables (X). The equation describes the line of best fit between the variables; the line may be straight (linear regression) or curved (curvilinear regression) and if more than one predictor variable is used the lines are multiple linear or multiple curvilinear regression lines respectively.

Lines of regression are lines of best fit, that is, the sum of the squares of the deviations, either horizontal or vertical, between the regression line and the points used in making up the line are at a minimum. The method of least squares is used for calculating the regression equations and two regression lines exist for each set of data, the regression lines of X on Y and the regression line of Y on X. The concept of regression analysis is closely related to that of correlation analysis and the divergence of the two regression lines is a measure of the degree of correlation between X and Y. Perfect correlation where $r = \pm 1$ results in the two regression lines having the same equation and thus coinciding and perfect non-correlation where $r = 0$ yields regression lines which intersect at right angles. The two regression lines intersect at the mean values of X and Y except in the case of perfect correlation.

The least squares method of fitting the regression line

to the data is superior to the subjective method of fitting the line by eye or personal judgment except in the case of perfect correlation where all points fall on a simple line, either straight or curved. In all other cases the least squares method is systematic and objective and gives a unique solution for the given data.

The linear regression model of Y on X is:

$$Y_x = b_0 + b_1X + e \quad \dots \quad \text{Equat.2.11}$$

the regression coefficients b_0 and b_1 being calculated as follows:

$$b_1 = \frac{n\sum XY - \sum X \sum Y}{n\sum X^2 - (\sum X)^2} \quad \dots \quad \text{Equat.2.12}$$

$$b_0 = \bar{Y} - b_1\bar{X} \quad \dots \quad \text{Equat.2.13}$$

where Y = response variable
X = predictor variable
e = error

The multiple linear regression model is:

$$Y_x = b_0 + b_1X_1 + b_2X_2 + e \quad \dots \quad \text{Equat.2.14}$$

the regression coefficients b_0 , b_1 and b_2 being calculated as follows:

$$(SSX_1^2)b_1 + (SSX_1X_2)b_2 = SSX_1Y \quad \dots \quad \text{Equat. 2.15}$$

$$(SSX_1X_2)b_1 + (SSX_2^2)b_2 = SSX_2Y \quad \dots \quad \text{Equat. 2.16}$$

$$\text{and } b_0 = \bar{Y} - b_1\bar{X}_1 - b_2\bar{X}_2 \quad \dots \quad \text{Equat. 2.17}$$

where SS indicates the corrected sums of squares.

Calculation of regression lines of variables which have a statistically significant coefficient of correlation gives rise to equations which may be acceptable as prediction models relating response variable to predictor variable (or variables).

Mean deviation (mean of absolute deviations), maximum deviation and average deviation (mean deviation with sign of deviation accounted for) are used to test regression equations

for acceptance as prediction models.

The most acceptable prediction model has the following features:

- (a) Mean deviation smaller than that for any other prediction models proposed and should, ideally, be zero.
- (b) Maximum deviation smaller than that for any other prediction models proposed and again, ideally, should be zero.
- (c) Average deviation of zero. A zero value indicates no bias towards undervaluation or overvaluation.

In practice the first two prerequisites are rarely met in the statistical analysis of the Marampa beneficiation data and, with the sampling and assaying techniques used and the natural variations which occur in milling conditions, are unlikely to be met, and an acceptable prediction model for Marampa grade control purposes would require the following features:

- (a) mean deviation < 0.4 Fe per cent
- (b) maximum deviation < 0.5 Fe per cent
- (c) average deviation < 0.05 Fe per cent

In practice the average deviation has been found to be acceptable with typical values being of the order of 0.008.

NOTE. Unless otherwise stated in the text, statistical significance is at the 95 per cent confidence level.

CHAPTER III

STATISTICAL ANALYSIS OF CONCENTRATE GRADE AND GRAINSIZE DATA

A. INTRODUCTION

The exact nature of the Marampa grade control problem is rather difficult to define since many operations which take place between the mining of the ore and the shipping of the concentrates have some influence on the quality of the final concentrate. In addition, the quantitative effects of most of these operations, particularly those taking place in the mill, are difficult, if not impossible, to measure.

The factors which influence concentrate quality may be divided into three broad categories;

(a) Geological factors concerned with the mineralogy, degree of deformation and grade of the ore.

(b) Mining factors, including mine planning, which cover the technical aspects of mining methods and transport of ore to the mill.

(c) Milling factors concerned with feed rate, pulp density, number of concentrate withdrawal ports and other technical factors involved in spiral beneficiation.

In order to determine more accurately the nature of the Marampa grade control problem and to indicate the more important factors influencing the quality of the Marampa concentrates a statistical analysis has been made of the grade variations of all the 127 hematite concentrate shipments despatched from Pepel over the eighteen month period January 1967 - June 1968, together with a statistical analysis of the grainsize distributions of 20 of these shipments. The more important aims of these statistical analyses were;

(a) To examine the variations of concentrate grade with time. The eighteen month period allows relatively long-term grade variations to be examined and evaluation of significant variations, if any, can be readily made. The main emphasis was on the variations of the hematite and quartz

contents of the shipments but variations in the other constituents were briefly examined.

(b) To examine the hematite distribution of the various screen fractions of the concentrate. A knowledge of the grainsize fractions which are relatively enriched or depleted in hematite may enable the results of the beneficiation test data to be more effectively interpreted and may even contribute to the design of more effective mining or milling procedures.

(c) To isolate any factors concerned with the geological control of the concentrate quality. If concentrate quality variations can be related to such geological factors then it may be possible to map these factors in the orebody thereby allowing the potential concentrate quality to be more accurately assessed.

B. STATISTICAL ANALYSIS OF CONCENTRATE SHIPMENT HEMATITE VARIATION

The mean grade of the iron ore concentrate shipments over the eighteen month period January 1967 - June 1968 is as follows (Table 3.1);

Constituent	Mean Grade (per cent)
Fe	64.8
SiO ₂	5.73
Al ₂ O ₃	0.85
Mn	0.17
Ti	0.10
P	0.008

Table 3.1: Mean grade of Marampa iron ore concentrate shipments over the period January 1967 - June 1968.

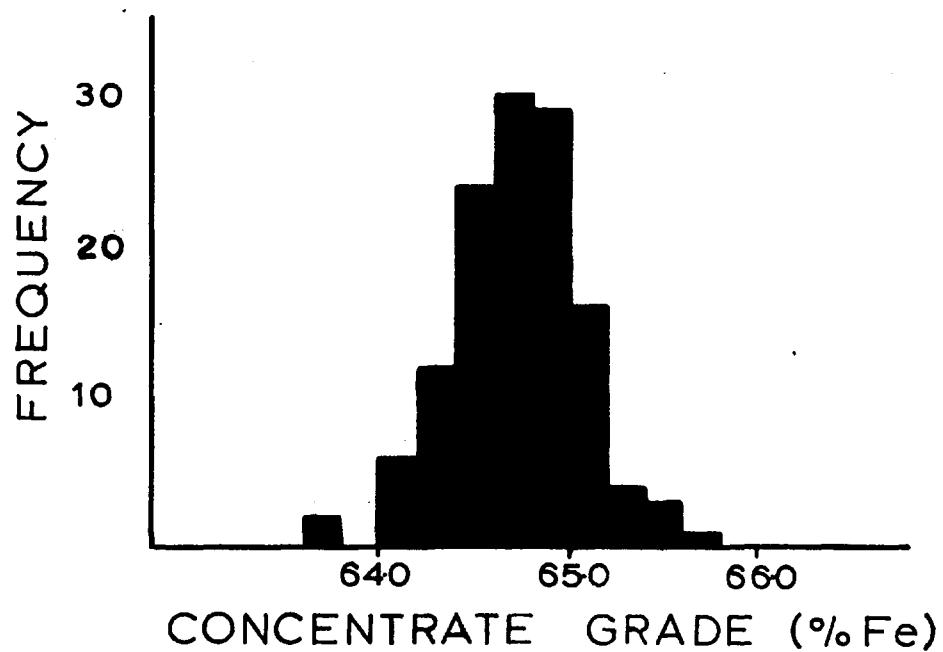
The iron grades of individual shipments range from 63.70 to 65.75 per cent Fe and have the following distribution (Table 3.2).

Grade interval (per cent Fe)	Frequency
65.65 to 65.85	1
65.45 to 65.65	3
65.25 to 65.45	4
65.05 to 65.25	16
64.85 to 65.05	29
64.65 to 64.85	30
64.45 to 64.65	24
64.25 to 64.45	12
64.05 to 64.25	6
63.85 to 64.05	0
63.65 to 63.85	2
< 63.65	0

Table 3.2: Frequency distribution of Marampa iron ore concentrate shipment grades over the period January 1967 - June 1968.

This distribution is more readily appreciated from Fig. 3.1 which expresses the frequency distribution in histogram form. A Chi-square (χ^2) test for goodness-of-fit (Rickmers and Todd, 1967) indicates that such a distribution is best described as normal. The Null Hypothesis proposed is that the concentrate grade distribution is not significantly different from a normal (Gaussian) distribution. Using values of;

Mean ($\bar{X}$)	= 64.8 per cent Fe
Standard deviation (s_x)	= 0.335
Number of samples (N)	= 127



Frequency histogram of iron content
of 127 concentrate shipments.

Fig. 3.1

The χ^2 test yields a value of;

$$\chi^2_{\text{calc.}} = 10.02 \text{ with 9 degrees of freedom}$$

Recourse to standard tables yields a critical value of $\chi^2 = 16.92$ at the 95 per cent confidence level.

$$\therefore \chi^2_{\text{calc.}} < \chi^2_{\text{critical}}$$

and the Null Hypothesis is accepted and there is no reason to suspect that the distribution of the concentrate shipment grades is other than normal.

Inspection of the assay values of the concentrate shipments reveals that the mean grade of the shipments has altered significantly over the eighteen month period under consideration. Table 3.3 describes the mean grade, standard deviation and number of shipment assays, for several concentrate constituents broken down into three six month periods.

Constituent	Jan-June 1967			July-Dec 1967			Jan-June 1968		
	Mean	St.dev	N	Mean	St.dev	N	Mean	St.dev	N
Fe	64.98	0.392	39	64.72	0.205	40	64.69	0.371	48
SiO ₂	5.40	0.465	35	5.80	0.452	40	5.92	0.456	48
Al ₂ O ₃	0.83			0.87			0.85		
Mn	0.23			0.15			0.12		

Table 3.3 Grade statistics of concentrate shipments sub-divided by time.

The decrease in the mean iron content of the shipments between the first six months of 1967 and the first six months of 1968 is only 0.29 Fe per cent which may appear, at first glance, to be negligible, but a Students t test applied to the data indicates that such a decrease is statistically significant at the 99 per cent confidence level. In other words, the decrease in mean iron content cannot be attributed to chance factors or sampling effects alone and that the grade decrease is real rather than apparent. The Student t test gave a calculated t value of 3.54 compared with the critical

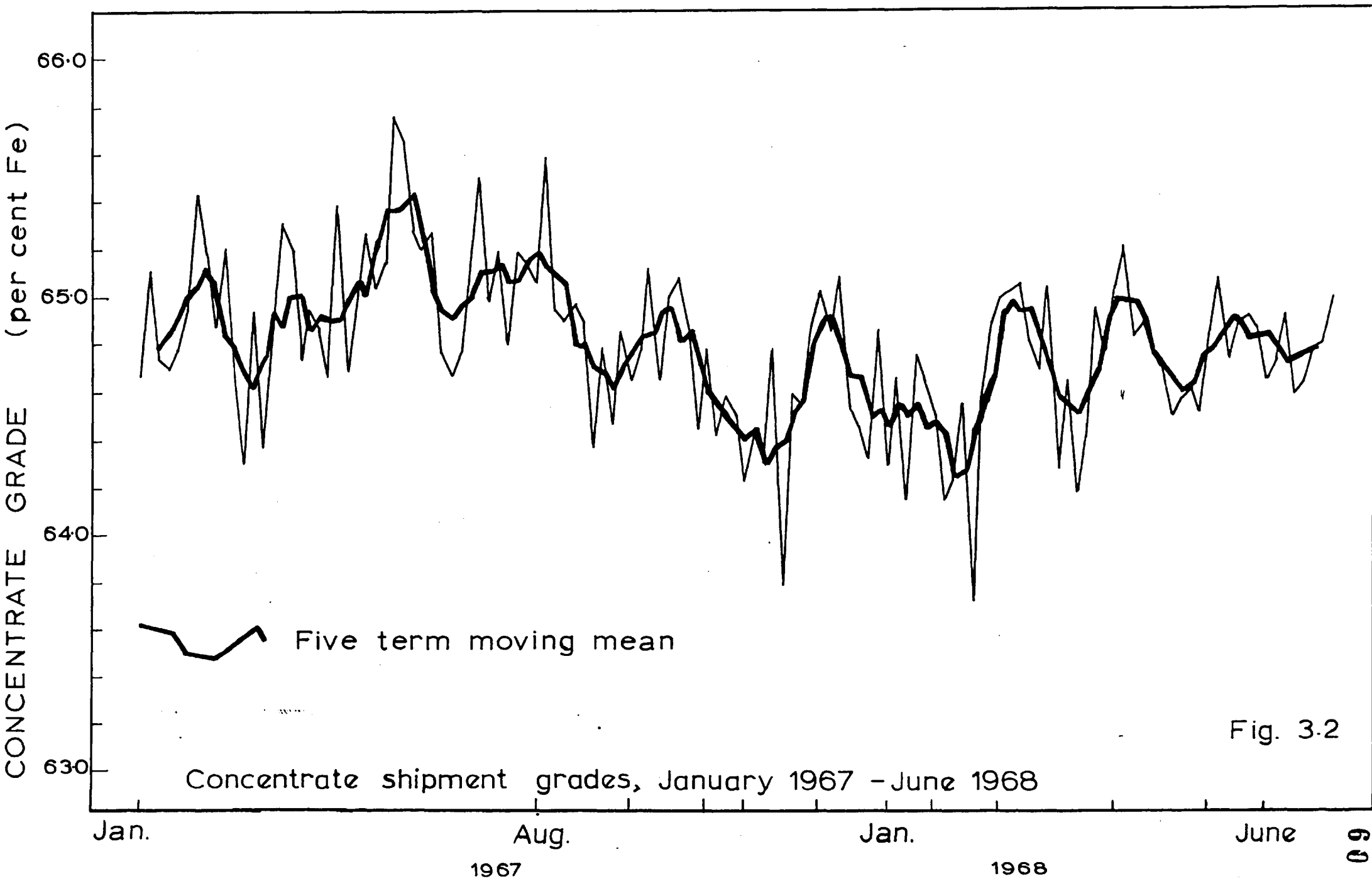
(standard table) value of 2.62.

Concentrate shipments during the first six months of 1968 totalled about 1,162,000 tons. The decrease in mean grade of 0.29 Fe per cent between January - June 1967 and January - June 1968 resulted, if other factors are considered as being unchanged, in a decrease or loss of more than 4500 tons of hematite which probably represents an income loss of many thousands of pounds, and unless this trend towards lower grade shipments is halted, considerable amounts of potential income will not be realised.

The variation of the concentrate shipment grade with time is graphically described by Fig. 3.2 which shows some intriguing features. Low grade shipments, those below about 64.4 per cent Fe, are almost invariably followed by a much higher grade shipment. The two most notable examples being shipment grades of 63.75 and 63.70 per cent Fe followed by shipment grades of 64.57 and 64.58 per cent Fe respectively.

The reason for the relatively large grade increase after a low grade shipment is difficult to explain in terms of ore grade variations or variable mill performance and it is suggested, not without some trepidation, that non-technical factors, particularly factors of management or personnel, may materially contribute to these grade differences. The suggestion is, that after low grade shipments occur, the detrimental financial effects of such shipments is recognized, probably by upper management, and the orders or recommendations necessary to combat the grade decrease are passed to lower management including the Mine, Mill, Planning and Port Superintendents. Mining, milling and concentrate blending are then more closely controlled and supervised and rapid improvements take place in the concentrate shipment grades.

This appears to be demonstrated by the five-term moving mean of the concentrate shipment grades which is shown superposed upon the shipment grade/time graph (Fig. 3.2). The five-term moving mean has a noticeably cyclical nature with periods ranging from about 4 to 12 weeks. The rate of increase from



low grade to high grade is usually much greater than the rate of decrease in grade from high to low and it is suggested that, once acceptable grade shipments are being produced, control and supervision of the mining, milling and blending procedures are relaxed and a gradual grade decrease occurs until another extremely low grade shipment is produced when once again the various production procedures are more closely controlled and supervised and the shipment grade rapidly increases.

It is very difficult to determine the validity of this theory regarding shipment grade variations and other explanations may include;

(a) The very low shipment grades may be due to incorrect sampling or inaccurate assaying.

(b) The cyclical nature of the five-term moving mean may be caused by various cyclical and periodically cumulative factors which favour the production of low grade concentrates.

(c) It was considered that the cyclical nature of the five-term moving mean may have been related to an alternative increase and decrease in the demand for concentrates. The Marampa mill is operated close to capacity level and an increased demand for concentrates would necessitate the mill feed rate being increased. An increase of mill feed rate above the optimum level is usually associated with a decrease in concentrate grade but correlation analysis of the monthly concentrate shipment grade and tonnage yields a non-significant coefficient of correlation and the above hypothesis does not appear to be valid.

C. STATISTICAL ANALYSIS OF HEMATITE GRAINSIZE OF CONCENTRATE SHIPMENTS

Grainsize analyses of twenty of the concentrate shipments yielded the following results (Table 3.4 overleaf)

Size Fraction B.S. mesh	Wt. %	Fe wt. %	Mean Grade % Fe
+ 44	16.9	17.3	66.1
- 44 + 60	12.5	12.8	66.8
- 60 + 85	25.8	25.5	64.0
- 85 +120	20.0	19.4	62.8
-120 +170	14.0	14.0	64.3
-170 +200	5.1	5.2	66.0
-200	5.7	5.8	65.2
Total	100.0	100.0	64.7

Table 3.4: Grainsize analyses of Marampa concentrate shipments during the period January 1967 - June 1968.

Plotting of cumulative weight per cent on arithmetical-probability paper indicates that the grainsize distribution of the concentrate shipments is approximately normal although the open-ended intervals (+44 and -200 B.S. mesh fractions) introduce irregularities into the otherwise straight line.

The distribution of iron in the various screen fractions is almost identical to that of the overall grainsize distributions; the correlation coefficient between the weight per cent and the Fe weight per cent in each of the seven fractions being always greater than +0.99 with the coefficient of determination always greater than 0.98 indicating that more than 98 per cent of the variation in iron distribution in each fraction can be attributed to variations in overall weight per cent. This extremely high correlation is to be expected in an iron ore concentrate in which 90 per cent of the total weight consists of iron oxides. As the concentrate grade increases so the distribution of iron tends to the overall grainsize distribution until at the limit, where the concentrate

consists entirely of iron oxide, the iron and total grainsize distributions coincide.

In spite of the very close similarity between the two distributions there are small but noticeably consistent deviations between the two distributions in certain screen fractions which can initially be explained by disproportionately high or low iron contents but which are fundamentally related to iron (and silica) distributions in the primary ore - functions of original deposition modified by post-depositional alterations of structure, climate and metamorphic history, and to inherent characteristics in the Humphreys spiral method of beneficiation.

From Table 3.4 it can be seen that;

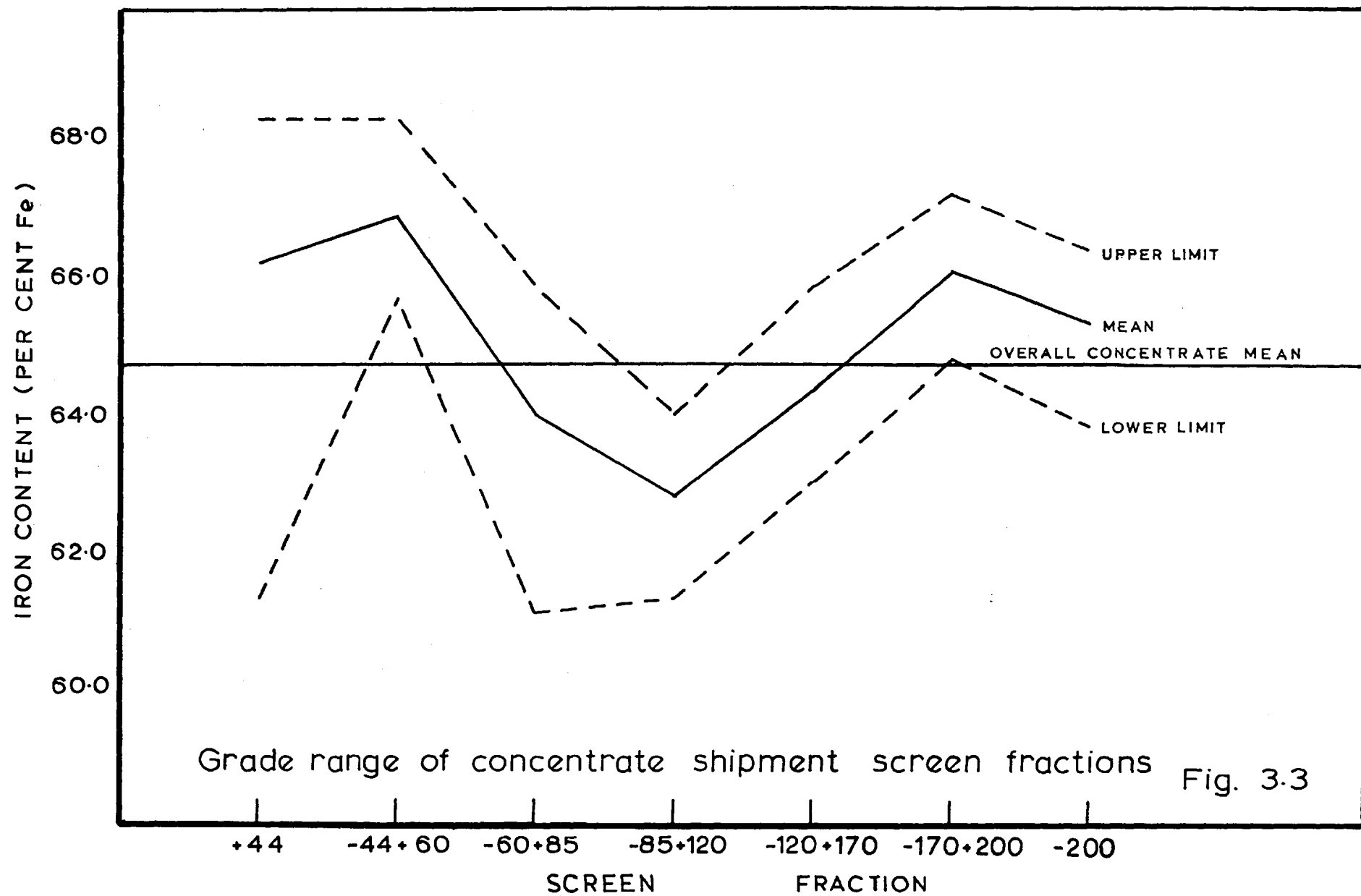
(a) Three major grainsize sub-divisions can be made on the basis of the contribution of iron to the final concentrate (Fig3.3).

(i) Coarse fraction, consisting of the +60 B.S. mesh fractions. This size range contributes 30.1 per cent of the iron from 29.4 per cent of the total weight.

(ii) Midsize fractions, consisting of the -60 +170 B.S. mesh fractions. This size range contributes 58.9 per cent of the iron from 59.8 per cent of the total weight.

(iii) Fine fraction, consisting of the -170 B.S. mesh fractions. This size range contributes 11.0 per cent of the iron from 10.8 per cent of the total weight.

(b) The highest iron grades of each screen analysis are either in the +60 mesh fraction or in the -170 +200 mesh fraction (Table 3.5) and iron grades in the -44 +60 mesh fraction are consistently about 2.0 Fe per cent higher than the overall concentrate grade, and reach as high as 68.3 per cent Fe which is equivalent to greater than 97.5 per cent hematite. This fact invalidates one theory of concentrate grade variation which holds that low grade concentrates are the result of excess amounts of coarse quartz which, the theory suggests, behaves similarly to hematite on the Humphreys spirals and



thus inhibiting the efficient separation of the two minerals. Although there is reason to believe that coarse quartz does behave similarly to hematite on the spirals the amount of such quartz in the Marampa ore is at present negligible. In fact, every -44 +60 mesh fraction of the twenty concentrate shipment samples examined had iron grades greater than the overall concentrate grade and the two coarse fractions, +44 and -44 +60 mesh, together average greater than 1.5 Fe per cent above the overall concentrate grades. These fractions contribute a disproportionately high amount of hematite to the final concentrate and it is apparently advantageous to have this fraction as large as possible.

(c) The three midsize range fractions, -60 +85, -85 +120 and -120 +170 B.S. mesh, which together constitute the bulk of the concentrates are all of consistently lower grade than the overall concentrate grade. The lowest grades of the seven fractions are almost exclusively restricted to the -85 +120 mesh fraction and in all twenty concentrate shipment samples examined, the iron grades of this fraction were below the overall concentrate grade (Table 3.5). Removal of material in this fraction would effect an overall grade increase of between 0.4 and 0.5 Fe per cent but at the prohibitive expense of a 20 per cent reduction in concentrate tonnage.

(d) The fine fraction, consisting of the two fractions -170 +200 and -200 B.S. mesh, have a passive role in the contribution of iron to the final concentrate although it is marginally beneficial; 10.8 per cent by weight contributing 11.0 per cent of the iron.

Table 3.5 (see overleaf)

Screen Fraction	Mean Grade % Fe	Highest Grade Fraction	Lowest Grade Fraction	Above concentrate grade	Below concentrate grade
+ 44	66.1	8	1	17	3
- 44 + 60	66.8	8	-	20	-
- 60 + 85	64.0	-	1	2	18
- 85 +120	62.8	-	17	-	20
-120 +170	64.3	-	1	6	14
-170 +200	66.0	4	-	19	1
-200	65.2	-	-	14	6
	64.7	20	20	78	62

Table 3.5 Screen fraction grade distribution for concentrate shipment grainsize analyses.

D. STATISTICAL ANALYSIS OF CONCENTRATE SHIPMENT IMPURITY VARIATIONS

The distributions of the five main impurities of the Marampa concentrates are graphically described by Figs. 3.4 and 3.5 (SiO_2) and Figs. 3.6a to d (P, Ti, Mn and Al_2O_3).

(a) Variation and distribution of silica

Coupled with the decrease in the mean iron content of the concentrate shipments is an increase in mean silica content from 5.4 to 5.9 per cent SiO_2 . Iron ore concentrates are essentially two-component systems, hematite plus quartz together forming greater than 98 per cent of the total weight of the concentrate and so a decrease in iron content is almost

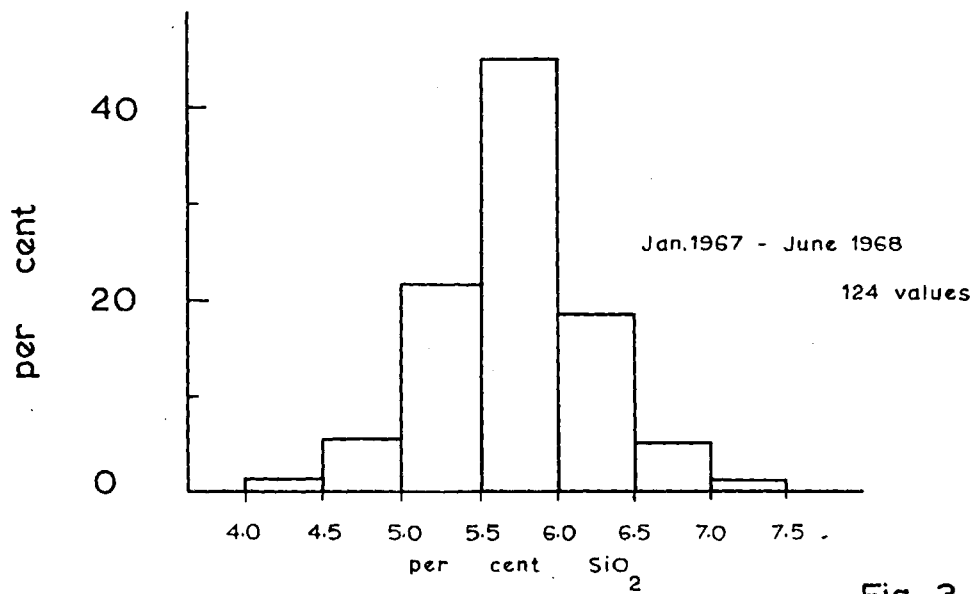
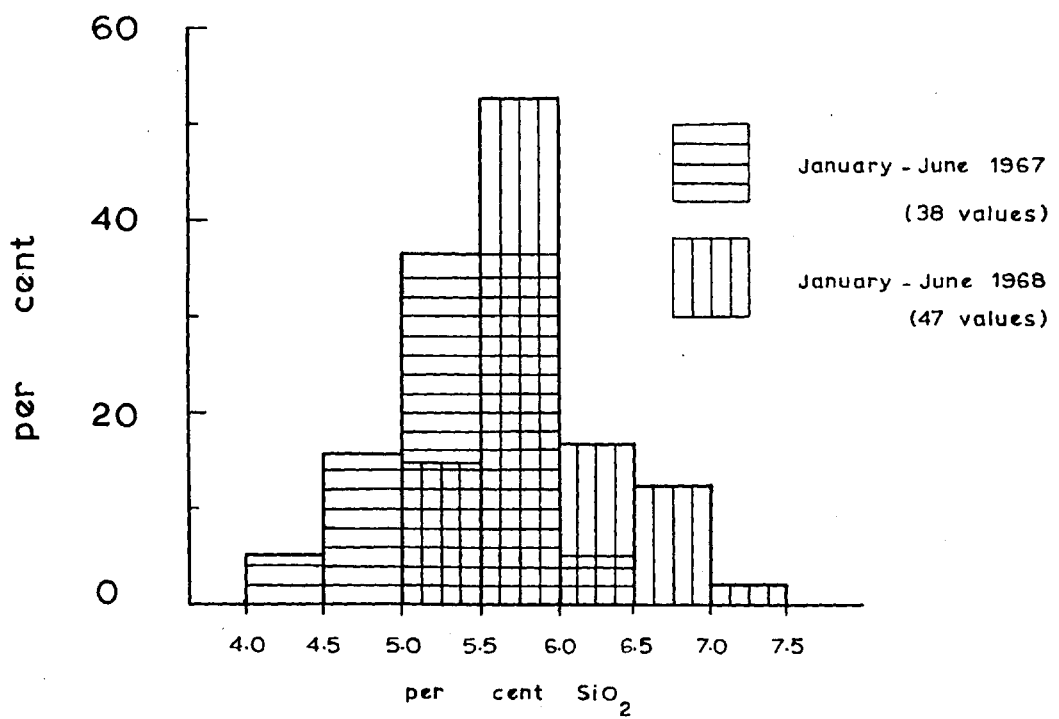


Fig. 3.4



SILICA DISTRIBUTION, CONCENTRATE SHIPMENTS

Fig. 3.5

invariably associated with a concomitant increase in the silica content. A Students t test confirms that the increase in silica content of the shipments is also statistically significant at the 99 per cent confidence level; $t_{\text{calc.}} = 5.09$, $t_{\text{crit.}} = 2.62$.

A study of the silica contents of the concentrate shipments indicates that the grade control policy has, as yet, failed to halt the increase in the silica contents of the concentrate shipments. Table 3.6 should be examined with the knowledge that the grade control objective is to hold the silica content of the concentrate shipments within the range 5.5 to 6.0 per cent SiO_2 .

The proportion of concentrate shipments with silica contents falling within the optimum range (5.5 to 6.0 per cent SiO_2) has steadily increased during the period January 1967 - June 1968 and this is, according to the grade control policy, a desirable trend. It will be noted however that this increase is at the expense of the proportion of shipments with silica contents of less than 5.5 per cent SiO_2 and that the proportion of shipments with silica contents greater than 6.0 per cent SiO_2 has also increased rapidly (see also Figs. 3.4 and 3.5).

The mean silica content of the shipments has increased steadily together with the maximum and minimum silica contents (Table 3.6) and the increase in the proportion of shipments with silica contents within the optimum range is associated with a greater increase in the proportion of shipments with silica contents which exceed the upper limit of the optimum range and it is concluded that the grade control policy has failed in its attempt to keep the mill feed, concentrate and concentrate shipments within the required specifications.

Table 3.6 (see overleaf)

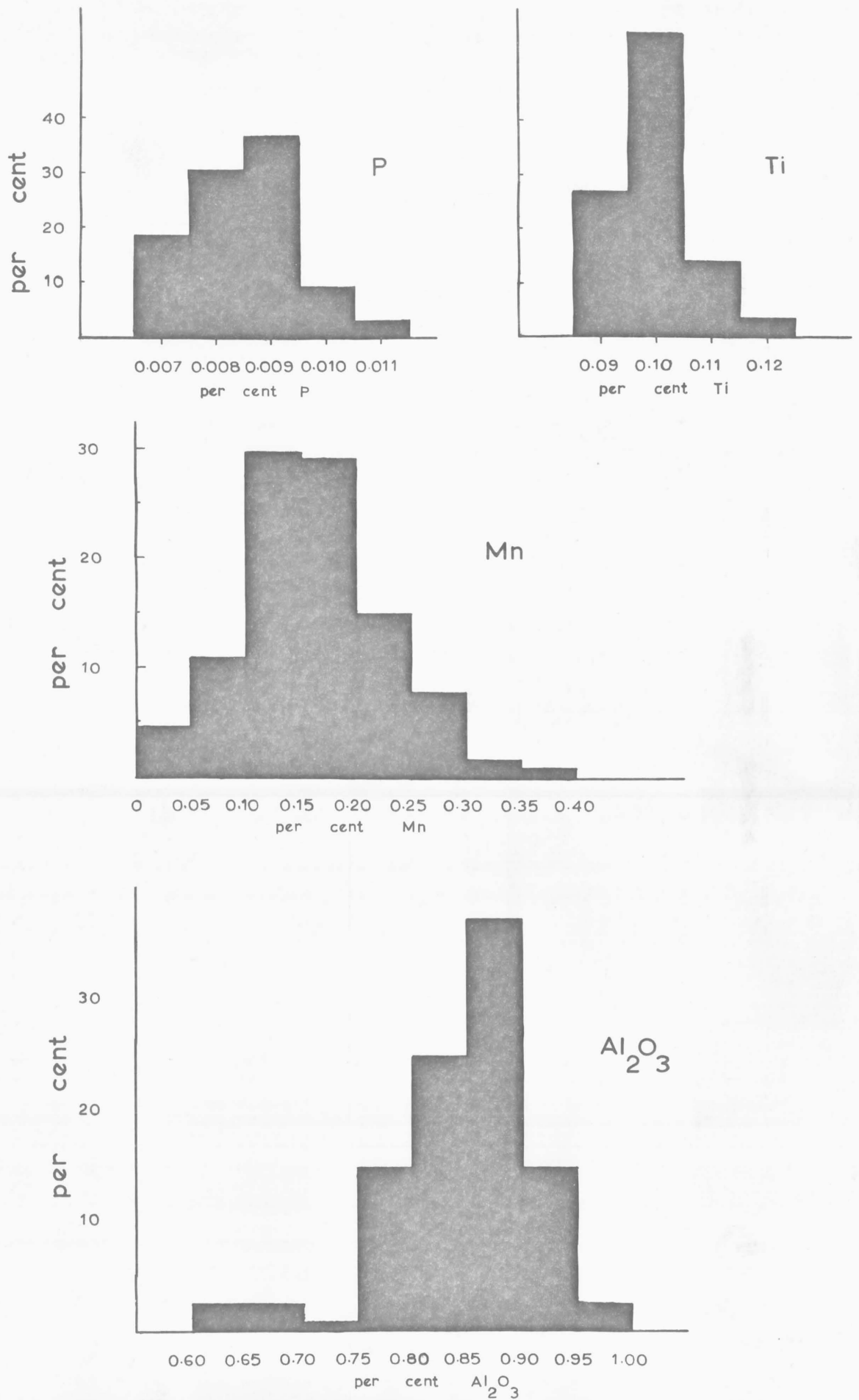


Fig. 3·6

HISTOGRAMS OF CONCENTRATE IMPURITIES

Concentrate shipments silica distribution	January-June 1967	July-Dec. 1967	January-June 1968
>6.0% SiO ₂	5.7 per cent	35.0 per cent	33.3 per cent
5.5 to 6.0% SiO ₂	37.1 " "	45.0 " "	54.2 " "
<5.5% SiO ₂	57.1 " "	20.0 " "	12.5 " "
Maximum SiO ₂ %	6.30	7.08	7.40
Minimum SiO ₂ %	4.28	4.66	5.17
Mean SiO ₂ %	5.40	5.80	5.92

Table 3.6: Distribution of concentrate shipment silica grades, January 1967 - June 1968.

It is considered that the concentrate shipments will continue to have relatively widely fluctuating silica contents until such time that the feed grade variations are damped (Chapter VII).

The increase in the mean silica grade of the concentrate shipments can be clearly observed in Fig. 3.5 which shows the distribution of silica contents of the concentrate shipments for the two periods, January - June 1967 and January - June 1968. It will be noted that over this eighteen month period there is a significant movement of grades from the low silica classes (-5.0 per cent SiO₂) to the mid-silica classes (5.0 to 6.5 per cent SiO₂) and high-silica classes (+6.5 per cent SiO₂).

(b) Variation and distribution of phosphorous

The phosphorous content of the Marampa concentrates is very low and stable, ranging in value from 0.007 to 0.011 per cent P (Fig. 3.6a).

(c) Variation and distribution of titanium

The Ti content of the Marampa concentrates is very stable, ranging in value from 0.09 to 0.12 per cent Ti with an average value of 0.10 per cent Ti (Fig. 3.6b).

(d) Variation and distribution of manganese.

The manganese content of the Marampa concentrates is at an acceptably low level and over the period January 1967 - June 1968 a significant decrease of the mean manganese content has occurred (Table 3.3). This decrease is considered to reflect a closer control of the mining operations in those areas of known high manganese concentration along the margins of synclines B and C. The manganese content of the shipments ranged from 0.03 to 0.38 per cent Mn (Fig. 3.6c).

(e) Variation and distribution of alumina.

The alumina content of the Marampa concentrates is relatively stable and average about 0.85 per cent Al_2O_3 with individual concentrate shipments containing from 0.60 to 0.97 per cent Al_2O_3 (Fig. 3.6d).

The phosphorous, titanium, manganese and alumina contents of the Marampa concentrates are acceptable and relatively steady and the main concern of the grade control policy at Marampa thus appears to rest with the control of the hematite or quartz and the control of the other constituents may be neglected unless extreme and detrimental variations occur. Control of the hematite content of the concentrate automatically confers control of the quartz content and vice versa, and the main mineral dressing and grade control investigations should be concerned with determining the causes of the relatively low hematite contents of the three midsize screen fractions making up the bulk of the concentrates.

A real decrease in concentrate grade has occurred over the eighteen month period January 1967 - June 1968 and while a decrease in mill feed grade could be invoked as a partial explanation, other statistical analyses (Chapter VII) have indicated that such a grade decrease is not apparent, at

least over the thirty feet height of a single mining bench.

E. CONTROLS OF CONCENTRATE QUALITY

The quality of the concentrates produced by spiral beneficiation is dependant upon a number of factors which can be subdivided into three main groups.

(a) Geological controls

These controls include the physical and chemical characteristics of the ore and, more especially, of the hematite. The geological controls can not be altered but their effects can be modified to a certain extent by blending of the ore or by mining the ore to a plan based on grade and physical criteria such as grainsize and grainshape. The grade of the ore is probably the single most important factor affecting concentrate grade but other factors, which are also geologically controlled, affect the susceptibility of the ore to spiral beneficiation.

Hematite grainsize (this chapter, Chapter IV and Chapter V) and grainshape (Chapters IV and V), together with ore grade, are considered to be the most important geological factors affecting concentrate quality and the effect of the variation of grade, grainsize and grainshape may be optimised by control of the mining methods and milling practise.

(b) Mining controls

The mining of the Marampa ore has, in the past, entailed the removal of one complete thirty feet level of the orebody before mining of the next level commenced but the grade control problem, coupled with the significant increase in ore hardness, has recently resulted in the simultaneous mining of as many as four different levels.

The Marampa system of mining involving the extensive use of conveyor belts is not suited to mining to a grade control plan based on geological criteria. Conveyor belts and ancillary equipment are bulky and can not be readily

relocated at short notice. This lack of flexibility in the mining operations has probably contributed to the grade control problem at Marampa and will be overcome when the phasing-out of the conveyor system in favour of rear-dump trucks is completed. The Marampa mining methods are described in more detail in Chapter I.

(c) Milling and Humphreys spiral controls

The Marampa grade control problem is inextricably involved with the problems, limitations and characteristics of spiral beneficiation. The effects of the geological controls of concentrate quality are simply the reflection of the response of such controls or characteristics to spiral beneficiation and geological factors which beneficially affect the spiral concentration of hematite may be detrimental if a different method of beneficiation were employed.

A full description of the Humphreys spiral and spiral beneficiation is given in Chapter IV but some aspects of spiral beneficiation and the Marampa beneficiation practice which arise from this study of concentrate characteristics and quality are now considered.

The number of concentrate withdrawal ports and the orientation of the concentrate splitters in these ports is of importance in regulating the concentrate grade and iron recovery (Chapter IV).

A five turn Humphreys spiral has fifteen concentrate withdrawal ports numbered one to fifteen from the top of the spiral. The top or No. 1 port is almost always blanked off since the heavy minerals are rarely sufficiently segregated at this point. Ports 2, 3 and 4 can invariably be opened wide or nearly so but below port No. 4 it is recommended (by Humphreys Engineering Co.) that ports be regularly blanked off to allow the build-up of a concentrate band of reasonable width. Technical literature received from the Humphreys Engineering Co. of the U.S.A., the manufacturers of the Humphreys spiral, gave a typical arrangement of concentrate

port openings on a five-turn spiral as follows;

B, 5, 7, 7, B, 5, B, B, 4, B, B, 3, B, B, 3

where,

B = blanked-off port

1 = minimum port opening

7 = maximum port opening

Wells (1968a) gave accounts of the splitter positions in his pilot plant beneficiation tests which were always carried out with eight open ports on the five-turn spiral. In addition, he states "in the pilot plant spirals, more blanks were used than in the mill spirals". Full scale spiral concentration at Marampa has therefore at least two more open ports than is recommended and, judging from the splitter positions in the pilot plant tests, the actual openings of the ports are quite different from those recommended. It is not known whether or not the splitter positions and number of open ports in the Marampa mill have been decided on the basis of earlier experience but little attention appears to be given at present to the control or maintenance of these, possibly critical, factors.

"Use of too many splitters frequently results in a drop of concentrate grade" (Snedden, pers. comm.). From the description given by Wells it is apparent that the Marampa milling practice does not conform with the recommended practice of the manufacturers and it is suggested that this malpractice may be significantly detrimental to the production of high grade concentrates at Marampa.

Snedden also points out that specular or micaceous hematite, as occurs at Marampa, may tend to act in a similar fashion to other micaceous minerals on the spirals and report as gangue although Canadian experience had shown that spiral treatment of specularite-magnetite mixtures usually results in higher recoveries of the specularite despite the tabular to micaceous habit of this mineral.

Analysis of the grade and grainsize distribution of the concentrate screen fractions indicates that overall

concentrate grade may be materially increased if the free quartz in the -85 +120 mesh fraction can be decreased. The twenty grainsize analyses made of concentrate shipments show that the -85 +120 mesh fraction had a mean grade of 62.8 per cent Fe and formed 20 per cent of the total weight of the concentrates. Calculations indicate that if the mean grade of the -85 +120 mesh fraction can be improved by removal of free quartz from this fraction then the overall concentrate grade will also be improved by the following amounts.

<u>-85 +120 mesh fraction</u>		<u>Overall concentrate</u>
62.8 per cent Fe	(From Table 3.4)	64.7 per cent Fe
64.0 " " "		64.9 " " "
65.0 " " "		65.1 " " "
66.0 " " "		65.3 " " "

Greater concentrate grades may be produced if the free quartz contents of the other midsize fractions is simultaneously reduced and mineral-dressing research into the more effective removal of the free quartz in the midsize fractions would be of material benefit in helping to resolve the Marampa grade control problem.

F. DISCUSSION

Hematite and quartz together constitute greater than 98 per cent of the Marampa concentrate shipments with manganese oxides, alumina (as muscovite), titanium (as ilmenite) and phosphorous (probably as apatite) being minor constituents. Grade control at Marampa is therefore only concerned with control of hematite or quartz, the control of one conferring automatic control of the other.

Examination of the assay values of the various screen fractions indicates that the coarse (+60 mesh) and fine (-170 mesh) fractions are both of relatively high grade and together average about 66.2 per cent Fe whereas the midsize fractions (-60 +170 mesh) tends to be of lower grade than the overall concentrate grade, averaging about 63.7 per cent Fe

(mean concentrate grade of the twenty shipment samples is 64.7 per cent Fe). A significant increase in concentrate grade could therefore be effected if greater control could be exercised on the amount of quartz passing into the midsize fraction of the concentrate.

Canadian experience of Humphreys spirals beneficiation of low grade iron ore suggests that large amounts of quartz in the mill feed need not necessarily be a critical factor in the production of a high grade concentrate on Humphreys spirals. Iron ores with grades of about 30 to 35 per cent Fe and greater than 50 per cent SiO_2 are readily beneficiated to a concentrate assaying 66.0 per cent Fe by the Quebec-Cartier Mining Company in Quebec (Thompson, 1963) although the recovery of iron is probably less than the 85 per cent recovery experienced at Marampa.

Several lines of approach are open to the reduction and control of the amount of quartz in the mill feed.

(a) Correlation and regression analysis of beneficiation data (Chapter IV) indicates that the grade of the mill feed ore is associated with the grade of the final concentrate; high mill feed grades (= low SiO_2 grades) favour the production of high grade concentrates. Selective mining on the basis of silica and alumina content rather than the iron content would undoubtedly favour an increase in concentrate grade but the loss of large amounts of ore being mined as waste would increase mining costs and decrease production. The use of alumina as well as silica content would enable a better estimate of quartz content to be made. Silicates other than quartz, notably muscovite, are reported in the silica assay and the muscovite content of the ore is probably better expressed by the alumina assay, the point being that muscovite is very effectively removed as tailings on the Humphreys spirals and an ore with high alumina and silica values may contain less quartz than an ore with a lower silica grade and a negligible alumina grade and thus eventually yield a higher grade concentrate.

This approach would be wasteful in terms of ore reserves and would result in an increase in mining costs and other, more cost-effective, methods of grade control should be sought.

(b) As the depth of mining below the original topographic surface of the deposit increases so the ore becomes harder and the amount of blasting necessary has increased over the last few years. Where the ore is hard and where the annealed texture of the hematite (Chapter V) is common, as in synclines B and C, then efficient liberation of the hematite is hampered. A point-count analysis of concentrate grainsize fractions shows that, with decreasing grainsize, the ratio of composite hematite-gangue grains to free, i.e. completely liberated, gangue decreases rapidly. Free quartz and mica in the coarse fraction is very readily removed as tailings on the Humphreys spirals but the composite hematite-gangue grains tend to behave as heavy minerals and report as concentrate. The ratio of composite hematite-gangue grains to free gangue grains in the -16 +44 B.S. mesh fraction is about 8.4:1 and this ratio decreases to 2.9:1 and 1.4:1 in the -44 +60 and -60 +85 mesh fractions respectively. The high values in the coarsest fractions is an expression of the resistance of hard ore and annealed ore to crushing and liberation of the hematite in the hammer mills. Finer grinding to, say, 95 per cent -60 B.S. mesh would ensure that most of the hematite would be completely liberated although there is a danger that finer grinding would increase the amount of hematite lost as tailings because of fine grainsize. Humphreys spirals pilot plant tests (Wells, 1968a) have shown that concentrate grades of 69.0 per cent Fe (greater than 98.5 per cent hematite) are readily achieved in the +44 mesh fraction. The mean grade of the +44 and -44 +60 mesh fractions of concentrate shipments are 66.1 and 66.8 per cent Fe respectively (Table 3.4). If finer grinding results in increased grade because of greater hematite liberation then an overall concentrate grade of 65.0 per cent Fe (from the shipment mean of 64.7 per cent Fe) could be achieved by an increase in the mean grade of the -60 mesh fraction by complete liberation of the hematite in

the +60 mesh fraction.

(c) Inspection of Table 3.4 and Fig. 3.3 indicates that the lowest grades of the concentrate screen fractions occur in the three midsize fractions -60 +85, -85 +120 and -120 +170 B.S. mesh, notably in the -85 +120 mesh fraction which has a mean grade of 62.8 per cent Fe and makes up 20.0 per cent (by weight) of the total concentrate. This latter fraction is invariably of lower grade than the concentrate of which it forms part and combined with the fact that it makes up one fifth of the entire concentrate suggests that a significant improvement in overall concentrate grade could be effected by reducing the amount of gangue in this fraction. Microscopic examination of the -85 +120 mesh fraction of the Marampa concentrate reveals that most of the gangue in this size range occurs as free quartz. The reason for the excess quartz in the midsize fractions is not fully understood but appears to be an inherent feature of spiral beneficiation (H. Snedden, pers. comm.), and quartz grains in these fractions are apparently less susceptible to removal as tailings, a phenomenon which may be related to ratio of surface area to specific gravity or some other physical attribute and it appears likely that alteration or more precise control of the present milling operations will be necessary to combat this inclusion of undesirable gangue. Snedden states that rejection of free gangue from these size ranges is usually improved by increasing feed pulp and wash water volumes, by using fewer concentrate splitters in the lower sections of the spirals and by selection of an optimum spiral loading for the grade and grainsize of the ore being processed.

G. SUMMARY

Statistical analysis of the grade distributions of all the 127 iron ore concentrate shipments dispatched over the eighteen month period January 1967 - June 1968, together with a statistical analysis of the grainsize analyses of 20 of these shipments, yielded the following information;

(a) The mean iron content of the shipments has decreased over the eighteen month period from 65.0 per cent Fe to 64.7 per cent Fe. In practical terms, over the first six months of 1968, this decrease in grade (other factors being equal) has meant a loss of about 4900 tons of hematite representing a direct income loss of many thousands of pounds.

(b) The mean silica contents of shipments has increased from 5.4 to 5.9 per cent SiO_2 .

(c) The mean manganese content of the shipments has decreased from 0.23 to 0.12 per cent Mn.

(d) The mean alumina, titanium and phosphorous contents of shipments have not altered significantly.

(e) A Chi-square test for goodness-of-fit indicates that the concentrate iron grades are normally distributed.

(f) The bulk (about 60 per cent) of the hematite concentrates occurs in the midsize fractions (-60 +170 B.S. mesh) and this grainsize range is over-endowed with free quartz gangue causing concentrate grades in this midsize range to be sub-standard.

(g) The coarse (+60 B.S. mesh) and fine (-170 B.S. mesh) grainsize fractions of the hematite concentrate are characterised by high iron contents.

(h) Reduction of the amount of free quartz in the midsize fractions would materially improve concentrate grades and suggests that the Marampa grade control problem may best be approached as a mineral dressing problem.

CHAPTER IV

STATISTICAL ANALYSIS OF MARAMPA BENEFICIATION DATA

A. HUMPHREYS SPIRAL BENEFICIATION

(a) Introduction

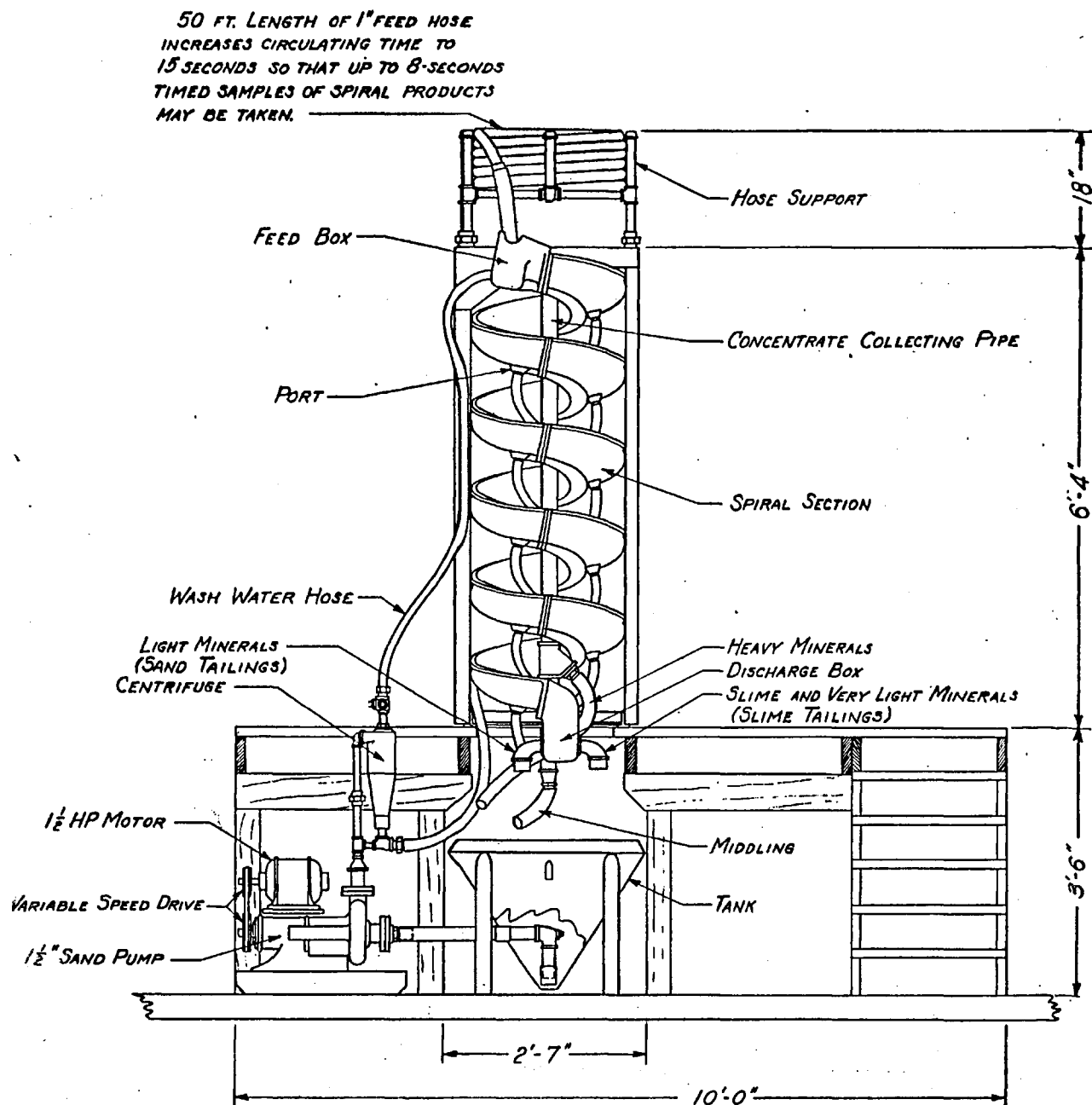
The concentration of hematite from the medium grade Marampa ore is effected by means of Humphreys spirals and a knowledge of the principles of spiral beneficiation and of the variables affecting spiral concentration is a prerequisite to an understanding of the problems involved in grade control at Marampa.

The Humphreys spiral concentrator is a device for separating minerals of different specific gravity or different shape and was first used commercially in 1943 to recover chromite from Oregon beach sands. It has been estimated (Thompson, 1969) that more than 25,000 Humphreys spirals were in operation by 1967 of which some 67 per cent were used for the beneficiation of iron ore.

The concentrator consists essentially of a feed box, a spiral trough of five complete turns arranged about a vertical axis, and a discharge box (Fig. 4.1). Pulp is introduced to the spirals from the feed box and, as it flows down the spirals, the heavier particles become concentrated in a ribbon along the inner margins of the spiral trough. Concentrate withdrawal ports are situated at regular intervals on the inner margins and the width of concentrate ribbon that is removed from the spirals can be altered by adjustment of stainless steel splitters set into the concentrate withdrawal ports. Less dense material tends to move towards the outer margins of the spiral and is discharged at the lower end of the spiral column. Wash water is introduced at the inner spiral margin in such a way as to flow across the spiral and assist in the separating process.

(b) Beneficiation on Humphreys spirals

The separating action in the Humphreys spirals combines



—Humphreys Spiral Concentrator Closed Circuit Test Unit

(from Manual of Operating Instructions,
Humphreys Bull. No. 10A, 1952)

Fig. 4.1

centrifugal action and sluicing and an element of dense-media separation in a partially controlled sink-float medium provided by the pulp. On the spirals the descending ribbon of pulp spreads across the width of the spiral trough, the dense material tending to settle towards the flat inner margins whereas the less dense material is swept around the inclined outer wall. The heavy mineral concentrate collected at the inner margins is removed from the spiral by adjustable, stainless steel splitters set in concentrate withdrawal ports and the less dense minerals continue down the spirals and are removed as tailings.

The separating action on the spirals is simple in outline but complex in detail. As the pulp flows down the spirals each particle is subjected to a centrifugal force which is tangential to the spiral trough, directly proportional to the square of the velocity of flow and inversely proportional to the radius at which the particle is situated (Gleeson, 1945). The centrifugal force piles the water against the outer wall of the spiral until the flowing stream reaches equilibrium between the two main forces acting; centrifugal outwards and gravitational downwards.

The velocity of the spiral stream decreases with depth from a maximum just below the surface to a minimum approaching zero at the water/spiral interface where frictional forces are at a maximum. The bottom layer of water is thus retarded by friction and has less centrifugal force acting upon it and tends to flow sideways towards the inner margin of the spiral, carrying with it the heavier particles. As the bottom layer flows towards the inner margin it displaces the upper layers which thus flow away from the inner margin and this action is assisted by introduction of wash water at regular intervals. The end result is a complex double spiral action combining spiral flow about a vertical axis and spiral flow about a spiral axis, and this double spiral action produces a lateral arrangement of particles of increasing densities across the spiral trough (Fig. 4.2).

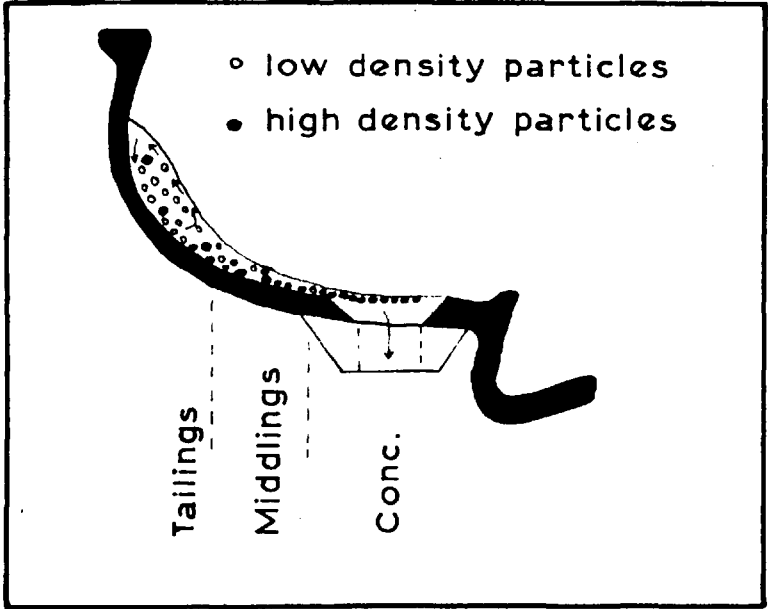


Fig. 4.2

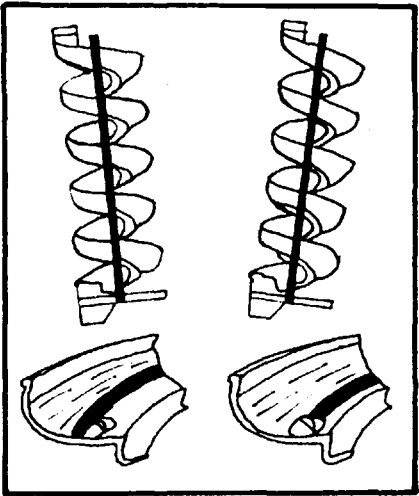


Fig. 4.3

(c) Requirements for optimum concentration

Optimum concentration requires the interaction of ore characteristics, spiral characteristics and operating conditions which include the following factors:

(1) Liberation of valuable minerals. The heavy minerals to be concentrated should be liberated from the usually less dense gangue minerals and this is usually accomplished by crushing and grinding to the necessary grainsize. If liberation is not complete the proportion of middlings will be too high and the grade of the concentrate and the recovery of the valuable minerals will suffer.

It will later be shown that liberation difficulties due to textural features of some types of Marampa ore contribute to the grade control problem at Marampa.

(2) Grainsize of valuable minerals. The heavy minerals to be concentrated and recovered should, ideally, fall within the size range -16 +150 B.S. mesh although beneficiation is practicable to -10 +200 B.S. mesh and under certain circumstances both the upper and lower size limits may be exceeded. The size of the gangue minerals is not usually critical.

The Iron Ore Company of Canada have found that recovery of iron decreases in the -200 mesh range (T.C. Murphy, pers. comm.) and if reduction to this size is necessary to effect liberation, other methods of beneficiation should be investigated.

There is usually no advantage in pre-sizing or classifying the ore before spiral treatment since better recovery is obtained on the fine fractions and poorer recovery on the coarse fractions, the final result on pre-sized feed thus being essentially the same as with unsized feed (Henry Snedden, pers. comm.). Apparently, with only coarse heavy mineral present the ore pulp loses fluidity, banking (sand-bar formation) becomes excessive, spiral capacity and heavy mineral recovery are markedly reduced and sub-grade concentrates are frequent. When only fine heavy mineral is present, pulp and wash water volumes can be reduced with a subsequent increase in recovery for the finest heavy mineral fractions.

(3) Specific gravity. The specific gravity of the heavy minerals should be significantly greater than that of the lighter gangue minerals by at least one specific gravity unit. In practice, the most efficient separations occur between light gangue minerals, such as quartz and feldspar of specific gravity about 2.65, and heavy minerals of specific gravity greater than 4.0, and it has been found that separation of gangue minerals with specific gravities greater than 2.7 from valuable minerals of still higher specific gravities is difficult. Separations between sphalerite (S.G. about 4.0), pyrite (S.G. about 5.0) and galena (S.G. about 7.5) are unsatisfactory and one of the reasons that sulphide minerals are rarely concentrated on Humphreys spirals is that a bulk concentrate is usually formed when more than one heavy mineral occurs in the same feed, and such a concentrate must then be re-treated, usually chemically, to separate the component heavy minerals.

The specific gravity differential of about 2.5 units between quartz (S.G. about 2.7) and hematite (S.G. about 5.2) makes hematitic ore usually most amenable to spiral beneficiation.

In certain cases, particularly the cleaning of coal, the gangue minerals have the higher specific gravity and thus report as tailings through what is normally the concentrate withdrawal ports, whereas the valuable minerals are collected at what is normally the tailings discharge.

(4) Grainshape. The effect of grainshape variations on the efficiency of heavy mineral spiral concentration is not yet well understood and may well be considerable. Significant shape differences may be utilised in separating minerals of similar specific gravities and Humphreys spirals have been used to separate micas from quartz and feldspar gangues. The thin flakey to tabular micas are swept around the outside of the spirals whereas the blocky and more equidimensional quartz and feldspar grains act as heavy minerals and, like the coal refuse, are removed to tailings through the concentrate withdrawal ports. It must be pointed out that for effective separation of micas from a quartz/feldspar gangue the micas

must be delaminated since there is no significant difference in spiral behaviour of books of mica and equidimensional grains of quartz or feldspar.

Much of the Marampa ore consists of micaceous to tabular hematite and it is apparent that such material may behave in two detrimental ways when undergoing spiral beneficiation:

(a) Micaceous hematite grains may act in the same manner as other, less dense, micaceous minerals because of the large surface area able to be presented to the descending pulp stream. In this case such hematite would tend to be removed as tailings. This may be the predominant tendency when the ore:water ratio is very low, i.e. when the spirals are under-loaded.

(b) Micaceous heavy mineral grains have a much greater tendency than equidimensional grains to settle on the spirals and bank-up or "sand-bar" to form obstructions on the spirals thereby disrupting the smooth streamline flow around the spirals. Micaceous hematite grains in contact with the spiral surface have a minimum surface area exposed to the descending pulp flow and a maximum surface area in contact with the spiral. If the spirals are overloaded, the frictional forces operating between the flat plates of hematite and the spiral surface may be sufficient to arrest the movement of the bulk of the material down the spiral. In such circumstances, the obstruction or sand-bar causes turbulent flow around the spirals and quartz and hematite are readily washed into the concentrate withdrawal ports producing lower grade concentrates. The turbulent flow may also carry hematite to the outside of the spiral from where it reports to tailings causing a loss in iron recovery. Gangue minerals are readily trapped among the hematite grains in the sand-bars and may eventually report as concentrate.

The grainshape of hematite in the Marampa ore synclines A and D is essentially micaceous whereas the hematite in synclines B and C is essentially equidimensional. The difference in hematite grainshape is very marked and is considered

to be at least a partial explanation of the fact that two feeds with identical iron content may yield concentrates with significantly different iron contents.

(5) Feed rate. The optimum feed rate to the Humphreys spirals depends upon the type of ore undergoing concentration and, for any particular ore, must be determined by test work in which feed rate is compared with heavy mineral recovery.

The feed rate will be higher for a coarsely ground ore than for a finely ground ore since a higher pulp density is needed to crowd coarse heavy minerals into the concentrate. The optimum feed rate varies from 0.5 to 2.5 short tons per spiral per hour, the average being about 1.5 tons per spiral per hour.

Spiral pulp and wash water volume rates are two of the most important operating variables and the optimum volumes should be determined by pilot plant tests. If the feed pulp or wash water volumes are too low, the pulp flow becomes sluggish and causes banking of heavy minerals all down the spirals with a resultant reduction in both concentrate grade and spiral capacity. If feed pulp or wash water volumes are too high the heavy minerals are swept wide of the upper concentrate ports causing banking in the lower spiral turns with a consequent decrease in heavy mineral recovery and a reduction in spiral capacity.

Spiral concentration is most efficient when the spiral is well loaded and the feed rate constant, and variations in feed rate can have adverse effects on both concentrate grade and heavy mineral recovery.

The rate at which wash water is introduced to the spirals should be kept fairly constant since the volume of wash water used markedly affects the position of the heavy mineral ribbon where it enters the concentrate port. The usual wash water input is about 6 to 8 gallons per minute but the actual amount for any particular operation must be determined by pilot plant tests.

(6) Axial orientation of spirals. The axis of the Humphreys spirals should be exactly vertical (Snedden, 1956). A tilted spiral has a reduced load capacity and will not produce a concentrate at the same grade and rate as a vertical spiral. With a tilted spiral there is an alternate increase and decrease in the effective pitch as the pulp descends the spiral trough. At the points of reduced pitch the heavy minerals may settle and bank and the concentrate will become diluted with gangue as it enters the concentrate withdrawal ports. At points of increased pitch heavy minerals will pass wide of the concentrate ports and result in an unnecessarily high grade concentrate with low recovery (Figure 4.3).

(7) Concentrate withdrawal ports and splitters. Efficient concentration depends upon the proper selection of the number of concentrate ports and the optimum orientation of the splitters.

A five-turn Humphreys spiral has fifteen withdrawal ports, three to each turn. The selection of the location and number of withdrawal ports has two objectives;

(a) To get maximum spiral capacity by removing as much concentrate as possible from the upper spiral turns. This is done by using the maximum splitter openings in the upper spiral turns. This not only increases spiral capacity but also improves heavy mineral recovery by including the coarse heavy minerals that tend to travel near the outer edge of the ribbon of concentrates.

(b) Obtain the maximum ratio of concentration per stage by blanking off some middle and lower ports to permit the build-up of a wider concentrate band between withdrawal ports.

Heavy minerals settle out rapidly on the spiral and should be removed as soon as practicable. A few ports with wide openings are usually more effective than many ports with narrow openings and the use of too many withdrawal ports frequently results in a reduction of both concentrate grade and spiral capacity.

In most operations the top or No. 1 port is blanked off

and ports 2, 3 and 4 can be opened completely or nearly so. Below port No. 4 it is advisable to blank off ports to allow a build-up of the ribbon of heavy minerals. A typical arrangement in a five-turn spiral would be the use of ports 2, 3, 4, 6, 9, 12 and 15 with splitter settings of 5, 7, 7, 5, 4, 3 and 3 respectively, where 7 is the maximum and 1 the minimum port openings, (Humphreys Eng. Co., pers. comm.). The number of ports and optimum openings should be determined by pilot plant tests although the standardization of these factors may be impracticable because of other independent variables including variations in ore type. The Iron Ore Company of Canada carried out numerous test programmes to determine the "best" settings of splitters and ports but failed to equate particular settings with particular ore types (Murphy, pers. comm.).

It is pertinent to point out that the "Ore Classification Tests" carried out at Marampa in the investigation of grade control (Wells, 1968a) were not undertaken with fixed port openings. The number of open ports on the first five-turn stage was eight and it was noted that this is less than that used in full scale milling. In addition the opening of the ports differed dramatically from that suggested by Humphreys Engineering, the ranges of port openings being as follows;

Port No	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Setting	B	2/7	1/5 $\frac{1}{2}$	B	4/6	2/5	B	3 $\frac{1}{2}$ /5 $\frac{1}{2}$	B/4 $\frac{1}{2}$	B/5	B	1/4	B	B	3/4 $\frac{1}{2}$

B = blank port 7 = maximum port opening 1 = minimum port opening.

It is not known whether the port openings used at Marampa were decided on the result of earlier spiral tests but it is considered that further pilot plant tests to determine the optimum port settings would be rewarded by improved concentrate grade and/or iron recovery.

Statistical investigations into the effects of ore characteristics, such as grainsize and feed grade, on

concentrate grade are made imprecise when milling variables which also affect concentrate grade are varied and it is suggested that milling variables be kept constant in any future pilot plant spiral tests.

(8) Feed grade. The feed grade itself is not a critical factor in spiral concentration but optimum results are achieved when feed grades are held constant or nearly so. Wide and frequent variations in feed grade necessitates frequent alterations to splitter settings and water supply to the spirals (Murphy, pers. comm.) although these alterations are apparently not carried out at Marampa, and this lack of mill flexibility may be contributing to the grade control problem at Marampa

It is relevant at this stage to point out that variations in spiral settings can only be carried out if there are no delays in the receipt of feed, concentrate and tailings sample assay results which otherwise become of historical interest only.

(d) Advantages of Humphreys spirals

One of the most important, though least appreciated, aspects of spiral beneficiation is that full scale mill tests may be carried out on a single spiral system, thus minimising the uncertainty which usually accompanies scaling-up operations from pilot plants or scale models.

Most of the advantages of Humphreys spirals are associated with the absence of moving parts which implies low power requirements and a minimal amount of maintenance because of only slight wear. Installation and operating costs are low, again because of low power requirements and the spirals occupy only about one-half the space of other beneficiation equipment of similar capacity. Labour costs should be low. One man may attend to about one hundred spirals (Hubbard et al, 1953) which at Marampa would produce about 800 tons of concentrate per eight hour shift.

(e) Fields of application of Humphreys spirals

Humphreys spirals are most cost-effective when used for large scale beneficiation of the order of thousands of tons of concentrates per day and they have found their greatest use in the beneficiation of iron ores. Many heavy minerals are amenable

to spiral beneficiation, particularly iron oxides and beach sands, although sulphide minerals are not suited to spiral concentration because of their extreme specific gravities and because flotation is generally cheaper and more efficient and can selectively separate the sulphides of copper, lead and zinc.

Less dense non-metallic minerals may also be separated from gangue of similar specific gravity on Humphreys spirals although in these cases (rock phosphate, coal and micas) the mechanics of separation depend upon shape differentials or the selective treatment of valuable minerals by suitable chemical reagents, rather than on distinct specific gravity differentials.

The following ores have been successfully concentrated on Humphreys spirals (Humphreys Eng. Co., 1960);

- iron ores (hematite, martite, and magnetite)
- beach sands (zircon, rutile, ilmenite and monazite)
- tungsten ores (wolframite and scheelite)
- micas (muscovite, biotite, lepidolite and vermiculite)
- rock phosphate, native copper, chromite, pyrite, barite, talc, coal, tantalite and columbite.

(B) STATISTICAL ANALYSIS OF MARAMPA BENEFICIATION DATA

(a) Introduction

Before a grade control scheme can be designed to combat the Marampa problem it is first necessary to have numerical information describing those measureable variables which may effect the grade of the hematite concentrate. Alterations to established mining or milling procedures are inevitably costly and cannot be prescribed on theories not founded on numerical descriptions. In addition any significant numerical information regarding concentrate grade is of greater use if it can be meaningfully related to measureable features of the ore in situ for only then can grade control systems be exercised on

the ore itself, i.e. variables which are found to be significantly associated with concentrate grade are of most use when they can be explained in terms of geological processes or milling procedures. If variables which appear to be significantly associated with concentrate grade cannot be explained in the above terms then the significance may only be fortuitous, in which case a grade control procedure based on these variables may fail to produce optimum concentrate grades or iron recovery.

This chapter describes the statistical analysis of pilot plant and full scale mill beneficiation data resulting from work carried out by the Marampa mineral-dressing staff.

(b) Description of Beneficiation data

The correlation and regression analyses are based on the results of two sets of Marampa beneficiation tests carried out by the Mineral Dressing staff of S.L.D.C. The tests were:

- (a) Pilot plant spiral tests. Thirteen tests were carried out on selected ore samples (Wells, 1968a).
- (b) Full scale mill tests (Wells, 1968b).
 - (i) 31 tests of silica and iron grainsize distributions.
 - (ii) 26 tests of total grainsize distributions.

(i) Pilot Plant Spiral Tests

A series of pilot plant spiral tests were carried out by the Mineral Dressing staff in order to:

- (a) Determine the "millability" characteristics of the Marampa ore.
- (b) Design a procedure for determination of the concentrate "potential" of the ore.
- (c) Compare the effectiveness of the two spiral systems, 5 + 3 turn and 5 + 5 + 5 turn.
- (d) Determine the effects of increasing the water:ore ratio. (Wells, 1968a).

Thirteen spiral tests were carried out, the ore samples being taken from synclines A (four samples), B (two samples), D (five samples) and with a further two samples from syncline "C/D included". The samples consisted of about 100 kilogrammes

of ore taken from channels 50 feet long across the strike of the orebody, and were initially screened on $1\frac{1}{2}$, $\frac{1}{2}$ and $\frac{1}{8}$ inch box screens with the $\frac{1}{8}$ inch material being subsequently hammered through the $\frac{1}{8}$ inch screen, after weighing, to simulate mill grainsize reduction. The samples were finally split to about 20 kilogrammes for the spiral tests and sub-samples screened on 44 and 120 B.S. mesh screens for grain size analysis. Further sub-samples of each of the +44, -44 +120 and -120 mesh fractions were assayed for iron and silica.

The pilot plant consisted of two 5-turn spirals and a physically separate 5-turn spiral making, in all, a 5 + 5 + 5 - turn system. The product of the first two 5-turn spirals formed the feed for the third 5-turn spiral and thus the pilot plant beneficiation was not a continuous operation as under full scale mill conditions. The pilot plant operating conditions also differed from those of the full scale mill in that, in practice, lump ore is crushed in an Aerofall mill and the $\frac{1}{8}$ inch material is further reduced in hammer mills. Also, the pilot plant tests were carried out with an ore:water ratio of 1:7 whereas the mill operates at a 1:6 ratio.

The effects of these differences between the pilot plant spiral system and full scale mill conditions are impossible to assess quantitatively with the data available but some generalised and qualitative observations are possible.

The crushing of lump ore in the Aerofall mill and the hammer mills may have a detrimental effect on hematite grainsize distributions by increasing the proportion of fines (-120 B.S. mesh) from that distribution occurring naturally in the ore. This may, however, be countered by the tendency of the Aerofall mill feed to be of slightly higher grade than that of the ore passing directly to the spirals. The Humphreys spiral method of beneficiation is largely dependant upon the grainsize of the hematite and, also, contracts for the sale of iron ore concentrates usually contain penalty clauses which limit the proportion of fines and therefore an increase in the proportion of hematite in the -120 mesh fraction may have a detrimental

effect on iron recovery and mine profit. The pilot plant tests results may therefore indicate a higher concentrate grade and/or iron recovery and smaller proportion of fines than would occur in full scale concentration.

Optimum conditions for spiral beneficiation include a smooth uninterrupted flow of the pulp down the spirals. This condition is not met in the pilot plant tests where the product of the first two 5-turn spirals is collected and then used as the feed for the third 5-turn spiral. The effect, if any, that this interruption has on the final concentrate grade and iron recovery is not known but is probably not beneficial.

(ii) Full Scale Mill Tests

Full scale mill tests were carried out in order to determine whether the results of the pilot plant spiral tests could be extrapolated to the prediction of concentrate grade from a knowledge of full scale mill feed data.

The performance of the mill over thirty-seven periods of eight hours (a single mill shift) was measured by sampling of feed from all eleven mill units. The feed was sampled hourly (a total of 88 samples per shift) and bulked to form one sample at the end of each shift. Sub-samples were screened on 8, 16, 44, 85, 120 and 200 B.S. mesh screens, each of the fractions being microscopically assessed for iron and silica content. The eight hourly concentrate grades were averaged to give a single value for each shift.

Six iron and silica grainsize distribution factors were examined:

- F₁ - per cent weight of total silica in the -44 +120 B.S. mesh fraction.
- F₂ - per cent weight of total silica in the +44 B.S. mesh fraction
- F₃ - " " " " " iron " " +44 B.S. " "
- F₄ - " " " " " silica " " -120 B.S. " "
- F₅ - " " " " " iron " " -44+120 B.S. mesh fraction
- F₆ - " " " " " " " " -120 B.S. mesh fraction

Six total grainsize statistics were examined:

+16, -16 +44, -44 +85, -85 +120, -120 +200 and -200 (all B.S. mesh). These were designated X_5 to X_{10} respectively.

The correlation and regression analysis carried out on the full scale mill test data is divided into two sections:-

- (a) Analysis of feed grade (Fe_f and SiO_{2f}), concentrate grade (Fe_c) and the iron and silica grainsize distributions (F_1 to F_6):- 31 tests.
- (b) Analysis of feed grade, concentrate grade and the total grainsize distributions (X_5 to X_{10}):- 26 tests.

C. CORRELATION AND REGRESSION ANALYSIS OF MARAMPA BENEFICIATION DATA

(a) Introduction

Correlation and regression analysis was carried out on the results of the beneficiation test data in order to ascertain the degree of association existing between concentrate grade and the various ore characteristics, particularly grainsize, feed grade and the iron and silica grainsize distributions. Significant correlations (at the 95 per cent confidence level) were further examined in order to separate the fortuitous or coincidental significant correlations from correlations which were both statistically and geologically significant. Geologically significant correlations were those which could be related to geological phenomenon, taking into account a knowledge of the theoretical behaviour of ore particles on the Humphreys spirals.

Regression analysis was undertaken on that data which had yielded significant correlation coefficients and the resultant regression equations were used as concentrate grade prediction models and predicted grade compared with observed grade in order to find an acceptable prediction model.

Mill variables including ore:water ratio, number of concentrate withdrawal ports and the orientation of splitters

in the ports were not fully taken into account, being both difficult or impossible to measure and difficult to interpret.

(b) Pilot Plant Spiral Tests

The results of the pilot plant spiral tests are given in Table 4.1

Sample	Feed Grade		F_1	F_2	F_3	$F_2 + F_3 - F_1$	Concentrate grade % Fe
	% Fe	%SiO ₂					
SA1	46.7	33.1	64.4	28.8	34.2	- 1.4	64.1
2	58.7	13.7	63.0	26.0	33.1	- 3.9	66.3
3	52.4	22.5	64.5	26.1	25.6	-12.8	64.2
4	51.2	23.8	74.1	15.7	24.5	-33.9	64.8
C/D1	41.1	38.4	33.6	60.9	18.2	+45.5	65.8
2	25.3	38.4+	24.5	72.3	23.3	+71.1	63.3
SD1	43.7	34.0	53.0	39.8	38.1	+24.9	64.8
2	45.0	32.4	42.6	51.7	41.3	+50.4	66.2
3	48.2	29.7	55.2	36.9	45.8	+27.5	64.9
4	51.8	22.6	50.4	40.6	38.3	+28.5	67.8
5	48.1	27.8	62.5	31.2	41.7	+10.4	67.5
SB1	38.6	38.4+	49.6	42.0	28.2	+20.6	61.7
2	33.5	38.4+	42.6	52.7	27.1	+37.2	61.9
Mean	45.0	30.2+	52.3	40.4	32.3		64.9

Table 4.1 Pilot plant spiral test results.

The statistically significant total correlation coefficients from this pilot plant spiral test data are: (Table 4.2)

	Fe(f)	F ₁	F ₂	F ₃	F ₂ +F ₃ -F ₁
Fe(c)	0.618 0.382 [≠]	N.S. ^{≠≠}	N.S.	N.S.	N.S.
Fe(x)	-	0.797 0.635	-0.827 0.684	N.S.	-0.748 0.56
F ₁	-	-	-0.996 0.992	N.S.	-0.967 0.935
F ₂	-	-	-	N.S.	0.955 0.912
F ₃	-	-	-	-	N.S.

[≠] lower figure is coefficient of determination.

^{≠≠} N.S. = Not statistically significant.

Table 4.2: Coefficients of correlation and determination for pilot plant spiral test results.

Feed grade (Fe_f) is the only single factor which is significantly related to concentrate grade (Fe_c) (Fig. 4.4), the correlation coefficient for these two variables being 0.618 with a coefficient of determination (r²) of 0.382 indicating that about 38 per cent of the variation in concentrate grade may be attributed to variation in the feed grade.

The 95 per cent confidence limits of r are about 0.07 and 0.87, i.e. 0.07 < p < 0.87, where p is the absolute or population coefficient of correlation. There is thus a statistically significant and positive correlation between feed grade and concentrate grade and this correlation is also intuitively recognised as having geological significance. A high grade feed has only minimal amounts of gangue minerals which must be removed to produce an acceptable concentrate and the spiral wash water is unable to become overloaded with gangue minerals which are thus readily removed as tailings.

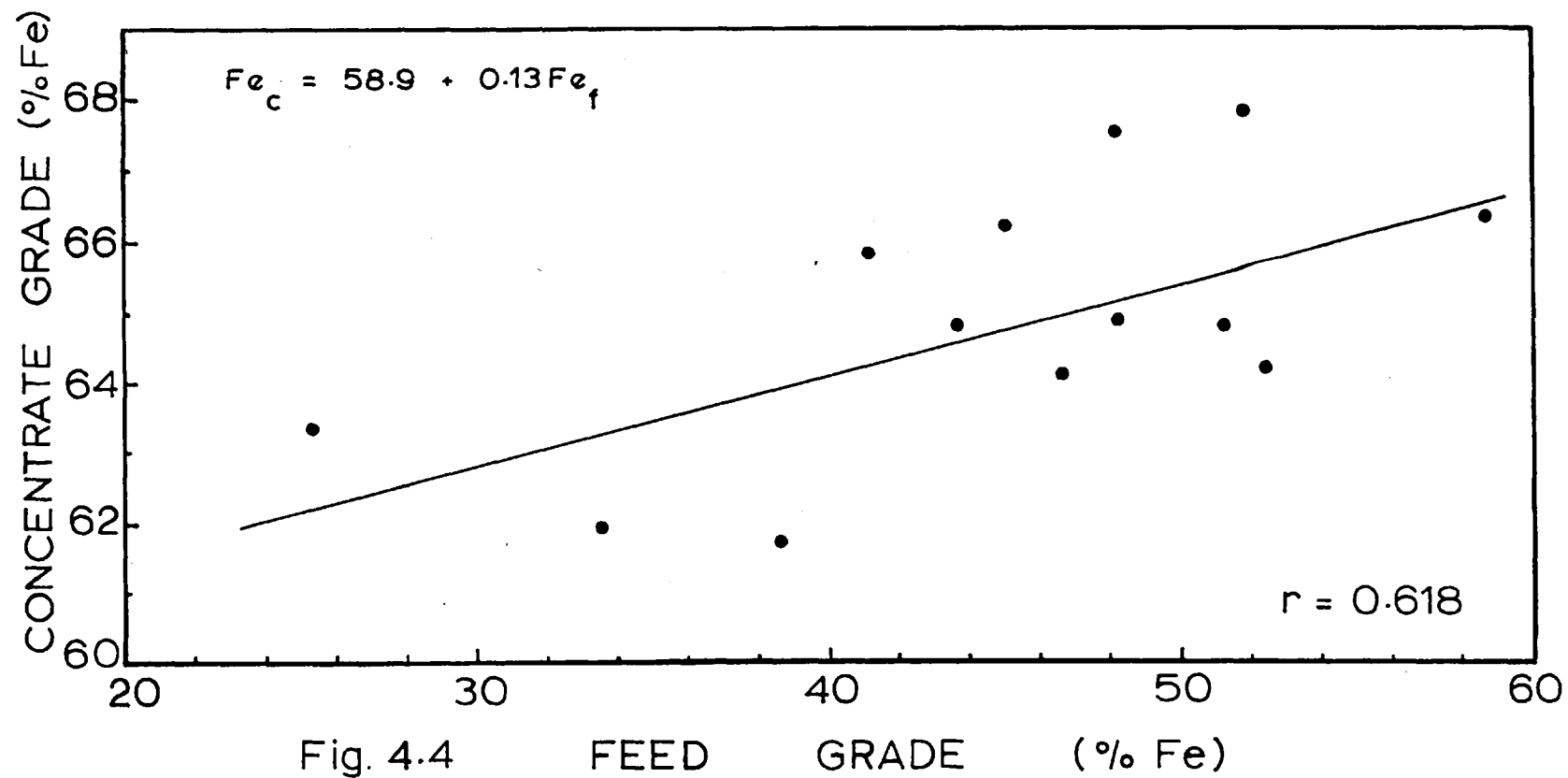


Fig. 4.4

The linear regression equation of concentrate grade on feed grade is

$$Fe_c = 58.9 + 0.13 Fe_f + e \quad \dots \quad \dots \text{Equat. 4.1}$$

where:

Fe_c = concentrate grade (per cent Fe)

Fe_f = feed grade (per cent Fe)

e = error (deviation: observed - predicted grade)

As noted in correlation analysis this statistically significant relationship also has geological significance but this regression equation is not of sufficient precision to allow its use as a prediction model as can be seen from the following figures.

Fe_f	Fe_c (predicted)	Fe_c (observed)	Deviation(e)
45.0	64.8	66.2	1.4
48.1	65.2	67.5	2.3
48.2	65.2	64.9	-0.3

Deviations of these magnitudes are too great for the equation to be used to predict concentrate grade with a knowledge of feed grade alone, the mean and maximum deviations being 1.2 and 2.3 Fe per cent respectively.

The range of concentrate grade which may be produced from similar or even identical feed grades is the major grade control problem at Marampa and must be related to physical and/or chemical characteristics of the ore and to variations in mill operating conditions.

Grainsize and grainshape are two factors which are known to affect ease of concentration on Humphreys spirals and in these preliminary tests three grainsize statistics were examined.

- (a) F_1 - per cent weight of total silica in the -44 +120 B.S. mesh fraction.

- (b) F_2 - per cent weight of total silica in the +44 B.S. mesh fraction.
- (c) F_3 - per cent weight of total iron in the +44 B.S. mesh fraction.

These grainsize statistics were chosen by the Mines staff on the results of earlier related tests on silica distribution and on a local knowledge of the behaviour of Marampa ore on the spirals. Coarse grained micaceous hematite is regarded as having a tendency to settle and bank on the spirals and tends to yield a low or even unacceptable grade concentrate, and thus the amount of ore in the +44 mesh fraction was considered to be one of the critical factors in beneficiation. Quartz content and grainsize distribution was also considered to be of importance since ".... when the quartz content was raised from 18.2 to 21.4 per cent there was a 2.2 per cent drop in concentrate grade" (Wells, 1968a). A scattergram of silica content of ore (SiO_2) against concentrate grade (Fig. 4.5) fails to substantiate this rather precise statement and, in fact, on the results of these present tests, it is equally accurate to state that an increase in quartz content from 23.8 per cent to 27.8 per cent gives an increase in concentrate grade of 2.7 Fe per cent, (Table 4.1 samples SA4 and SD5). Although it is correct that increasing silica content of the ore is related to decreasing iron content of the concentrates, the relationship is a generalised trend and not an absolute cause and effect relationship. In addition, it must be remembered that an increase in the silica content of the ore is naturally associated with a decrease in the iron content since the two variables sum to a constant or approximately so, and it is probably more accurate to state that low iron grade and high iron grade ores tend to yield low grade and high grade concentrates respectively.

The correlation coefficient (r_{y3}) describing the degree of association between the concentrate grade and the percent weight of total iron contained in the +44 B.S. mesh fraction (F_3) has a value of 0.464 which is well below the value of 0.55 which must be exceeded for this correlation to have significance at the 95 per cent confidence level. In addition,

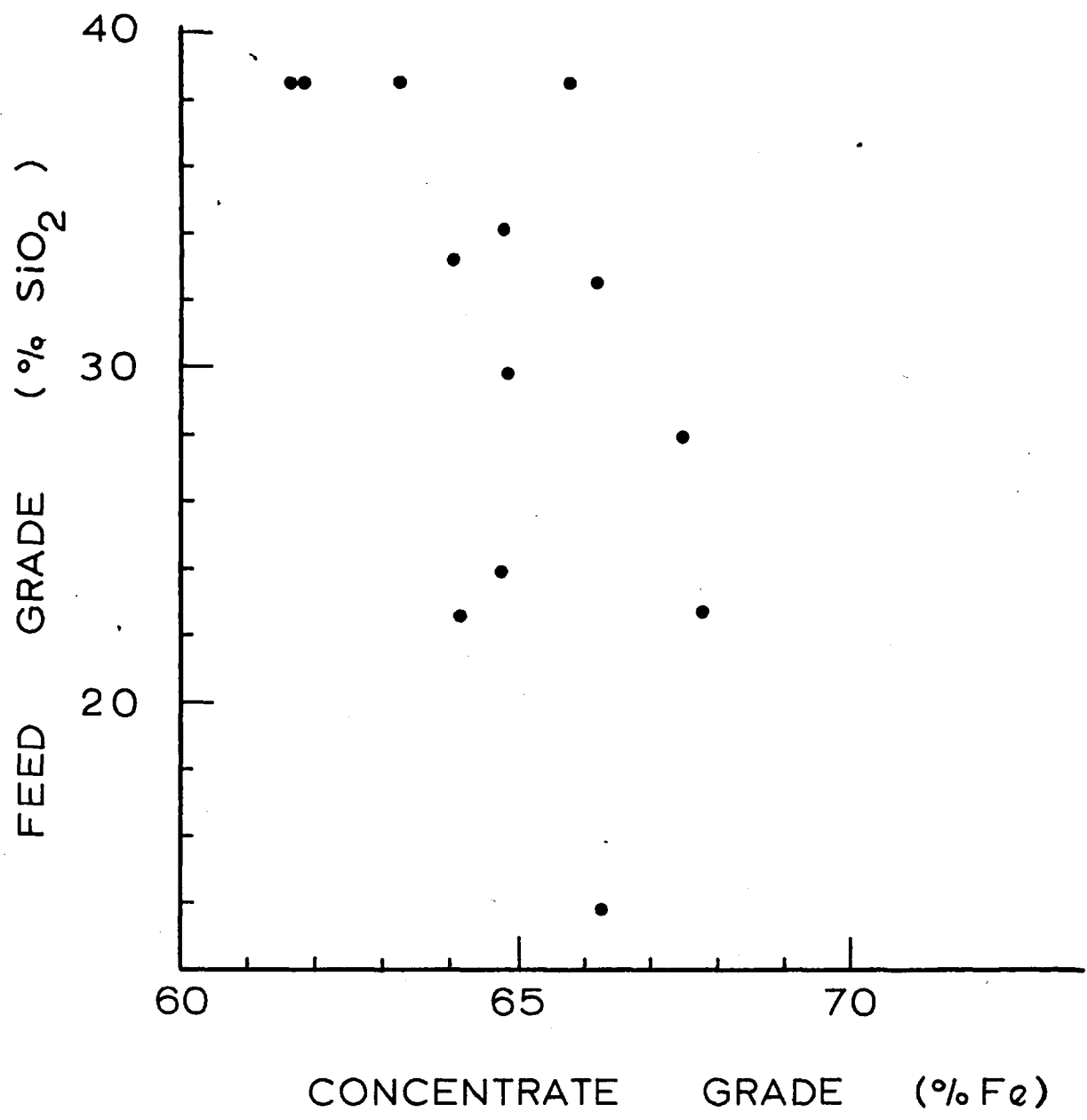


Fig. 4.5

when the effects of feed grade are accounted for in a partial correlation coefficient calculation, the effects of F_3 on concentrate grade is even less marked with $r_{Y1:3}$ having a value of 0.314. The value of $r_{Y1:3}$ which must be exceeded to have significance at the 95 per cent confidence level is, for first order partial correlations, 0.58. The multiple linear regression equation relating F_3 and Fe_f to Fe_c yields mean and maximum deviations of 1.2 and 2.2 Fe per cent respectively and it is considered, from the results of the pilot plant spiral tests that concentrate grade is independent of the per cent weight of total iron contained in the +44 mesh fraction.

It has been suggested (Wells, 1968a) that the concentrate grade and the per cent weight of total silica contained in the -44 +120 mesh fraction (F_1) are inversely related, i.e. a high Fe_c is associated with low values of F_1 . Correlation analysis of these two variables shows that the reverse is true although the relationship is not statistically significant. The coefficient of correlation has a value of 0.217 indicating a positive relationship and a partial correlation coefficient, which holds the effects of feed grade constant, has a value of 0.58 which is marginally significant at the 95 per cent confidence level but not significant at the 99 per cent confidence level. The fact that the partial correlation coefficient is marginally significant suggests that small but significant variations in concentrate grade, caused by variation in F_1 , may be masked by the overriding effects of feed grade variations.

Examination of the test results also shows that the five lowest values of F_1 are associated with three of the lowest concentrate grades which again suggests a direct rather than an inverse relationship between Fe_c and F_1 and it is suggested that F_1 has only a marginal effect, if any, on the value of Fe_c . This conclusion is strengthened by the fact that the multiple linear regression equation describing the relationship of Fe_c , Fe_f and F_1 yields mean and maximum deviations of 0.9 and 2.9 Fe per cent respectively, both values being unacceptable for a prediction model.

The per cent weight of total silica contained in the

+44 B.S. mesh fraction(F_2) has no significant relationship with concentrate grade, the correlation coefficient being - 0.23. The effect of F_2 on concentrate grade when the feed grade is held fixed is, however, positive and significant and indicates that for constant or similar feed grades an increasing value of F_2 is associated with increasing concentrate grade. The first order partial correlation coefficient for this association is $r_{Y2:1} = 0.64$ with a coefficient of determination of 0.407 indicating that about 40 per cent of the variation in concentrate grade at constant feed grade may be attributed to variations in the value of F_2 . This difference in significance is again attributed to the masking effects of feed grade variations on the smaller, but possibly significant, effects that F_2 has on concentrate grade.

The significance of this partial correlation coefficient may be interpreted as being due to easier transportation and separation of large quartz grains in the pulp because of the large surface area exposed to the descending flow of water on the spirals. This size fraction(+44 mesh) of the quartz distribution may be the optimum for the separation of quartz from hematite since the per cent weight of silica contained in the other size fractions does not have any significant relationship with concentrate grade. Microscopic examination of polished sections of ore reveals that quartz rarely occurs in the micaceous habit common to hematite in synclines A and D and the combination of low specific gravity and large surface area causes coarse quartz grains to be efficiently separated from hematite.

The significant association of Fe_c and F_2 is not, however, of sufficient magnitude to enable the two factors to be involved in an acceptable prediction model.

The value of the empirical combination $F_2 + F_3 - F_1$ in conjunction with feed grades has been suggested as a concentrate grade predictor (Wells, 1968a), and, that with constant feed grades, increasing values of $F_2 + F_3 - F_1$ are associated with increasing concentrate grades. Correlation analysis fails to

substantiate this association although the partial correlation coefficient between Fe_c and $F_2 + F_3 - F_1$, independent of feed grade, approaches statistical significance with a value of 0.53. The total correlation coefficient for Fe_c and $F_2 + F_3 - F_1$ is - 0.186 indicating an inverse relationship. The relatively large difference between these two types of correlation coefficient may be explained by the fact that there is a significant negative correlation ($r_{15} = -0.748$) between feed grade and $F_2 + F_3 - F_1$ which masks the influence of $F_2 + F_3 - F_1$ on concentrate grades when feed grades vary.

The values of $F_2 + F_3 - F_1$ plotted against feed grade were used to draw up a concentrate grade potential diagram (Wells, 1968a, Graph 2) which related feed grade, concentrate grade and the value of $F_2 + F_3 - F_1$ into five zones of potential concentrate grade as follows:

- (a) Less than 64.0 per cent Fe
- (b) Greater than or equal to 64.0 per cent Fe and less than 64.5 per cent Fe.
- (c) Greater than or equal to 64.5 per cent Fe and less than 64.7 per cent Fe.
- (d) Greater than or equal to 64.7 per cent Fe and less than 65.0 per cent Fe.
- (e) Greater than 65.0 per cent Fe.

When the original data is applied to this graph, only four (of thirteen) samples yielded a concentrate grade which falls within the predicted range and it is apparent that this graph fails completely in its attempt as a prediction model and cannot be used for grade control purposes. The multiple linear regression equation describing the behaviour of Fe_c , Fe_f and $F_2 + F_3 - F_1$ yields unacceptable mean and maximum deviations of 0.9 and 2.3 Fe per cent respectively, the equation being:

$$Fe_c = 53.6 + 0.23Fe_f + 0.04 (F_2 + F_3 - F_1) \dots \text{Equat. 4.2}$$

The pilot plant tests were also designed to examine the effects, on concentrate grade, of increasing the number of spirals from a 5 + 3-turn to a 5 + 5 + 5-turn system. The results were:

Sample	A [Ⓢ]	B [Ⓢ]	A-B	(A-B) ²
	per cent. Fe			
SA1	62.7	64.1	-1.4	1.96
2	67.5	66.3	1.2	1.44
3	63.4	64.2	-0.8	0.64
4	65.1	64.8	0.3	0.09
SD1	61.1	64.8	-3.7	13.69
2	64.6	66.2	-1.6	2.56
3	64.7	64.9	-0.2	0.04
4	66.6	67.8	-1.2	1.44
5	65.0	67.5	-2.5	6.25
SB1	64.1	61.7	2.4	5.76
2	59.0	61.9	-2.9	8.41

Ⓢ A = concentrate grade from 5 + 3-turn spiral system

Ⓢ B = concentrate grade from 5 + 5 + 5-turn spiral system

From the above table,

$$\bar{A} = 63.98 \text{ per cent Fe} \quad \bar{B} = 64.93 \text{ per cent Fe}$$

$$s_A = 2.39$$

$$s_B = 1.99$$

Examination of the means of the concentrate grades from the two spiral systems suggests that the 5 + 5 + 5-turn system is superior to the 5 + 3-turn system and this is what is generally expected according to the principles of spiral beneficiation. It will be noted however, that in three (of eleven) cases the concentrate grade from the 5 + 5 + 5-turn system is lower than that from the 5 + 3-turn system and statistical analysis of the results gives no reason to suspect that the 5 + 5 + 5-turn spiral system is more cost-effective than the simpler 5 + 3-turn system.

Two Students t tests carried out on the means and on the eleven pairs of concentrate grades do not confound Null Hypotheses of;

(a) no significant difference between the two means, (b) no significant difference among the paired data. On the results of these eleven pilot plant tests it cannot be stated with certainty

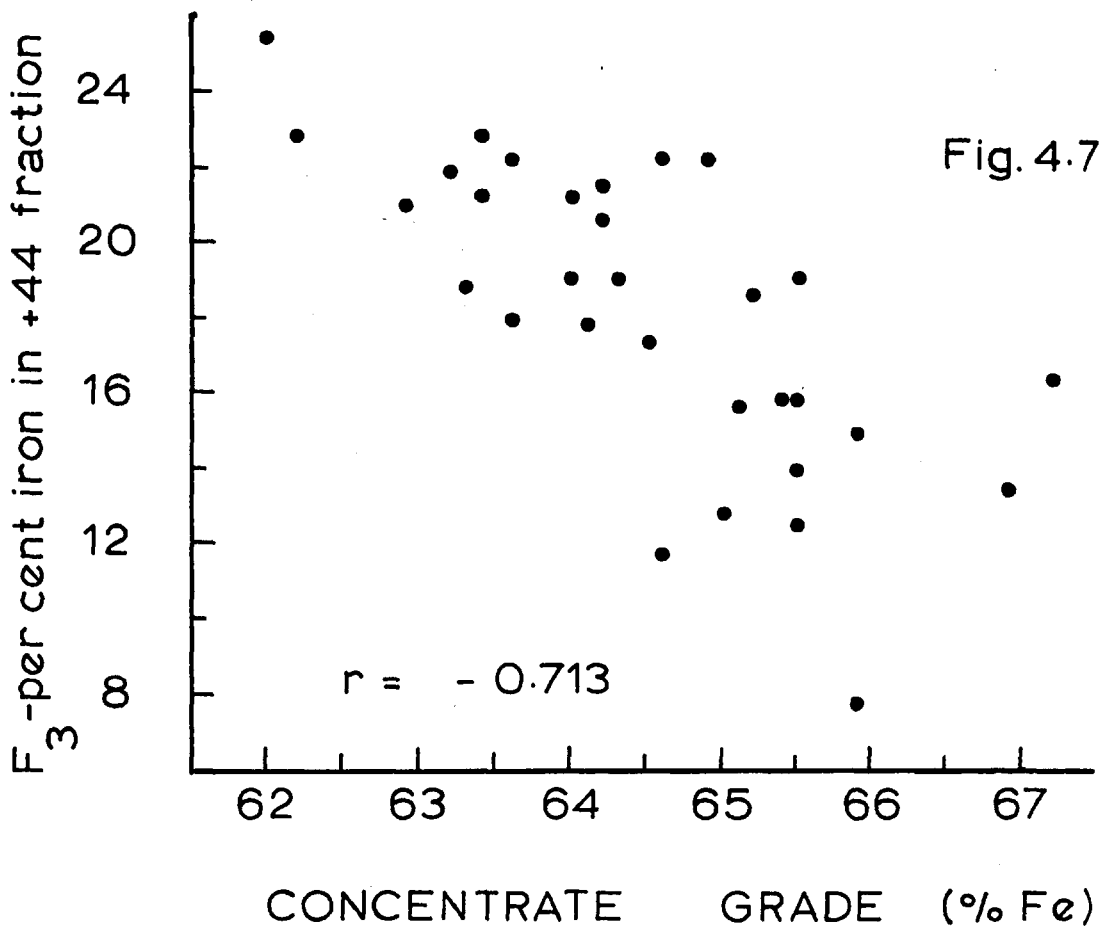
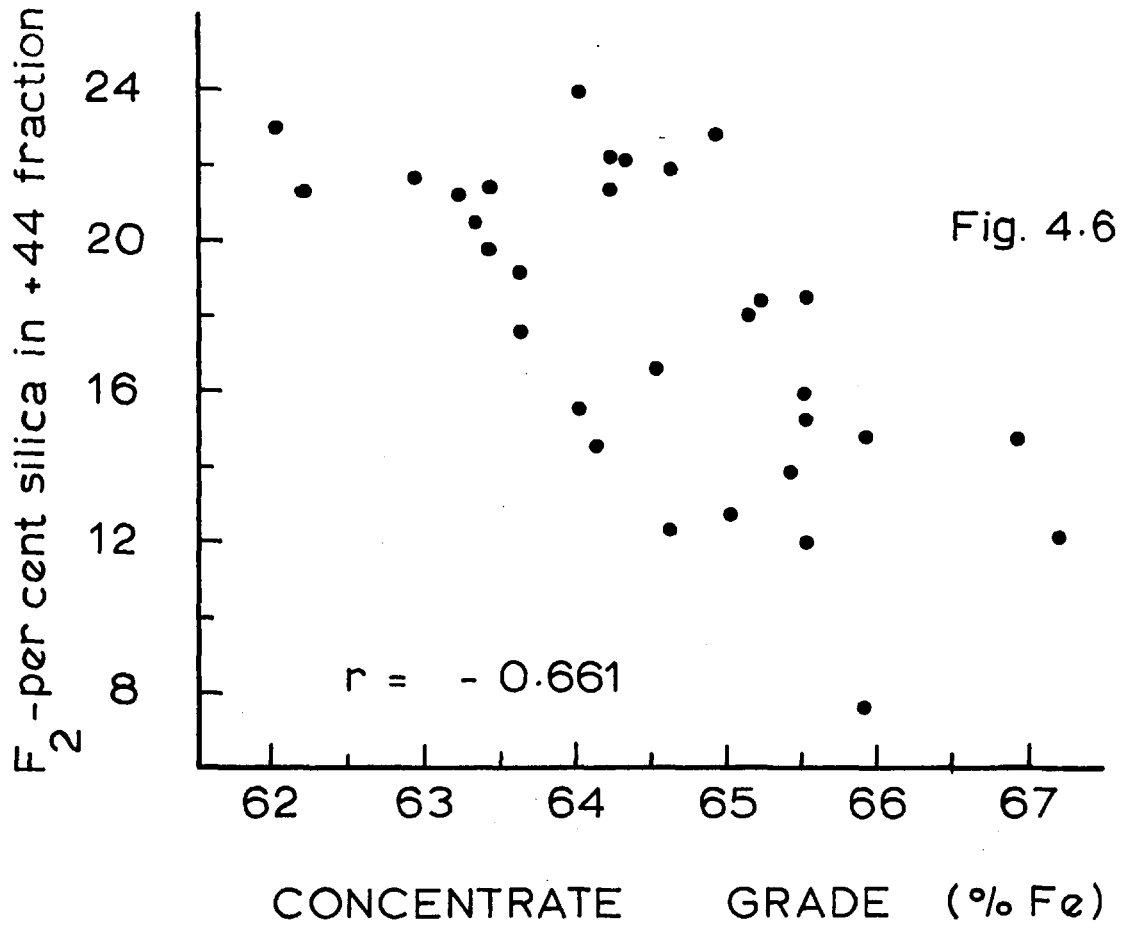
that the 5 + 5 + 5-turn spiral system produces concentrate grades consistently and statistically significantly higher than those produced on the 5 + 3-turn system. A tendency certainly appears to exist for the higher concentrate grades to be associated with the greater number of spiral turns but on the limited amount of evidence available it would be premature to recommend a complete change from 5 + 3-turn to 5 + 5 + 5-turn spiral system in the Marampa mill.

The pilot plant tests indicated that increasing the water:ore ratio from 6:1 to 7:1 was beneficial to the production of high grade concentrates. The increased amount of water probably acts against the buildup of micaceous hematite on the spirals and reduces the tendency of such hematite to form "sandbars". It has subsequently been found that a similar increase in the water:ore ratio in full scale mill operations also has a beneficial effect (E.G. Harvey, pers.comm) and this higher water:ore ratio of 7:1 is to be adopted throughout the Marampa mill.

(c) Conclusions.

This series of pilot plant spiral tests failed in its attempt to provide a satisfactory prediction model for concentrate grade from a knowledge of feed grade and the three grain-size statistics but the results do indicate that variations in certain factors, notably feed grade and F_2 , appear to be associated with sympathetic variations in concentrate grade. No single ore statistic is so significantly related to concentrate grade to allow its use as a predictor. Feed grade is the only single statistic to have a statistically significant correlation with concentrate grade and it is expected that any prediction equation which may arise from the results of further spiral tests will relate concentrate grade to feed grade and other ore characteristics.

Silica distributions are apparently more critical than corresponding iron distributions in determining final concentrate grade but it is suggested that the results of this series of spiral tests have only a qualitative application



and can only be used to indicate in which directions any further spiral tests should proceed.

The best estimate of potential concentrate grade obtainable from these results is given by the multiple linear regression equation:-

$$\text{Fe}_c = 47.5 + 0.29\text{Fe}_f - 0.11\text{F}_2 \quad \dots \quad \text{Equat. 4.3}$$

but this equation is not of sufficient precision to allow its use as a prediction model.

Variation in the number of concentrate withdrawal ports, orientation of concentrate splitters, ignorance of the between-syncline grainshape, insufficient sub-division of iron and silica grainsize distributions and insufficient samples have combined to mask any fine trends which may exist. Between-syncline hematite grainshape variations were not accounted for and the effects of grainshape variation on spiral beneficiation may well account for the mean deviation of 0.9 Fe per cent reported for the most efficient prediction equation.

The effects of variation in mill operating conditions (feed rate, number of concentrate ports etc.) are difficult to quantify and any further spiral tests should be carried out with all mill variables held fixed.

It is perhaps unfortunate that two of the most important ore synclines, in terms of contribution to amount of mill feed, were so poorly represented in this series of pilot plant spiral tests. Syncline C was not represented at all and syncline B was represented by only 2 samples, and this may be the reason why there is little correlation between pilot plant and full scale test results.

(c) Full Scale Mill Tests

(i) Iron and Silica Distribution

The statistically significant total correlation coefficients for the 31 full scale mill tests involving the six iron and silica grainsize distribution are as follows (Table 4.3)

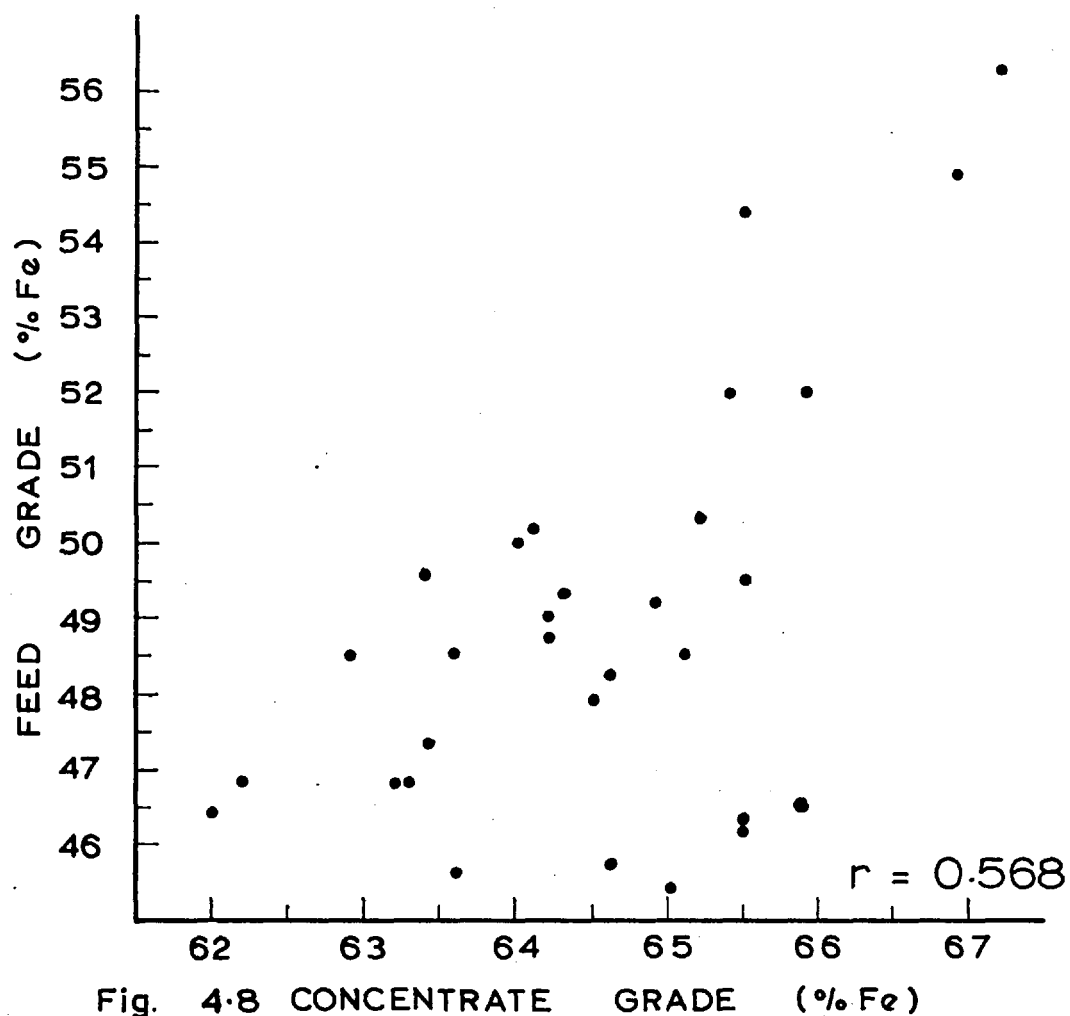


Fig. 4.8 CONCENTRATE GRADE (% Fe)

FULL SCALE MILL TESTS

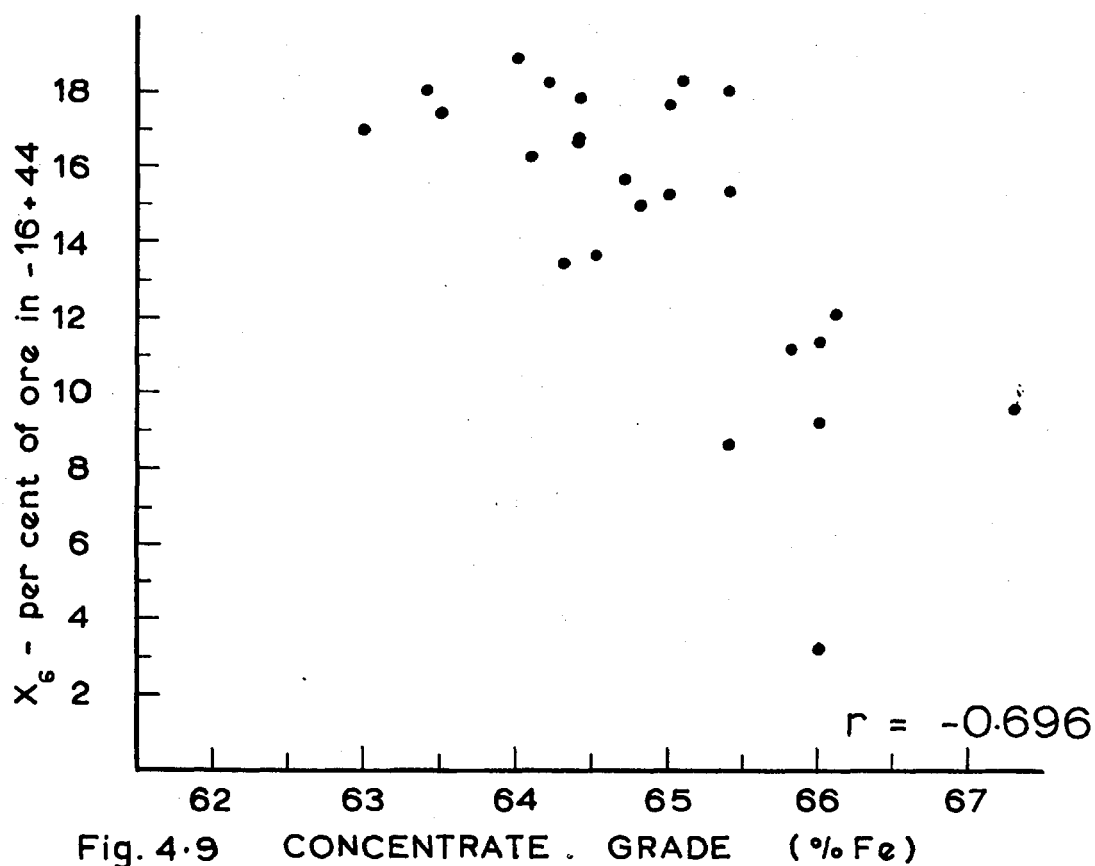


Fig. 4.9 CONCENTRATE GRADE (% Fe)

	Fe _f	F ₂	F ₁	F ₄	F ₃	F ₅	F ₆	F ₁ - (F ₂ +F ₃)	F ₂ +F ₃
Fe _c	0.568 0.323 ^{##}	-0.661 0.437	N.S. ^{##}	N.S.	-0.713 0.508	N.S.	N.S.	0.685 0.469	-0.704 0.496
Fe _f	-	N.S.	N.S.	N.S.	N.S.	0.578 0.334	N.S.	N.S.	N.S.
F ₂	-	-	-0.352 0.124	-0.61 0.372	0.902 0.814	-0.103 0.011	-0.614 0.377	-	-
F ₁	-	-	-	-0.526 0.276	N.S.	0.747 0.558	-0.364 0.133	-	-
F ₄	-	-	-	-	-0.60 0.36	-0.533 0.284	0.864 0.746	-	-
F ₃	-	-	-	-	-	N.S.	-0.661 0.437	-	-
F ₅	-	-	-	-	-	-	-0.65 0.423	-	-

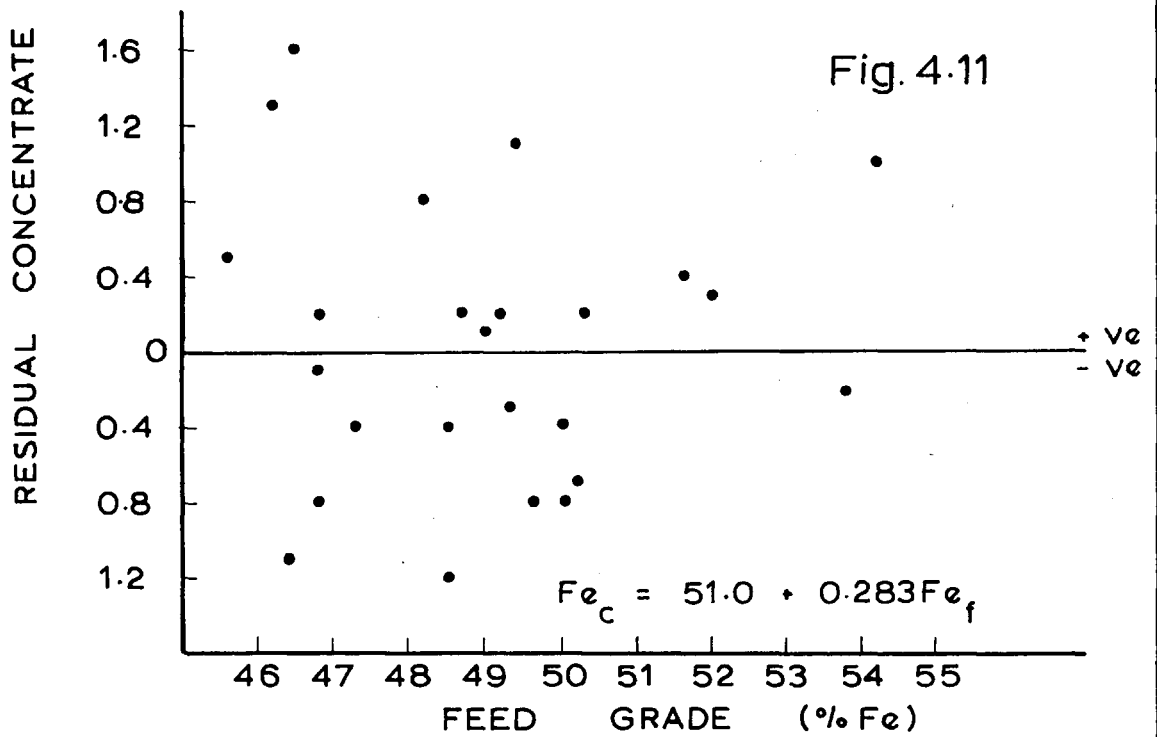
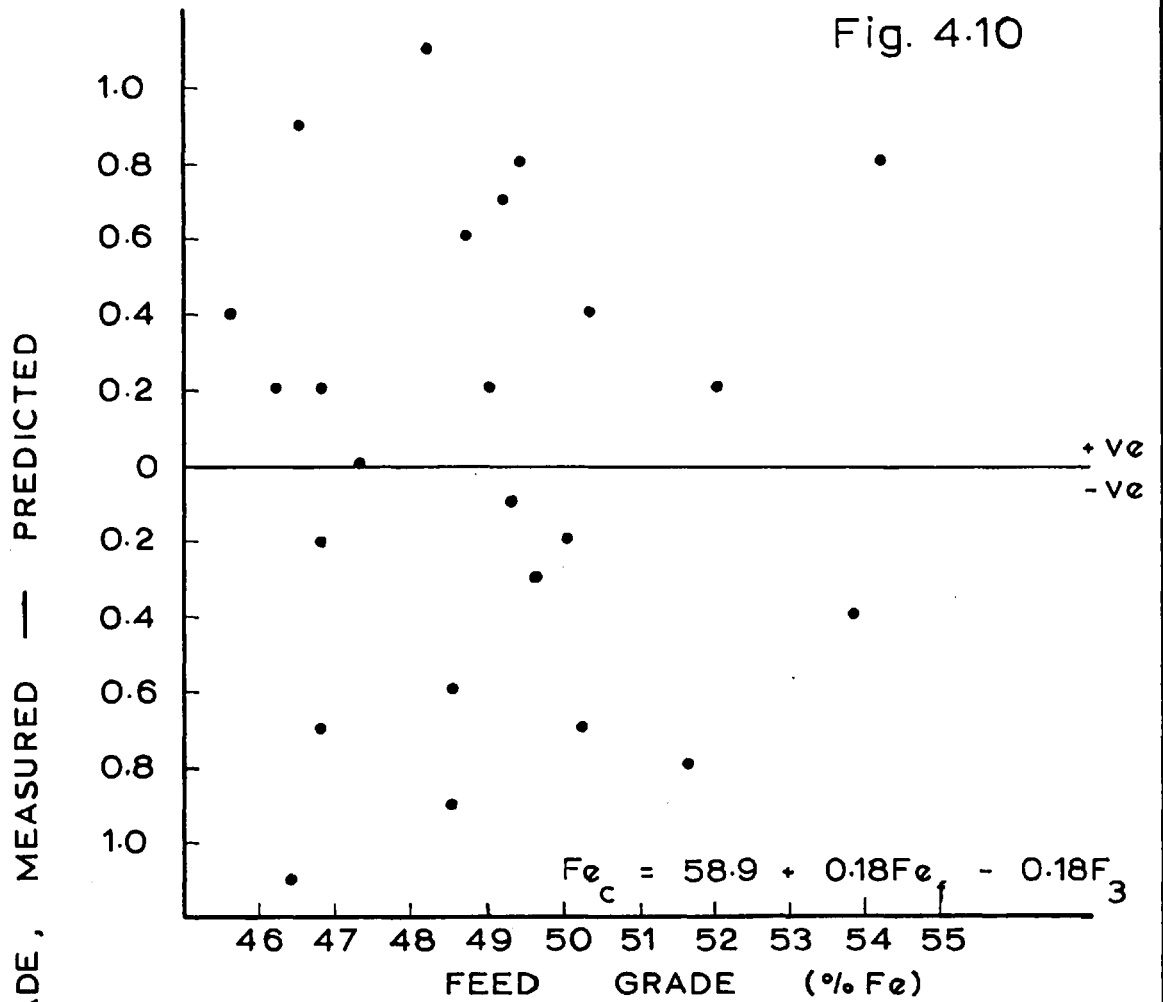
Table 4.3

^{##}Lower figure = coefficient of determination (r^2)

^{##} N.S. = not statistically significant

Concentrate grade (Fe_c) is significantly associated with feed grade (Fig 4.8), F₂, F₃, F₁ - (F₂ + F₃) and (F₂ + F₃), the most important single factor, as measured by the magnitude of coefficient of determination, being F₃, the per cent weight of total iron contained in the +44 mesh fraction. Concentrate grade and F₃ are inversely related, high concentrate grades being associated with low percentages of total iron in the +44 mesh fraction (Fig. 4.7).

This observation agrees with other theoretical and practical knowledge of the behaviour of particles on the Humphreys spiral and it is recognized (Humphreys Eng. Bull., No. 20, 1960) that above a certain size, particles on the spirals may behave as light minerals regardless of specific gravity. The coefficient of determination for the association of concentrate



FULL SCALE MILL TESTS

grade and F_3 is 0.508 indicating that about 51 per cent of the variations observed in concentrate grade may be attributed to variations occurring in F_3 . This value is much greater than that calculated for the pilot plant tests and greater than that for concentrate grade and feed grade which was the only significant factor affecting concentrate grade in the pilot plant tests. The multiple linear regression equation describing the relationship of Fe_c , Fe_f and F_3 is:-

$$Fe_c = 58.9 + 0.18Fe_f - 0.18F_3 \quad \dots \quad \dots \quad \text{Equat. 4.4}$$

and this equation yields mean and maximum deviations of 0.6 and 1.3 Fe per cent respectively when applied to the original data and thus, while this equation may serve as a crude warning model, it is not of sufficient precision to be used as a prediction model for grade control purposes.

This prediction model has been further studied by examination of residual concentrate grades and their association with feed grade. The residual concentrate grade is simply the observed (assay) concentrate grade minus the expected (predicted) concentrate grade. The scattergram (Fig. 4.10) graphically describes the variation of residual concentrate grade with feed grade and it can be seen that the magnitude and direction of the residual concentrate grade is independent of, and bears no systematic relationship with, feed grade which suggests that the relationship between Fe_c , Fe_f and F_3 is linear rather than non-linear.

A similar examination of the equation;

$$Fe_c = 51.0 + 0.283Fe_f \quad \dots \quad \dots \quad \dots \quad \text{Equat. 4.5}$$

also shows that residual concentrate grades are apparently independent of feed grade (Fig. 4.11)

Feed grade variations do not significantly affect the relationship between concentrate grade and F_3 as the partial correlation coefficient between the two, with feed grade held fixed, has a value of - 0.709 and a coefficient of determination which indicates that about 50 per cent of concentrate grade variations are attributable to F_3 variations.

The per cent weight of total silica contained in the +44 mesh fraction (F_2) is also inversely related to concentrate grade (Fig 4.6) and, like F_3 , the degree of association is little altered when the effect of feed grade variation is held fixed, since the partial correlation coefficient for these factors is $r_{Fe_c F_2 : Fe_f} = -0.641$.

The multiple linear regression equation describing the relationship of Fe_c , Fe_f and F_2 is very similar to that describing the relationship of Fe_c , Fe_f , and F_3 and this is explained by the high degree of association ($r = 0.902$) between F_2 and F_3 . Similarly, F_1 and F_5 (the -44 +120 mesh silica and iron distributions) and F_4 and F_6 (the -120 mesh silica and iron distributions) also have high correlation coefficients which are reflections of the sympathetic grainsize relationship existing between hematite and quartz in the Marampa ore. This similarity of hematite and quartz grainsize distributions is probably the result of response to the metamorphism of the ore and not a primary feature.

Conclusion

Regression and correlation analysis of iron and silica distributions indicates those factors which appear to be significantly related to concentrate grade but fails to provide a sufficiently precise prediction model for grade control purposes, and it is considered that other, as yet unmeasured, factors are responsible for the deviations between predicted and observed concentrate grades.

The best estimate of potential concentrate grade obtained from this series of full scale mill tests is given by:-

$$Fe_c = 58.9 + 0.18Fe_f - 0.18F_3$$

The mean and maximum deviations of 0.6 and 1.3 Fe per cent respectively indicate that this equation is a more efficient prediction model than any proposed from the results of the pilot plant spiral tests.

(ii) Total Grainsize Distribution

Humphreys spirals are often most efficient when operating with a classified feed and when the bulk of the feed is within the optimum grainsize range 0.1 to 1.0mm. Analysis of total grainsize distribution may indicate which size fractions, if any, affect the concentrate grade and whether the feed should be classified in order to produce acceptable quality concentrates.

Total correlation analysis yields the following significant correlation coefficients (Table 4.4)

	Fe _f	SiO _{2f}	X ₅ + 16	X ₆ -16+44	X ₇ -44+85	X ₈ -85+120	X ₉ -120+200	X ₁₀ -200
Fe _c	0.646 0.417*	-0.654 0.428	N.S.	-0.696 0.484	N.S.	0.573 0.328	N.S.	N.S.
Fe _f	-	-0.886 0.785	N.S.	-0.476 0.226	N.S.	0.364 0.132	N.S.	N.S.
SiO _{2f}	-	-	N.S.	0.573 0.328	N.S.	0.448 0.201	N.S.	N.S.

Table 4.4

* Lower figure = coefficient of determination.

Concentrate grade is apparently sensitive to variations in the proportions of two size fractions, -16 +44 (Fig. 4.9) and -85 +120 B.S. mesh and, as shown by earlier correlation analysis, the coarse fraction (+44 mesh) is not amenable to spiral beneficiation and it is suggested that reduction in the amount of +44 mesh ore may significantly improve the resultant concentrate grade. Reduction in the amount of +44 mesh ore would involve preliminary screening at, say, 16 B.S. mesh (1.00 mm) and rod mill comminution before spiral treatment. At present the ore is initially screened at $\frac{1}{8}$ inch (3.2 mm) before spiral treatment.

The partial correlation coefficient between Fe_c and the per cent of total weight in the -16 +44 mesh fraction, with the effects of feed grade held constant, is also negative and

significant although somewhat reduced to $r = 0.58$, and it is postulated that, under the present mill operating conditions, the Marampa hematite in the +44 mesh fraction tends to behave as gangue despite the specific gravity differential.

The regression equation describing the behaviour of Fe_c , Fe_f and the amount of material in the -16 +44 mesh fraction is the most efficient prediction model for concentrate grade obtained from the Marampa beneficiation tests having mean and maximum deviations of 0.5 and 1.1 Fe per cent respectively. The equation is:

$$Fe_c = 58.0 + 0.18Fe_f - 0.13X_6 \quad \dots \quad \dots \quad \text{Equat. 4.6}$$

where

Fe_c = concentrate grade (per cent Fe)

Fe_f = feed grade (per cent Fe)

X_6 = per cent weight of total ore in the -16 +44 B.S. mesh fraction.

The fact that this equation gives the minimum mean deviation and the minimum maximum deviation of all linear and multiple linear regression equations calculated for significant correlation coefficients from Marampa beneficiation test results is accepted as evidence of its superiority as a first order prediction model although for grade control purposes it is only marginally acceptable, and it is apparent that concentrate grade cannot be accurately predicted from a knowledge of size distribution and feed grade alone.

A scattergram of mill feed grade (Fe_f) and the proportion of mill feed contained in the -16 +44 B.S. mesh fraction (X_6) has been superimposed with three concentrate grade isocons (Fig. 4.12) derived from the prediction model;

$$Fe_c = 58.0 + 0.18Fe_f - 0.13X_6$$

These isocons described the relationship between Fe_f and X_6 for the three concentrate grades 64.0, 64.5 and 65.0 per cent Fe.

The open circles (Fig. 4.12) represent observed concentrate grades (from the full scale mill tests) which fall on the opposite side of the isocon, representing the concentrate target grade

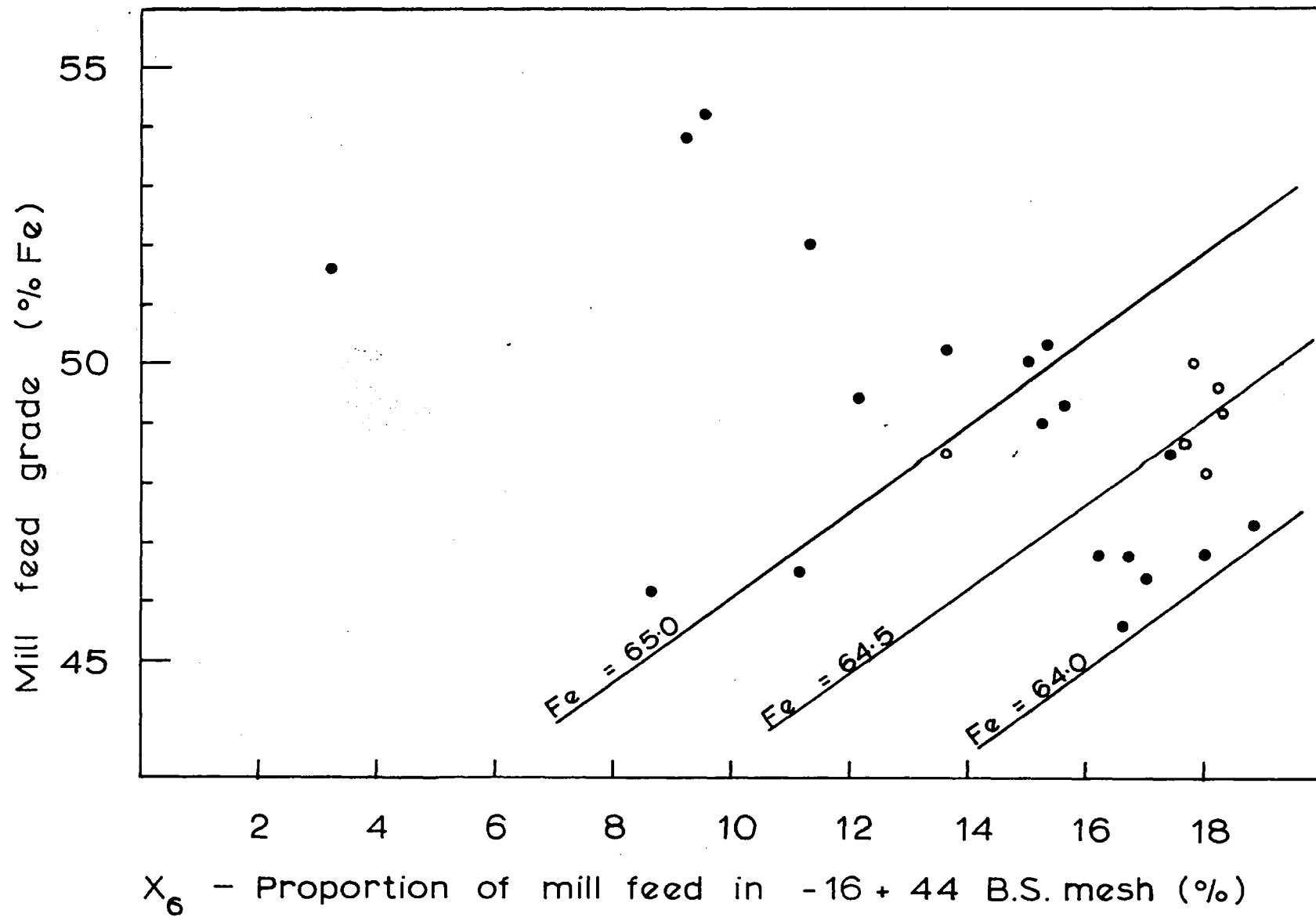


Fig. 4.12

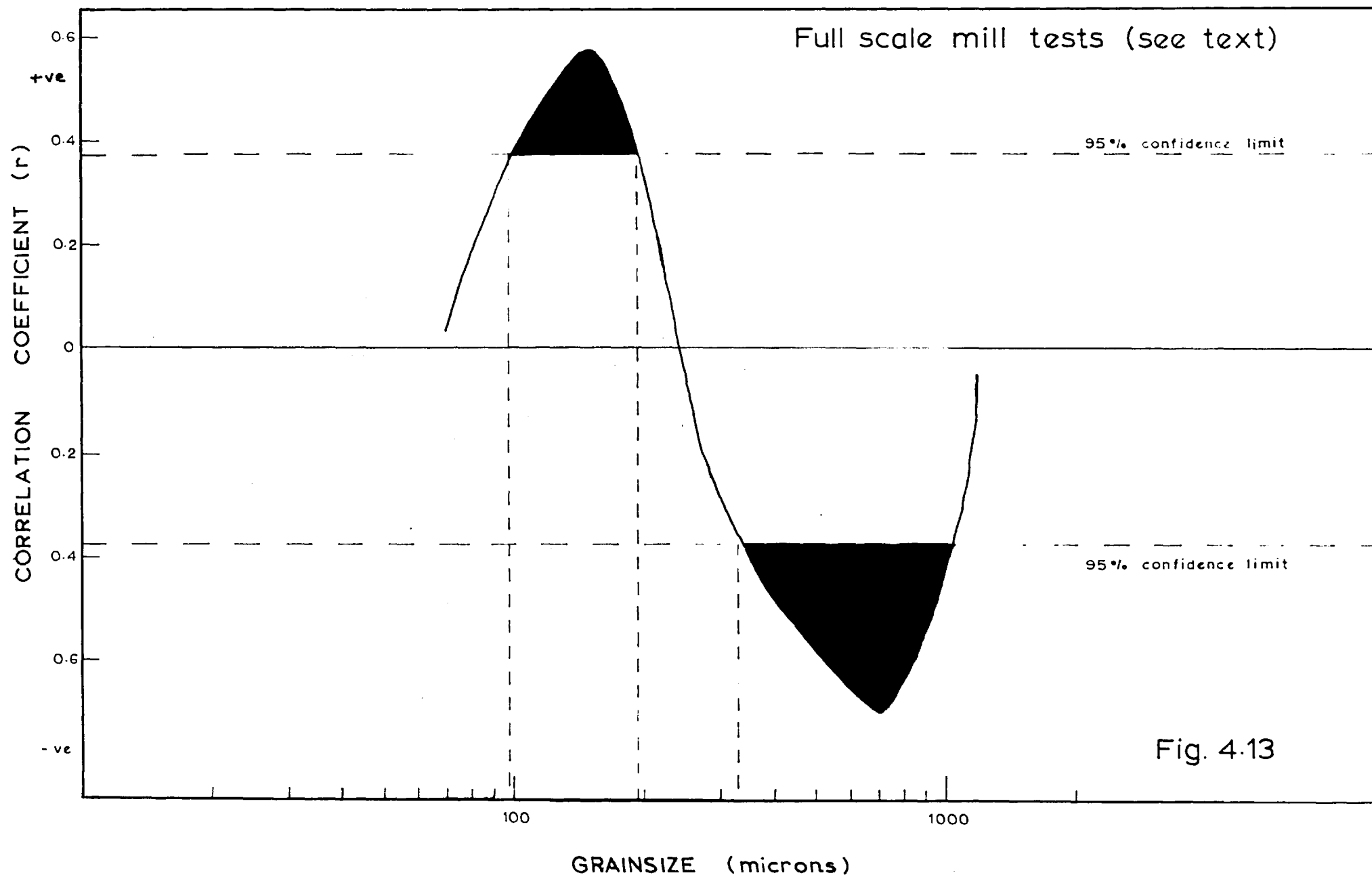
of 64.5 per cent Fe, to which they were assigned by the prediction model, i.e. they are observed grades of greater than 64.5 per cent Fe which were predicted (from a knowledge of Fe_f and X_G) to be less than 64.5 per cent Fe, and vice versa. The black circles represent observed concentrate grades falling on the same side of the 64.5 per cent Fe as the predicted grades.

It will be noted that in 20 of 26 cases, i.e. between 75 and 80 per cent, the prediction model assigns the predicted grade to the correct side (whether greater than or less than 64.5 per cent Fe) of the concentrate target grade. Although precise concentrate grade prediction is not feasible with this prediction model it may find use as a guide as to whether the concentrate grade will be substandard (less than 64.5 per cent Fe) or acceptable (greater than or equal to 64.5 per cent Fe).

This prediction model is the most efficient model available from the Marampa beneficiation tests but is not of sufficient precision to allow its use as an absolute prediction model and it is suggested that other, unmeasured, factors are responsible for the deviations between expected (predicted) and observed (measured) concentrate grades.

The proportion of ore contained in the -85 +120 mesh fraction is significantly related to the concentrate grade ($r = 0.573$, $r(\text{part.}) = 0.475$) indicating that high grade concentrates are associated with large proportions of ore in this size fraction, and for Marampa ore this is the optimum grain-size range for successful beneficiation on Humphreys spirals and also suggests that reduction of coarse ore to at least 90 per cent -60 mesh would result in higher grade concentrates.

A plot of the total correlation coefficients between concentrate grade and the proportion of ore in the various grainsize ranges against grainsize yields a sinusoidal curve on arithmetic - logarithmic paper, (Fig. 4.13), and indicates more precisely that the optimum grainsize range for spiral beneficiation of Marampa ore is between 100 and 200 microns, about the size range -85 +150 B.S. mesh. This is based on a 95 per cent confidence level cut-off for correlation coefficients and it appears that ore containing at least 90 per cent



-60 mesh material will produce acceptable concentrates at the optimum recovery. Concentrate grade is detrimentally affected by ore with increasing amounts of +60 mesh material although this graph must be interpreted with caution since the per cent weights of each size fraction sum to a constant (100 per cent) and the system is thus closed and the value of the statistically non-significant correlation coefficients are probably meaningless. In addition a certain degree of autocorrelation exists in the data since some of the grainsize fractions contribute disproportionately large amounts to the mill feed, but it is probably safe to say that the optimum grainsize range for the spiral beneficiation of Marampa ore is between 100 and 250 microns and that the concentrate grade is adversely affected by increasing amounts of ore coarser than about 350 microns (+44 B.S. mesh). One explanation of this effect of grainsize may be related to the grainshape differential between the two groups of ore synclines, synclines A and D and synclines B and C. The hematite of synclines B and C is finer grained and equidimensional whereas that of synclines A and D is coarser grained and micaceous. The equidimensional hematite is more easily concentrated on the Humphreys spirals than is the micaceous hematite and this produces a partition in concentrate grades;

- (a) synclines A and D = coarse and micaceous = lower concentrate grades
- (b) synclines B and C = fine and equidimensional = higher concentrate grades

D CONCLUSION

Correlation and regression analysis of Marampa beneficiation data indicates that hematite concentrate grades are significantly associated with;

- (a) Feed grade of ore, both iron (Fe_f) and silica (SiO_2_f).
- (b) Per cent weight of ore in the -16 +44 (X_6) and in the -85 +120 (X_8) B.S. mesh fractions.
- (c) Per cent weight of total iron in the +44 B.S. mesh fraction (F_3).

The two most significant factors being feed grade (Fe_f), F_3 and X_6 .

Increasing amounts of +60 mesh ore are associated with decreasing concentrate grades and it is suggested that reduction of the ore to 90 per cent -44 mesh before spiral beneficiation would materially improve concentrate grades. It appears that the detrimental effect of the -16 +44 B.S. mesh ore is due to the particle size alone rather than the specific gravity or composition of the particles and suggests that hematite in this size range behaves as a light mineral on the Humphreys spirals.

The most efficient prediction model for concentrate grade arising from the test data is;

$$Fe_c = 58.0 + 0.18Fe_f - 0.13X_6$$

where,

Fe_c = concentrate grade (per cent Fe)

Fe_f = feed grade (per cent Fe)

X_6 = per cent weight of total ore in the -16 +44
B.S. mesh fraction

The minimum mean and minimum maximum deviations of 0.5 and 1.1 Fe per cent yielded by this prediction model are too great to allow the equation to be used for grade control purposes. It is considered that these deviations are caused by various combinations of;

- (a) Grainshape differentials.
- (b) Between - and within - mill unit feed rate variations.
- (c) Variations in number of concentrate withdrawal ports.
- (d) Variations in orientation of concentrate port splitters.

The above sources of variation are effective at all times and a unique prediction model for concentrate grade is probably not feasible but a warning system prediction model which would indicate whether a particular ore will yield a low, average or high grade concentrate could conceivably be evolved with further pilot plant tests incorporating a finer

grainsize distribution subdivision and a close study of hematite grainshape.

It is not known whether or not the results of the beneficiation tests are directly applicable to concentrate grade prediction from a knowledge of the in situ characteristics of the ore since the full scale mill tests, which have provided the most efficient prediction model, were carried out on blends of ore from several ore synclines and it is recognized that between-syncline grainsize distributions and grainshape are significantly different. This suggests that comprehensive pilot plant spiral tests would be necessary in order to design a within-syncline concentrate grade prediction model.

The response of the concentrate grade to changes in the significant variables may well be non-linear in which case correlation analysis on raw data would yield non-significant associations without data transformations. Graphical plots of the variables against concentrate grade however, fails to suggest major non-linear associations and it appears probable that unmeasured variables may play a more prominent part in concentrate grade variation than has hitherto been recognized.

A gross analysis of full scale concentration may be carried out by considering input variables (feed grade) and output variables (concentrate grade) with the mill acting as an amplifier, the two variables being correlated by time series analysis.

CHAPTER V

THE MINERALOGY OF THE MARAMPA ORES, CONCENTRATES AND TAILINGS

A. INTRODUCTION

World demand for steel increased substantially after the Second World War and certain steel-making countries, which had previously met their iron ore requirements from domestic sources, became concerned at the apparent depletion of their iron ore reserves. This apparent shortage of iron ore had two main effects:

- (a) Exploration for new deposits was intensified.
- (b) Metallurgical research was stimulated in order to utilise low grade ores and resulted in advancements in pelletising and sintering technology.

World iron ore reserves have since increased enormously and in 1965 were put at about 248,000 million tons (Mining Journal, 11-10-68) compared with an earlier figure of 84,580 million tons in 1954 (U.N. Survey of World Iron Ore Resources). Many new high grade direct shipping iron ore bodies are now being exploited and the lifting of the Australian embargo on the export of iron ores in the early 1960's coupled with the emergence of Japan as a major iron and steel producer [1968 figures; 2.2 million tons of domestic iron ore and steel production of about 68 million tons, (Mining Annual Review, 1969)] have resulted in the present iron ore market being characterised by generally low prices and an excess of supply over demand. Japan, being a large buyer of iron ore, is able to dictate the price to a certain extent and it is most important for iron ore producers to reduce production costs in order to compete successfully and to maximise profits.

Marampa has been in production since before the Second World War and has operated for most of its life in an economic environment of iron ore demand. The beneficiation costs of producing the Marampa concentrates are probably low by world standards but in order to compete successfully on the world

market the concentrate grade and iron recovery must remain high.

In the past the Marampa ore has simply been mined, beneficiated and sold but with the long term state of the iron ore markets favouring iron ore buyers, together with technical factors relating to Marampa mining methods and ore characteristics (increasing ore hardness for example) systematic microscopic examination of the Marampa ores with regard to ore-dressing properties appears to be overdue and should be introduced before the concentrate grade and recovery of iron further decreases and cost of concentrate production increases. With this in mind the Marampa ores concentrates and tailings have been microscopically examined, from the ore-dressing, rather than the academic, point of view.

The grainsize, grainshape and the distribution and association of the various ore and gangue minerals are best observed by reflected light examination of polished sections or transmitted light examination of thin sections of the particular ore, concentrate or tailings. In addition to the semi-quantitative and qualitative description of the particular specimen, various qualitative conclusions can be reached regarding the origin and history of the ore deposit and, most important, information may be gained which may be applicable to ore-dressing or beneficiation of the ore, i.e. the microscopic examination may have direct economic implications.

Chemical analysis of an ore gives an indication of the absolute proportions of the valuable ore metals but fails to indicate the proportions of the valuable minerals which can be recovered during beneficiation. The microscopic examination of the ore reveals the physical attributes of the ore minerals and indicates to what size the ore must be crushed to yield optimum recovery. This is a particularly important aspect of the examination of low to medium grade ore deposits when it is considered that crushing and grinding are often the single most expensive operations in ore beneficiation and any technique which assists in reducing these costs enables production costs to be lowered and ultimate profits to be increased.

Systematic microscopic examination of ore specimens and samples from various parts of the orebody indicate in what form the ore minerals occur and may enable more efficient milling methods to be introduced. Similarly, intergrowths and exsolution textures are revealed and recommendations may be made as to their effective removal if widespread and deleterious. Valuable mineral inclusions in gangue minerals are readily observed and may be recovered if suitable treatment of tailings can be suggested from the microscopic evidence.

Polished and thin sections of Marampa ore, concentrate and tailings specimens and samples have been microscopically examined in order to determine:

(a) Grainsize distributions of hematite in the various ore synclines. The absolute grainsize of liberated hematite has an optimum range within which spiral beneficiation is most efficient, i.e. gives the optimum concentrate grade and iron recovery coarse material, either as large discrete hematite grains or as composite grains, tend to act as gangue minerals on the Humphreys spirals and eventually report as tailings. In a similar manner, at the opposite end of the size range, very fine hematite, mainly that less than 100 microns diameter, also tends to report as tailings. Microscopic examination of the ore can indicate the presence of very fine hematite, either original fine-grained or secondary fine-grained hematite which has formed as a result of brecciation, and coarse grained hematite and if present in a sufficiently large volume the mill operating conditions, particularly the feed rate and spiral settings, may be altered to combat the loss of the very fine or very coarse grained hematite.

(b) Grainshape distribution of hematite in the various ore synclines. The shape of the individual hematite grains undergoing Humphreys spiral beneficiation influences the final hematite recovery and flat micaceous or tabular grains are believed to be only poorly susceptible to efficient spiral beneficiation, whereas the optimum grade and recovery of hematite concentrates probably occurs with equidimensional to spherical grains. Micaceous grains lie flat on the spirals

and have a minimum surface area exposed to the downward flow of the pulp. In these circumstances the flat grains tend to settle on the spirals and the movement of the pulp may be insufficient to overcome the frictional forces operating. The settling of the flat grains causes banking on the spirals and a barrier is eventually formed which causes a dramatic increase in the amount of turbulent flow down the spirals. Turbulent flow is anathema to efficient spiral beneficiation and causes a reduction in grade of the final concentrate. Microscopic examination of the ore reveals the presence and proportions of micaceous grains and with systematic mapping of the shape of hematite grains it may be possible to plan long term changes in milling practice.

(c) Distribution of detrimental intergrowths in ore minerals. Exsolution intergrowths, particularly those of the titaniferous minerals rutile and ilmenite, commonly occur in iron ore deposits and, if present above a certain amount, may be detrimental to the sale of hematitic ores.

(d) Grainsize and grainshape of hematite and gangue minerals recovered in concentrate and tailings. The microscopic examination of tailings indicates the form of hematite which is not amenable to spiral beneficiation and also indicates the efficiency of beneficiation by revealing whether hematite with dimensions within the optimum size and shape ranges is regularly reporting as tailings.

B. MICROSCOPIC EXAMINATION OF MARAMPA ORE

Mechanical grainsize analysis and megascopic structural features enable the Marampa ores to be subdivided into two main groups and the validity of this subdivision has been corroborated by microscopic examination of the ores.

Earlier structural interpretation of the Marampa orebody (Kennedy and Thomson 1962) indicated that the four major ore synclines are all part of the same original ore zone, but grainsize analysis and microscopic examination of ore specimens tends to refute this theory and instead suggests that two

separate and distinct ore zones may originally have been present. The first ore zone, possibly the older of the two, is now represented by synclines A and D; the second ore zone is now represented by synclines B and C. The ores of synclines A and D are sufficiently similar in terms of mineralogy, hematite grainsize and grainshape, and apparent degree of deformation to be considered and described together; similarly for the ores of synclines B and C. Although the evidence suggests two original ore zones, it is possible that differential deformational stress produced the present structures and textures observed in the open-pit and in microscopic examination respectively.

The two types of ore are considered under the following headings:

- (a) the ores of synclines A and D.
- (b) the ores of synclines B and C.

(a) Microscopic Examination of the Ores of Synclines A and D

The ores of synclines A and D are almost invariably coarse grained, hematite-quartz-muscovite schist (Fig. 5.1C) with accessory cordierite, tourmaline and manganese oxides. The ores are distinctly foliated and complexly deformed (Fig. 5.1A and 5.1D) and have undergone at least two periods of deformation.

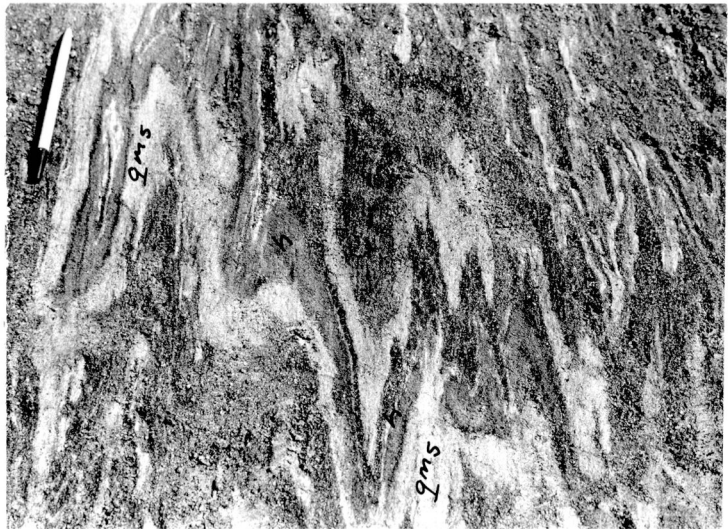
The hematite is dominantly euhedral with well defined crystal faces and sharp angular grain boundaries. Individual hematite grains are tabular to micaceous and have an extremely high degree of both dimensional and lattice preferred orientation (Fig. 5.2E). The crystallographic and morphological axes of the hematite are coincident and exhibit a very strong tendency towards parallelism (Fig. 5.1B). The high degree of preferred orientation and the micaceous habit of the hematite are the characteristic features of the ores of synclines A and D.

The hematite has a typical grainsize in the range 100 to 500 microns but the maximum and minimum grainsize (long axes) are greater than 2mm and less than 5 microns respectively. The very fine grained hematite, which occurs mainly as minute

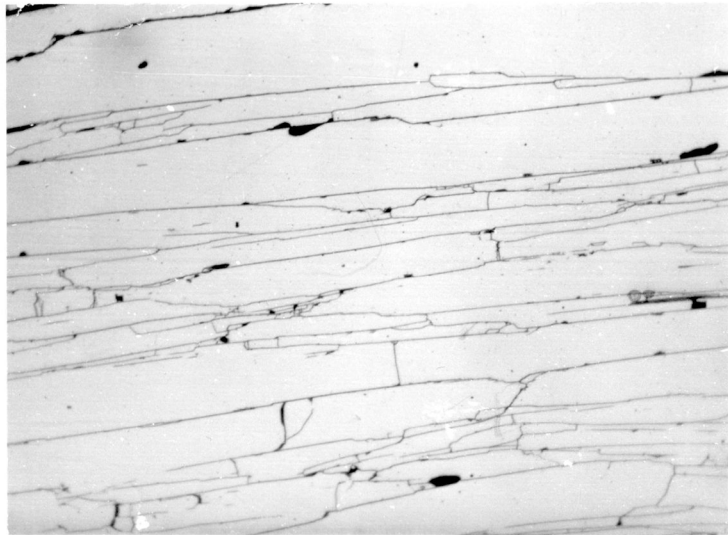
FIG. 5.1

THE ORES OF SYNCLINES A AND D

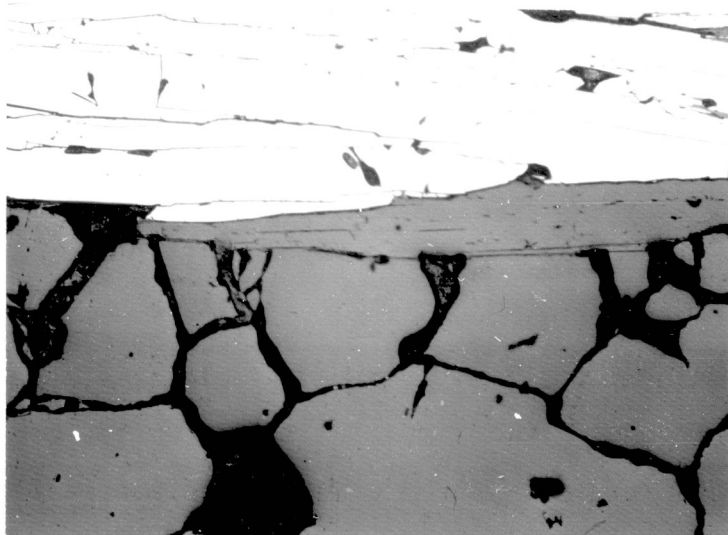
- (A) Complexly deformed quartz-muscovite schist (qms) and hematite (h). Pen approximately six inches long.
- = (B) Micaceous hematite with extremely well developed dimensional preferred orientation. Reflected and plane polarised light. Scale bar 200 microns.
- (C) Hematite (white), quartz (grey) and muscovite (grey, micaceous, between hematite and quartz). Note well developed dimensional preferred orientation of hematite and muscovite and complete lack of dimensional preferred orientation of quartz. Reflected and plane polarised light. Scale bar 200 microns.



A



B



C

Fig. 5.1

Fig. 5.1 (continued)

THE ORES OF SYNCLINES A AND D

- (D) Complexly deformed quartz-muscovite schist and hematite.
- (E) Well developed dimensional preferred orientation of hematite. Note fractures and slight bending about the axial plane of the fold. Reflected and plane polarised light. Scale bar 200 microns.

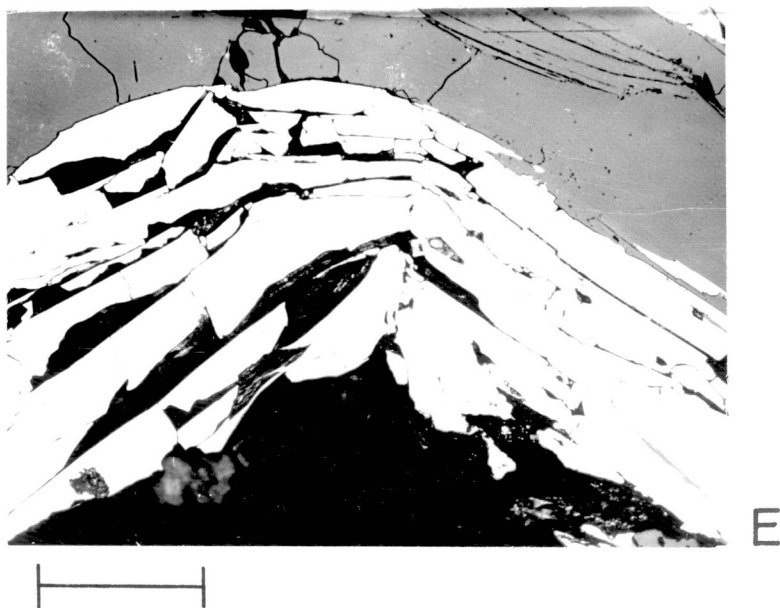
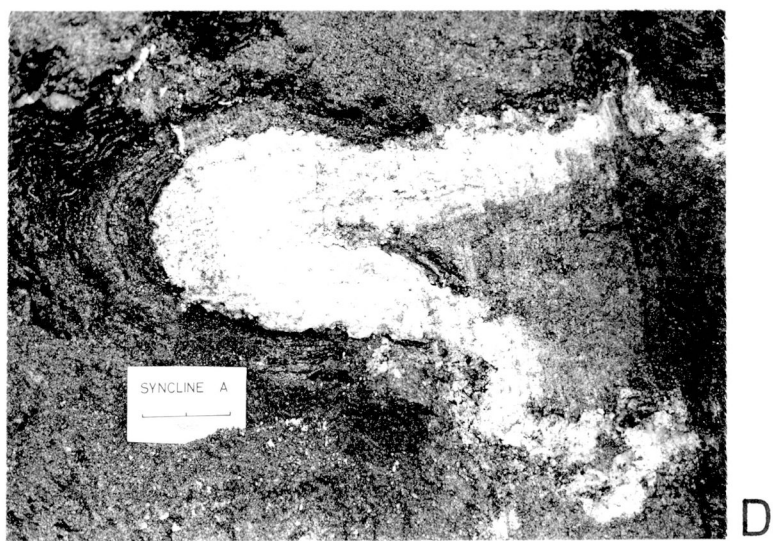


Fig. 5.1

inclusions in quartz grains, is usually highly euhedral, rhombic to dodecahedral and is equidimensional rather than tabular or micaceous, and often exhibits deep red internal reflections under oil immersion. Independent mechanical grainsize analysis of ore from syncline A (Table 5.1) indicates that most of the hematite grains are coarser than 85 B.S. mesh.

Fraction		Iron Distribution
B.S. Mesh	Microns (μ)	(per cent)
+ 44	+ ~~350~~	25.5
- 44 + 60	-350 +252	19.0
- 60 + 85	-252 +177	20.6
- 85 +120	-177 +125	15.8
-120 +170	-125 + 88	9.2
-170 +300	- 88 + 53	6.0
-300	- ~~53~~	3.9

Table 5.1 Grainsize distribution of iron in syncline A

Secondary deformation twinning of hematite is common in the ores of synclines A and D with the twinned hematite apparently associated with dragfold locations. The twinning is polysynthetic and conjugate deformation twins are common (Fig. 5.3D). The twinning takes three major forms;

- (a) many long and narrow polysynthetic twins.
- (b) smaller number of thick twins probably formed by lateral growth and coalescence of adjacent twins (Spry, 1969).
- (c) very numerous disjointed micro-twins aligned parallel to the long axis of the micaceous hematite. These micro-twins occur in swarms as disjointed rods and are only readily visible under high power (X 600) with crossed nicols and may be a physical expression of the hematite translation slip planes.

Hematite twins are commonly wedge-shaped and conjugate twins usually show slight displacement at their intersections.

FIG 5.2

THE ORES OF SYNCLINES A AND D

- (A) Low grade ore. Equidimensional quartz (white, grey) and scattered micaceous hematite (black). The rare muscovite (white and micaceous) and the hematite are parallel and exhibit well developed dimensional preferred orientation. Transmitted light. Crossed nicols. Scale bar 1.00 mm.
- (B) Diamond drill core, location and depth unknown but presumed to be from synclines A or D. The presence of calcite (large grain lower left) indicates that the specimen was recovered from well below the present open pit. Micaceous to tabular hematite (black) and sub-parallel muscovite. Transmitted and plane polarised light. Scale bar 1.00 mm.
- (C) Equidimensional quartz (white, grey) hematite (black) and muscovite (acicular, white). Hematite and muscovite are sub-parallel. Transmitted light. Crossed nicols. Scale bar 1.00 mm.

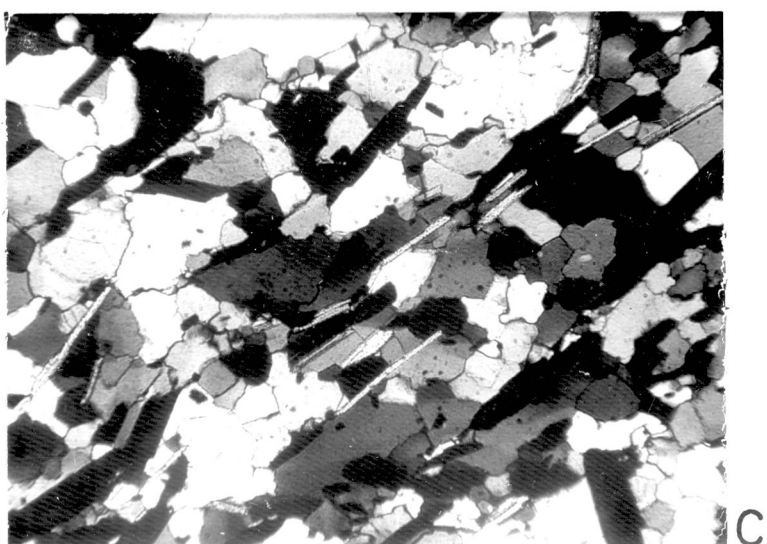
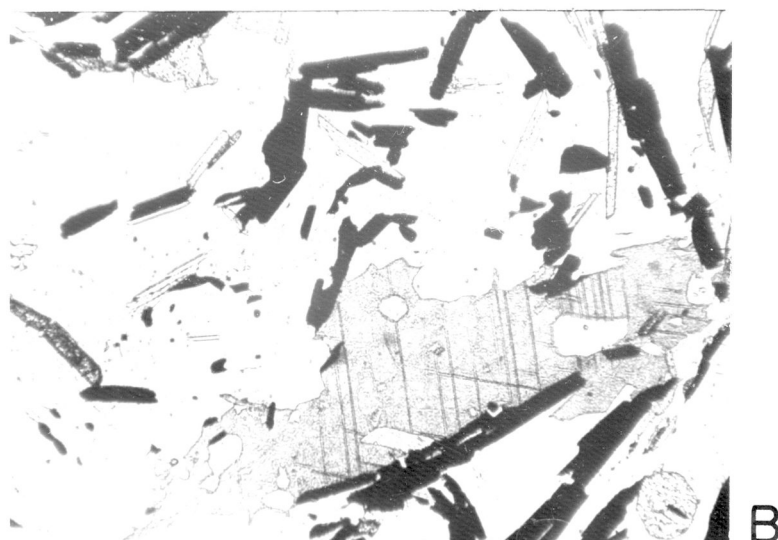
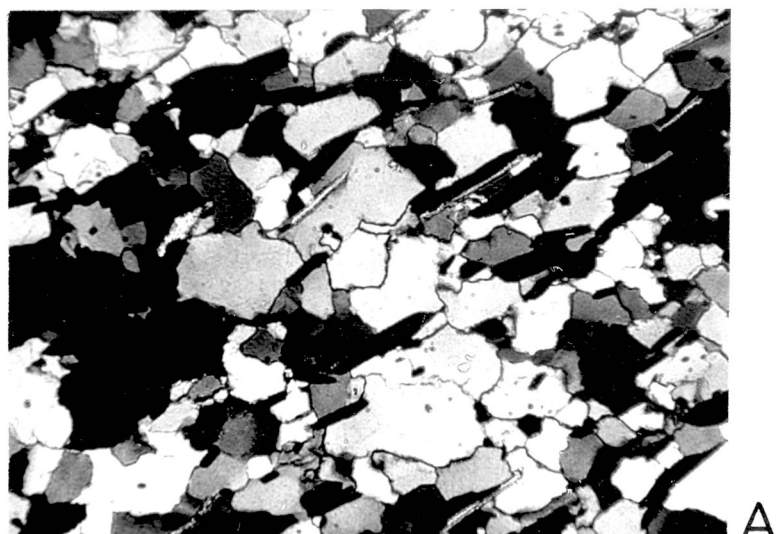


Fig. 5.2

Fig. 5.2 (continued)

THE ORES OF SYNCLINES A AND D

- (D) Tabular hematite (black) muscovite (grey) and quartz (white). Low grade ore. Note non-parallelism of muscovite and hematite. Transmitted and plane polarised light. Scale bar 1.00 mm.
- (E) Well developed dimensional and lattice preferred orientation of hematite. Note twinning of hematite near centre. Reflected light. Nicols partially crossed. Scale bar 200 microns.
- (F) Hematite (black), quartz (grey, white) and muscovite (grey, mottled). Note undulose extinction in quartz and slightly sutured quartz grain boundaries. Transmitted light. Crossed nicols. Scale bar 1.00 mm.

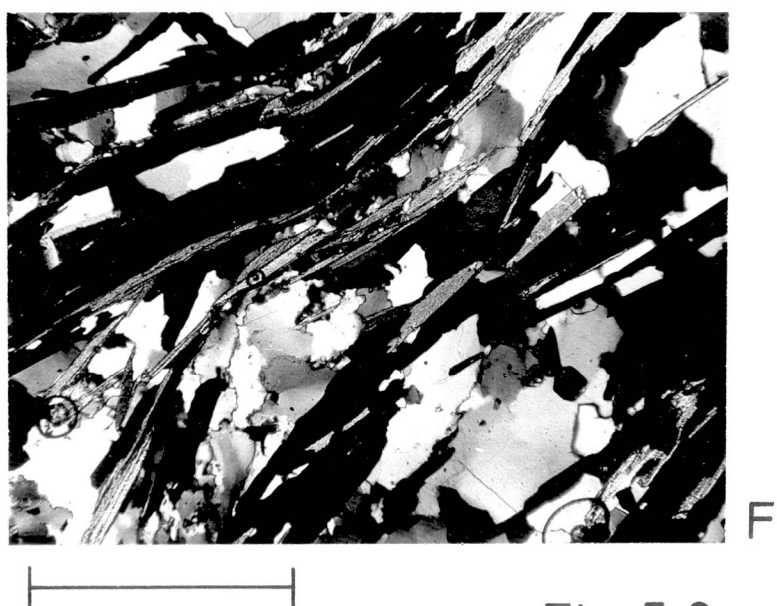
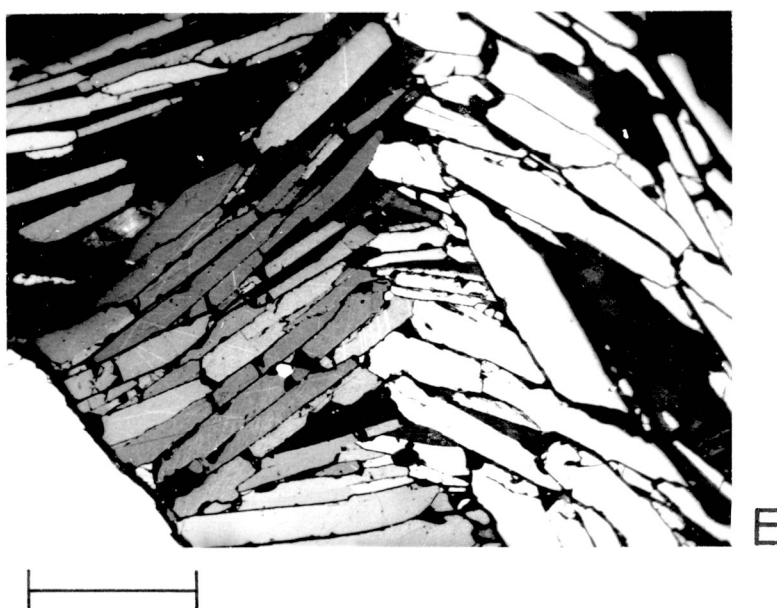
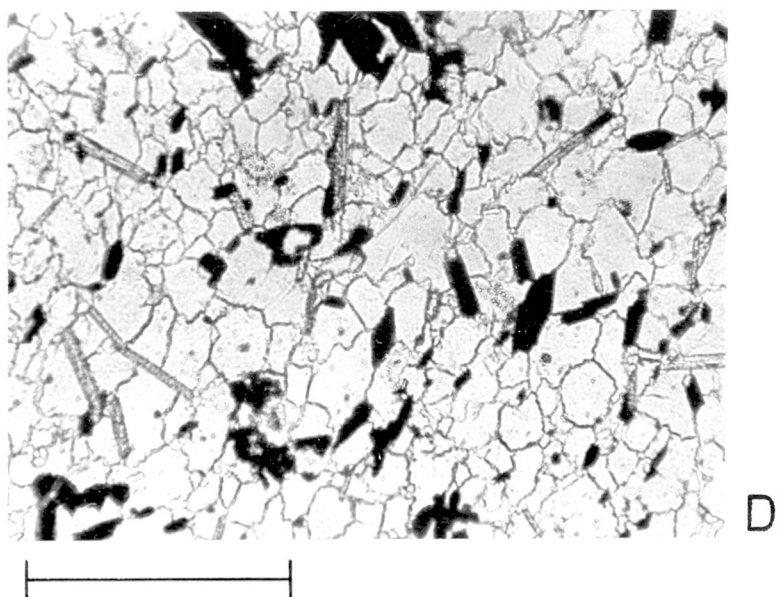


Fig. 5.2

Gangue minerals in the ores of synclines A and D are, in order of decreasing amount, quartz and muscovite and accessory cordierite and tourmaline, probably schorlite.

Muscovite is similar to the hematite in that it has a high degree of preferred orientation of morphological and crystallographic axes (Figs. 5.2C,F). Quartz is usually anhedral and occurs as roughly equidimensional grains (Fig. 5.2A,D) concentrated in lensoid masses or coherent bands. Mechanical grainsize analysis and microscopic examination of quartz indicates the quartz grainsize distribution is similar to that of the hematite although the quartz rarely occurs with a micaceous habit.

Where single quartz grains occur within a mass of micaceous hematite the quartz is occasionally elongated parallel to the long axis of the hematite grains and tends to acquire a tabular to micaceous habit (Fig. 5.3D).

Quartz and cordierite in thin section are characterised by widespread undulose extinction under crossed nicols. Many grains of quartz and cordierite are composed of deformation bands which are narrow regions of slightly different orientations within a single grain and which are indicative of strong deformation. Quartz, being essentially equidimensional, does not exhibit dimensional preferred orientation and lattice preferred orientation appears to be present to a slight degree only. Cordierite is distinguishable from quartz only by its negative biaxial interference figure, quartz being uniaxial positive. Cordierite grains are occasionally observed to be surrounded by reaction rims, apparently of mica or chlorite.

In rare cases hematite grains have deformed plastically without rupture and these occurrences, like those of quartz and cordierite, are marked by undulose extinction and the presence of deformation bands under crossed nicols (Fig. 5.3E). Such internal deformation weakens the inherent strength of the hematite grains which may then readily disintegrate into very fine grains when undergoing beneficiation. If deformation has proceeded too far, in situ brecciation or cataclasis of

FIG. 5.3

THE ORES OF SYNCLINES A AND D

- (A) Cataclasis or in situ brecciation of hematite (white). Note that the brecciation yields tabular to acicular hematite sub-grains. Most of these subgrains report as tailings during spiral beneficiation. Reflected and plane polarised light. Scale bar 200 microns.
- (B) Similar to (A). Note the extremely fine grained sub-grains of hematite. Reflected and plane polarised light. Scale bar 200 microns.
- (C) Cataclasis of hematite. Many of the sub-grains have long dimensions of less than 10 microns. Reflected and plane polarised light. Scale bar 50 microns.

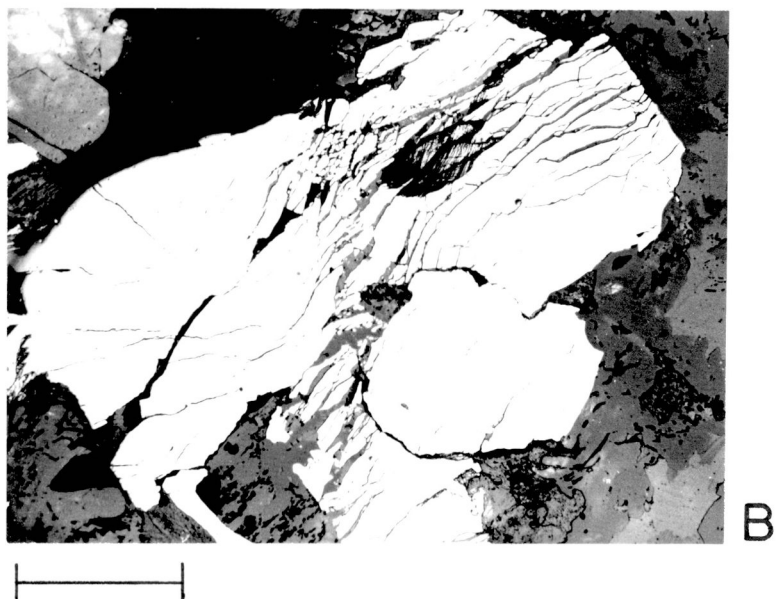
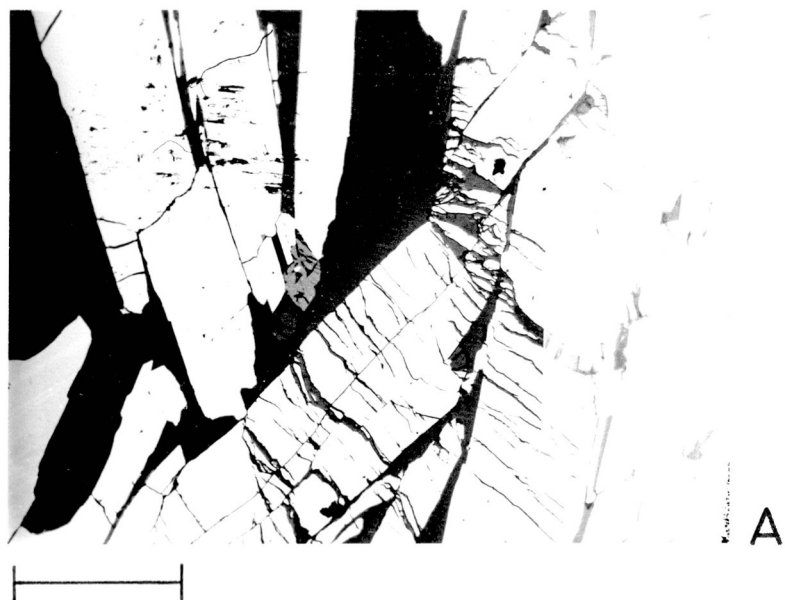
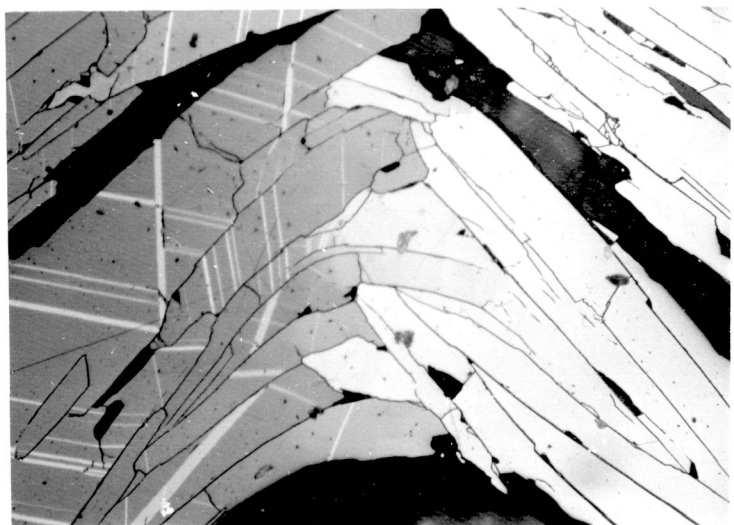


Fig. 5.3

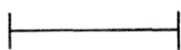
Fig. 5.3 (continued)

THE ORES OF SYNCLINES A AND D

- (D) Conjugate deformation twinning of hematite. Twinning apparently restricted to one side of the fold. Note well developed dimensional : and lattice preferred orientation and slight bending of some hematite grains. Reflected light. Nicols partially crossed. Scale bar 200 microns.
- (E) Internal deformation of hematite. Large hematite grain (top) showing undulatory extinction, slight bending and fractures. Reflected light. Nicols partially crossed. Scale bar 200 microns.
- (F) Separation and partial rotation of hematite grains during deformation. Reflected light. Nicols partially crossed. Scale bar 200 microns.



D



E



F



Fig. 5.3

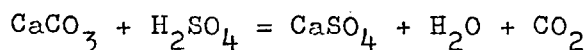
the hematite may similarly result in the formation of many, very fine grained, acicular to tabular grains (Figs. 5.3A,B and C) and this fact has important economic implications since Humphreys spiral beneficiation of hematite is most efficient when the hematite grains are within a certain grainsize range, the lower limit of which is about 100 microns (Rehwal, 1966). The fine grains produced as a result of internal deformation and brecciation are almost certain to be lost to tailings during beneficiation.

Dragfolds are common in the ores of synclines A and D, on both megascopic and microscopic scales. The megascopic dragfolds are emphasised and most easily observed where folding has occurred of alternate bands of pale coloured gangue and dark hematite. In polished section the microscopic folding is seen to have taken place by a separation and partial rotation of adjacent hematite grains and by fracturing of individual hematite grains along the axial plane of the fold (Figs. 5.1E and 5.3F). The location of the dragfolds is apparently associated with the presence of bands or lensoid groups of quartz grains and it is probable that hematite/quartz grain boundary strengths are weaker than those of hematite/hematite and these discontinuities therefore serve as preferred centres for the initiation of deformation and the site of dragfolds. In the cases where quartz is not apparently associated with dragfolds, incipient fractures in hematite are commonly observed parallel to the axial plane of the folds.

Dragfold crests are often square as a result of the resistance of the hematite to bending or fracturing, the square ends being formed by partial rotation of hematite grains.

Microscopic examination of a thin section of diamond drill core from an unknown location has revealed the presence of calcite (Fig. 5.2B) scattered tourmaline (schorlite) and rare garnet (almandine) at depth. Although the location of the diamond drill hole is unknown the presence of dragfolds and the micaceous habit of the hematite suggests an origin in synclines A or D.

The presence of calcite in the ore would be of benefit to the milling operations at Marampa, particularly as the ore becomes harder with increasing depth. At present, the mining operations are still above the hard unleached ore expected at depth and lump ore is crushed in an Aerofall mill which requires that the ore be dry before crushing. The drying is achieved by preheating the ore as it passes into the Aerofall mill and the high temperature necessary is supplied by the burning of high sulphur oil. Release of sulphur oxides produces sulphuric acid which corrodes launders and thickeners and reduces the effectiveness of the thickeners by depressing flocculation and aiding suspension of the material in the thickeners. Calcite in the ore should prove to be beneficial by tending to neutralize the acid by the formation of calcium sulphate according to the equation:



Calcite has not been observed in samples or specimens collected from the present open pit and it is probable that leaching of the ore has removed all carbonate from above the zone of weathering.

The calcite noted in the drill core is polysynthetically twinned, often with conjugate twins. It is anhedral and often includes or surrounds grains of all other constituents of the ore. No undulose extinction was noted in the calcite even where it contained quartz grains with well developed undulose extinction and this fact suggests that the calcite may be a post-tectonic addition to the ore.

The tourmaline and garnet crystals noted in the drill core specimen are both considered to be post-tectonic or at least formed after the main deformation of the orebody. The tourmaline is prismatic, pleochroic, pale olive to colourless and occasionally has pale to dark brown cores which probably consist of a mass of iron oxide inclusions (Heinrich, 1965). The tourmaline is almost invariably fractured normal to the long axis and on the basis of its prismatic habit, colour and host rock it is considered to be schorlite, the iron tourmaline.

The tourmaline crystals do not exhibit the high degree of preferred orientation common to the hematite and muscovite in the same specimen and it therefore appears probable that the tourmaline is either late syntectonic or post-tectonic.

Almandine garnet has been observed but only rarely. It occurs as colourless, isotropic dodecahedral crystals with high relief. A few fractures are usually present. The well defined grain boundaries and the lack of rotational (snowball) structure or other deformation immediately suggests that the garnet is post-tectonic although the fracture may indicate a later mild deformation. The garnet is specified as almandine on the basis of the total isotropism, nature of the host rock and on the lack of black manganese staining which would otherwise indicate spessartine.

Dragfolds, twinning of hematite, undulose extinction of hematite, quartz and cordierite, strongly preferred orientation of hematite and muscovite and the fracturing and brecciation of hematite, are all attributed to permanent deformation and at least two periods of deformation have occurred.

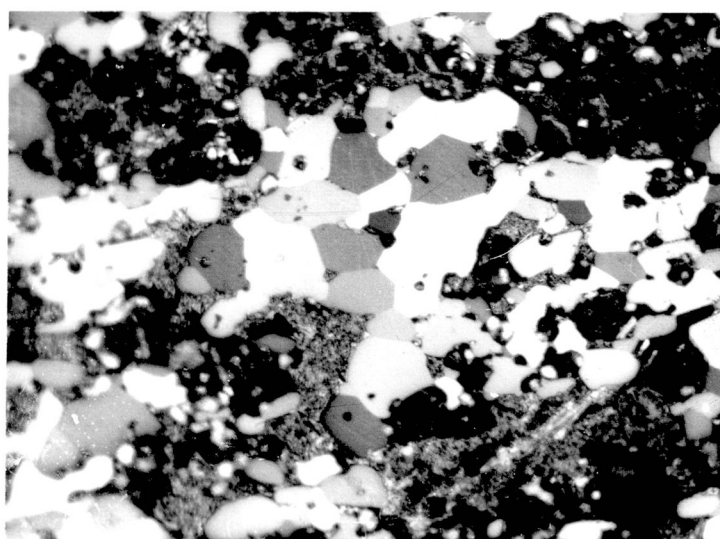
(b) Microscopic Examination of the Ores of Synclines B and C

The ores of synclines B and C are significantly different from those of synclines A and D but are themselves sufficiently similar to be considered and described together. The field evidence, particularly an inspection of the degree of deformation and grainsize analysis also indicates marked similarities between the ore of synclines B and C. Banding of the ores by the alternation of dark, blue-black hematite bands and light-coloured quartzitic gangue shows little evidence of the drag-folding so common to the ores of synclines A and D, and individual hematitic bands can be traced along strike over a much greater distance than similar bands in the other two major ore synclines. Mechanical grainsize analysis of the syncline B ore shows that most of the hematite occurs in the minus 85 B.S. mesh fractions (Table 5.2, compare with Table 5.1)

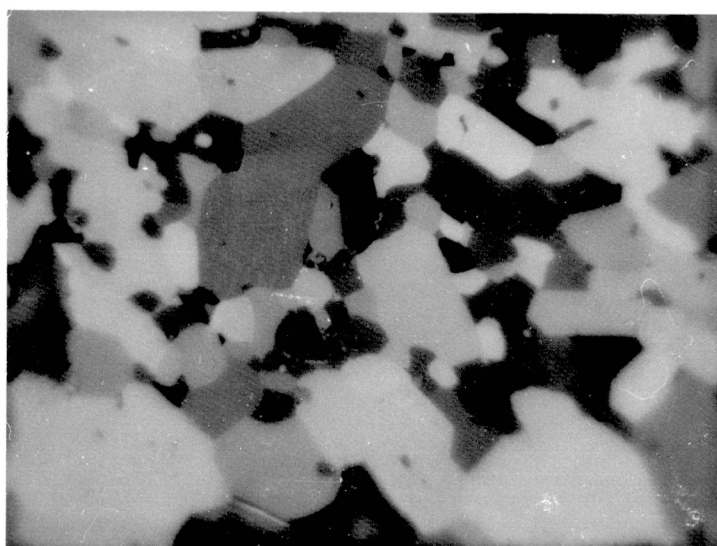
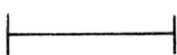
FIG. 5.4

THE ORES OF SYNCLINES B AND C

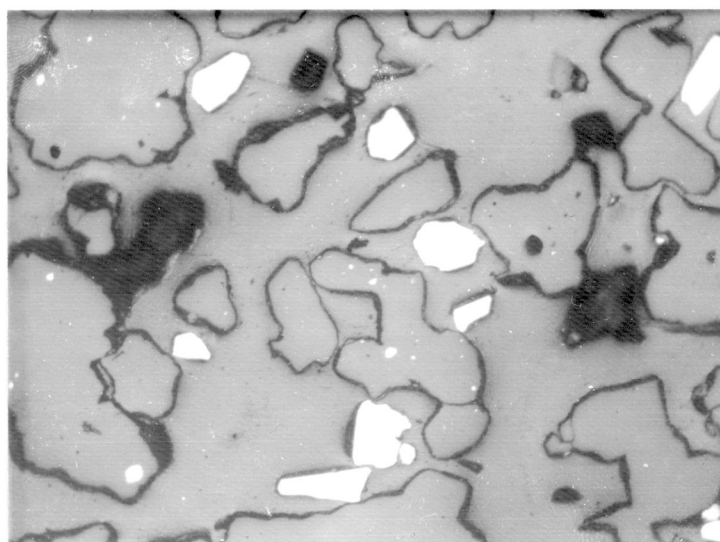
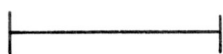
- (A) Annealing textures in hematite (white, grey). Note straight grain boundaries and 120 degree triple points. Reflected light. Nicols crossed. Scale bar 200 microns.
- (B) Similar to (A) but higher magnification. Reflected light. Nicols crossed. Scale bar 50 microns.
- (C) Hematitic quartz "sand", syncline C. Hematite (white), quartz (grey). Hematite generally tabular to equidimensional. Reflected and plane polarised light. Scale bar 200 microns.



A



B



C



Fig. 5.4

Fig. 5.4 (continued)

THE ORES OF SYNCLINES B AND C

- (D) As for (C) but higher magnification.
Note many minute hematite inclusions in quartz. This hematite is totally irrecoverable being too fine for spiral concentration even if liberated from the quartz. Reflected and plane polarised light. Scale bar 50 microns.

- (E) Relatively simple banding of quartz-muscovite schist (white) and hematitic ore (dark). Compare with Fig. 5.1A.

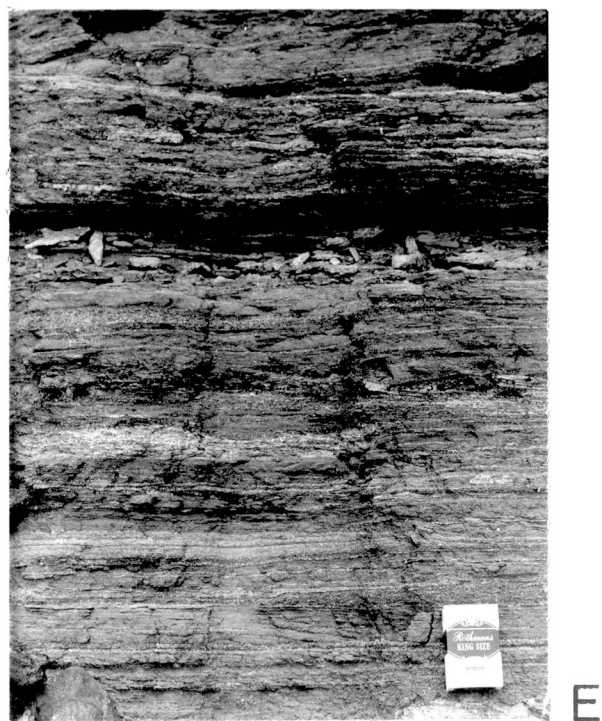
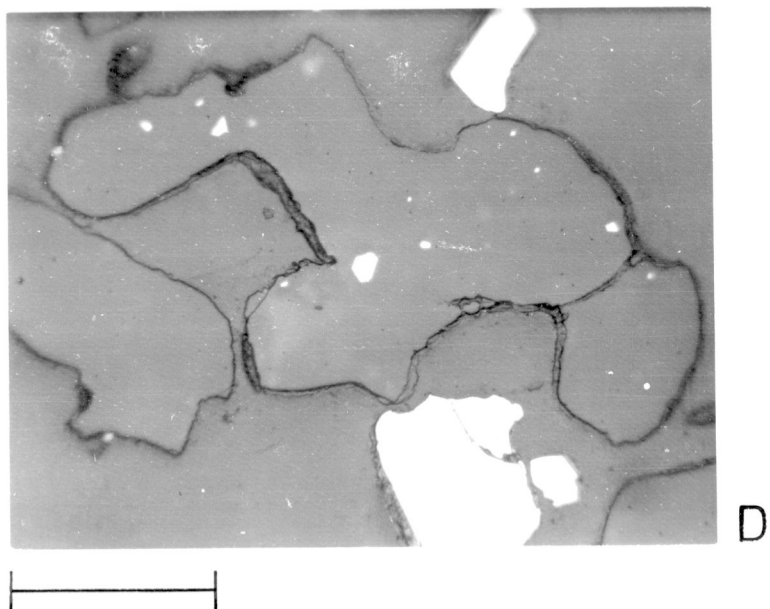


Fig. 5.4

Fraction		Iron Distribution (per cent)
B.S. Mesh	Microns (μ)	
+ 44	+350	10.9
- 44 + 60	-350 +252	11.8
- 60 + 85	-252 +177	21.6
- 85 +120	-177 +125	24.3
-120 +170	-125 + 88	16.3
-170 +300	- 88 + 53	10.4
-300	- 53	4.7

Table 5.2 Grainsize distribution of iron in syncline B

Microscopic examination of polished sections of ore from synclines B and C supports the evidence of the mechanical grainsize analysis. The hematite is fine-grained, euhedral to subhedral and equidimensional but with occasional crude dimensional and lattice preferred orientation of the more elongate hematite grains (Figs 5.5 A and D).

Annealing textures are common among the hematite grains with 120 degree triple points very conspicuous in most ore specimens from synclines B and C (Figs 5.4 A and B). The annealing texture is only visible under crossed nicols such is the intimate contact between adjacent hematite grains. What appear to be large hematite grains are seen, under crossed nicols, to be composed of a number of smaller sub-grains exhibiting annealed texture and this occurrence of hematite, if widespread, will affect the milling characteristics of the ore, particularly with regard to the effective liberation of hematite passing on to the Humphreys spirals. Large composite grains tend to behave as gangue when undergoing spiral beneficiation and, in addition, the composite grains often contain included quartz grains. If the annealing texture causes considerable amounts of large composite grains to remain intact

FIG. 5.5

THE ORES OF SYNCLINES B AND C

- (A) Equidimensional to slightly elongate hematite. Slight dimensional preferred orientation. Compare with Fig 5.1B. Reflected and plane polarised light. Scale bar 200 microns.
- (B) Banding of hematite (white) and manganese oxides (grey, centre). Note that the annealed texture of the hematite is not readily observed with plane polarised light. Reflected and plane polarised light. Scale bar 200 microns.
- (C) Conjugate deformation twinning in hematite. Same section as (A) but with nicols crossed. Reflected light. Nicols crossed. Scale bar 200 microns.

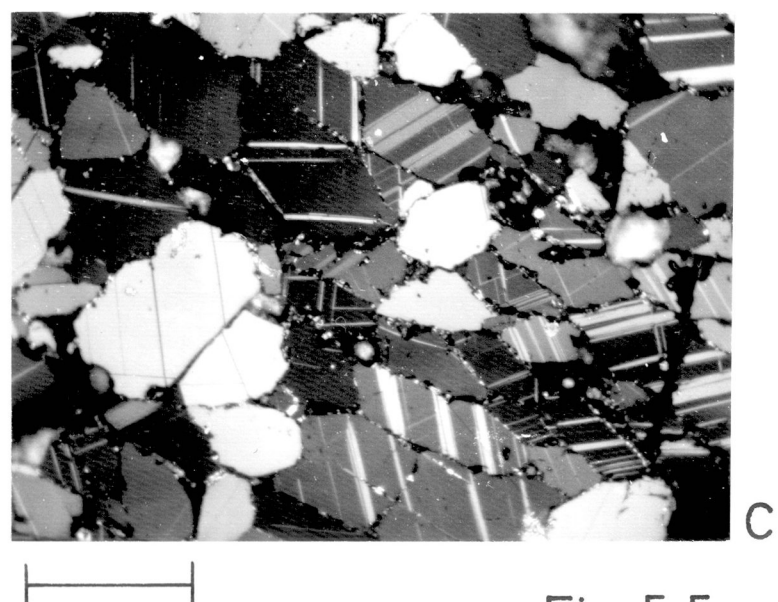
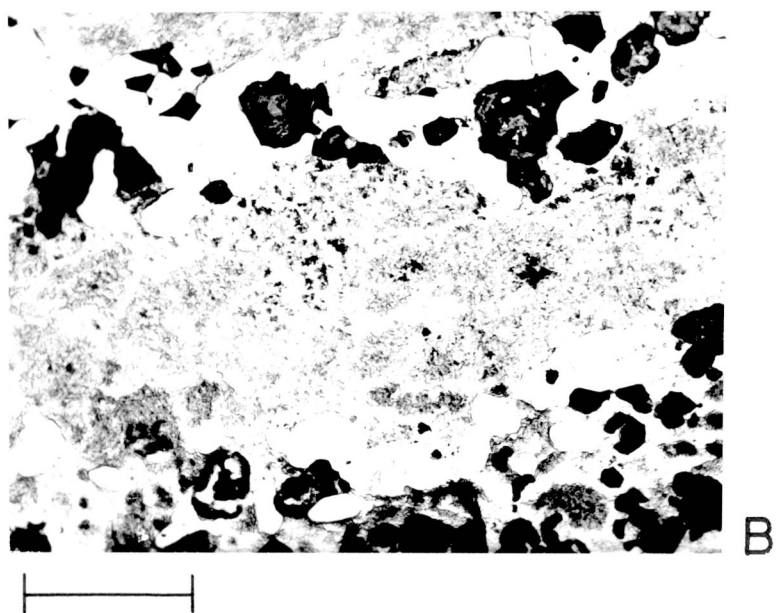
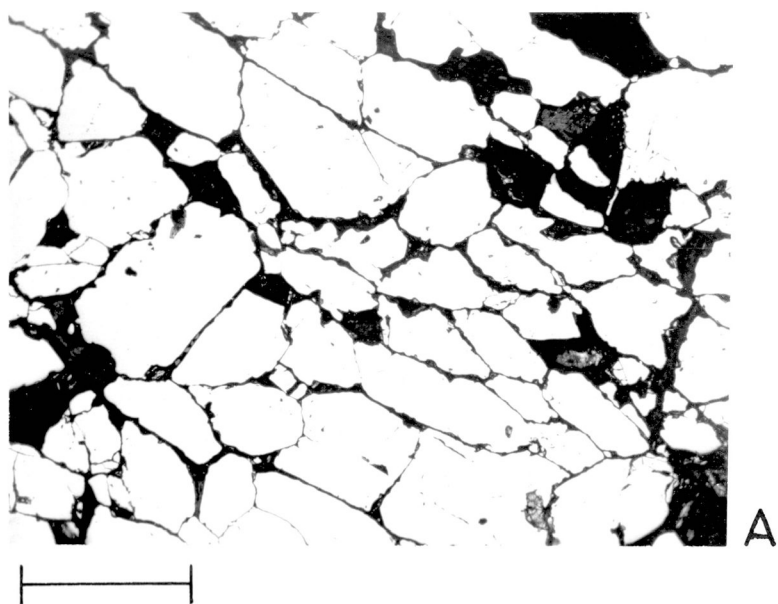


Fig. 5.5

Fig. 5.5 (continued)

THE ORES OF SYNCLINES B AND C

- (D) Low grade ore. Dimensional preferred orientation of tabular to equidimensional hematite (black). Transmitted and plane polarised light. Scale bar 1.00 mm.
- (E) Equidimensional quartz (white to grey) with undulose extinction. Equidimensional hematite (black). Two generations of muscovite;
(i) small grains with well developed dimensional and lattice preferred orientation,
(ii) large grain oriented normal to (i).
Transmitted light. Crossed nicols.
Scale bar 1.00 mm.
- (F) Equidimensional hematite (black) and quartz (white, grey). Muscovite (grey) with well developed dimensional and lattice preferred orientation. Transmitted light.
Nicols crossed. Scale bar 1.00 mm.

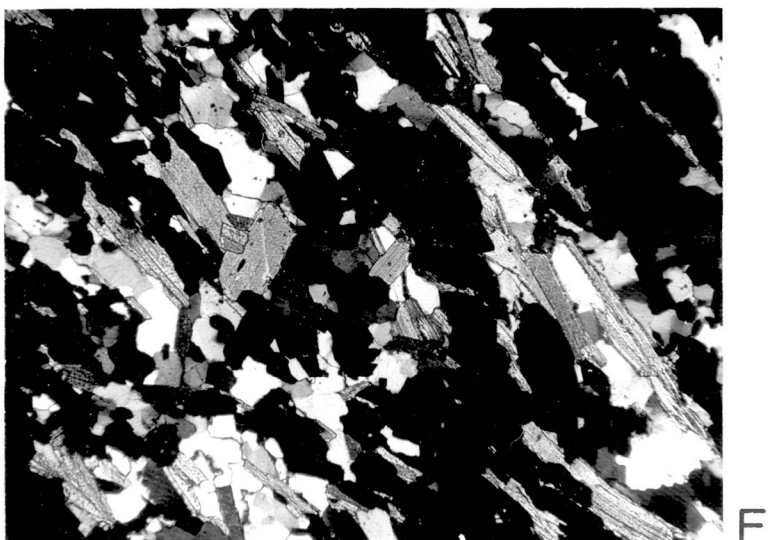
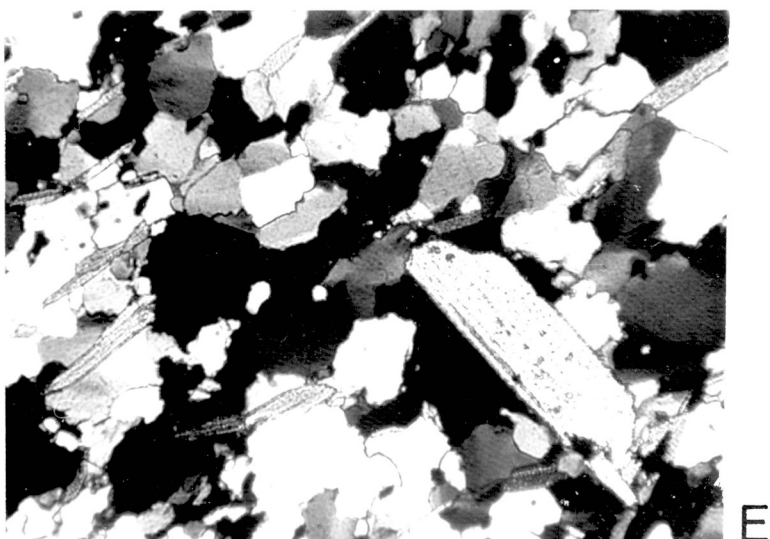
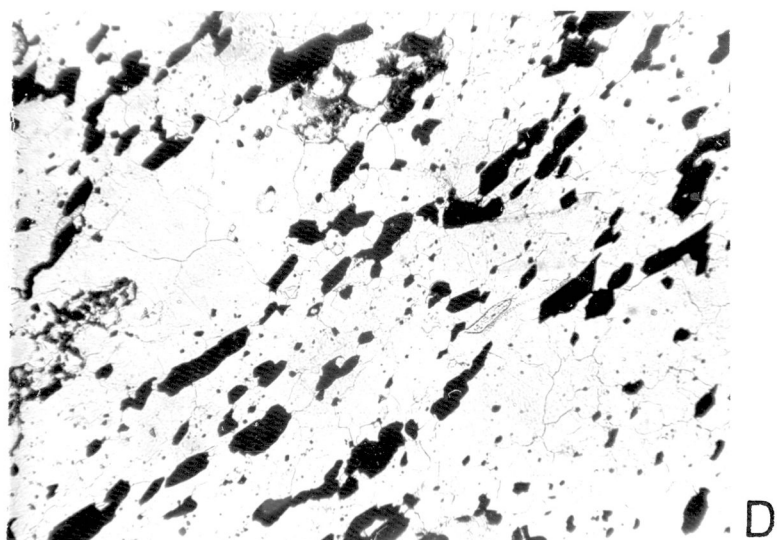


Fig. 5.5

after the secondary hammer mill grainsize reduction, then serious reductions may occur in concentrate grade and/or iron recovery. The strength of these annealing texture grain boundaries appears to be enhanced if quartz or other gangue minerals are absent.

Hematite does not occur in the micaceous habit common in the ores of synclines A and D although slight elongation of hematite grains does occasionally occur. This elongation is accompanied by a slight dimensional preferred orientation but lattice preferred orientation is recognised only with difficulty.

Microscopic foliation in synclines B and C is most obvious in the presence of muscovite (Fig. 5.5F) which almost invariably has well developed dimensional and lattice preferred orientation. The alternate banding of dark hematitic and light gangue bands (Fig. 5.4E) which is obvious in hand specimens, is not so obvious in thin or polished section with quartz not apparently restricted to well defined bands or lenses as in synclines A and D but occurring mainly as scattered single grains or irregular masses of grains.

Quartz always occurs as subhedral to anhedral equidimensional grains (Fig. 5.5E) with no preferred orientation, either dimensional or lattice, and often contains many minute (less than 10 microns diameter) inclusions of euhedral hematite. This occurrence of hematite is very noticeable in pale, blue-grey, completely friable, hematitic quartz "sand" (Figs. 5.4 C and D) which is seen to occur in small (usually less than 10 feet) lenses in syncline C. A brief point-count analysis of this material yielded the following figures:

quartz grains with hematite inclusions	68 per cent
quartz grains without hematite inclusions	24 " "
discrete hematite grains	8 " "

Most of the hematite inclusions are less than 10 microns in diameter and many of these are estimated to be less than 1 micron in diameter. It is also estimated that those quartz grains with hematite inclusions contain an average of about five inclusions per grain, the inclusions being randomly distributed with no evidence of zoning or other orderly arrangement

of hematite within the quartz. Although these inclusions are minute, the great number of them introduces errors, possibly serious, in the reporting of assay grades which, in ore of this type, always over-values the recoverable iron content of the ore. The inclusions of hematite are not recovered by the spiral beneficiation of the ore because the quartz reports as tailings and even if the ore was crushed or ground sufficiently to liberate the hematite inclusions, the very fine grainsize of this hematite would ensure its loss as tailings.

C. MICROSCOPIC EXAMINATION OF MARAMPA TAILINGS

Polished grain mounts of Marampa tailings have been microscopically examined under reflected light and several features of the hematite recovered as tailings warrant discussion in the light of milling practice.

Most of the hematite occurring in the tailings (Fig. 5.6C) occurs as extremely fine grains which are certainly below the minimum grainsize requirement for Humphreys spiral beneficiation; i.e. less than 100 microns, although grains falling within the optimum grainsize range are occasionally observed. The fine hematite grains are usually acicular, wedge-shaped or tabular which suggests that cataclasis or internal deformation of the hematite has resulted in the release of fine grains by breakdown of initially larger grains during crushing by Aerofall or hammer mills. As the ore becomes progressively harder with increasing depth of mining, a greater proportion of ore will require crushing to liberate the hematite and this may result in increased loss of hematite during milling.

Large hematite grains recovered from tailings are usually found to consist of a number of smaller sub-grains occurring together as a coherent group as a result of the "annealing" of the hematite (Fig. 5.6D). This annealing texture is only present in the ores of synclines B and C. Excessive amounts of such ore may be difficult to beneficiate profitably since it tends to be resistant to liberation of the hematite because of the inherent strength of the hematite/hematite grain boundaries in ores with the annealed texture.

FIG. 5.6

CONCENTRATE, TAILINGS AND WASTE.

- (A) Ferromax (>68 per cent Fe. Tabular to micaceous hematite (white) with grain of limonite (centre). Note the general uniformity of the hematite grainsize. Reflected and plane polarised light. Scale bar 200 microns.
- (B) Concentrate, Hematite (white) and quartz (grey). Note, much larger range of hematite grainsize. Reflected and plane polarised light. Scale bar 200 microns.
- (C) Tailings, Hematite (white), quartz (grey) and muscovite (grey, acicular). Note presence of many fine grained and acicular hematite grains. Reflected and plane polarised light. Scale bar 200 microns.

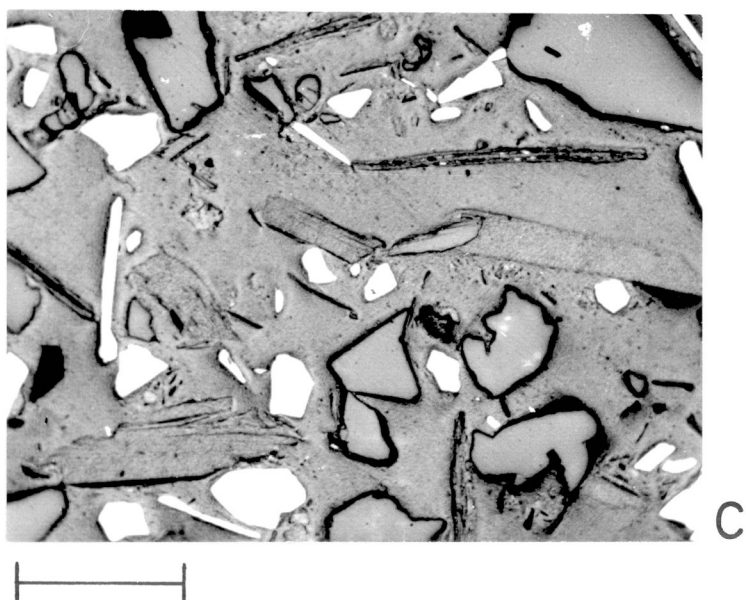
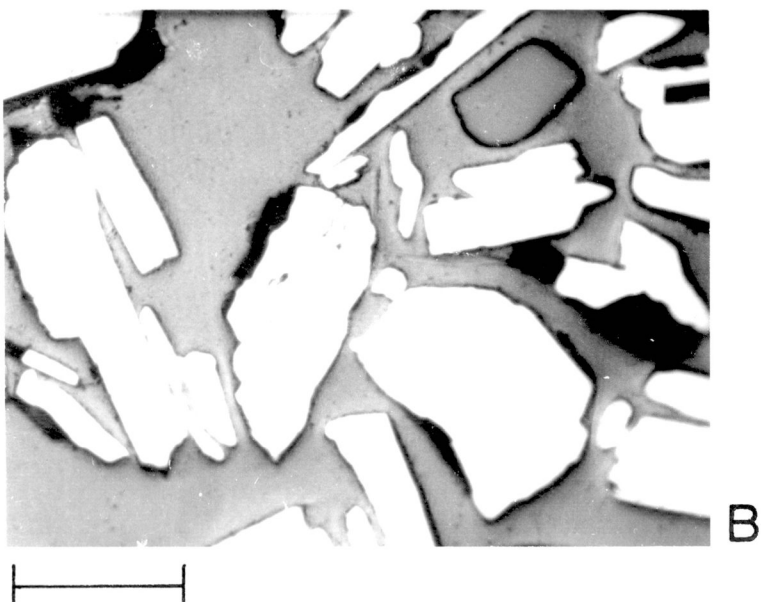
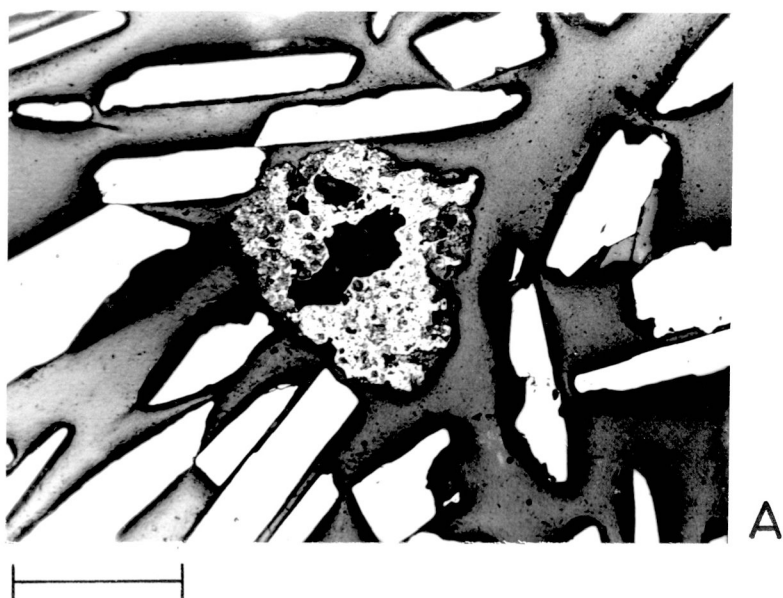


Fig. 5.6

Fig. 5.6 (continued)

CONCENTRATE, TAILINGS AND WASTE

- (D) Tailings. Note large composite grain (upper left) of annealed hematite and quartz. Reflected and plane polarised light. Scale bar 200 microns.

- (E) Waste from B/C anticline. Quartz-muscovite schist. Mainly muscovite with well developed dimensional and lattice preferred orientation. Note larger grains which may be second generation muscovite. Transmitted and plane polarised light. Scale bar 1.00 mm.

- (F) Waste. Quartz-muscovite-hematite schist. Hematite (black), quartz (white) and muscovite (grey). Transmitted and plane polarised light. Scale bar 1.00 mm.

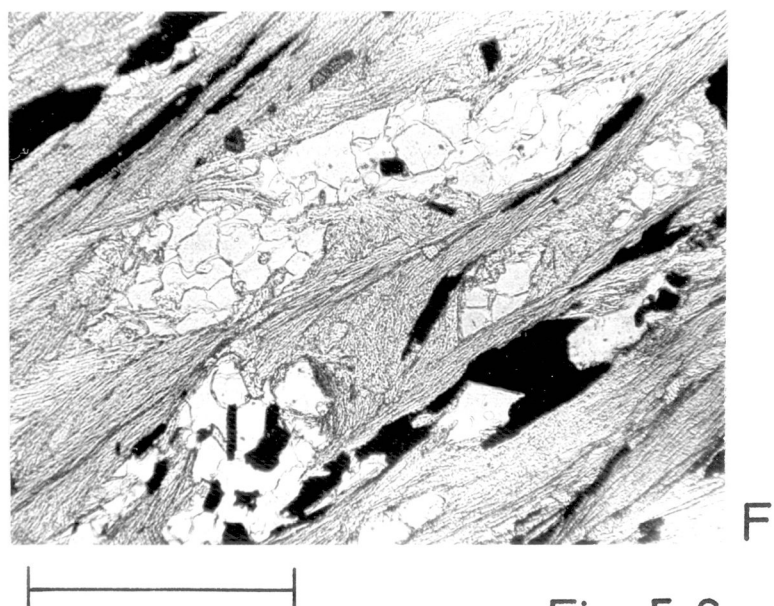
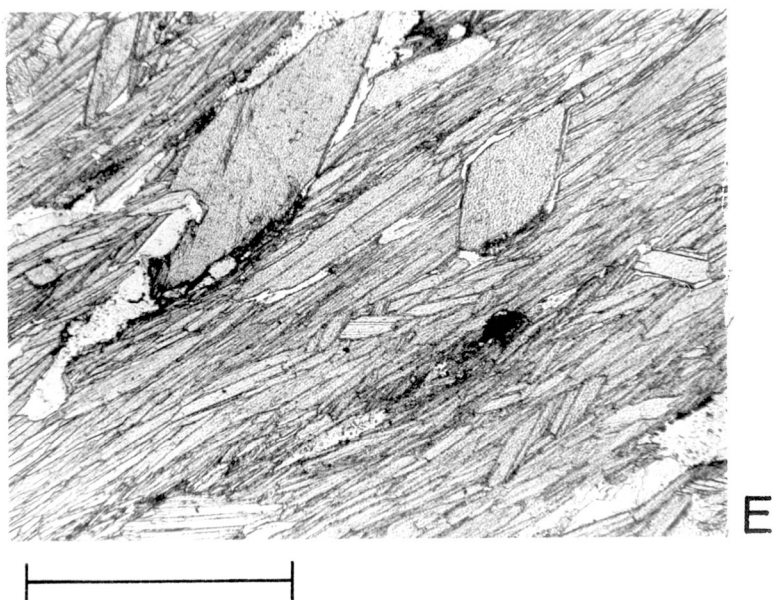
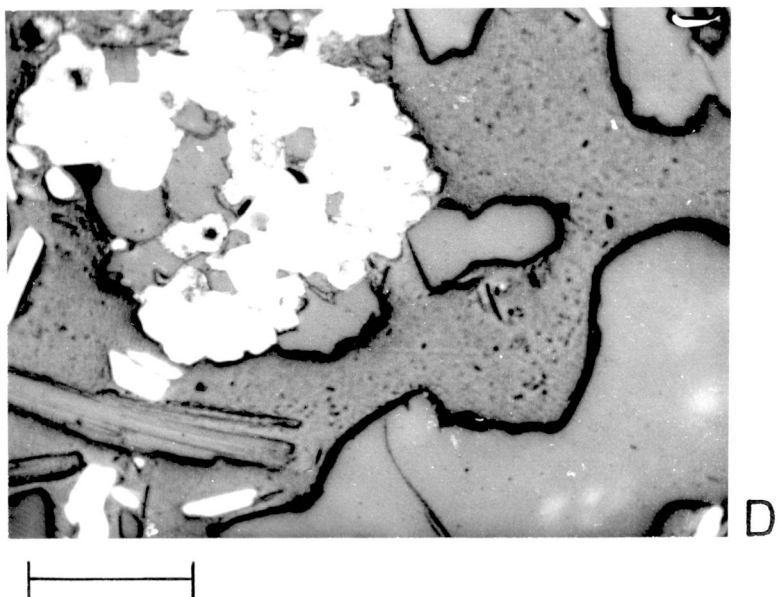


Fig. 5.6

Marampa tailings consist essentially of the following minerals (in order of decreasing amounts); quartz, hematite, muscovite and others including manganese oxides and limonite.

The quartz commonly contains minute (less than 10 microns diameter) euhedral hematite grains and the muscovite occasionally contains minute hematite inclusions arranged in an orderly fashion along or parallel to the cleavage traces. Hematite in either of these two forms is invariably lost during beneficiation and all iron assays of ore should be accepted only as an absolute maximum indication of the iron content.

Many hematite grains which fall within the optimum size range for spiral beneficiation are observed to be acicular or wedge-shaped and it appears probable that the shape of the hematite grains, as well as size, is an important factor in effective beneficiation of the Marampa ores. Acicular or wedge-shaped grains of hematite may be formed by shattering of initially larger grains during primary and secondary crushing, by cataclasis of hematite or by disruption of hematite along boundaries initiated by internal deformation which has taken place without cataclasis.

The most common grainsize of hematite occurring in the Marampa tailings is in the range 30 to 60 microns. Irregular and shattered grain boundaries are indicative of the crushing and fracturing during size reduction.

Composite hematite grains are not uncommon in the tailings, but as noted earlier, they are restricted to the annealed ores of synclines B and C. Ores from synclines A and D, which are dominantly schistose with micaceous to tabular hematite, are readily disrupted and the hematite is liberated without intensive crushing or grinding.

D. MICROSCOPIC EXAMINATION OF MARAMPA CONCENTRATE

Microscopic examination of various grainsize fractions of Marampa iron ore concentrates (Figs 5.6 A and B) confirms that the coarse fractions consist mainly of micaceous to tabular

hematite such as occurs in synclines A and D, and composite grains of annealed hematite, with or without included quartz and muscovite, such as occurs in synclines B and C. The finer fractions are composed mainly of roughly equidimensional hematite grains characteristic of synclines B and C together with fine acicular grains from synclines A and D and from the crushing of equidimensional grains.

E. ECONOMIC IMPLICATIONS OF THE MICROSCOPIC EXAMINATION
OF THE MARAMPA MATERIALS

The iron ore deposit at Marmapa is a hematite - quartz - muscovite schist with subordinate manganese oxides (lithiophorite and others) cordierite, tourmaline (schorlite), garnet (almandine) and very rare magnetite. The hematite occurs in two distinct physical forms;

(a) Coarse grained micaceous to tabular.

(b) Fine grained equidimensional.

The ores corresponding to (a) and (b) occur in synclines A and D and in synclines B and C respectively and suggests that two distinct ore zones were originally deposited rather than a single ore zone as advocated by Kennedy and Thompson. The two types of ore are sufficiently different to suggest that Humphreys spiral beneficiation of the ore may be best carried out by separate milling of the two types although serious objections to this could be raised on the grounds that the mining method is not sufficiently flexible to deal with such a situation. Milling experience has shown that the coarse micaceous ore tends to settle and bank on the spirals whereas the finer grained ore is more easily treated but tends to yield a higher loss of hematite to tailings.

Hematite inclusions in quartz and muscovite are relatively widespread and such hematite is invariably lost to tailings. It is totally irrecoverable and mine iron assay grades should only be taken as an estimate of the maximum iron content of the ore and should not be used indiscriminately to predict the recoverable amount of iron.

In situ brecciation and internal deformation of hematite grains tends to yield very fine grained hematite during the pre-spiral treatment stages and those areas of the orebody, particularly in synclines A and D, in which complex deformation is known to have occurred, may be expected to yield a disproportionately high amount of fine, acicular hematite which is usually irrecoverable by spiral beneficiation.

Twinning of hematite, which is common in all four ore synclines (Fig. 5.3D and 5.5 C), tends to act as a reinforcing agent of the grain, and twinned grains are probably more resistant to disintegration during the crushing stages than the untwinned grains. The twinning is related to deformational stress and appears to be associated with the initiation of dragfolds particularly in synclines A and D and is therefore of secondary origin. Twinning of hematite in synclines B and C may be deformational in origin but may also be related to grain growth during recrystallization which has resulted in the widespread annealing texture, in which case the twinning may be primary.

The presence of schorlite and almandine, the iron-bearing tourmaline and garnet respectively, adds to the overvaluation of the iron content of the ore. Although they are apparently not present in large amounts they both contain appreciable iron, some 16 per cent and 34 per cent Fe for schorlite and almandine respectively, and where the presence of these two minerals becomes significant the iron assay of the ore may be a misleading estimate of the hematite content with consequent errors in ore reserve estimation and grade control.

Manganese minerals occur mainly on the margins of synclines B and C and are considered to be genetically related to an original depositional feature which, if correct, supports the hypothesis of two original ore zones since, if only a single ore zone was originally deposited, manganese minerals should also be expected to occur along the margins of synclines A and D. X-ray powder crystallography indicates that the manganese material is mainly lithiophorite, a hydrated lithian

manganese oxide with approximate composition given by the formula $(\text{Li,Al})\text{MnO}_3 \cdot \text{H}_2\text{O}$. The lithiophorite is extremely fine grained and cannot be effectively examined microscopically. It is a secondary mineral formed probably by the oxidation of a primary manganese silicate or carbonate, and it occurs mainly in discrete bands alternating with bands of hematite (Fig. 5.5B). The fine grain size of the lithiophorite ensures that it reports mainly as tailings and only the fine fraction (-300 mesh) of the hematite concentrate is relatively enriched in manganese. Grain size analysis of ore from synclines B and C shows that the manganese content of the screen fractions is lowest for the range -44 +85 B.S. mesh and progressively increases with decreasing grain size. The manganese content of syncline A ore averages less than 0.1 per cent Mn with 0.03 per cent Mn being a typical value. The mean manganese content of the syncline B ore is, however, often greater than 0.5 per cent Mn and is usually greater than 3.26 per cent Mn within about 50 feet (Fig. 5.5B) of the syncline ore/waste contact, 3.26 per cent Mn being the upper limit of detection of the Marampa X-ray fluorescence spectrometer. Examination of polished sections reveals that much of the ore on the margins of syncline B exceeds 10 per cent Mn and may occasionally be as high as 20 per cent Mn. At present the manganese content of the concentrate shipments averages about 0.16 per cent Mn and since this amount gives no cause for concern, a subdivision of ore types on the basis of manganese content is probably not warranted since the fine grain size of the manganese minerals ensures that they usually report as tailings. The distribution and form of the manganese minerals at depth below the present pit is completely unknown.

The annealing texture common to the ores of synclines B and C has two possibly detrimental effects on the beneficiation of the Marampa ore.

(a) The individual hematite grains are not readily liberated when the annealing texture is present to any great extent and the "cementing" effect of the annealing may yield composite hematite grains of a sufficiently large size to

behave as gangue and thus eventually report as tailings. This loss of hematite does not necessarily decrease the concentrate grade but would probably result in a decrease in iron recovery.

(b) Gangue minerals, particularly quartz, are often included among the composite annealed hematite and passage of this material to the concentrate would tend to reduce the concentrate grade without affecting the iron recovery.

These two effects of the annealed texture have not been quantitatively assessed and may be insignificant but they have been observed to take place in full scale milling of the ore and thus cannot be dismissed.

The presence of calcite in the ore at depth may eventually prove to be beneficial by neutralising the effects of the sulphuric acid produced during Aerofall milling. Calcite has not been reported or observed from the present open pit and may, in fact, not appear in the mill feed until mining has progressed to a much greater depth. Since the calcite was observed only in one polished section of diamond drill-core from an unknown locality and depth it would be incorrect to comment further on the distribution of calcite in the Marampa orebody until more detailed drilling has been carried out.

The main gangue minerals, quartz and muscovite, present in the Marampa ores are both particularly suitable for separation from hematite on Humphreys spirals. The quartz and muscovite have a specific gravity of about 2.6 compared with a value of 5.2 for hematite. In addition, the quartz is roughly equidimensional and the muscovite is, of course, micaceous. These physical characteristics are theoretically ideal and a very efficient separation of hematite and gangue should take place in the Marampa mill. Examination of mill analysis sheets however, reveals that on occasions the concentrate grade falls below 60 per cent Fe and is often below 62 per cent Fe. This latter figure corresponds with a hematite and gangue content of about 89 per cent and 11 per cent respectively. Given that low grade concentrates are almost invariably associated with low grades in the midsize ranges (-60 +170 B.S. mesh) because

of the poor separation of hematite and quartz in this size range, it is difficult to account for concentrate grades of less than 62 per cent Fe being caused by factors other than inefficient milling practice or ignorance of the quantitative milling behaviour of mill feeds with different shape characteristics. Low grade mill feeds alone are not sufficient to produce such low grade concentrates although the unfortunate combination of low feed grade, large amounts of hematite inclusions in quartz and large proportions of coarse micaceous hematite could readily result in low grade concentrates.

Electron-probe microanalysis (E.P.M.A.) of hematite from the various ore synclines indicates that the hematite grains are essentially pure Fe_2O_3 with manganese and titanium being present in negligible amounts, certainly less than one per cent. E.P.M.A. of the manganese oxide indicated a manganese content of about 39 per cent which supports the conclusion that the manganese is mainly in the form of lithiophorite (38 per cent Mn).

X-ray diffraction and microscopic studies indicate that the anticlinal bodies of waste between the synclinal, orebodies are composed almost entirely of quartz and muscovite (Fig. 5.6E) although hematite (Fig. 5.6F) is not uncommon and fuchsite (chrome variety of muscovite) and minnesotaite (iron silicate) were detected in very minor amounts.

CHAPTER VI

MINE SAMPLING AND COMPARISON OF SAMPLING TECHNIQUES

(A) INTRODUCTION

The value of the accuracy and reliability of sampling methods are often poorly appreciated in the mining industry when it is considered that large capital investments may be made on the basis of an analysis of ore grades, the accuracy and reliability of which are themselves dependant upon the sampling method employed. Too often, when ore reserves are estimated, the applicability and reliability of the sampling method is not questioned and ore reserves are quoted as "X tons at a grade of Y per cent", with no confidence limits quoted for either the tonnage or the grade. As pointed out by Gy (1968), "large financial risks may arise from sampling errors", but this fact is often overlooked by finance groups and mine operators.

This chapter is concerned with the investigation of some sampling methods employed at Marampa and in particular, the reliability of the Marampa sampling methods are assessed both quantitatively, by analysis of assay values, and qualitatively.

(B) SAMPLING TECHNIQUES EMPLOYED AT MARAMPA

The Planning Department at the mine is responsible for the collection of samples and for the plotting, processing and interpretation of the sample assay data. The preparation and assaying of the samples is undertaken by the Mines Laboratory, an adjunct to the Planning Department.

Three ore sampling techniques are employed at the mine;

- (a) Groove (channel) sampling.
- (b) Augerdrill sampling.
- (c) Face shovel sampling.

(a) Groove (Channel) Sampling

Ore reserve estimation and grade control at Marampa are based on analyses of assay values of what are locally known as groove samples and what are in fact crude forms of channel samples. The samples are collected along sample lines 200 feet apart, The sampling technique is as follows;

Sample lines at right angles to the average strike of the ore horizons (about N 35°E) are cleared to bedrock and swept clean. A 100 feet tape is laid along the cleared line, ten feet sections are marked off and polythene sample bags laid alongside each section. A narrow (less than two inches wide) groove is pickaxed out along the line to a depth ranging from $\frac{1}{2}$ to 2 inches depending upon the hardness of the ore. The loosened material from each ten feet section is then scraped by hand to the centre of the section, placed in a headpan and hammered to reduce the size of the larger particles. It is then perfunctorily mixed by hand, a rough cross marked on the sample and alternate quarters roughly scooped out and discarded. This process of volume and particle size reduction is repeated until about 1 to 2 kilogrammes of sample remain to be placed in the sample bag for eventual assaying.

The advantages of this method of sampling are;

(a) Low cost. The unit cost with this sampling method is about 2 to 4 shillings and may be as low as one shilling inclusive of labour, transport and supervision.

(b) Ease of collection. The rate of sample collection depends mainly upon the amount of overburden to be cleared from the sample lines and collection of fifty samples per day is readily achieved, i.e. sampling of 500 feet across strike.

(c) Under ideal conditions of soft, friable and vertically dipping ore the technique is expected to give a good estimate of ore grade.

Under certain circumstances however, it is envisaged that this method of sampling may lead to discrepancies in the

valuation of the ore by over-representation of soft friable ore and hard fissile ore, and by under-representation of hard non-fissile ore, the overall effect depending upon the relative proportions of the different ore types. As a rule, the soft friable ore contains a greater proportion of quartz-muscovite schist than the hard ore and is consequently of lower grade. The inclusion of disproportionate amounts of the various ore types in the samples is due to the natural tendency of the samplers to collect too much soft or friable material, this being most easily loosened, and too much hard fissile material which tends to break into particles with greater length than the width of the sampling groove. Hard non-fissile material is often only scratched and unless shattered is always under-represented.

The effects of contamination on the final assay values are unknown but may, at times, be considerable and variable since only cursory attention is paid to the cleaning of equipment and the active avoidance of contamination.

The main objection to this method of sampling is that, where dips are sub-vertical or variable, it gives an unrealistic indication of ore grades at depth, i.e. to the foot of the bench, and in the extreme case of horizontal bedding gives absolutely no indication of the underlying grade and, consequently, the overall grade. This is because the across-strike grade variations are much greater than the along-strike variations and the sampling of horizontal strata in which is usually an across-strike direction becomes equivalent to along-strike sampling with a consequent drastic reduction in grade variance.

The Marampa orebody is structurally complex, only the broad structural outlines being known and no detailed geological maps are available nor is any routine mine mapping being undertaken, so that surface sampling data cannot be projected down-dip with any certainty in order to give a more realistic indication of the absolute ore grade.

This sampling method is suited to the sampling of soft, friable, non-fissile material with constant and, ideally,

vertical dips.

(b) Augerdrill Sampling

Augerdrill sampling for ore reserve calculation purposes was recently introduced at Marampa in order to combat the masking effects of sub-horizontal strata on the underlying ore. The drilling is carried out on a rectangular grid, the dimensions of which vary according to the amount of detail required, but are usually 100 by 50 feet or 200 by 50 feet, the longer dimension being measured along strike. The holes are drilled to about 33 feet to enable a single mining level to be completely sampled but drilling may continue to about 63 feet to enable sampling of two mining levels.

Seven samples are collected from each 33 feet hole. The first sample represents an interval of about three feet and the remaining six samples each represent an interval of five feet corresponding to the length of an auger drillrod. The drilling is done by a track-mounted, diesel powered, drill rig using six-inch diameter auger rods. The drilling rate is dependant upon the hardness of the ore but operating in friable ore a 33 feet hole may be drilled and sampled within one hour.

Drilling is the only efficient way in which subsurface grade information may be collected ahead of mining operations and the information gained is invaluable for purposes of grade control. It is doubtful however whether the positions and grades of individual samples within each hole should be accepted for detailed grade control planning since accurate sampling is not possible. The interval represented by the first auger rod in each hole is dependant upon the height of the rig above the position of the hole and this may vary considerably due to local topographic irregularities. Contamination in auger drillholes is almost unavoidable particularly below the water table where recovery is poor and contamination becomes extreme. Efforts to delineate blocks of ore within the height of a single mining level by use of five-foot auger samples are considered to give misleading information and a more practical approach would be to bulk the samples to produce one sample for

each hole, to close the drilling grid to a 50 by 25 feet pattern and to abandon each hole if the water table is reached before the 33 feet level. Forward planning would be given a systematic base if holes were continued to depths covering two or three mining levels, i.e. to 63 to 93 feet, during the latter parts of the dry season, although water table variations may still interfere with such drilling.

Auger drillhole assay data has been used to examine vertical variations in ore grade (Chapter VII.)

(c) Face Shovel Sampling

Twice daily face shovel samples collected at six a.m. and six p.m. gave limited short term grade information concerning ore mined during the previous shift and ore to be mined during the forthcoming shift.

Various sampling techniques in addition to those employed by the Marampa Planning Department were employed during the course of this investigation in order to test the effectiveness of the routine mine sampling and to determine if more efficient methods could be introduced. The most efficient sampling method is that which gives the best estimate of mill feed grade, taking into account ease and cost of sampling, i.e. it is most cost-effective. The additional sampling techniques included;

(a) Channel sampling.

(b) "Posthole" sampling, both random and systematic.

(d) Channel Sampling

This method was almost identical to the groove sampling technique of the Planning Department except that the channels were of constant width, about three inches, and the depth of the channel was constant for each ten feet section, generally being in the range $\frac{1}{2}$ to 1 inch. The channels were cut with hammer and chisel and every precaution was taken to guard against contamination and to ensure that the width and depth were kept constant for each section.

These channel samples were used as a standard against which other sampling methods could be compared by comparison of the assay values, the value as a standard arising from the relative lack of contamination and the accuracy with which they were collected. In addition these channel samples were systematically reduced in volume by coning and quartering and by riffing, this not being the case with the groove samples and augerdrill samples which are only crudely reduced in volume.

To enable comparison of channel and groove samples the two types of samples were collected as close together as possible and although a potential sampling error is introduced by the slight lateral displacement (usually less than 3 inches) of the two types of samples, inspection of the lithological variations between the groove and channel samples indicates that the errors incurred would probably be no greater than those incurred had the groove samples been collected from within the earlier formed channels.

(e) Posthole Sampling

Posthole samples consist of rectangular blocks of ore with in situ areal dimensions of 12 inches (across strike) by 6 inches and thickness of two or more inches. Two types of posthole sample were collected;

(a) Systematic posthole samples were taken from the centre of the ten feet channel samples.

(b) Random posthole samples had their location on the sample line selected in a random manner but were otherwise similar in all respects to the systematic posthole samples. For each ten feet section of sample line one number was drawn, without replacement, from a list of numbers, each number representing one foot of the sample line. Posthole sampling is therefore stratified random sampling.

Posthole sampling is more rapid and therefore much cheaper than channel sampling but it does not give the higher degree of ore type representation afforded by channel sampling.

(C) COMPARISON OF SAMPLING TECHNIQUES

(a) Channel and groove sampling

In order to assess the accuracy and reliability of the Marampa method of channel sampling (groove sampling), the assay values of channel and groove samples taken over 27 identical ten feet sections on three of the ore synclines (A, B and C) were statistically examined by;

(a) A Students t test (Chapter II) for significant differences between the means of the two groups (channel and groove) of assays.

(b) A Students t test for significant differences between the pairs (channel and groove samples from the same ten feet section) of sample assays.

The pairs of adjacent samples were collected along ten feet sections for a distance of 90 feet in each of the three ore synclines A, B and C. For comparative purposes the channel sample assays were assumed to give the best estimate of ore grade at any particular sample location and were thus used as the criteria with which to compare the groove sample assays. The basis for this assumption includes;

(a) Channel samples are more representative of the ore along the ten feet section than are the groove samples. The greater width and the constant width and thickness of the channel samples ensures that the softer and more friable ore, which is usually of lower grade, is not over-represented.

(b) Contamination of samples from walls of channel and from other sources was kept to a minimum. Groove sampling, as practised at Marampa, is open to contamination from several sources particularly during sample volume and particle size reduction because of the ignorance of the samplers of the principles of sampling.

The iron assays of the channel and groove samples are (Table 6.1)

	SYNCLINE A			SYNCLINE B			SYNCLINE C		
	Channel % Fe	Groove % Fe	Deviation C - G	Channel % Fe	Groove % Fe	Deviation C - G	Channel % Fe	Groove % Fe	Deviation C - G
	44.3	43.2	1.1	54.1	54.4	-0.3	48.2	45.6	2.6
	⌘	15.4	N.A.	53.4	54.5	-1.1	34.8	40.0	-5.2
	38.8	37.4	1.4	45.1	47.5	-2.4	42.6	43.3	-0.7
	44.1	36.0	8.1	50.2	53.1	-2.9	45.0	45.1	-0.2
	44.3	45.6	-1.3	51.5	48.7	2.8	45.6	44.8	0.8
	43.9	40.4	3.5	49.3	54.3	-5.0	47.7	46.5	1.2
	44.1	42.2	1.9	53.8	52.8	1.0	54.8	55.4	-0.6
	42.3	39.6	2.7	49.4	49.4	-	56.4	57.4	-1.0
39.0	39.2	-0.2	43.4	45.4	-2.0	49.2	49.9	-0.7	
Mean	42.6	40.45	2.53	50.02	51.12	1.94	47.14	47.57	1.44

Table 6.1: Comparative groove and channel sampling assay values.

⌘ sample assay not received.

Inspection of the assay results indicates that neither sampling method gives assay values which are consistently greater than those given by the other method, i.e. the two methods are apparently unbiased with respect to each other, with individual groove sample assays being greater than the comparative channel sample assays in the ratio 12:11, with three of the pairs of assays differing by less than 0.3 Fe per cent which is about the assumed reproducibility range of the assaying technique (D. Barnes, pers. comm.)

The channel sample assays from syncline A are noticeably, but not significantly (in the statistical sense, at the 95 per cent confidence level), greater than those of the groove samples with a difference between means of 2.2 Fe per cent (42.6 against 40.4 per cent Fe). This difference may be partially explained by the effect of the greater amount of soft and friable quartz-muscovite schist present in syncline A. Groove sampling tends to include an over-representative amount of this easily loosened material and in these circumstances the ore tends to be undervalued.

In order to determine whether or not the two sampling methods yield identical assay values, and by doing so to confirm the validity of the groove sampling method, statistical tests of significance are carried out using two variants of the Students t test. The two Null Hypotheses proposed are;

Hypothesis (a): No significant difference exists between the means of the two groups of sample assays. If this Null Hypothesis is accepted, then, intuitively, the two sampling methods yield concordant results and the groove sampling method is accepted as being valid for the sampling of the Marampa ore for the purpose of ore reserve estimation. This hypothesis is tested by equations 2.1 and 2.2

Hypothesis (b): No significant difference exists between the assay values of the two samples (channel and groove) taken at identical locations. Acceptance of this Null Hypothesis implies that groove sampling, as practiced at Marampa, is suitable for the collection of samples for the purpose of grade

control. This hypothesis is tested by equation 2.3.

The statistics of the assay values of the samples taken by the two sampling methods are (from Table 6.1);

Channel samples: $\bar{X}_c = 46.74$ per cent Fe, $s_c^2 = 28.73$

Groove samples: $\bar{X}_g = 46.61$ " " " $s_g^2 = 36.48$

$$n_c = n_g = 26, \quad v_c = v_g = 25$$

Mean deviation = 1.95 Fe per cent

Average deviation (sign accounted for) = 0.14 Fe per cent

Hypothesis (a):

Substitution of the appropriate values in equations 2.2 and 2.1 yields;

$$t_{50} = 0.08$$

The critical (standard table) values of t_{50} at the 95 per cent confidence level is 2.00,

$$\therefore t \text{ (calculated)} < t \text{ (critical)}$$

and the Null Hypothesis of no significant difference between the assay means of the samples taken by the two methods is accepted. It is thus concluded, from the data available, that there is no reason to suspect that groove sampling yields mean assay values significantly different from those given by channel sampling, and groove sampling is therefore valid in the conditions under which it is employed at Marampa. Inspection of the assay means given by the two sampling methods suggests, and the objective statistical test confirms, equivalence of the means.

Hypothesis (b):

Substitution of the appropriate values in equation 2.3 gives;

$$t_{25} = 0.26$$

The critical value of t_{25} at the 95 per cent confidence level is 2.06.

$$\therefore t \text{ (calculated)} < t \text{ (critical)}$$

and the Null Hypothesis of no significant difference between the assay values of the two samples (channel and groove) taken at identical locations is accepted. The implication behind

the acceptance of this Null hypothesis is that, since no significant difference has been shown to exist between channel sample grades and groove sample grades when sampling identical sections, then groove sampling is an acceptable method for collection of samples to be used for grade control purposes in which the location of the sample, as well as the iron content, is important.

(i) Discussion

The results of the statistical tests carried out on the channel and groove sample assay data indicate that groove sampling is an acceptable method of sampling the Marampa ore. This does not necessarily imply that groove sampling is the best method of sampling the Marampa ore but that the method, for all its apparent sources of error and contamination, yields similar results to the more accurate and reliable channel sampling method.

Certain physical features of the Marampa ore are expected to affect groove sampling in such a way as to yield non-representative samples thereby influencing ore reserve estimation or other use to which the samples are put. Variations in hardness and fissility of the ore are detrimental to the groove sampling method but do not affect channel sampling.

Bands of soft and friable quartz-muscovite schist which are common in the ore of syncline A are probably always over-represented by groove sampling with a resultant undervaluation of the ore. This is due to the ease with which this material is loosened by the pickaxe during sampling. The samplers have a natural tendency to include a disproportionately large amount of the quartz-muscovite schist and comparison of the groove and channel sample assays from syncline A (Table 6.1) would appear to support this observation. Six (of eight) channel samples have higher iron contents than the comparable groove samples with absolute differences ranging from 1.1 to 8.1 Fe per cent. This suggests a tendency towards undervaluation of the syncline A ore with groove sampling although the differences are not statistically significant.

The ores of synclines B and C are more regularly banded, contain fewer large bands of quartz-muscovite schist and appear to have suffered less deformation than the ore of syncline A. The hematitic bands are harder and much more coherent than those of syncline A and are also slightly fissile. These bands, when broken during groove sampling, tend to yield tabular plates of hematitic material which are longer than the width of the groove, and in these circumstances groove sampling has a tendency towards overvaluation of the ore. Statistical analysis of the assay data from the channel and groove sampling of synclines B and C reveals no significant difference between the means of the assays of the two types of samples but the fact that twelve (of seventeen) groove samples have a higher iron content than the comparable channel samples does suggest a tendency towards overvaluation of the ore with variations in fissility.

Groove sampling, as practiced at Marampa, is an adequate sampling method for use in ore reserve estimation and grade control and is to be preferred to channel sampling because of its better cost-effectiveness. Under conditions of uniformly soft and (ideally) vertically dipping ore, groove sampling is expected to give a good estimate of ore grade but where ore hardness and fissility vary and where the ore does not dip uniformly, then the potential sampling errors inherent in the method may become dominant with a resultant lack of accuracy and reliability in the assay results.

(b) Systematic posthole, random posthole and groove sampling

The assay values of samples taken by various sampling methods on syncline B (Chapter VII) have been utilised in order to compare systematic posthole, random posthole and groove sampling methods when sampling at identical locations.

This involves comparison of the assay results of channel sampling together with the assay results of two posthole samples taken one above the other at the centre of the channel sample. This is in effect a study of the grade variations between two vertically adjacent samples and one effect of the random post-hole sampling method is that in only eight cases do the

locations of the three sampling methods coincide. Table 6.2 gives the iron assay values of the three sampling methods taken at these eight locations, arranged in an analysis of variance (ANOVA) table.

LOCATION	1	2	3	4	5	6	7	8	METHOD TOTALS
SAMPLE METHOD									
CHANNEL	39.6	52.0	48.3	42.0	30.2	33.8	37.3	46.7	329.9
SYSTEMATIC POSTHOLE	38.3	52.0	52.2	35.0	52.7	45.9	47.2	19.6	342.9
RANDOM POSTHOLE	33.5	58.1	49.8	38.0	50.2	45.7	46.9	16.0	338.2
SAMPLE TOTALS	111.4	162.1	150.3	115.0	133.1	125.4	131.4	82.3	GRAND TOTAL 1011.0

Table 6.2: Assay value (per cent Fe) of three types of samples from eight locations. Syncline B.

To determine whether or not the three sampling methods yield significantly different assay values when sampling identical sections, an analysis of variance (Chapter II) is carried out.

The ANOVA calculations proceed as follows

Grand total (T), (sum of all assay values)	=	1011.0
Number of samples(N)	=	24
Correction Factor (C.F.) = T^2/N	=	42588.4
Total sum of squares	=	2510.2
Between sample location sum of squares	=	1405.4
Between sample method sum of squares	=	10.8

These figures are used to draw up a summary ANOVA table (Table 6.3 overleaf)

SOURCE OF VARIATION	SUM OF SQUARES	DEGREES OF FREEDOM	VARIANCE ESTIMATE
BETWEEN METHODS	10.8	2	5.4
BETWEEN LOCATIONS	1405.4	7	200.8
RESIDUAL	1094.0	14	78.1
TOTAL	2510.2	23	

Table 6.3 Summary ANOVA table for Table 6.2 data

The Null Hypothesis is proposed that there is no significant difference between the assay results given by the three sampling methods. The hypothesis is tested by Snedecors Variance Ratio or F test in which the variance estimate based on the sampling methods degrees of freedom is compared with the residual variance estimate;

$$\begin{aligned} F &= \frac{\text{methods variance estimate}}{\text{residual variance estimate}} \\ &= 5.4/78.1 \end{aligned}$$

$$\therefore F = 0.07$$

The critical (standard table) value of F at the 95 per cent confidence level with 2 degrees of freedom in the numerator and 14 degrees of freedom in the denominator is;

$$F_{0.05}(2,14) = 3.74$$

The calculated value of F is therefore much less than the critical value and the Null Hypothesis is accepted and there is no reason to suspect, at the 95 per cent confidence level, that the different sampling methods give rise to significantly different assay values.

A more surprising result perhaps, is that a Null Hypothesis that no significant differences occur between the

assay results of the several sample locations is also accepted although the calculated value of F is noted to be approaching a critical value as expected.

$$F = \frac{\text{sample location variance estimate}}{\text{residual variance estimate}} \\ = 200.8/78.1$$

$$\therefore F = 2.57$$

Critical values of F at the 90 and 95 per cent confidence levels are;

$$F_{0.1}(7,14) = 2.19 \text{ and } F_{0.05}(7,14) = 2.76$$

The Null Hypothesis is therefore accepted at the 95 per cent confidence level but confounded at the 90 per cent confidence level.

This ANOVA therefore indicates that, when sampling identical sections, either one of the two main sampling methods (channel and posthole), may be used and will give compatible mean assay values, and therefore, by extrapolation, either method is valid for ore reserve estimation purposes with one important proviso. Posthole sampling, by its very nature, yields assay values with significantly greater ranges and hence, larger variances than channel sampling. This is due to the fact that the smaller dimensions of the posthole samples make it extremely more likely that it could be collected completely within an iron-rich ore zone or an iron-deficient zone of the orebody. Examination of Table 6.2 confirms this interpretation; sample location 5 yielded post hole sample assay values of greater than 50 per cent Fe whereas the channel sample grade is only 30.2 per cent Fe. Conversely, at sample location 8, the posthole assay values are less than 20 per cent Fe whereas the channel sample grade is 46.7 per cent Fe. It is obvious that in these two cases, the posthole samples, being of much smaller dimensions, were collected within zones of iron-rich and iron-poor ore respectively, and the channel samples, being taken over a much greater length, included both iron-rich and iron-poor hands with a resultant smoothing effect on the overall grade.

As noted above, posthole sampling and channel sampling assay values are compatible and either method would be valid for ore reserve estimation purposes. The proviso on this statement is that the posthole sampling at 10 foot intervals along lines 200 feet apart is not valid for grade control purposes, where sample location, as well as sample grade, is important, because of the large range of misleading values which arise as a result of the relatively small dimensions of the posthole samples.

To determine the number of posthole samples necessary to give the same standard error of the mean as the channel samples the following calculation is made.

$$\begin{aligned} S_{Ec} &= S_{Ep} \\ \therefore \frac{s_c}{\sqrt{N_c}} &= \frac{s_p}{\sqrt{N_p}} \\ \therefore N_p &= (s_p \sqrt{N_c} / s_c)^2 \end{aligned}$$

Substituting the following values of,

$$\begin{aligned} s_c &= 8.81, \quad s_p = 11.553, \quad N_c = 70 \\ \therefore N_p &= 120.38, \quad \text{i.e. } 121 \end{aligned}$$

Over identical sample traverses, 121 posthole samples would be required to give the same standard error of the mean as 70 channel samples and this would require sampling at intervals of about six feet, or at five feet to simplify subsequent calculations. This would, of course, increase the costs of sampling and assaying and would not provide any additional useful information and it is suggested that posthole sampling at this density is less cost-effective than the channel sampling and is therefore not warranted.

(c) Groove sampling and systematic posthole sampling

The channel sample assay values from a rectangular area 140 by 200 feet (Fig. 6.1) within the main test block on syncline B were used to compare the mean grades given by the

A 7	6 d	sample	6 c	lines	6 b	A 6
northern			boundary			syncline B
33.4	96	3.26	45.8	80	3.26	38.1 62 3.26 28.0 42 - 42.0 12 3.26
37.5	95	3.26	47.1	79	3.26	36.1 61 3.26 35.0 41 3.26 41.5 11 3.26
40.5	94	3.26	42.9	78	3.26	42.6 60 3.26 42.3 40 3.26 39.5 10 3.26
33.9	93	3.26	20.0	77	-	42.1 59 3.26 43.5 39 3.26 40.1 9 3.26
20.4	92	-	39.6	76	2.75	42.8 58 3.26 30.2 38 - 15.0 8 -
37.2	91	3.26	34.8	75	2.09	35.1 57 2.53 42.5 37 1.57 43.5 7 1.74
43.1	90	1.57	37.3	74	3.26	34.3 56 3.26 40.9 36 3.20 24.0 6 -
40.0	89	2.14	47.6	73	0.93	25.4 55 - 37.3 35 3.26 44.2 5 2.57
49.5	88	0.70	51.0	72	1.72	51.7 54 0.77 44.7 34 2.11 48.3 4 0.90
49.6	87	1.37	49.8	71	1.64	52.7 53 1.08 51.7 33 1.25 54.7 3 1.36
49.3	86	0.69	45.3	70	3.02	52.9 52 0.91 48.8 32 1.12 52.0 2 1.29
47.8	85	0.63	44.7	69	1.40	52.4 51 1.37 52.1 31 1.05 55.3 1 1.09
46.7	84	0.96	45.8	68	1.09	51.6 50 0.94 47.8 30 1.93 54.8 14 1.10
53.4	83	0.84	49.7	67	1.22	46.8 49 1.38 54.1 29 0.82 53.3 15 1.00
			50.5	66	0.59	47.1 48 1.16 51.4 28 1.61 48.7 16 2.33
			48.8	65	0.71	47.9 47 0.82 49.0 27 0.64 49.7 17 0.76
			33.8	64	0.71	53.7 46 0.79 53.7 26 0.61 48.9 18 0.61
• 53.4	Fe per cent					46.7 45 0.65 50.3 25 0.69 32.5 19 0.96
83	sample no. (SPT series)					25.2 44 - 35.0 24 1.17 41.3 20 0.98
0.84	Mn per cent					43.6 43 0.59 50.8 23 0.57 50.0 21 0.76
Channel samples						39.6 22 0.60

— 50' —

northern			boundary			syncline B		
21.0	94	3.26	51.2	79	3.26	45.3 62 3.26	35.0 21 3.26	
45.6	93	3.26	35.5	78	3.26	35.0 61 3.26 36.8 41 3.26 48.6 20 3.26		
38.4	92	3.26	35.7	77	2.57	28.2 60 3.26 40.7 40 3.26 49.3 19 3.26		
27.0	91	3.26	33.6	76	1.45	48.0 59 3.26 36.3 39 3.26 12.2 18 3.26		
14.0	90	3.26	42.4	75	2.36	35.4 58 3.26 52.7 38 1.38 4.8 17 -		
32.0	89	3.26	44.5	74	1.22	45.4 57 1.72 40.1 37 3.26 47.4 16 -		
31.4	88	3.26	47.2	73	0.71	32.6 56 3.26 46.9 36 1.08 52.2 15 0.98		
39.4	87	3.26	47.4	72	1.02	34.1 55 2.18 37.9 35 3.26 25.8 14 3.26		
51.3	86	0.55	50.3	71	1.42	44.2 54 0.66 52.3 34 0.96 52.2 13 0.99		
43.9	85	1.95	52.8	70	2.22	53.6 53 1.71 41.2 33 0.53 57.5 12 1.32		
44.5	84	0.78	45.7	69	2.86	55.4 52 0.72 43.5 32 2.06 52.0 11 1.17		
52.8	83	0.89	39.0	68	0.60	45.7 51 1.94 56.0 31 1.62 59.9 10 0.33		
19.6	82	-	48.4	67	0.56	52.3 50 1.05 50.7 30 2.36 55.2 9 0.89		
41.0	81	1.72	52.8	66	0.87	52.6 49 2.26 57.8 29 0.79 59.0 8 0.52		
			47.3	65	0.45	54.9 48 0.48 52.7 28 1.57 47.7 7 1.13		
			46.5	64	0.67	47.7 47 0.49 43.9 27 0.81 45.2 6 0.45		
			45.9	63	0.74	49.5 46 0.41 56.4 26 0.52 47.3 5 0.62		
• 41.0	Fe per cent					50.5 45 0.41 49.1 25 0.45 15.6 4 -		
81	sample no. (SPP series)					9.4 44 - 49.2 24 0.56 48.6 3 0.86		
1.72	Mn per cent					44.6 43 0.52 55.9 23 0.40 50.2 2 0.34		
Posthole samples							38.3 1 0.50	

TEST BLOCK - SYNCLINE B

Fig. 6-1

two sampling methods. The rectangular area included all five sampling lines and was bounded by the four channel samples SPT 96,83,11 and 16. Two comparisons are made;

- (i) Comparison of the mean grade given by the two sampling methods.
- (ii) Comparison of paired sample (channel and posthole from same location) assay values to determine whether either method results in overvaluation of the grade.

The assay values of the two sampling methods for each of the five lines is summarised in Table 6.4.

The two Null Hypotheses proposed are;

Hypothesis (a): No significant difference between the mean grades given by the two sampling methods.
This hypothesis is tested by means of equations 2.1 and 2.2

Hypothesis (b): No significant difference between the individual (paired) grades given by the two sampling methods.
This hypothesis is tested by means of equation 2.3.

Hypothesis (a) From Table 6.4

$$\begin{aligned}\bar{X}_c &= 43.35 & s_c^2 &= 77.62 \\ \bar{X}_p &= 43.07 & s_p^2 &= 133.47 \\ N_c &= N_p & &= 70 \\ V_c &= V_p & &= 69\end{aligned}$$

The subscripts c and p referring to channel (groove) and systematic posthole samples respectively. Note the high value of the posthole sample assay variance.

Substitution of the appropriate ratios in equations 2.2 and 2.1 gives:

$$t_{138} = 0.16$$

The critical (standard table) value for t_{138} at the 95 per cent confidence level is;

$$t_{138} = 1.98.$$

∴ $t(\text{calculated}) < t(\text{critical})$ and the Null Hypothesis

Table 6.4: Summary of channel sample and systematic posthole sample assay values, syncline B.

LINE	A7			A6d			A6c			A6b			A6		
	C ^{Fe}	P ^{Fe}	C-P ^{Fe}	C	P	C-P	C	P	C-P	C	P	C-P	C	P	C-P
33.4	21.0	12.4		45.8	51.2	- 5.4	36.1	35.0	1.1	35.0	36.8	- 1.8	41.5	48.6	- 7.1
37.5	45.6	-8.1		47.1	35.5	11.6	42.6	28.2	14.4	42.3	40.7	1.6	39.5	49.3	- 9.8
40.5	38.4	2.1		42.9	35.7	7.2	42.1	48.0	-5.9	43.5	36.3	7.2	40.1	12.2	27.9
33.9	27.0	6.9		20.0	33.6	-13.6	42.8	35.4	7.4	30.2	52.7	-22.5	15.0	4.8	10.2
20.4	14.0	6.4		39.6	42.4	- 2.8	35.1	45.4	-10.3	42.5	40.1	2.4	43.5	47.4	- 3.9
37.2	32.0	5.2		34.8	44.5	- 9.7	34.3	32.6	1.7	40.9	46.9	- 6.0	24.0	52.2	-28.2
43.1	31.4	11.7		37.3	47.2	- 9.9	25.4	34.1	- 8.7	37.3	37.9	- 0.6	44.2	25.8	18.4
40.0	39.4	0.6		47.6	47.4	0.2	51.7	44.2	7.5	44.7	52.3	- 7.6	48.3	52.2	- 3.9
49.5	51.3	-1.8		51.0	50.3	0.7	52.7	53.6	- 0.9	51.7	41.2	10.5	54.7	57.5	- 2.8
49.6	43.9	5.7		49.8	52.8	-3.0	52.9	55.4	- 2.5	48.8	43.5	5.3	52.0	52.0	-
49.3	44.5	4.8		45.3	45.7	- 0.4	52.4	45.7	6.7	52.1	56.0	- 3.9	55.3	59.9	- 4.6
47.8	52.8	-5.0		44.7	39.0	5.7	51.6	52.3	- 0.7	47.8	50.7	- 2.9	54.8	55.2	- 0.4
46.7	19.6	27.1		45.8	48.4	- 2.6	46.8	52.6	- 5.8	54.1	57.8	- 3.7	53.3	59.0	- 5.7
53.4	41.0	12.4		49.7	52.8	- 3.1	47.1	54.9	- 7.8	51.4	52.7	- 1.3	48.7	47.7	1.0

^{Fe}C channel sample, per cent Fe : P posthole sample, per cent Fe : C-P channel sample grade minus posthole sample grade

of no significant difference between the means is accepted.

The statistical comparison of the assay values of the two types of samples therefore indicates that the mean grades given by the two sampling methods are essentially identical and the observed difference (0.28 Fe per cent) could readily have arisen by chance.

Hypothesis (b) can be tested by two methods;

- (i) Students t distribution test (equation 2.3)
- (ii) Wilcoxon's Signed Ranks test (Chapter II)

This second test has the advantage of being distribution-free and it is not necessary that the sample grades be normally distributed.

The Students t test is based on the equation;

$$t_v = \frac{\bar{d} \sqrt{n}}{S_d} \quad \dots \quad \dots \quad \text{Equat 2.3}$$

From Table 6.4

$$\begin{aligned} \bar{d} &= 0.275 \\ S_d &= 8.752 \\ n &= 70 \end{aligned}$$

Substituting these values in equation 2.3 gives

$$t_{69} = 0.26$$

The critical value of t_{69} at the 95 per cent confidence level is 2.00,

$$\therefore t \text{ (calculated)} < t \text{ (critical)}$$

and the Null Hypothesis is accepted and there is no reason to suspect that the two sampling methods give rise to significantly different assay values when sampling similar sections of ore.

In order to free the comparison of paired assay values from the effects of the normal distribution, a Wilcoxon's Signed Rank test can be carried out on the paired data. Wilcoxon's Signed Rank test is based on the fact that if no significant differences occur between the two values making up each pair of values, then the sum of the differences between the two will tend to zero.

Application of Wilcoxon's Signed Rank test gives rank sums of 1188 and 1227. The smaller rank sum is too great, at the 95 per cent confidence level, to suspect that significant differences exist between the channel and posthole sample assay grades and the Null Hypothesis is accepted.

(e) Systematic and random posthole sampling

These two types of samples are physically identical but are collected from different locations. The systematic posthole samples, as the name suggests, are taken at systematically located positions along the sample line whereas the random posthole samples are taken along the sample line at locations decided by random sampling methods.

A Student's t test involving equations 2.1 and 2.2 is used to determine whether the mean grades given by the two methods are significantly different. Table 6.4 gives the assay values of the systematic posthole samples; the mean grade, variance and number of these samples being;

$$\bar{X}_{sp} = 43.07 \quad s_{sp}^2 = 133.47 \quad n_{sp} = 70$$

The mean grade, variance and number of random posthole sample assays, from the same rectangular area of the ore, are;

$$\bar{X}_{rp} = 40.22 \quad s_{rp}^2 = 181.66 \quad n_{rp} = 72$$

Substituting these values in equations 2.2 and 2.1 gives,

$$t_{140} = 1.35$$

Recourse to standard tables gives a critical value of $t_{140} = 1.98$ at the 95 per cent confidence level.

$$\therefore t(\text{calculated}) < t(\text{critical})$$

and the Null Hypothesis is accepted, and there is no reason to suspect that random posthole and systematic posthole sample assays yield significantly different mean values (at the 95 per cent confidence level) when sampling a similar area or block of ore. The tests are somewhat hampered by the wide range of individual assay values and the consequent large variances but, on the information available, there is no reason to suggest that the observed difference (2.85 Fe per cent)

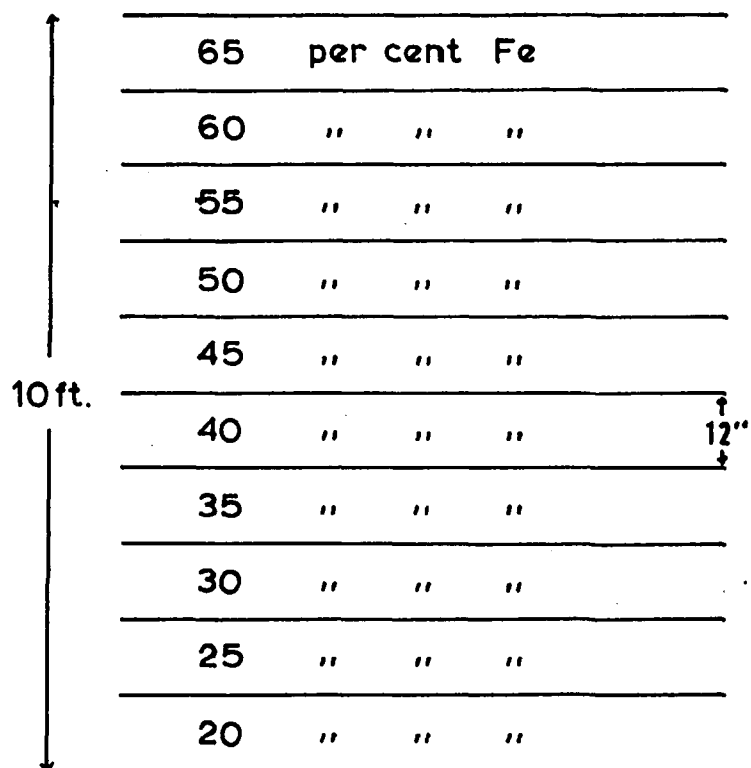


Fig. 6-2 Hypothetical ore section

between the mean grades of the systematic and random posthole samples could not have been due to chance factors alone.

Random posthole sampling is considered to be inadequate for the purposes of grade control as can be seen with reference to Fig. 6.2.

A channel sample taken across the hypothetical section of banded ore (Fig. 6.2) would have a mean grade of 42.5 per cent Fe but a random posthole sample taken from within the same ten feet section could have any grade between 20 and 65 per cent Fe and would obviously often give a misleading estimate of the ore grade over the ten feet section. Ideally, the size of the samples should bear some systematic relationship to the rate or method of mining; in the Marampa case a ten feet or twenty feet channel sample is more realistic than a single posthole sample.

(D) SUMMARY

(a) The present channel (groove) sampling method employed at Marampa is acceptable for purposes of ore reserve estimation and grade control and is to be preferred to other methods because of its better cost-effectiveness.

(b) The mean assay values given by the three sampling methods, channel (groove), systematic posthole and random posthole, are essentially the same for identical blocks of ore and the observed differences are most likely to be caused by chance factors.

(c) The smaller dimensions of the posthole type of sample renders them more likely to fall completely within an iron-rich or iron-poor band of ore, than the channel samples. As a result, while the mean assay grades of the two main types of samples are essentially similar, the variances are significantly different and this is due to the extreme range of values which occurs with the posthole samples. The channel samples, being collected over a greater length, are more likely to include both iron-rich and iron-poor bands of ore which results in a

"smoothing" effect on the ore grades and a consequent reduction in the variance.

(d) The posthole samples are, theoretically at least, adequate for ore reserve estimation purposes and random posthole sampling methods would yield mean grades similar to the channel sample grades. However, grade control, in the context of mining, requires that a sample assay value be associated with a definite location and have some correlation with nearby sample locations, and in this respect random posthole samples are quite inadequate for grade control purposes.

(e) The large variances of the posthole type of sample implies that the absolute or true grade of the deposit can not be estimated with any great degree of accuracy with the use of this sampling technique. The channel sample assays give a mean value with a much smaller variance and greater confidence can be given to the estimated range of values within which the absolute grade can be expected.

(f) Sampling methods, such as posthole sampling, which yield large ranges and high variances are best used on a systematic rather than a random pattern in order to reduce the effect of the different sample weighting according to "area" of influence and it is therefore suggested that random posthole sampling (at least at 10 feet intervals on sample lines 200 feet apart) would be an unacceptable sampling method to be employed at Marampa.

CHAPTER VII

ANALYSIS AND INTERPRETATION OF HORIZONTAL AND VERTICAL ORE GRADE DISTRIBUTION AT MARAMPA

(A) INTRODUCTION

Prediction models of ore grade variations, both horizontal and vertical, are based on statistical analyses of ore grade distributions and this chapter describes the type of ore grade distribution and the spatial distribution of ore grades which occur at Marampa.

The determination of the type of grade distribution is based on an analysis of assay values of 1350 channel samples taken on the 560 feet level during the 1967 channel sampling programme and includes ore grades (greater than 30.0 per cent Fe) from all 14 sample traverses.

The spatial distribution of ore grades was studied in two sections;

(a) Horizontal grade distribution was based on an analysis of channel and posthole sample assay values collected from an intensively sampled block of ore on syncline B.

(b) Vertical grade distribution was based on an analysis of auger drillhole data and on channel sampling data from two mining levels.

An area, some 200 feet by 220 feet, on the northern margin of syncline B, was intensively sampled using three sampling methods, in order to determine certain numerical features of the orebody pertaining to grade distribution and trends, if any, of mineralization, optimum sample spacing, optimum sampling method for ore reserve estimation and for grade control (Chapter VI), across-strike and along-strike grade variation and distance over which sample grades are associated (area of influence).

Most of these features have economic implications which, if successfully interpreted and introduced into mine planning,

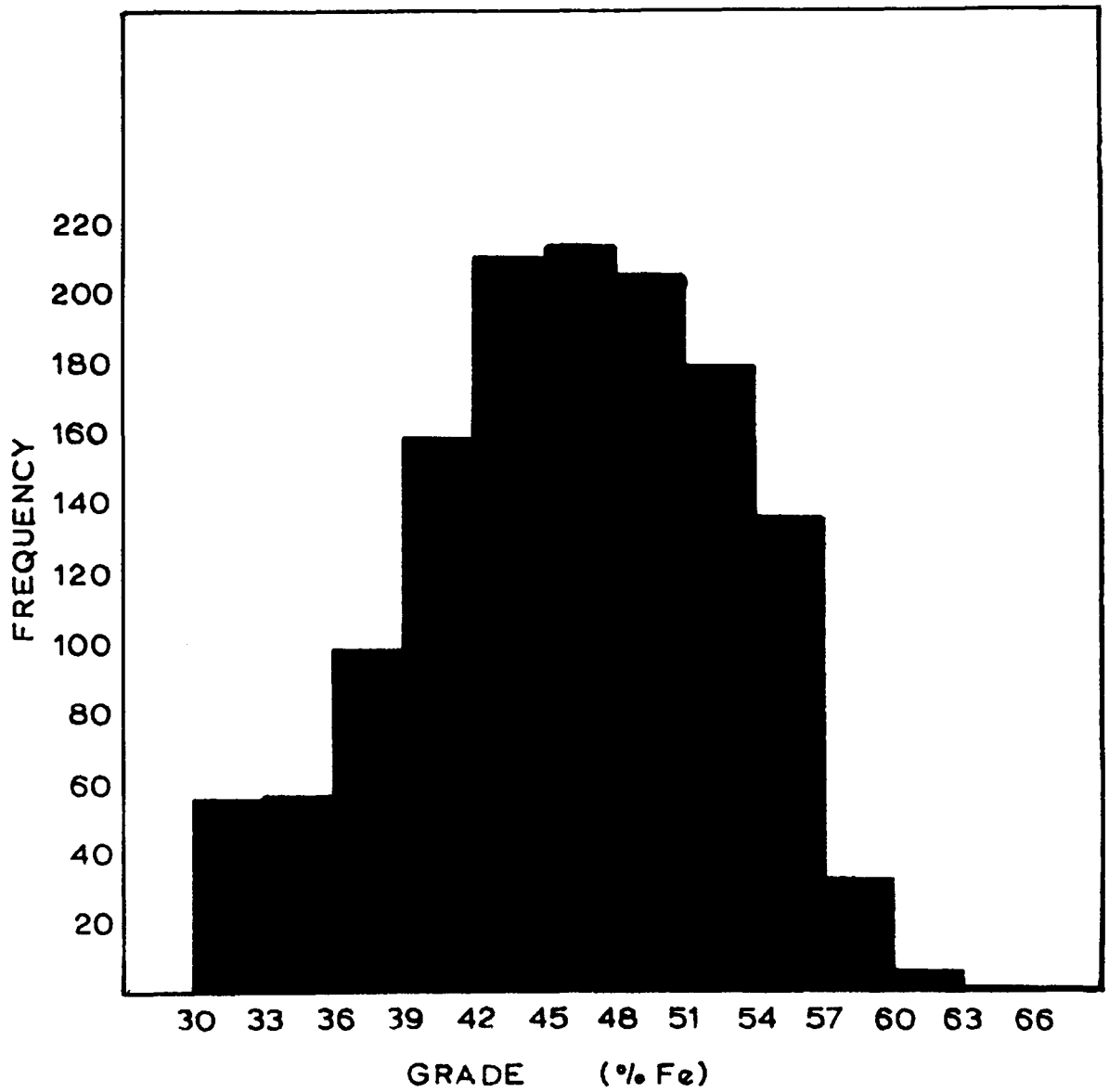
may result in an improvement in iron recovery and/or concentrate grade. If such long term aims are not achieved, then at least there will exist a body of information concerning grade distribution and sampling techniques which may eventually be relevant in the design of more cost-effective procedures for ore reserve estimation and grade control.

The ultimate use of such a comparative sampling programme depends on the valid extrapolation of the results from the syncline B test block to the entire Marampa orebody and while it is considered that a successful extrapolation to all parts of the four major Marampa ore synclines is probably not feasible, because of unmeasured factors such as ore hardness and differential deformation, the analysis of the sampling data from this initial area will provide much fundamental information.

(B) STATISTICAL ORE GRADE DISTRIBUTION AT MARAMPA

The frequency distribution of ore grades of 1350 channel samples taken along 14 traverse lines during the 1967 channel sampling programme are tabulated in Table 7.1 and graphically described by Figs. 7.1 and 7.2.

The Marampa ore grade distribution is best described as Gaussian or normal although certain discrepancies, particularly in the class intervals 30.0 to 32.9 and 54.0 to 56.9 per cent Fe, are such that a Chi-square test for normality fails at the 95 per cent confidence level. The discrepancy in the lower class may be due to truncation of ore grades at 30 per cent Fe. The frequency of ore grade occurrences in these two class intervals are higher than would be expected from a Gaussian distribution and a geological interpretation suggests that the original grade of the Marampa deposit may have been within the range 25 to 35 per cent Fe which would at least partially explain the similar frequencies in the two lower class intervals 30.0 to 32.9 and 33.0 to 35.9 per cent Fe and the relatively large increase in frequency in the following class interval 36.0 to 38.9 per cent Fe (Fig 7.1). The original grade of the deposit has been modified by leaching to produce grades in excess of 50 per cent Fe. There are thus



Grade distribution, 1350 channel samples

Fig. 7.1

Class Interval % Fe	Frequency	Cumulative Frequency	Per cent	Cumulative per cent
> 68.9	0			
66.0 - 68.9	1	1350	0.07	100.00
63.0 - 65.9	1	1349	0.07	99.92
60.0 - 62.9	6	1348	0.44	99.85
57.0 - 59.9	32	1342	2.37	99.41
54.0 - 56.9	136	1310	10.07	97.04
51.0 - 53.9	179	1174	13.26	86.96
48.0 - 50.9	205	995	15.18	73.70
45.0 - 47.9	213	790	15.78	58.52
42.0 - 44.9	210	577	15.56	42.74
39.0 - 41.9	158	367	11.70	27.18
36.0 - 38.9	98	209	7.26	15.48
33.0 - 35.9	56	111	4.15	8.22
30.0 - 32.9	55	55	4.07	4.07

Table 7.1: Grade distribution of 1350 channel samples.

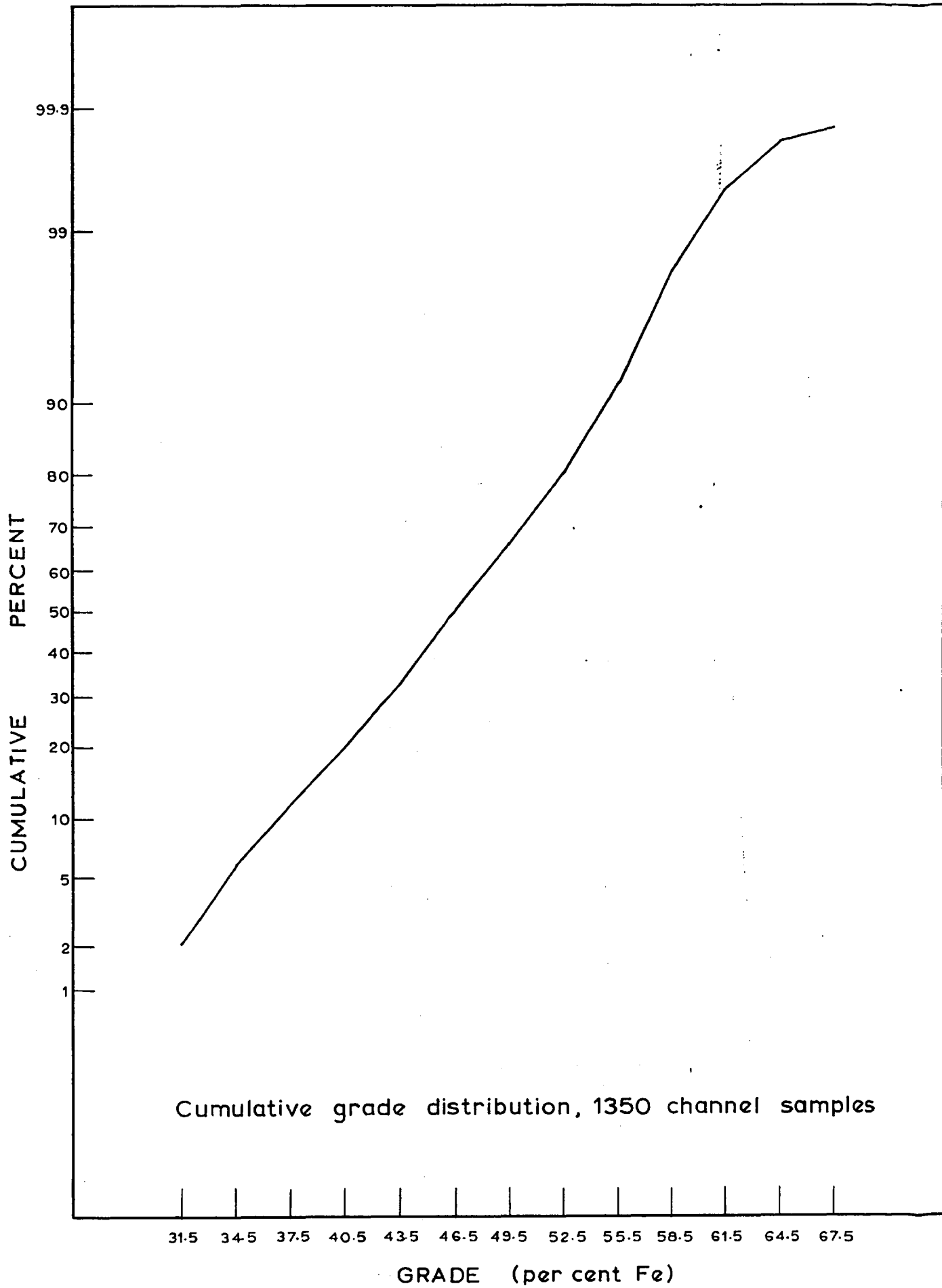


Fig. 7-2

two basic ore "types", leached and unleached, and thus two ore grade populations exist. This may account for the slight deviation of the cumulative grade distribution (Fig. 7.2) from a straight line and this line may, in fact, be the resultant of two straight lines, one for the high grade leached ore and one for the lower grade unleached ore. The more highly leached ore around the periphery of the orebody may contribute a disproportionately high number of grades in the 54.0 to 56.9 per cent Fe class interval. Similarly, the unleached ore away from the margins of the orebody contributes a large number of ore grades to the lower grade class intervals and it is probably significant that the sample traverse lines with the highest and lowest mean grades occur at the margin and centre of the orebody respectively (Fig. 7.3).

A Chi-square test for normality of ore grade distribution on each of the sample traverses indicated that only 5 (of 12) failed the normality test at the 95 per cent confidence level and, of these 5, two were accepted as normal at the 99 per cent confidence level. The three traverse lines, A6, A8 and A9, which failed the test for normality at the 99 per cent confidence level, lie towards the centre of the orebody and it is suggested that the ore in the vicinity of these traverses may contain areas of leached (traverse line A6) or unleached (traverse lines A8 and A9) ore in sufficient amounts to weight the ore grade distribution so as to fail the Chi-square test. It is also possible that these three sample traverse lines with the apparently anomalous grade distributions may be reflecting anomalous features of the original ore deposition.

The mean grade and standard deviation of the Marampa orebody calculated from the assay values of the channel samples collected from the 560 feet mining level is 45.9 per cent Fe and 6.72 respectively with 95 per cent confidence limits on the mean of 0.37 Fe per cent, i.e. the mean grade is 45.9 $\pm$ 0.4 per cent Fe.

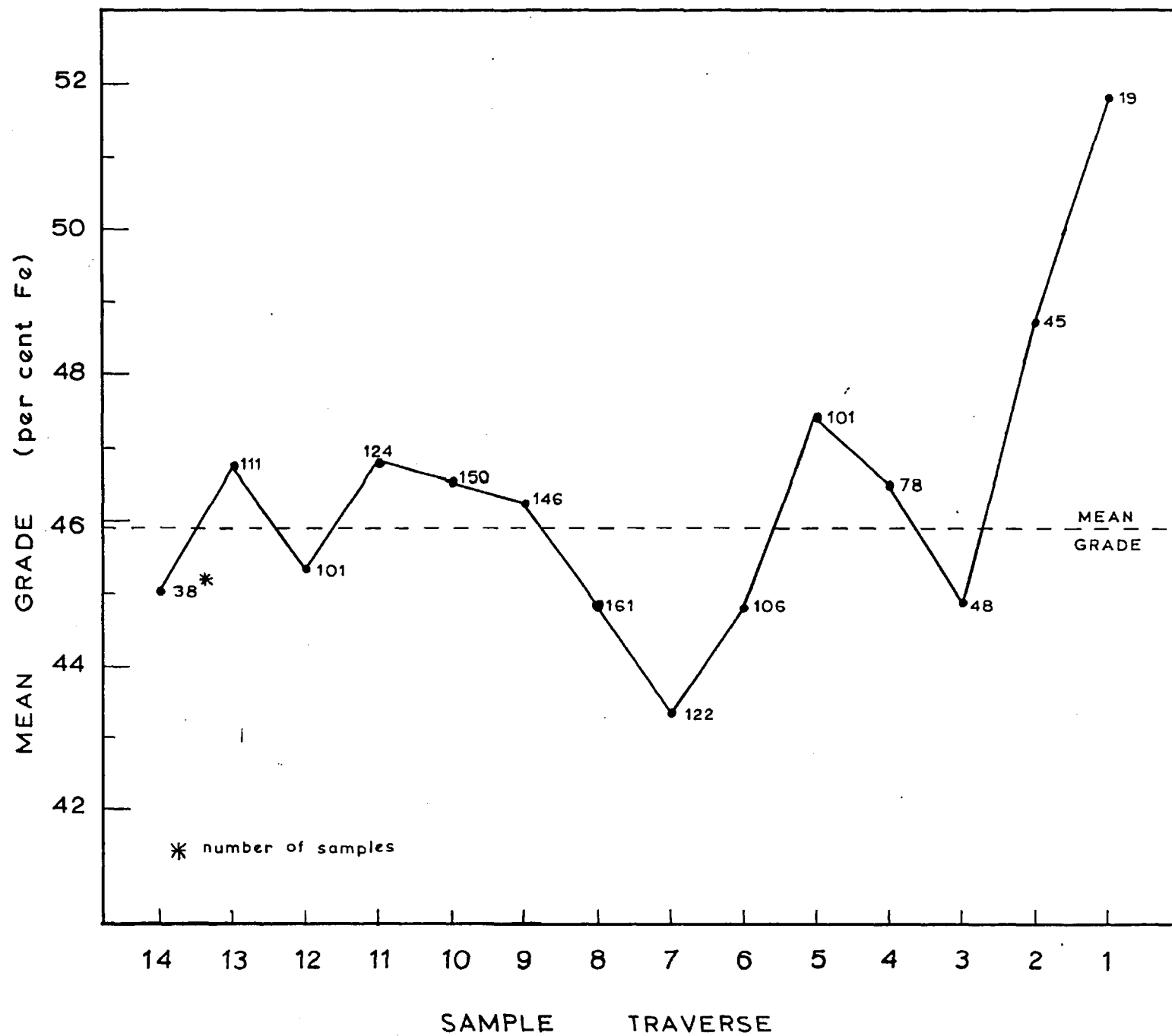


Fig. 7.3

(C) ANALYSIS AND INTERPRETATION OF SYNCLINE B ASSAY DATA

(a) Introduction

The block of ore on which the comparative sampling programme was carried out was situated on the northern margin of syncline B, between the mine planning grid lines A7 and A6, and formed the surface of the 560 feet level mining bench. Five sample lines were constructed across-strike at along-strike intervals of fifty feet, the two end lines coinciding with the planning grid lines A7 and A6. The length of each line which was sampled varied according to the variations in the boundary between syncline B ore and anticline B/C waste; the shortest line was A7 at 140 feet and the longest was A6 at 226 feet, the lengths of the other three lines A6b, A6c and A6d being intermediate between these two extremes.

Three sampling methods were used on each of the five sample lines;

(i) Channel (groove) sampling. This is the sampling method used by the Marampa Planning Department to obtain samples for ore reserve estimation and grade control purposes. The actual method has been described in detail elsewhere (Chapter VI) but a brief summary will be given.

Sample lines are cleared to bedrock across strike and a groove is pickaxed out to a depth ranging from one-half to two inches depending on ore hardness. The loosened ore from each ten feet section is scraped by hand to the centre of the section, placed in a headpan and hammered to reduce the size of the larger particles. It is then perfunctorily mixed by hand, a rough cross marked on the sample and alternate quarters roughly scooped out by hand and discarded. This process of size and volume reduction is repeated until about one to two kilograms of sample remain to be placed in the sample bags for later assaying.

This sampling method is only a crude form of channel sampling but statistical tests indicate that, for all its apparent sources of error and contamination, the method yields satisfactory results.

(ii) Systematic posthole sampling. The term "posthole" is used for want of a more descriptive term. This sampling method consists of taking tabular samples 12 by 6 by ($\frac{1}{2}$ to 1) inches from the centre of the ten feet section of the earlier described channel samples.

(iii) Random posthole sampling. This sampling method is identical in all respects to the systematic posthole sampling except that the location of the sample is determined by random sampling methods with the provision for one sample for each ten feet of sample line; i.e. a sample line two hundred feet long will be represented by twenty random posthole samples, the positions of which along the line are determined by random sampling methods.

During the mining of the block of ore, conveyor samples were collected in an attempt to correlate the surface channel and posthole sampling results with the actual mill feed grade but several technical factors concerned with the mining of the block weighed against the production of meaningful analytical results. In particular, the block of ore was not mined according to usual mine practice because of variations in the hardness of the ore. Much of the ore could not be won directly by the face shovels and bulldozers were used to loosen the ore and push it down to the shovels. As a result, the usual thirty feet face was not developed at any stage during the removal of the block and at any one time, ore being loaded on the conveyor belt may have been from any part of the 200 by (about) 180 feet area of the block.

(b) Trends of mineralization and continuity of ore grades.

Geological factors and contour plans of assay values suggests that the trend of mineralisation at Marampa, as expressed by least grade variation in a particular direction, is parallel to the strike of the orebody. On geological grounds it would be expected that grade variations would be at a minimum within a single depositional unit which is usually taken to represent deposition under similar or constant conditions.

Grade variations along strike and down dip in such a unit would tend to be smaller than variations across strike. A boundary between lithological units implies, among other things, that changes in depositional conditions may have occurred and grade variations between lithological unit, i.e. across strike, would be expected to be much greater than grade variations within each unit where depositional conditions are inferred to have been essentially constant.

Distinct lithological banding at Marampa has been largely obscured by tectonic events including shearing and minor faulting, but the trend of mineralisation may be determined numerically by analysis of variance of assay grades to determine whether grade variations are greater along or across-strike.

An ANOVA carried out on the assay grades of the channel samples taken from the rectangular area within the test block on syncline B is based on the following data (Table 7.2).

ALONG STRIKE ACROSS STRIKE	A7	A6d	A6c	A6b	A6	ACROSS STRIKE TOTALS
	33.4	45.8	36.1	35.0	41.5	191.8
	37.5	47.1	42.6	42.3	39.5	209.0
	40.5	42.9	42.1	43.5	40.1	209.1
	33.9	20.0	42.8	30.2	15.0	141.9
	20.4	39.6	35.1	42.5	43.5	181.1
	37.2	34.8	34.3	40.9	24.0	171.2
	43.1	37.3	25.4	37.3	44.2	187.3
	40.0	47.6	51.7	44.7	48.3	232.3
	49.5	51.0	52.7	51.7	54.7	259.6
	49.6	49.8	52.9	48.8	52.0	253.1
	49.3	45.3	52.4	52.1	55.3	254.4
	47.8	44.7	51.6	47.8	54.8	246.7
	46.7	45.8	46.8	54.1	53.3	246.7
	53.4	49.7	47.1	51.4	48.7	250.3
ALONG STRIKE TOTALS	582.3	601.4	613.6	622.3	614.9	T = 3034.5

Table 7.2: Channel sample assay grades (per cent Fe), Syncline B.

The ANOVA calculations yield the following results (Table 7.3)

SOURCE OF VARIATION	SUM OF SQUARES	D.F	VARIANCE ESTIMATE	F RATIO
ALONG STRIKE	78.77	4	19.69	0.63
ACROSS STRIKE	3658.60	13	281.43	9.00
RESIDUAL	1626.69	52	31.28	
TOTAL	5364.06	69		

Table 7.3: Summary ANOVA, raw data from Table 7.2

The Null Hypotheses proposed are;

(a) No significant variations in grade exist along strike.

(b) No significant variations in grade exist across strike.

These hypotheses are tested by comparing the variance estimates with the residual variance estimates as follows;

Hypothesis (a):

$$F = \frac{\text{along strike variance estimate}}{\text{residual variance estimate}} \\ = 19.69/31.28$$

$$\therefore F = 0.63$$

$$F_{0.05}(4,52) = 2.52 \text{ and } F_{0.01}(4,52) = 3.65$$

$\therefore F(\text{calculated}) < F(\text{critical})$ at both the 95 and the 99 per cent confidence levels and the Null Hypothesis for along-strike grade variations is accepted.

Hypothesis (b):

$$F = 281.43/31.28$$

$$\therefore F = 9.00$$

$$F_{0.05}(13,52) = 1.92 \text{ and } F_{0.01}(13,52) = 2.50$$

$\therefore F(\text{calculated}) > F(\text{critical})$ at both the 95 and 99 per cent confidence levels and the Null Hypothesis is confounded. An F value of 9.00 could have arisen by chance less than once in one hundred and there is therefore a significant variation in ore grades across-strike.

The objective ANOVA confirms the geological interpretation and indicates conclusively that across-strike grade variations are much greater than along-strike grade variations

and this may be carried a step further and used to support the reason for the relatively large along-strike separation of the sampling traverses at present used at Marampa. It is apparent that, if grade variations along-strike are modest and grade variations across-strike are large, a square sampling grid is an inefficient method of sampling the orebody.

A similar ANOVA series carried out using samples located at 50 feet centres, i.e. a square grid, yields similar results although the magnitude of the calculated value of F varies widely.

The results of this analysis of variance may be interpreted in geological terms as being confirmation of the existence of an originally sedimentary deposit which has since been tectonically disturbed mainly by shearing parallel to the regional strike. If large displacements had occurred across-strike the ANOVA would tend to yield results suggesting similarity of along- and across-strike grade variations.

The trend of mineralisation at Marampa has very well defined directional properties and may be said to be strongly anisotropic with the greatest variations in grade occurring across the strike of the orebody.

The ANOVA technique could be adapted to similar grade information on other types of orebodies to determine trends of mineralisation and can be undertaken on either micro- or macroscopic scales.

Given that the Marampa orebody was deposited as a sedimentary sequence and has been subsequently metamorphosed and structurally deformed, several points concerned with continuity of grade from any single location may be considered.

(a) Grade variations along strike should be less variable than grade variations across strike. This has been confirmed by the earlier ANOVA carried out on the syncline B test block assay data.

(b) This ANOVA indicates that the ore grades are strongly anisotropic, i.e. the magnitudes of the ore grades have a

preferred orientation.

(c) Little transverse faulting or other across-strike displacements appear to have taken place at Marampa. Although no distinctive marker beds exist for greater than about one hundred feet, no major displacements have been observed on ore/waste contacts, and only minor displacements, measured in inches, have been observed (and only rarely) in the ore synclines themselves. In addition, the ANOVA of the syncline B test block would have reflected any major displacements by indicating isotropism rather than anisotropism of ore grades.

(d) Correlation analysis (Chapter II) of the syncline B test block assay data also confirms that ore grades along strike are much more uniform than ore grades across strike. Table 7.4 gives the total correlation coefficients between the assay values of the five sample lines.

SAMPLE LINE	A6d	A6c	A6b	A6
A7	N.S.	0.5888 [Ⓜ] 0.3467 ^{ⓂⓂ}	0.6755 0.4563	0.5622 0.3160
A6d	-	N.S.	0.7441 0.5537	0.8521 0.7261
A6c	-	-	0.6676 0.4457	N.S.
A6b	-	-	-	0.8068 0.6509

N.S. denotes not significant at the 95 per cent confidence level

[Ⓜ] denotes correlation coefficient (r)

^{ⓂⓂ} denotes coefficient of determination (r^2)

Table 7.4 Total correlation coefficients and coefficients of determination for assay values of five sample lines, syncline B.

The critical values of the correlation coefficient above which statistical significance is attained at the 95 and 99 per cent confidence level, are 0.5324 and 0.6614 respectively.

Significant correlations exist between almost all of the possible pairs of sample line assay values indicating that variations across strike in one part of the orebody are associated with similar variations further along strike, i.e. the grade variations which occur along sample line A7 are associated with the grade variations which occur along sample line A6 (see Table 7.4).

Correlation analysis of along-strike assay values from the syncline B test block indicate that there is virtually no meaningful grade relationships between adjacent ore zones or lithological units. Discrete lithological units reflect, among other things, the physico-chemical conditions prevailing during the time of deposition; changes in these conditions are reflected by lithological changes which, if severe, are expressed as significantly different lithological units usually separated by a well defined bedding plane. Within each lithological unit the ore grade remains essentially similar whereas between the lithological units the differences in ore grade may be extreme and the ore grades in the two units may be independent. In a bedded ore deposit, as at Marampa, the grade variations within each lithological unit are less than those between the units and correlation analysis of assay results of two sample lines across the strike of the orebody would be expected to yield correlation coefficients which were positive, large and statistically significant. Conversely, correlation analysis of the assay values of two sample lines parallel to the strike of the orebody would be expected to tend to zero as illustrated in Fig. 7.4 which describes part of a hypothetical orebody with lithological units and ore grades as shown.

Correlation analysis of the assay data of the hypothetical orebody yields the following results;

(a) Correlation analysis of sample lines A and C gives correlation coefficient of +1.0, i.e. perfect, positive linear correlation.

Fig. 7.4

	A	B	C			
1	30	30	30	per cent	Fe	
2	40	40	40	"	"	"
3	50	50	50	"	"	"

Hypothetical ore section

A B C	sample lines
1 2 3	lithological units

Fig. 7.5

A	B			
30	5	per cent	Fe	
40	6	"	"	"
50	7	"	"	"

(b) Correlation analysis of lithological units 1 and 3 gives a zero correlation coefficient, i.e. perfect non-correlation indicating no meaningful relationship between the two assay groups.

The style of ore grade distribution in the syncline B test block at Marampa is similar to that described by Fig 7.4 although modifications have been effected by later deformation and leaching. The optimum separation of samples at Marampa can be estimated from a knowledge of the along and across-strike ore grade distribution and the degree of deformation of various parts of the orebody, after assuming an originally sedimentary deposit.

The optimum distance between the sample lines, which are aligned normal to the strike, may be determined by recourse to correlation analysis which defines the maximum separation of such sample lines as being somewhat less than the distance at which no significant correlation exists between the assay results of two series of samples taken over similar sections of the orebody. Examination of Table 7.4 indicates that significant associations exist between sample lines separated by up to 200 feet along strike. Table 7.5 (below) gives the correlation coefficients of Table 7.4 arranged in order of increasing separation of the sample lines.

Separation of Sample lines	Correlation Coefficients (r)
50 Feet	0.6676 ^{**}
	0.8068 ^{**}
	N.S. (2)
100 Feet	0.5888 [*]
	0.7441 ^{**}
	N.S. (1)
150 Feet	0.6755 ^{**}
	0.8521 ^{**}
200 Feet	0.5622

N.S. denotes not significant at the 95 per cent confidence level

* denotes significant at the 95 per cent confidence level

** denotes significant at the 99 per cent confidence level

Table 7.5 Correlation coefficients of Table 7.4 arranged in order of increasing sample line separation, syncline B.

Significant correlation at the 99 per cent confidence level occurs at sample line separations of up to 150 feet although it must also be noted that non-significant correlation coefficients exist between sample lines separated by only 50 feet. The single correlation coefficient obtained from the syncline B test block at the 200 feet separation is significant at the 95 per cent but not at the 99 per cent confidence level and correlation analysis undertaken on sample data from the Marampa mine sampling plan at separations of 200 feet indicate that non-significant correlation coefficients at this spacing are the rule rather than the exception. It is thus suggested that the present mine practice of a 200 feet separation may be too great to allow meaningful interpolation between the two sample lines. The 200 feet separation of the sample lines implies that ore grades are extrapolated to one hundred feet on either side of each sample line. This being the case it would be expected that the assay values of line A6 and A7 would be significantly associated with the assay values of line A6c which is midway between the lines A6 and A7. Examination of Table 7.5 shows that significant correlation coefficients do not necessarily occur at these separations which suggests that separations of 200 feet between the sample lines may yield independent assay values for each line.

Correlation analysis alone is not sufficient to indicate continuity of grade between sample locations or sample lines since the correlation coefficient is a measure of the relative changes rather than absolute values. Consider Fig. 7.5 which shows the assay results of a hypothetical ore sampling programme.

Correlation analysis of the assay values of the two sample lines (A and B) indicates perfect linear correlation between the two assay groups with $r = +1.0$, but it is clear that the perfect correlation coefficient is due to identical relative changes within the sample lines rather than identity of grade between the sample lines. The optimum guarantee of ore grade continuity is therefore given probably by a combination of highly significant and positive correlation coefficients and highly non-significant differences between mean grades of

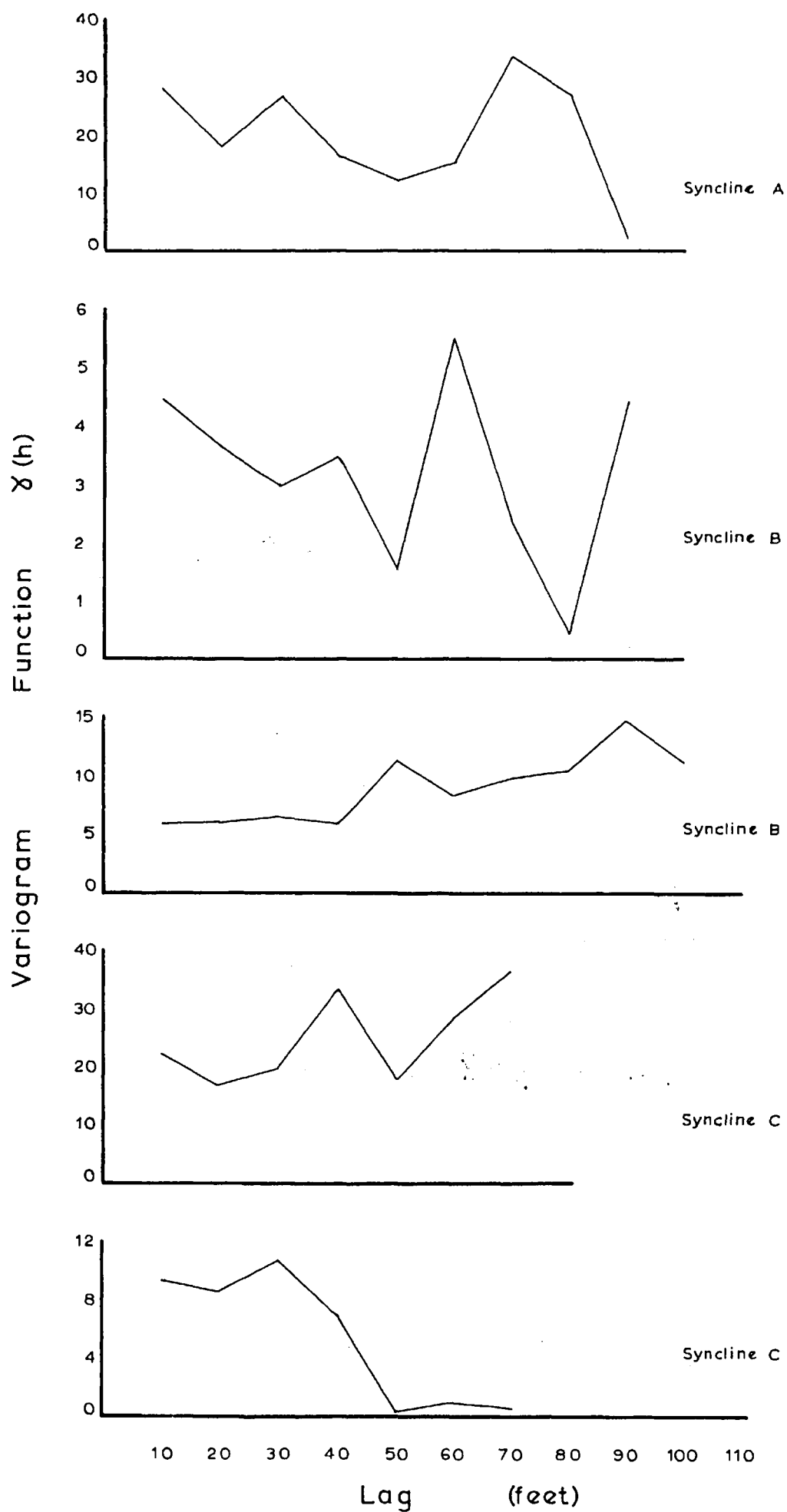
assays over the sample line.

The spacing or length of channel samples across the strike of the Marampa orebody must be determined by a combination of theoretical and practical factors and a consideration of the purpose of the sampling. Ore samples are taken for two main purposes; (i) ore reserve estimation, and (ii) grade control, and the different purposes require that the sampling techniques be somewhat different. For ore reserve estimation of the entire orebody, the actual locations of the samples need not be known since the analysis of the assay values gives a single mean value with confidence limits based on the standard deviation and the number of samples. Conversely, for purposes of grade control the actual location of the sample is a prerequisite to the analysis of the assay values.

A brief study was made of the along-strike ore grade variation within single and composite lithological units in order to examine what may be termed the fundamental grade variation which is a function of both the original depositional grade and post-depositional modifications brought about mainly by climatic effects. The fundamental grade variation is expressed by the standard deviation or coefficient of variation and provides a rough indication of the minimum grade variations which could be expected when mining the ore along the strike of the orebody.

Discrete lithological units represent deposition during periods of relatively uniform chemical and physical conditions. Changes in these conditions favour changes in lithology or cessation of deposition and it is expected that the grade variations occurring within a single lithological unit would be significantly less than grade variations occurring between composite lithological units, e.g. grade variations between across-strike channel sample assay values.

A number of single and composite lithological units were channel sampled along strike at intervals of ten feet. The deformation of the Marampa orebody tends to mask or destroy recognizable marker horizons and in practice it is difficult



VARIOGRAMS — ALONG STRIKE

Fig. 7-6

to trace a single lithological unit for more than 100 feet along strike and the apparent thickness of these units varies considerably even over relatively short distances, particularly in synclines A and D.

The along-strike grade variations occurring within five single and composite lithological units are presented in Table 7.6 and graphically described by the variogram functions of Fig. 7.6. For comparative purposes the syncline B test block coefficients of variation are also presented.

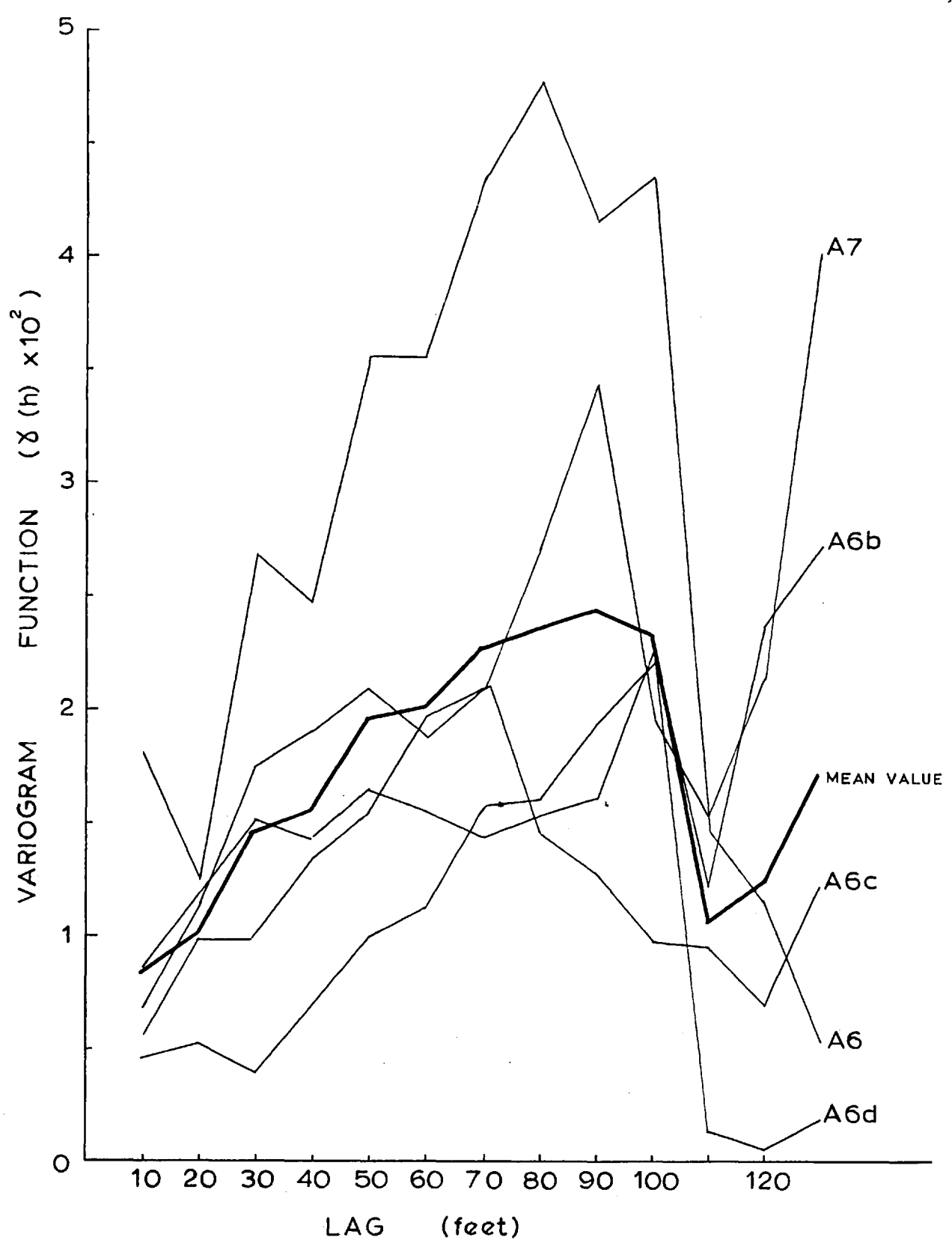
With reference to Table 7.6 and Fig. 7.6 it can be seen that the grade variations occurring within the single and composite lithological units are much less than those occurring across-strike (see also Fig. 7.7). The variograms constructed from the along-strike assay data yields random variogram models with pronounced nugget effects which suggests that, for the statistical analysis of along-strike sampling data, the use of standard statistical techniques based on probability theory is justified. In this context it is interesting to point out that random variograms arise from small random variations in high grade mineralization such as occur within single lithological units, as well as from extreme variations such as occur in very low grade mineralization with high nugget effect (Blais and Carlier, 1968).

The variograms of Fig. 7.6 should be compared with those of Fig. 7.7 which describe the across-strike grade variations of the syncline B test block. It will be seen that the variogram functions ($\gamma(h)$) of the across-strike grade variations are much greater than those of the along-strike grade variations and this is taken as further confirmation of the marked grade anisotropy of the Marampa orebody.

The variograms of Fig. 7.7 are best described as transitive variograms, the high nugget effect being a function of the sample size. The variogram function (mean value, Fig. 7.7) increases steadily to a maximum at a lag of about 90 feet and at distances greater than 90 feet the values of the variogram function appear to be random. According to Serra (1966)

Syncline	Mean Grade % Fe	Range Fe %	Std.Dev.	Coeff. of Var. %	Distance feet	Remarks
A	44.9	11.9	3.20	7.1	90	Composite lithological unit
B	50.8	3.6	1.31	2.6	90	" " "
B	50.5	5.3	1.95	3.9	100	Single lithological unit
C	41.3	8.7	3.36	8.1	70	" " "
C	50.6	5.8	1.89	3.7	70	" " "
B	43.4	Syncline B test block		12.4		Average of 14 along-strike traverses.
B	43.4			20.1		Average of 5 along-strike traverses

Table 7.6: Results of along-strike sampling of single and composite lithological units.



VARIOGRAMS FOR SAMPLE TRAVERSES, SYNCLINE B

Fig. 7.7

independent sampling data is obtained when sampling intervals are at least 95 per cent of the variogram range, in this case about 90 feet. Sampling of the Marampa orebody at across-strike intervals of about 90 feet may yield independent sample assay data adequate for the estimation of the mean grade of the orebody but would not be of any use for grade control purposes and it is considered that standard statistical techniques are best suited for the analysis of the Marampa grade data.

(c) Variations in estimated mean grade.

(i) Variations due to differential sample line spacing.

The Marampa ore sampling programme is carried out on an elongated rectangular grid 200 feet (along strike) by 10 feet (across strike) and it is apparently assumed that ore grades at one sample location can be safely extrapolated along-strike one hundred feet on either side. As noted earlier, this assumption is based on valid geological grounds and is statistically supported by the ANOVA of sampling data.

One aspect of grade control and ore reserve estimation is the determination of the optimum sample spacing, which is that spacing or density of samples which gives the optimum information at least cost or least number of samples.

The assay values of the five lines of channel samples on the syncline B test block may be utilised to investigate what differences, if any, are given by various combinations of between- and within-sample line assay values. The mean grades given by the various combinations are compared with the mean ore grades given by the two end-members of the five sampling lines, i.e. lines A7 and A6. These two lines coincide with the Marampa planning grid lines used for collection of samples for purposes of ore reserve estimation. Table 7.7 (overleaf) gives the mean and standard deviation of assay values for various combinations of the five sample lines.

	A7	A6d	A6c	A6b	A6
A7	41.59 [#] 8.803 ^{##}	42.28 8.339	43.07 8.029	43.02 7.968	42.76 10.325
A6d	-	42.96 8.121	43.39 8.189	43.70 7.503	43.44 9.995
A6c	-	-	43.83 8.540	44.14 7.698	43.88 10.149
A6b	-	-	-	44.45 7.061	44.19 9.588
A6	-	-	-	-	43.92 11.875

= mean grade ## = standard deviation

Table 7.7: Mean and standard deviation of assay values for various combinations of sample lines.

The values of the two comparison lines, A6 and A7, are a mean of 42.76 per cent Fe and a standard deviation of 10.325.

The line A6c is midway between the two end lines and would on geological grounds, be expected to have a mean grade similar to that of the two comparison lines and this is confirmed by a Students t test carried out on the relevant assay values.

The mean grade and standard deviation of the entire test block, given by all five sample lines, are 43.35 per cent Fe and 8.817 respectively. A Students t test indicates that these values are not significantly different from those given by the two end lines alone and therefore, in this area of the orebody at least, sampling along lines separated by 50 feet would not yield significantly different values from those given by sampling on lines separated by 200 feet, and would thus be unnecessary.

The standard deviation of groups of sample assay values

taken from lines separated by increasing distances indicates that the standard deviation decreases with decreasing sample line separation (Fig 7.9) but that little significant decrease occurs below a separation of about 150 feet.

The data used to draw up Fig 7.9 was taken only from the test block on syncline B and cannot claim to be representative of the entire orebody. It does however, tend to support what is intuitively recognized as being an inherent feature of ore mineralization and that is; the difference in grade between any two samples tends to be directly proportional to the distance between the samples. This concept may be more easily visualized by considering the two extremes of the sample interval scale. Two samples taken from within the same ore mineral grain will have essentially identical grades whereas the grades of two samples, one taken from within an orebody and one taken from outside the orebody, will have very little similarity. Certain conclusions concerning ore grade distribution may be drawn from the changes which occur in the standard deviation (Fig. 7.9) with increasing separation of sample lines. Matheron (1963) has based his geostatistical theory on the hypothesis that classical statistics, which are based on random sampling, are not usually valid for the statistical analysis of ore bodies where the samples are neither random nor independent but are often correlated over various distances.

(ii) Variations due to differential sample spacing

The channel sample assay data of Table 7.2 was used in order to determine the variations in mean grade arising as a result of a wider separation of samples along the length of the sample lines. Two sample separations were considered:

(a) Channel samples at 20 feet intervals. Alternate sample assay values yield the following means and standard deviations. (Table 7.8 overleaf)

		$\bar{X}$	s.d.
level	A	47.0	7.0
	B	45.0	6.5
	C	43.0	6.0
	D	41.0	5.5

Fig. 7.8

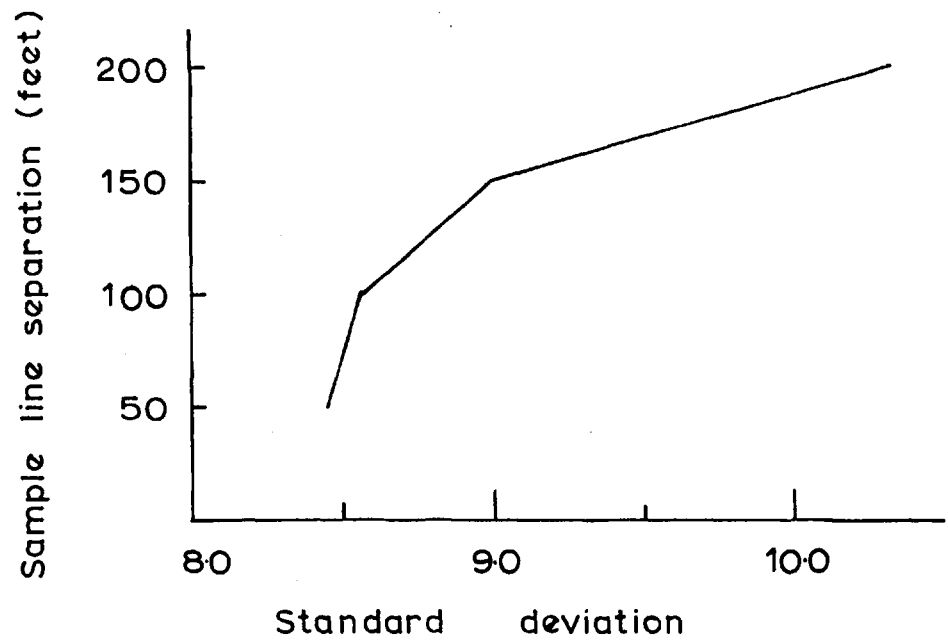


Fig. 7.9

Sample Line	Mean (Per Cent Fe)		Standard Deviation		N
	a	b	a	b	
A6	47.51	40.33	6.633	15.21	7
A6b	45.17	43.73	7.591	7.014	7
A6c	41.51	46.14	10.017	6.723	7
A6d	43.96	41.96	4.521	10.960	7
A7	40.41	42.77	10.472	7.417	7
TOTAL	43.71	42.99	8.078	9.604	35
OVERALL	43.35		8.81		70

Table 7.8: Mean grades and standard deviations of channel sample assay data; 20 feet sample separation, syncline B.

Two means and standard deviations exist with the 20 feet sample separation because there is two possible starting points for ten feet samples at 20 feet intervals.

Relatively large variations exist between the two means for some sample lines but in these cases the difference in mean grade is almost invariably caused by one or more extremely low assay values which indicate that the sample consisted mainly of quartz-muscovite schist with little hematite. The inclusion of low assay values gives rise to large standard deviations and application of the Student t distribution test to the values has indicated that no significant differences occur between the mean values calculated using the two sets of assay data.

(b) Similar tests were carried out with sample separations of thirty feet and, again, no significant differences were detected between the mean grades although mean grade differences of greater than 4.0 Fe per cent occurred.

(iii) Summary

An increase in channel sample separation to 20 or 30 feet

does not result in significant changes in mean grade and it appears feasible that mine sampling at these sample separations would give valid results. Due to the low unit cost of channel sampling however, it is suggested that, instead of increasing the sample separation so as to include an unsampled ten feet interval between channel samples, the channel samples should be taken over a twenty feet section rather than the ten feet section as at present. In this way, the same grade information would be available except that each sample would relate to a twenty feet section which is about the absolute minimum width to which the mine shovels could possibly work, and the amount of sample assaying is decreased by fifty per cent. Sampling over twenty feet sections also reduces the range of grades by incorporation of a larger amount of across-strike ore thus diluting or smoothing out the effects of anomalous bands of iron-rich or iron-deficient ore. When sampling across-strike in a sedimentary or bedded ore deposit, the range of assay values, and hence the standard deviation, is a function of sample size and the across-strike sample length should always be greater than the width of discrete ore bands or zones except when the sample size or length is lithologically controlled rather than based on an arbitrary sample length. At Marampa, where the complex interaction of tectonic events have obscured the original bedding and distinctive marker horizons do not exist for more than one hundred feet along strike, it is not practicable to sample to lithological boundaries and it appears probable that increasing the sample length to 20 or even 30 feet and decreasing the sample line separation to 100 feet would yield more meaningful results for both ore reserve estimation and grade control, than the present sampling grid.

(D) STATISTICAL ANALYSIS OF VERTICAL ORE GRADE VARIATIONS
AT MARAMPA

(a) Introduction

Assay plans of the various ore synclines prepared during the earlier stages of mining on Masaboin Hill indicate that average ore grades from samples taken towards the original

topographic surface of the orebody were significantly higher than the average ore grade encountered during present mining operations and it is suggested that leaching of the orebody by meteoric water has resulted in the enrichment of the surface ore by abstraction of silica. Earlier assay plans indicate average ore grades of greater than 55 per cent Fe as against the present average ore grade of less than 50 per cent Fe and this decrease in grade would appear to support the hypothesis of silica abstraction by leaching. The leaching effects are expected to decrease with increasing depth below the original surface and this, in turn, is expected to result in the ore containing increasing amounts of silica with a concomitant decrease in average ore grade and an increase in ore hardness.

Three major mining problems will eventually arise if the average ore grade and ore hardness alter as predicted.

(a) The lower grade ore will tend to yield lower grade concentrates as predicted by correlation and regression analysis of beneficiation data (Chapter IV) and this will have adverse effects on the sale of the concentrates with a probable decrease in potential profit, if the detrimental effects of the lower grade mill feed cannot be neutralised.

(b) Mining costs will increase because of the additional ore which will require beneficiation to produce a given amount of concentrate. The cost increase will mainly be in the form of capital costs and depreciation of additional earthmoving equipment. It is estimated, for example, that with the present facilities and concentrate production, a decrease in average ore grade from 47 to 40 per cent Fe would require an increase in hilltop ore production of at least 2000 tons per day (Fig.7.10).

(c) An increase in ore hardness implies that radical changes will become necessary in the method of mining and in ore beneficiation. At present the bulk of the ore is mined directly by face shovels and little blasting is necessary. An increase in ore hardness will necessitate the use of large scale blasting which would further increase the mining costs by involving more operations in the handling of the ore and by

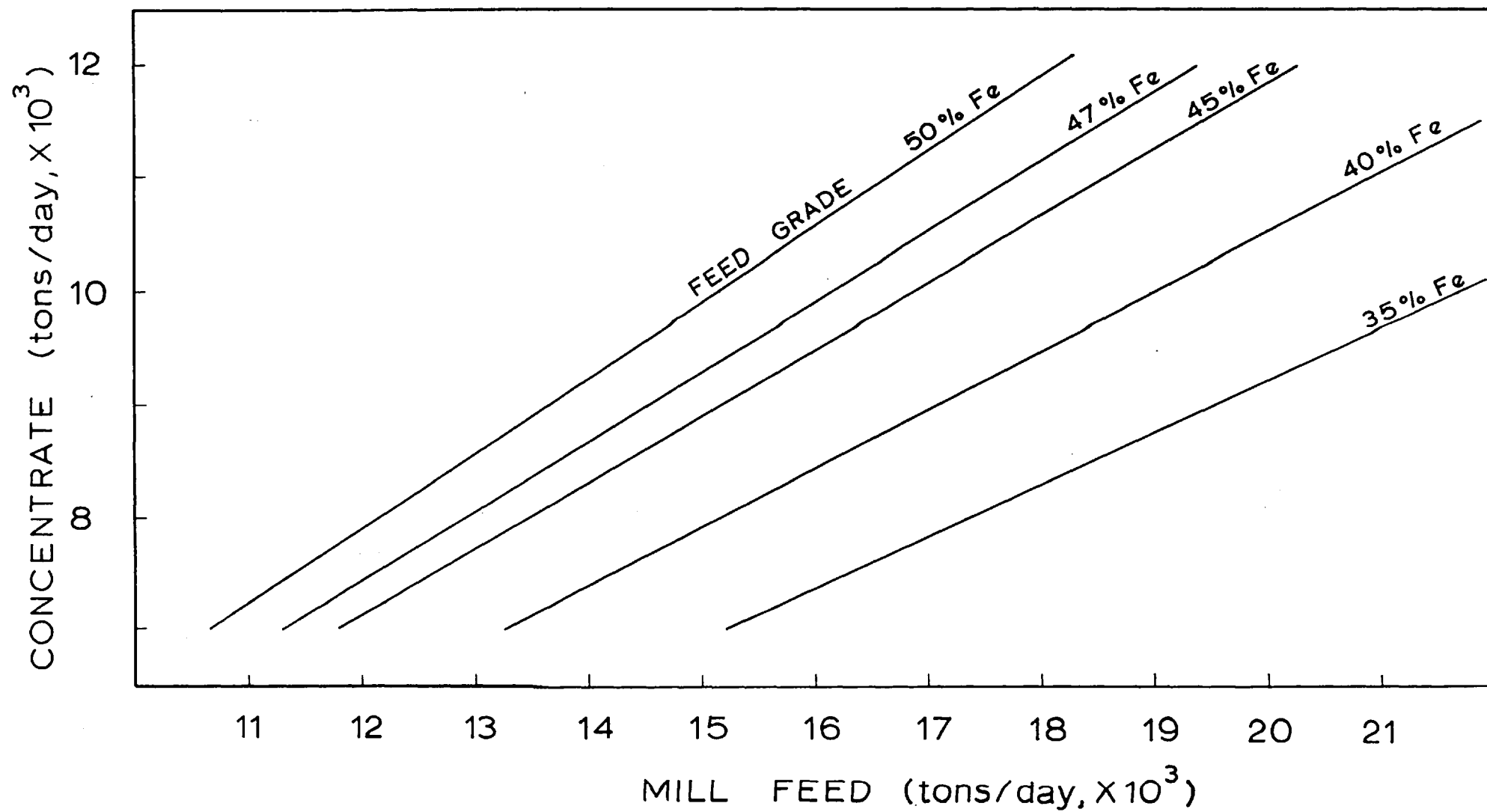


Fig. 7-10

expenditure in the form of blasthole drills, explosives and crushing equipment. In addition, greater amounts of ore will require crushing, liberation of hematite from gangue will become increasingly difficult and two stage size reduction may become essential for all ore instead of for the small proportion as at present. Spiral beneficiation of heavy minerals is sensitive to the degree of liberation of the valuable minerals and variations in hematite liberation is expected to have a detrimental effect on iron recovery and concentrate grade.

Without a knowledge of the vertical variations in ore grade, mine planning is hindered, ore reserve estimation becomes less confident and inferred ore reserves may be overvalued, all these points having economic implications that may adversely effect the potential mine profit.

An increase in ore hardness, which is predicted from the silica leaching hypothesis, is becoming increasingly evident in various parts of the orebody as mining progresses and this may represent the first stages of widespread ore hardness at depth.

Vertical variations in ore grades may be examined, in a gross manner at least, by testing for statistically significant differences between ore grades of groups of samples collected from different mining levels, although at Marampa this is complicated by the fact that ore grades are only available over vertical distances of about thirty feet due to the absence of systematic sub-surface grade information.

Assay results from samples of two types have been statistically examined in order to determine whether or not ore grades decrease, as predicted, with increasing depth below the original topographic surface of the orebody. The two sample types are;

- (a) groove (channel) samples
- (b) auger drill samples

The channel sample grade information was taken from the results of four sampling traverses on the 560 and 530 feet mining levels of synclines A, B and C and was selected so that

each sample location on the 560 feet level corresponded to a sample location on the 530 feet level. In this way it was hoped that true vertical grade variations could be examined with the effects of along-strike and across-strike grade variations minimised. Errors are introduced by the fact that the samples on the two mining levels are not necessarily stratigraphically correlated, except where dips are vertical, because of the complex and variable structure of the orebody. Collection of samples which were stratigraphically correlated over the thirty feet height of a single mining bench was attempted but proved to be impracticable. The distribution of the channel samples over three synclines and four traverse lines (strike length of about 800 feet) is considered to be sufficient to give a representative coverage of the entire orebody and ore grade variations detected from these samples may be extrapolated to describe the overall vertical ore grade variations.

The auger drill hole sample assays were taken from the results of a drilling programme on syncline A which was undertaken to obtain sub-surface ore grade information after realization of the fact that channel sampling of relatively flat-lying ore on syncline A would not necessarily provide an efficient estimate of the ore grade at depth. The drilling was carried out on a rectangular grid, 100 feet by 50 feet, the longer dimension being measured along strike and the holes were bottomed at about 33 feet. Seven samples were recovered from each drillhole; a three-foot sample from the interval 0 to 3 feet, and six five-foot samples from the interval 3 to 33 feet, each five-foot sample corresponding to the length of a single auger drillrod. For the purpose of testing for vertical ore grade variations only the first sample (0 to 3 feet) and last sample (28 to 33 feet) assays were considered.

(b) Statistical tests for examination of vertical ore grade variation

Three statistical tests were carried out on the assay data in order to determine whether or not the observed vertical ore grade variations were statistically significant.

(a) A Students t distribution test (Chapter II) for significant differences between the means of the channel sample assays from the two mining levels and for significant differences between the means of the ore grades from the two groups of auger-drilling samples.

(b) A Students t distribution test for significant differences in paired ore grades of the channel samples from the two mining levels and from the paired ore grades (top and bottom sample assays) from the auger-drilling samples. Paired ore grades consist of either;

(i) Ore grades of channel samples taken from similar locations on the 560 and 530 feet levels, or

(ii) Ore grades of the top (0 to 3 feet) and bottom (28 to 33 feet) samples from a single auger-drillhole.

(c) A Wilcoxon's signed rank test for significant differences in the paired ore grades of the auger-drilling samples.

The Null Hypotheses proposed for each test are that no significant differences exist between the means of ore grades from the two levels or that no significant difference exists between paired ore grades.

(c) Statistical analysis of vertical ore grade variation

(i) Channel samples - difference between means

The mean assay values of the channel samples from the two mining levels are (Table 7.9)

Level	Mean (% Fe)	Std.Dev.	N
560 feet	47.7	6.22	246
530 "	46.8	6.34	246

Table 7.9: Mean and standard deviation of channel sample assays from two mine levels.

The statistical significance of the observed difference of 0.9 Fe per cent between the two groups of assay values is tested by a Students t test using equations 2.2 and 2.1.

The data in Table 7.9 gives:

$$\begin{aligned}\bar{X}_1 &= 47.7 & \bar{X}_2 &= 46.8 \\ n_1 &= n_2 & &= 246 \\ S_p &= 6.28\end{aligned}$$

Substituting in equations 2.2 and 2.1 gives:

$$t_{490} = 1.59$$

The critical value for t_{490} at the 95 per cent confidence level is

$$t_{490} = 1.96$$

$$\therefore t_{\text{calculated}} < t_{\text{critical}}$$

and the Null Hypothesis of no significant difference between the means of the ore grades for samples taken from the two mining levels is accepted. On the basis of these sample assays it is concluded that there is no significant decrease in mean ore grade over the thirty feet height of a single mining bench although there does appear to be a tendency towards such a decrease, the mean grade for the lower bench being less than the mean grade for the upper (560 feet level) bench. It is pertinent at this stage to point out that although a statistically significant decrease in mean ore grade has not been shown to exist over the height of a single mining bench, this does not preclude a significant decrease in mean ore grade over greater vertical distances or over similar thirty feet distances at a greater depth below present mining operations. Calculations indicate that a difference of greater than 1.1 Fe per cent between the two means would approach the level at which a statistically significant decrease in mean ore grade could be implied.

(ii) Channel samples - differences between paired ore grades

A Students t test is also used to determine whether or not statistically significant differences occur between the ore grades from the two mining levels by examination of the differences in ore grades from samples taken from similar locations on the two mining levels. The Null Hypothesis is tested by

equation 2.3.

The relevant values are;

$$\bar{d} = 0.82$$

$$S_d = 6.91$$

$$N = 246$$

Substituting in equation 2.3 gives;

$$t_{245} = 1.86$$

The critical value for t_{245} at the 95 per cent confidence level is 1.96.

$$\therefore t_{\text{calculated}} < t_{\text{critical}}$$

and the Null Hypothesis of no significant differences between the ore grades from the two mining levels is accepted.

Calculations suggest that an average difference ($\bar{d}$) between the paired ore grades of 0.9 Fe per cent would be sufficient to indicate a statistically significant difference at the 95 per cent confidence level.

(iii) Auger drillhole samples - difference between means

The iron assays of the top (0 to 3 feet interval) and bottom (28 to 33 feet interval) samples from the auger drilling programme were compared by means of a Students t test.

$$\bar{X}_1 = 41.61 \text{ per cent Fe} \quad \bar{X}_2 = 39.51 \text{ per cent Fe}$$

$$s_1^2 = 46.24 \quad n = 36 \quad s_2^2 = 20.61$$

Substituting in equations 2.2 and 2.1 gives; $t_{70} = 1.54$

The critical value of t_{70} at the 95 per cent confidence level is 2.0

$$\therefore t_{\text{calculated}} < t_{\text{critical}}$$

and the Null Hypothesis of no significant difference between the grades of the two series of auger drillhole samples is accepted.

It is to be noted however that there is an apparent mean ore grade decrease of 2.1 Fe per cent over the height of a single bench (41.61 per cent Fe for mean of upper sample assays against a mean of 39.51 per cent Fe for the lower sample assays).

Thus, although a statistically significant decrease in mean ore grade has not been proved, there does exist a tendency towards a decrease in mean ore grade with increasing depth and it should

also be noted that positive grade variations, indicating a decrease in ore grade with increasing depth, outnumber the negative deviations, which suggest a grade increase with increasing depth, by 22 to 14, and this again suggests a tendency towards a decrease in mean ore grade with increasing depth below the original topographic surface of the orebody.

(iv) Auger drillhole samples - difference between paired ore grades (a)

The differences between the assay values of the upper and lower samples from each drillhole are examined by the Students t test (equation 2.3).

The relevant values are;

$$\begin{aligned}\bar{d} &= 2.10 \\ s_d &= 7.80 \\ n &= 36\end{aligned}$$

Substituting in equation 2.3 gives;

$$t_{35} = 1.62$$

The critical value of t_{35} at the 95 per cent confidence level is 2.03.

$$\therefore t_{\text{calculated}} < t_{\text{critical}}$$

and the Null Hypothesis of no significant difference between the grades of the top and bottom sample from each auger drillhole is accepted.

(v) Auger drillhole samples - difference between paired ore grades (b)

Wilcoxon's Signed Ranks test was applied to the paired sample data for the auger drillhole samples in order to examine the sensitivity of the Students t test. The paired drillhole sample assay values gave the following results;

$$\begin{aligned}n &= 36 \\ R &= 227 \text{ (smaller rank sum)}\end{aligned}$$

Since n is greater than 25 the value of n and R are substituted in equation 2.6 to give;

$$Z = 1.67$$

This value of Z is not significant at the 95 per cent confidence

level and it is concluded that the deviations between the grades of the top and bottom samples of each drillhole could have arisen by chance. The Students t test and Wilcoxon's Signed Ranks test are shown to be compatible when applied to the same data.

(d) Discussion

A significant decrease in mean ore grade with increasing depth below the original topographic surface of the orebody has not been proved with the existing grade information but a tendency towards such a grade decrease may be inferred from both the channel sampling and auger drillhole sampling assay data as shown below.

	Upper level	Lower level	Grade decrease
	per cent Fe		Fe per cent
Channel sampling	47.7	46.8	0.9
Auger sampling	41.61	39.51	2.1

The comparison of channel sampling assay data covered a strike length of 800 feet on three of the four major ore synclines at Marampa. This coverage is considered to be sufficiently extensive to enable horizontal extrapolation of the results of the statistical tests to the entire orebody at the 530 feet level. In a similar manner, the auger drilling results may be extrapolated the length of syncline A, but probably not to other ore synclines.

The evidence therefore suggests that no decrease in mean ore grade is to be expected with increasing depth of mining at Marampa but this conclusion, while valid for the test data, is not considered to be valid for the orebody as a whole or for all future mining operations and could be misleading if accepted without qualification and further investigations.

The main objections to the results of the statistical tests is that sample assay data is only available for the

thirty feet separation of two vertically adjacent mining levels. The test data was collected from only two levels in the mine, the 560 and 530 feet levels and the Null Hypotheses of no significant decrease in mean ore grade with increasing depth can only be accepted for this thirty feet separation at these two particular levels. The fault lies in the meagre amount of grade information available, not with the power of statistical tests. Comprehensive assay plans of the orebody have only recently been produced and mill feed assay sheets prior to 1966 have been destroyed and it is thus not possible to compare quantitatively the grade changes that have taken place since mining began at Marampa.

Over a sixty feet vertical separation between the 560 and 500 feet levels, for example, a significant decrease in mean ore grade may be detectable and it may happen that while a significant difference in mean ore grade between any thirty feet separations may never be detected, the cumulative ore grade difference between successive 30 feet levels may indeed be very significant. This is illustrated in Fig. 7.8 which summarises the results of a hypothetical sampling programme carried out on four mine levels each having a thirty feet vertical separation. The mean grade ($\bar{X}$) and standard deviation (S.d.) of the sample assays from each level are given.

A Students t test indicates that no significant difference exists between the mean ore grades of immediately adjacent levels but that a statistically significant difference exists between the means of ore grades of levels separated by sixty feet or greater. Such may be the position at Marampa. The tendency towards a decrease in mean ore grade with increasing depth which has been shown to exist over 30 feet heights, may be cumulative and may be statistically significant over heights greater than 30 feet.

Mining operations in various parts of the orebody, particularly in synclines B and D, have indicated that hard unleached ore is becoming increasingly common but it is not known with any degree of certainty to what depths the leached

ore extends. The hard ore encountered in present mining operations are probably "outcrops" of the completely unleached ore which is expected to be present at some depth below the present mining levels. The grade variations in the vicinity of the boundary between leached and unleached ore are unknown but both these variations and the actual position of the surface between the leached and unleached ore should be determined for purposes of long-term mine planning.

The leached/unleached ore boundary surface is expected to be irregular because of the irregularities of the original topographic surface of Masaboin Hill and location of the original (pre-mining) drainage system, and ore grades below this surface are not expected to decrease significantly with further increase of depth but are more likely to correspond with the original depositional grade of the ore with superimposed metamorphic modifications.

The changes in mining and milling practice at Marampa which would be prompted by a sudden increase in ore hardness and/or decrease in mean ore grade should stimulate investigations into determining at what depth below the present pit hard unleached ore may be expected and what the grade and milling characteristics of such ore will be. Relevant information on these points is restricted to about 130 diamond drillholes put down about 1960 for structural geology purposes. The ore recovered from most of these holes was not assayed and the holes were not drilled to a regular grid, and of the 130 drillholes only some 40 were sited in positions which would yield useful information for present mining purposes had the ore intersections been assayed. The leached ore encountered during this drilling programme failed to core effectively but a hypothetical leached/unleached ore boundary was inferred on the basis of core recovery; the ore was considered to be leached if no core was recovered and unleached when core was recovered. Unfortunately, no core from this drilling programme was retained and the milling characteristics can only be inferred.

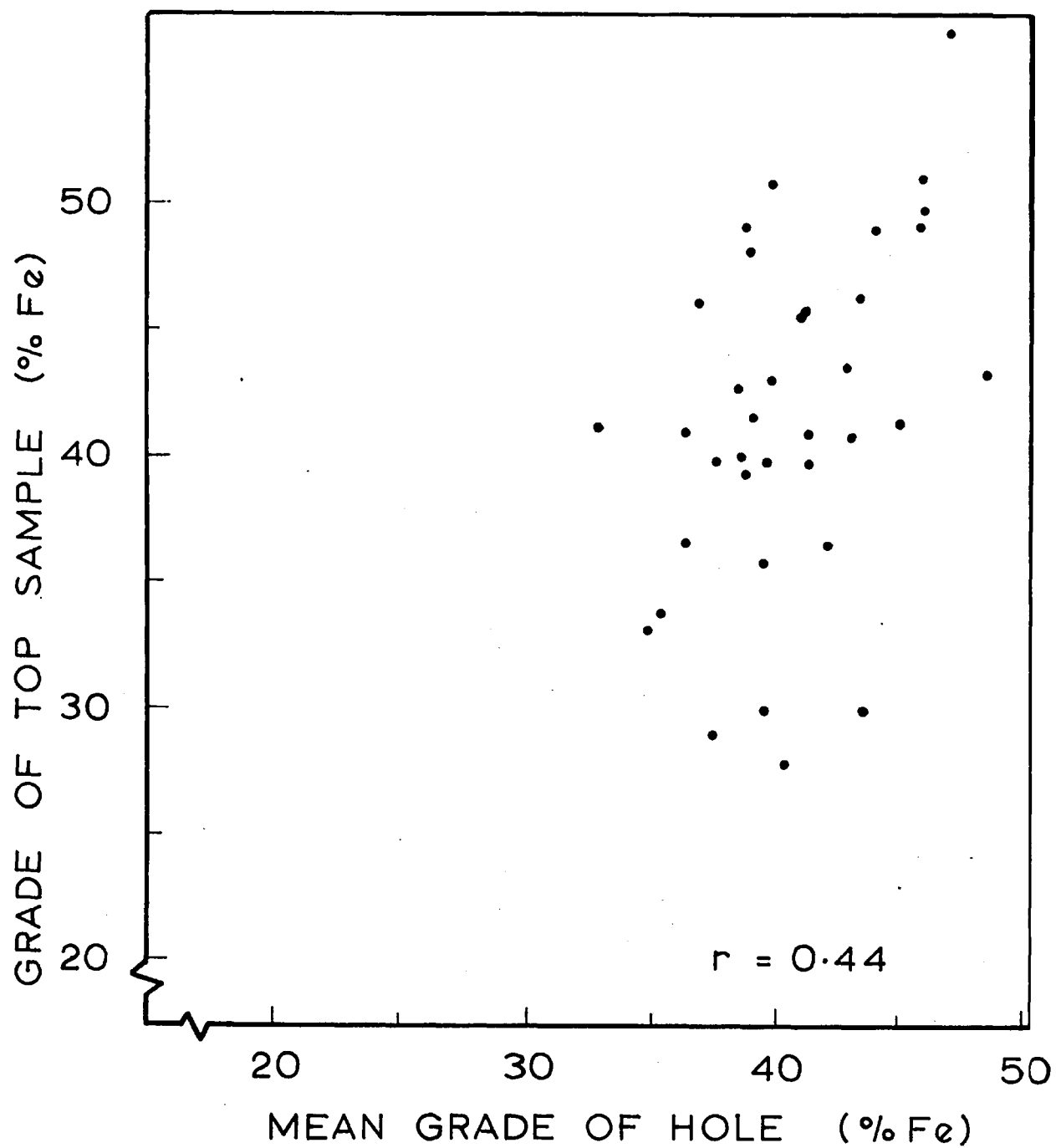
(c) Auger drilling for subsurface ore grade information

The syncline A auger drilling programme was carried out in order to determine the subsurface grade distribution in those areas of the syncline where it had been considered that the ore was relatively flat-lying and that the regular channel sampling had failed to give an efficient estimation of the sub-surface ore grade.

Correlation analysis undertaken on assay results from the drillholes indicates that the grade of the top (0 to 3 feet interval) sample of the auger drillhole has a statistically significant association with the overall mean grade of the drillhole, and this fact, together with the fact that there appears to be no significant difference between the mean grades of the top and bottom (28 to 33 feet interval) samples suggests that auger drilling over the depth of a single mining bench fails to give additional useful grade information. This may be caused by the ore not being flat-lying as originally suggested and examination of the ore faces in the vicinity of the drilling programme indicated that ore dips varied considerably and were often vertical.

The correlation analysis yielded a positive correlation coefficient (r) of 0.444 (Fig. 7.11) and a coefficient of determination of 0.197 indicating that about 20 per cent of the variation in grade of the top samples could be attributed to variations in overall grade of the drillhole samples. The correlation coefficient is significant at the 99 per cent confidence level which, for the number of samples used, has a threshold value of 0.418.

Where a significant correlation exists between the top and bottom samples of a drillhole it is suggested that continuation of such drilling serves no practical purpose and that surface channel samples will yield similar grade information and will be extremely more cost-effective. Where the top and bottom samples are not significantly correlated then it may be assumed that the depth of the bottom sample exceeds the "area" of influence of the top sample and the grade of the top sample



Auger drilling assay values

Fig. 7-11

cannot then be extended to the depth of the bottom sample. In such circumstances channel sampling would not be sufficient and sub-surface sampling would be necessary.

(B) CONCENTRATE GRADE IMPROVEMENT BY REDUCTION IN FEED GRADE VARIATION

One of the problems related to concentrate grade variation is the variation in the grade of the mill feed entering the mill and the variations in the shape of the hematite grains on the spirals. Sudden grade variations or changes in the shape of the hematite grains passing on to the humphreys spirals causes fluctuations in concentrate grade (and iron recovery) because the mill operating conditions, including feed rate, wash water rate and orientation of withdrawal port splitters, have been designed so as to treat an "average" type of ore and these conditions will not at all times be the optimum conditions for treating the range of ore types which occur at Marampa.

De Gast (1968) pointed out that "controlled uniformity of the mill feed grade invariably improves metallurgical recovery and milling costs, as sudden fluctuations will not occur in the materials balance". This statement holds true for Marampa particularly when it is recalled that the concentrate grade is directly related to the feed grade (Chapter IV).

Several approaches are open to the production of a reasonably constant grade mill feed but hematite from different areas of the orebody are characterised by significantly different grainshapes. To recapitulate, hematite from synclines A and D is coarse and micaceous whereas hematite from synclines B and C is finer-grained, equidimensional and is commonly annealed. As pointed out in Chapter IV the shape of the valuable minerals is a critical factor in spiral beneficiation and even if a constant feed grade was attained by careful blending, it is considered that the significant hematite grainshape differentials would ensure that optimum concentrate grade and iron recovery would be difficult to achieve.

A more effective approach to achieving a reasonable uniformity of feed grade and hematite grainshape may be to mine and treat ore from areas of fairly constant grade, e.g. blocks of ore with sample assay values between, say, 40 and 50 per cent Fe, from within a single ore syncline or from two synclines with similar characteristics, i.e. mine from only synclines A and/or D or from only synclines B and/or C but not from all synclines simultaneously. Mining from synclines having similar ore characteristics has the effect of minimising the between-syncline grainsize and grainshape differential and the mining of blocks of ore of fairly constant grade minimises the effect of feed grade variation. This may be illustrated with reference to Figs. 7.12 to 7.14 inclusive which describe the following;

Fig. 7.12: Isopleth plans of Fig. 6.1 with isopleth values of 30, 40 and 50 per cent Fe.

Fig. 7.13: Isopleth plans of the 3-term moving means of Fig. 6.1 assay values.

Fig. 7.14: Isopleth plans of the 5-term moving means of Fig 6.1 assay values.

If it is assumed that the interpolation of assay values between the 50 feet separation of the sample lines is valid then it can be seen (Fig. 7.12) that the trend of the mineralisation, as expressed by least grade variation in a particular direction, is roughly normal to the sample traverse lines and parallel to the strike of the orebody. This was suggested by analysis of variance (this chapter) and is here confirmed.

Mining of blocks of ore characterised by similarity of grade could be carried out in three divisions based on ore grade;

- (a) Blocks of ore with mean grade greater than or equal to 50 per cent Fe.
- (b) Blocks of ore with mean grade greater than or equal to 40 per cent Fe and less than 50 per cent Fe.
- (c) Blocks of ore with mean grade greater than or equal to 30 per cent Fe and less than 40 per cent Fe.

Figs. 7·12 – 7·14

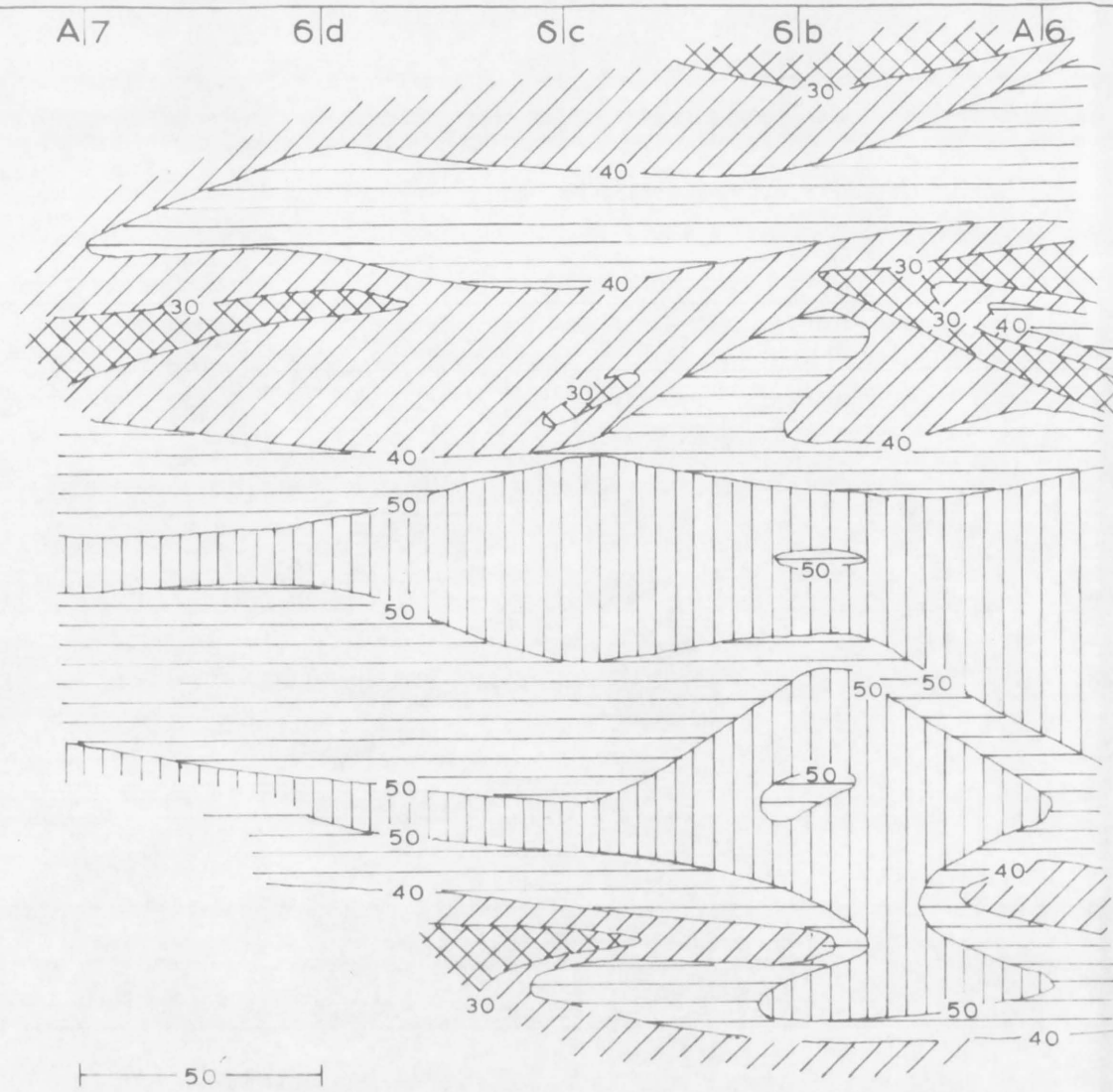


Fig. 7-12

INDIVIDUAL ASSAYS

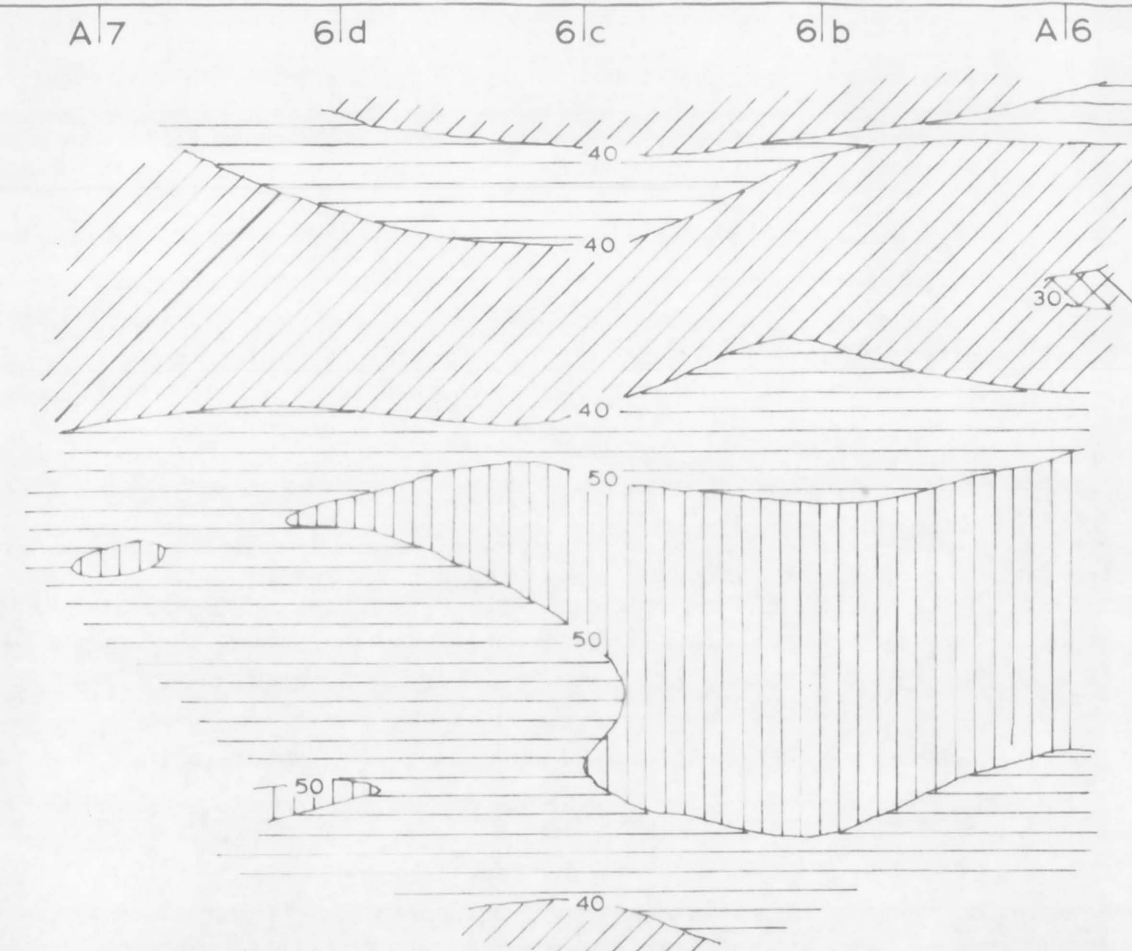


Fig. 7-13

3 TERM MOVING MEAN

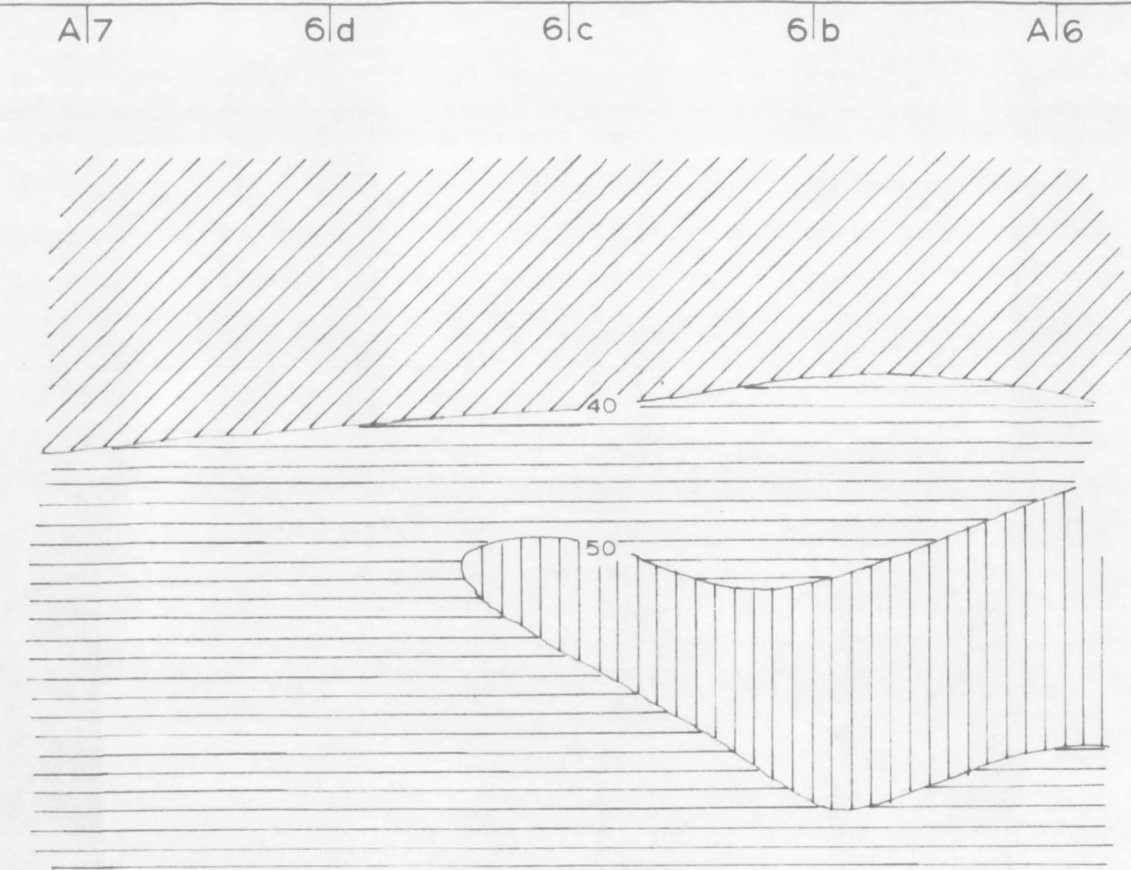
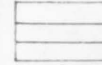


Fig. 7-14

5 TERM MOVING MEAN

Legend

-  + 50 per cent Fe
-  40-50 per cent Fe
-  30-40 per cent Fe
-  - 30 per cent Fe

TEST BLOCK , SYNCLINE B

Channel sample data (upper)
Posthole sample data (lower)
raw data from Fig. 6-1

Blocks within the ore with mean grade of less than 30 per cent Fe to be rejected as waste.

These particular grade divisions are arbitrary and could be altered if experience of mining and milling such blocks dictated otherwise.

Mining of blocks of ore to grade isopleths based on individual assay values (Fig. 7.12) would appear to be impracticable because of the narrow separation of and tortuous boundaries between, the various blocks but a mining plan following the isopleths of the 3-term or 5-term moving means (Fig. 7.13 and 7.14) would probably prove effective in reducing the grade variations of mill feed although such a scheme would also involve major reorganisation of mining and milling practice as will be later explained.

A measure of relative grade variation is given by the coefficient of variation which is expressed as a percentage and is given by;

$$v = \frac{100s}{\bar{X}}$$

where,

s = standard deviation of sample values

$\bar{X}$ = mean value of sample values

Calculation of the coefficient of variation of the various mine and mill products yields the following results (Table 7.10).

	Mean grade per cent Fe	Standard deviation	Coefficient of variation
Channel samples	47.0	6.72	14.3
Mill feed samples	47.8	3.1	6.5
Concentrate samples	64.5	1.0	1.6
Concentrate train	64.5	0.8	1.2
Concentrate ship	64.5	0.32	0.5

Table 7.10: Mean, standard deviation and coefficient of variation of various mine and mill products at Marampa.

These calculations indicate that blending processes, both deliberate and accidental, are active throughout the mining and milling operations. Blending takes place on the mine by mining of ore from different locations and subsequent combination on the conveyor belts and by active mixing of ore by bulldozer at the entrance to the ore hoppers. Further blending takes place in the mill feed bunkers, at the concentrate dewatering classifiers and at the concentrate stockpile. Reclamation of concentrate from the stockpile affords further blending and final blending takes place at the port stockpile and during loading operations.

The effects of blending can be seen in the steady decrease in standard deviation and coefficient of variation and it is considered that mining of blocks of ore of fairly constant grade from within synclines of similar ore characteristics would reduce still further the variations occurring in feed grade and concentrate grade.

The mean, standard deviation and coefficient of variation for the various blocks of ore based on isopleths of 30, 40 and 50 per cent Fe are presented in Table 7.11 for both the 3 and the 5-term moving means and also for the individual sample assays.

	Ore grade intervals	Mean % Fe	Standard deviation	Coefficient of variation
3-term moving mean blocks	> 50% Fe	51.8	2.38	4.6
	40.0 to 49.9% Fe	45.6	5.32	11.7
	< 40.0% Fe	35.1	7.96	22.7
5-term moving mean blocks	> 50% Fe	52.2	2.33	4.5
	40.0 to 49.9% Fe	46.1	6.28	13.6
	< 40.0% Fe	36.4	7.80	21.4
Individual samples		43.3	8.70	20.1

Table 7.11:

The standard deviation of the individual sample assays is seen to be greater than that for any of the blocks of ore based on 3 or 5-term moving means although the relative variability, as expressed by the coefficient of variation, indicates that the -40 per cent Fe blocks of ore tend to be slightly more variable in grade than the entire block taken as a whole. The large difference between the standard deviations of the various grade blocks is simply a reflection of the range of values occurring in each block; the samples making up the -40 per cent Fe blocks have assay values ranging from 15.0 to greater than 40 per cent Fe and so have the largest standard deviation whereas the samples falling within the +50 per cent Fe blocks have a much smaller range of values and, consequently, a much smaller standard deviation.

The standard deviations and coefficients of variation of the blocks of ore based on 3 and 5-term moving means are essentially similar (Table 7.11) and because it would be easier to mine to the plan outlined by the 5-term moving mean, it is suggested that a mining plan based on the 5-term moving mean isopleth plan (Fig. 7.14) would effectively reduce the grade variations of the mill feed. Mining to this plan would involve working that ore with mean grade greater than 50 per cent Fe and setting the mill variables (feed rate, wash water rate, orientation and number of withdrawal port splitters) to concentrate ore of this type remembering that the grainshape variations have been minimised by mining only one ore type at any one time.

When mining of the +50 per cent Fe ore is completed the 40 to 49.9 per cent Fe blocks would be mined and the mill variable altered accordingly, and eventually the -40 per cent Fe ore would be mined and treated.

Mining and milling the Marampa ore to a plan based on this general scheme should be practicable but would involve extensive Humphreys spiral pilot plant tests on ore from each of the four synclines and expansion of the blending facilities. Mill feed from within the +50 per cent Fe blocks of ore would yield significantly higher grade concentrates than mill feeds

from the blocks of ore with mean grade less than 40 per cent Fe although systematic alteration of the mill operating conditions would probably go some way towards lessening the effects of the large feed grade differences. The blending necessary if a scheme of this nature was undertaken may best be carried out along similar lines to the blending method employed at Sept Iles by the Iron Ore Company of Canada (Kirkland and Waring, 1964) in which trainloads of ore are made up by assembling a number of ore wagons of varying grade (each wagon or small group of adjacent wagons being samples). At the loading port shipments of the required tonnage and grade are made up by choosing from the available wagons those which, when combined, give the required grade. A modification of this scheme at Marampa might involve stockpiling the different grade concentrates in separate areas of the stockpile and, when loading the trains, to combine the different grade concentrates in proportions calculated to give the required grade.

In connection with the effect of feed grade variations on concentrate grade it is relevant to comment briefly on the Marampa grade control policy and the way in which the decrease in concentrate grade has assumed major proportions.

Prior to the mid-1960's grade control at Marampa appears to have been neglected but for a very good reason. Grade control was simply not a problem and the mine had little trouble in producing a consistently high grade concentrate. The ore was almost completely friable and of high grade and the mill experienced no difficulty in treating the ore. In recent years however, as the ore became harder, mill feed grade decreased and the concentrate production increased, the problem of maintaining a high grade concentrate has increased and has resulted in a significant decrease in concentrate grade.

During 1967 and early 1968 three S.L.D.C. memoranda concerning grade control were issued and from these memoranda it is apparent that it was recognized that fluctuations in concentrate grade were associated with, among other factors, feed grade fluctuations and various suggestions were put forward

for reducing the feed grade variations. These included suggestions concerning methods of mining, milling and blending and various investigations were to be made into ore characteristics and their effect on concentrate grade.

Two of the memoranda, entitled Grade Control Policy (S.L.D.C. reference, Mining, Policy and Planning, GM4/10-65) and Concentrate Grade Control (S.L.D.C. reference, Mining, Policy and Planning, GM4/10-66) gave the 1968 grade control objectives as;

- (a) Mean iron grade of mill feed to be equal to mean iron grade of proved ore reserves.
- (b) Mean silica grade of mill feed to fall within the range 23 to 26 per cent SiO_2 .
- (c) Mean concentrate grade to be 64.75 per cent Fe and 5.75 per cent SiO_2 , the silica content to be kept within the range 5.0 to 6.5 per cent SiO_2 .
- (d) Silica content of concentrate shipments to be kept within the range 5.5 to 6.0 per cent SiO_2 .

Although the sale of the concentrates is based on the iron and moisture contents which are naturally to be kept as favourable to the Company as possible, it is the silica content which is of most interest to the consumer and it is the silica content which was to be the most closely controlled constituent of the concentrates.

Few of the investigations recommended in the Concentrate Grade Control memorandum appear to have been undertaken or, if they were undertaken, the results have been inconclusive or not acted upon. This conclusion is arrived at by examination of the mill feed grade variations, the concentrate grade variations and the variations in the concentrate shipments grades during the period January 1967 - June 1968. In addition it is noted that during 1968 the concentrate target grade was successively reduced from 64.75 to 64.65 to 64.50 per cent Fe, i.e. the concentrate target grade was not being attained so in order to combat this problem the target grade was simply reduced rather than approaching the problem from a technical point of view

involving mining, milling and blending factors.

The variations in mill feed grade, concentrate grade and concentrate shipment grade mentioned above are;

Mill feed grade variations;

- (a) Fe; from less than 40 to greater than 60 per cent Fe.
- (b) SiO_2 ; from less than 5 to greater than 38.4 per cent SiO_2 . The silica content of the feed rarely falls within the optimum range of 23 to 26 per cent SiO_2 .

Concentrate grade variations;

- (a) Fe; from less than 60 to greater than 68 per cent Fe.
- (b) SiO_2 ; from 1.0 to greater than 8.2 per cent SiO_2 .

Concentrate shipment grade variations;

- (a) Fe; from 63.70 to 65.75 per cent Fe. (See also Table 3.3 and Fig. 3.2).
- (b) SiO_2 ; 4.28 to 7.40 per cent SiO_2 . (See also Figs. 3.4 and 3.5).

CHAPTER VIII

INVESTIGATION INTO ASPECTS OF THE ACCURACY OF ASSAYING AT MARAMPA

A. INTRODUCTION

Ore reserve estimation and grade control are most important in the economics of the mining industry and are based on an analysis of assay grades. The whole purpose in the development and operation of a mine is to realise a financial profit and at present, with mines becoming larger and capital investments becoming enormous, the importance of accurate and reliable collection, preparation and assaying of mine ore samples is assuming major proportions.

Some aspects of the analytical facilities at Maramapa have been investigated after certain anomalous assay results had been revealed.

(a) A study of 132 channel sample assays made during 1968 revealed that, when the iron determinations were corrected to hematite, greater than 42 per cent of these sample assays totalled more than 100 per cent after the hematite, manganese, silica and alumina values were summed. e.g. Fe_2O_3 (68.9%) + Mn (0.8%) + SiO_2 (>38.4%) + Al_2O_3 (3.1%) = >111.2%. A similar study of a random sample of assay results reported during 1967 indicated that less than 1 per cent of the assays exceeded 100 per cent when similarly summed.

While it is realised that the iron may be present in forms other than as pure hematite and that more accurate corrections could probably be made, it is suggested that detailed analysis of assay grades which are biased or anomalous will lead to incorrect and possibly misleading interpretations particularly if the analytical errors are concentrated in the iron determinations.

(b) Examination of the operations in the Maramapa laboratory revealed certain disturbing features in the reporting and presentation of assay results. These features included; falsification of assay results, reporting of non-existent sample

numbers, loss of samples, incorrect conversions made from spectrometer printout sheets, incorrect transfer of data from sample register to sample sheets and the transposition of manganese and alumina values in sample registers and sample sheets.

Many of these errors can be corrected by assay of duplicate samples and by personal examination of the sample register, sample sheets and spectrometer printout sheets but they should not be allowed to occur in the first instance.

B. STATISTICAL ANALYSIS OF CONTROL SAMPLE ASSAY VALUES

Assaying is the third and final stage of sample treatment after sample collection and the preparation of samples for assaying. The reliability of each of the stages of sample treatment is heavily dependant upon the accuracy and reliability of each of the preceeding stages and this is particularly so in the case of assaying where errors in each of the two preceeding stages are inherited and may be the cause of assay bias even before assaying takes place. Even the most reliable and accurate assaying technique is inadequate if sample collection and preparation of samples for assaying is undertaken without regard to the relevant principles of sampling and sample preparation. In a similar manner, carefully collected and prepared samples may be rendered worthless if the analytical or assaying technique fails to provide accurate and reliable results.

No assaying technique yet devised is able to yield results with absolute precision and assaying techniques are usually a compromise between precision and reproducibility.

Reproducibility is a measure of the range of values yielded by an assaying technique when assaying samples with identical composition, e.g. when assaying the same sample many times or when assaying sub-samples of a larger sample. Assaying techniques with high reproducibility yield a narrow range of values, i.e. a low standard deviation, whereas those of low reproducibility yield a wide range of values, i.e. a high standard deviation, when assaying samples of identical grade.

Precision, in the context of assaying, is a measure of the ability of the assaying technique to yield results close to the absolute or true value. Assaying techniques with high precision give assay results very close to the absolute value whereas those with low precision give assay results which are numerically distant from the absolute or true value.

Reproducibility and precision are not necessarily synonymous although high precision does imply high reproducibility. The most efficient assaying technique gives assay results of high precision but assay techniques of low precision may be acceptable if the reproducibility of the technique is high since the addition or subtraction of constants or other suitable manipulation or transformation of assay data may be sufficient to correct the initial assay value to the true value. Assaying techniques should, ideally, be regularly checked for precision and reproducibility by the assaying of standard or control samples, the grades of which are accurately known.

Variation in analytical precision and reproducibility caused by inherent errors in the assaying technique may be investigated by two main methods:

- (a) Preparation and assaying of Craven control samples and mathematical and graphical analysis of the assay results.
- (b) Assaying of other standard or control samples and statistical and graphical analysis of the assay results.

(a) Craven Control Samples

Craven control samples (Craven, 1954) consist of a series of carefully prepared samples containing various and accurately determined proportions of the valuable constituent, in this case, hematite. The samples are prepared by mixing end-members of the sample series in various proportions thus allowing the expected grade of the intermediate members of the series to be calculated using a knowledge of the assay values of the two end-members and the proportions of each

in the various intermediate members. The two end-members are designated as high and low corresponding to a high or low content of the valuable constituent.

Cravens method of estimation of the accuracy of assaying consists of assaying members of the prepared sample series thereby giving two estimates of the grade of each sample, an expected (calculated) grade and the assay grade. An assaying technique with absolute precision would yield assay grades corresponding exactly with the calculated grades, assuming no preparation or calculation errors, and a plot of one against the other would yield a straight line passing through the origin at an angle of 45 degrees (Fig. 8.2). In practice however, no assaying technique has absolute precision and a plot of the two grade estimates would only approximate a straight line, the deviation of the individual points from this line being a measure of the precision of the assaying technique.

Mathematical processing of the assay values coupled with the known proportions of the two end-members in each sample enables estimates of the high ($\hat{H}$) and low ($\hat{L}$) samples to be made and these estimates are used to give a calculated measure ($\hat{A}$) of the grade of the sample. The sum of $(A - \hat{A})^2$ for each sample series is used to determine the standard deviation of the series. Assay grades (A) are plotted against calculated grades ($\hat{A}$) for each sample series and grade limits of two or three standard deviations are drawn parallel to the line $A = \hat{A}$ which describes the results of a perfect assaying technique with which assay grade equals calculated grade. Plotted points falling outside the standard deviation limit are anomalous and automatically suspect. If the points regularly fall within the standard deviation limits the assaying technique is accepted as being reliable.

The method and the mathematical and graphical analysis of the Craven procedure are more fully described by Craven (1954).

(b) Application of Craven Control Samples at Marampa

The precision and reproducibility of the Marampa assaying technique (X-ray fluorescence spectrometry), was investigated

with a series of six Craven control samples which were prepared so as to cover the range 35 to 70 per cent Fe, i.e. a range from marginal ore containing less than 40 per cent Fe to high grade hematite concentrates containing about 68 per cent Fe.

The samples consisted of mixtures of the two end-members, Ferromax and quartz, calculated to cover the desired range of values. Ferromax is a high grade hematite concentrate containing greater than 97.5 per cent hematite. Five samples of Ferromax and five of quartz were assayed for iron by wet chemical methods to determine the iron content of the end-members to be used in calculation of the grade of the intermediate members of the sample series. These assay results were:

Sample No.	Ferromax (% Fe)	Quartz (% Fe)
1	68.7	1.2
2	68.0	1.3
3	68.8	0.9
4	68.6	1.0
5	68.2	1.2
Mean	68.46	1.12

The mean grade of the Ferromax and the quartz were thus taken as 68.5 and 1.1 per cent Fe respectively.

The quartz (crushed to - 44 B.S. mesh) and the Ferromax were mixed in the following proportions (Table 8.1) to give 2000 grams of each sample mixture with the indicated grade.

Table 8.1 - see overleaf

Sample Mixture	Proportions		Calculated grade (per cent Fe)
	Ferromax	Quartz	
A	1	Nil	68.5
B	15	1	64.3
C	6	1	58.9
D	4	1	55.0
E	2	1	46.0
F	1	1	34.8

Table 8.1: Craven control samples, relative proportions of two end-members and the calculated grade.

Ten sub-samples of each of the six sample mixtures were prepared and one sub-sample from each of the six groups were forwarded for assay at various intervals over a period of about seven weeks. The iron assay only is used in the investigation of the accuracy of assaying, mainly because iron is the valuable constituent but also because the control samples are prepared only with the amount of iron and quartz controlled and since the sum of the assay values is, theoretically, a constant, the other constituents should also be relatively well controlled although the absolute values remain unknown.

The Craven control samples were forwarded for assay together with routine ore and concentrate samples and were numbered in such a way that no indication of iron content was evident.

(c) Results of the Craven Control Sample Programme

Results of the Craven control sample programme carried out at Marampa are summarised in Tables 8.2 and 8.3.

On the basis of the Craven control sample technique the Marampa assaying facilities appear to be satisfactory when the assay data for the sixty control samples (6 sample groups by 10 sub-samples) are considered and by extrapolation it could be concluded that the Marampa assaying technique yields reliable

SAMPLE GROUP	A		B		C		D		E		F	
CALCULATED GRADE	68.5		64.3		58.9		55.0		46.0		34.8	
DATE ASSAYED	Sample No.	% Fe	Sample No.	% Fe	Sample No.	% Fe	Sample No.	% Fe	Sample No.	% Fe	Sample No.	% Fe
4.12.68	SPC 129	68.9	SPC 138	65.0	SPC 152	60.2	SPC 158	57.9	SPC 174	51.6	SPC 183	39.0
12.12.68	126	68.2	144	63.6	153	60.5	162	58.8	169	51.2	180	43.6
14.12.68	134	68.2	142	65.6	155	60.6	159	56.8	172	51.0	179	44.5
15.12.68	135	68.2	145	65.1	151	59.4	160	56.8	173	50.1	185	42.3
20.12.68	132	68.9	143	65.2	147	60.4	163	58.0	175	50.5	184	41.2
31.12.68	131	68.5	140	64.7	150	60.1	156	57.0	166	50.9	178	42.6
8. 1.69	130	69.0	136	63.9	148	58.9	165	57.6	168	51.0	182	37.9
13. 1.69	128	68.8	137	61.6	149	61.0	157	58.0	167	52.3	181	42.4
18. 1.69	127	69.1	139	64.8	154	60.4	161	57.8	171	51.8	176	40.0
21. 1.69	133	67.0	141	66.6	146	59.7	164	64.6	170	50.6	177	41.5

Table 8.2 : Craven control sample assay values.

assay results for all ore and concentrate samples, but certain anomalous features of the results, which would otherwise suggest biased assay results, are not explained fully by the Craven procedure.

The Marampa assaying facilities are found to give reasonably consistent results, i.e. the reproducibility of the technique is high, but serious discrepancies occur between the assay grades and the calculated grades particularly in the range of values 30 to 50 per cent Fe. The reported assay grade is almost invariably higher than the grade calculated for the various sample mixtures and the discrepancy increases with decreasing iron content.

All the weighted (Craven) assay values ($\hat{A}$) fall within the limits of two standard deviations when plotted against the reported assay grades (A) and this, according to Craven, is indicative of reliable assaying. The weighted assay results are however calculated from a knowledge of the reported assay grades rather than from a knowledge of the original calculated grade and thus the Craven method is apparently not suited for the detection of uniform errors in which the deviations between assay grade and calculated value are similar for a particular grade sample, but is generally applicable for determining relative concentrations, i.e. it detects grade differences rather than absolute grades. The Craven method estimates the high ($\hat{H}$) and low ($\hat{L}$) grades and for the Marampa data the high sample estimate is similar to the grade of the high sample calculated from the assay values of the initial Ferromax samples but the estimate of the low sample is significantly greater than the value given by the mean assay value of the five quartz samples and this is taken to indicate a failure of the Craven method to detect uniform assaying bias.

(d) Statistical Analysis of the Craven Control Sample Assay Values

The Craven control sample assay results have been examined by statistical methods in an attempt to determine whether or not the observed grade deviations are statistically significant and, if so, to devise mathematical or graphical

correction techniques in the form of regression equations.

Assay grades reported for the Craven control samples are, in almost all cases, greater than the grades calculated for the various sample mixtures. This is shown in Table 8.3 and in Fig. 8.1 which clearly demonstrates that the mean deviation increases with decreasing iron content of the various samples.

Sample Mixture	Calculated Grade per cent Fe	Mean Assay Grade per cent Fe	Mean Deviation Fe per cent
A	68.5	68.5	0.46
B	64.3	64.6	1.07
C	58.9	60.1	1.22
D	55.0	58.3	3.27
E	46.0	51.1	5.10
F	34.8	41.5	6.70

Table 8.3: Craven Control sample grades and mean deviations.

A "perfect" assaying technique is one which yields assay values identical to the calculated grade, the relationships between these two grades being expressed by the equation,

$$Fe_A = Fe_C$$

the subscripts A and C referring to assay and calculated grades. This degree of accuracy is probably never attained and slight deviations between Fe_A and Fe_C are the rule rather than the exception, in which case the above equation is modified to,

$$Fe_A = Fe_C + e$$

where e = random error.

A graph (Fig. 8.2) of assay grade against calculated grade for the sixty Craven control samples indicates a significant divergence from the ideal relationship and examination of the graph allows subdivision of the assay results into three grade ranges as follows;

- (a) Lower range from 30 to 58 per cent Fe.

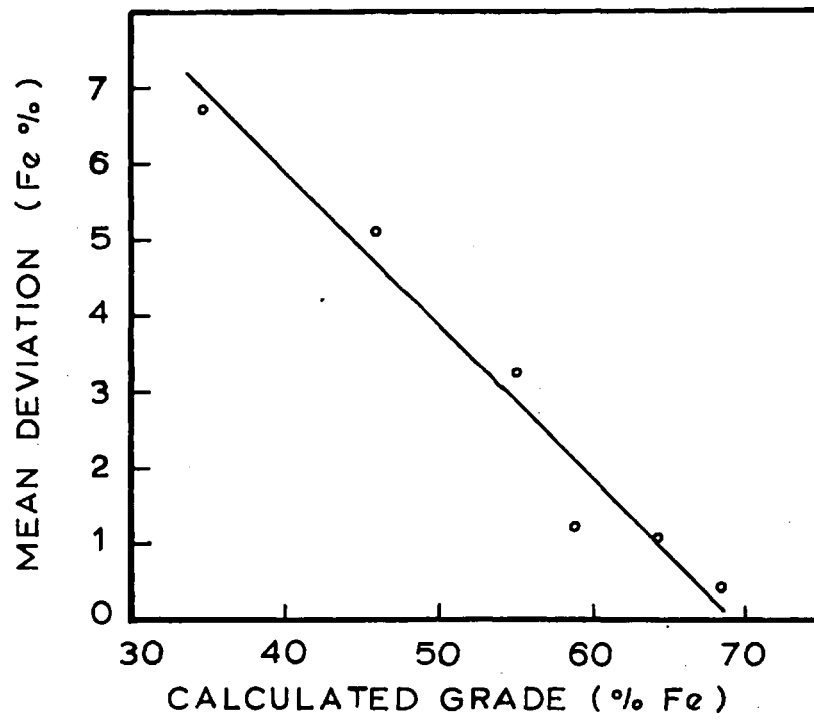


Fig. 8-1

(b) Transition range from 58 to 60 per cent Fe.

(c) Upper range from 60 to 70 per cent Fe.

Each of these ranges is characterised by a unique linear relationship between expected or calculated grade and the reported assay grade and the upper and lower ranges coincide with concentrate and feed grade ranges respectively. Iron grades in the transition range are rare at Marampa, being exceptionally low for concentrate grades and exceptionally high for hilltop ore grades or mill feed grades.

Regression analysis of the assay data, both calculated and reported, has been used to determine the lines of best fit for the various ranges and are as follows (Fig. 8.2).

(a) Line a: describes the relationship between measured and calculated grades for a "perfect" assaying technique,

$$Fe_C = Fe_A$$

(b) Line b: describes the relationship between measured and calculated grades for all sixty Craven control sample assays,

$$Fe_C = 1.24 Fe_A - 16.7$$

(c) Line c: describes the relationship between measured and calculated grades for the sample groups A, B and C which correspond with the concentrate grade range,

$$Fe_C = 1.08 Fe_A - 5.4$$

(d) Line d: describes the relationship between measured and calculated grades for the sample groups D, E and F which correspond with the feed grade range,

$$Fe_C = 1.13 Fe_A - 11.5$$

In all four equations Fe_C and Fe_A represent the calculated or "true" grade and the measured or assay grade respectively, both being expressed in per cent Fe.

The observed bias towards overvaluation may have arisen from one or more of three major sources;

(a) Errors in the initial wet chemical assaying of the five Ferromax and five quartz end-members used to make up the six groups of control samples. It is most improbable that the Ferromax samples were overvalued since the mean value of

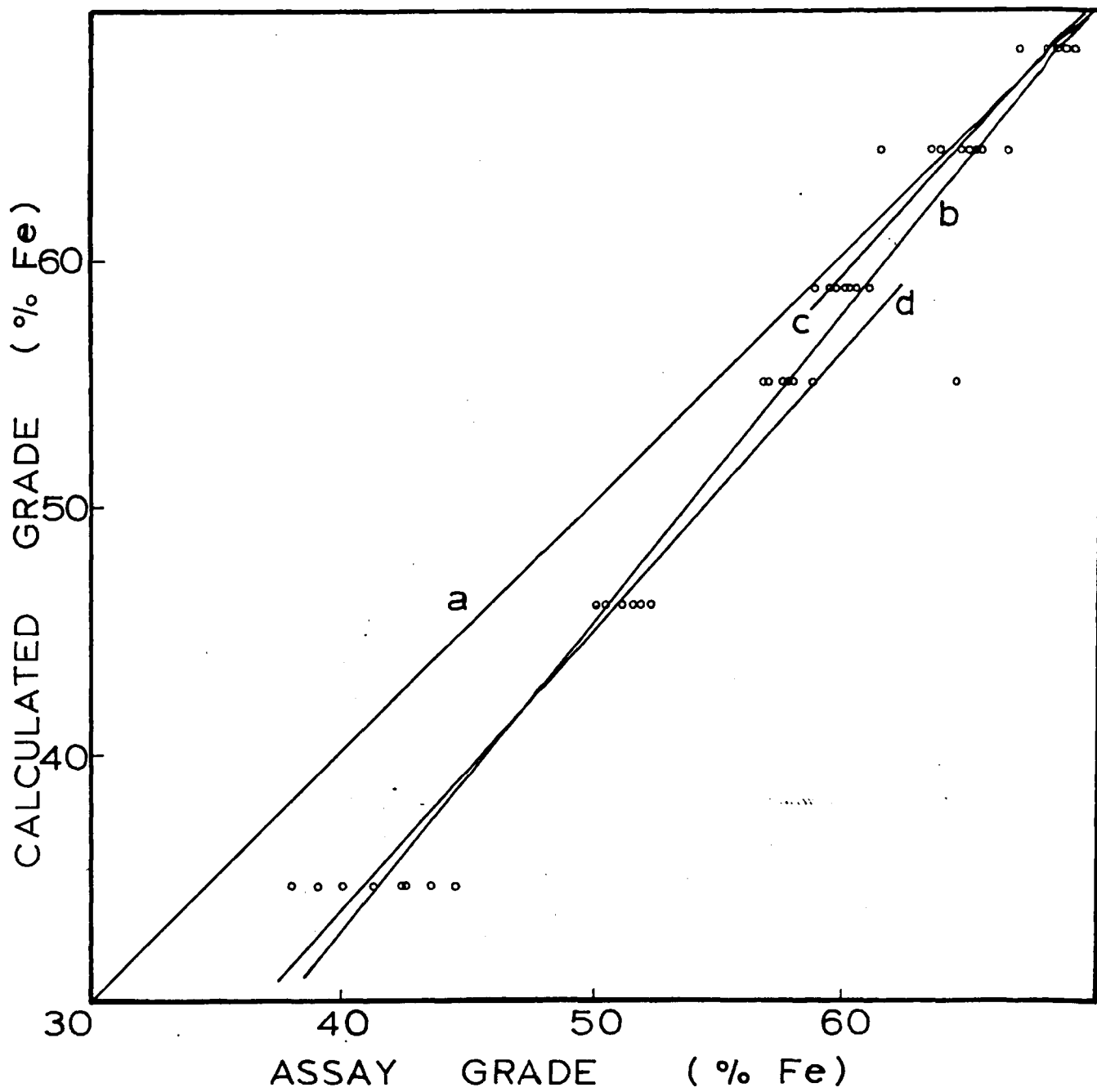


Fig. 8.2

68.5 per cent Fe obtained for the five samples is a very typical value and the small range of values suggests that the five samples were fully representative of the bulk Ferromax sample. If the value of 1.1 per cent Fe is assumed to be the correct value for the mean iron content of the five quartz samples then in order for sample group F to attain its mean value of 41.5 per cent Fe would necessitate the Ferromax having the theoretically impossible grade of greater than 80 per cent Fe. It is thus concluded that the mean value of 68.5 per cent Fe is a reliable estimate of the iron content of the Ferromax.

In a similar manner, the value of 1.1 per cent Fe is accepted as a reliable estimate of the iron content of the quartz which was handpicked from syncline B on a visual estimate of no apparent hematite. The presence of iron as indicated by the mean grade of 1.1 per cent Fe is attributed to contamination during crushing and sample preparation. For sample group F to attain its mean assay grade of 41.5 per cent Fe would require a mean grade of 14.5 per cent Fe for the quartz. Such an iron content would be readily observed but the quartz used showed very little discolouration or apparent contamination.

(b) Incorrect sampling and preparation of control samples. This also is considered to be improbable. The weighing of the sample mixtures was correct to the nearest 0.1 grams and the proportions of Ferromax and quartz making up each sample group were carefully mixed before splitting into the ten sub-samples.

(c) Bias in assaying technique. This is considered to be the most probable source of error and further discussion is based on the hypothesis that the Marampa sample preparation and assaying facilities yield biased assay results which favour over-valuation of the iron content of feed and, to a lesser extent, concentrate samples.

A Students t test (Chapter II) carried out on the six sample groups indicates that the observed deviations of assay means from calculated grades are significantly different at the 99 per cent confidence level for the four sample groups

C to F inclusive.

The Null Hypothesis proposed for this t test is that there is no significant difference between the calculated grades and the mean assay grades for each group. The hypothesis is tested by the equation (Rickmers and Todd, 1966 pp. 88),

$$Z = \frac{(\bar{X}_i - C_i)\sqrt{N}}{s_i}$$

where,

$\bar{X}_i$ = mean assay grade of each sample group

C_i = calculated grade of each sample group

s_i = standard deviation of each group of sample assays

N = number of sub-samples in each assay group (10)

Z = calculated value which is distributed as Students
t with $N-1$ degrees of freedom

This equation is applied to the experimental data with the following results.

Sample Mixture	$\bar{X}_i$	C_i	s_i	Z
A	68.5	68.5	0.62	-
B	64.6	64.3	1.35	0.70
C	60.1	58.9	0.62	6.12
D	58.3	55.0	2.29	4.56
E	51.1	46.0	0.65	24.81
F	41.5	34.8	2.04	10.39

The number of degrees of freedom for this test is $N - 1 = 9$. A one-tailed test to determine whether or not $\bar{X}_i$ is significantly greater than C_i indicates that a calculated t value greater than 2.82 is significant at the 99 per cent confidence level. Inspection of the calculated t values reveals that the four groups C to F inclusive have t values significantly greater than the critical (standard table) value of 2.82 and thus the Null Hypothesis is rejected and it is concluded that the assay values of the four sample groups C to F are significantly

greater than the expected values calculated from a knowledge of the grades of the two end-members and the proportion of these end-members in each of the sample groups. Although a tendency towards over-valuation occurs in sample group B the observed differences between calculated and assay grades are not of sufficient magnitude to suspect that the deviations are due to any factor other than chance.

If it is accepted that;

(a) the sampling and wet chemical assaying of the initial end-members, and,

(b) the preparation of the six sample groups, and,

(c) the principle of control sample assaying,

are all valid, then the conclusion reached is that the method of iron determination employed for the assaying of Marampa samples yields biased results which, for hilltop ore samples and mill feed samples at least, will result in over-valuation of the orebody with all the attendant economic implications concerning ore reserve estimation and grade control.

Silica determinations made by the Marampa spectrometer are not consistent (D. Barnes,* pers. comm.) and this has been verified by the silica assay results of the Craven control samples which indicate that silica assaying at Marampa is of low reproducibility. The absolute accuracy of silica determinations is difficult to assess because of the analytical "shadow" for values between 8.2 and 12.6 per cent SiO_2 and for values greater than 38.4 per cent SiO_2 , but a plot of reported silica grades against calculated silica grades suggests that silica values are undervalued to the extent of between 2 and 4 SiO_2 per cent, i.e. a feed grade silica assay of, say, 20 per cent SiO_2 has a true value of about 23 per cent SiO_2 .

Undervaluation of the silica grades and overvaluation of the iron grades leads to overvaluation of the ore reserves with a resultant failure of the mill to attain the concentrate target figures of both grade and tonnage.

* Chemist, Sierra Leone Development Co., Marampa.

(c) Conclusion

The Craven control sample method for estimating the accuracy of assaying is not of sufficient sensitivity to detect significant analytical deviations such as occur at Marampa and standard statistical tests, including the Students t test, are better suited for quantitative investigations.

Iron grades, particularly in the range of the hilltop ore samples and mill feed samples, are overvalued to an extent which increases with decreasing iron content. The overvaluation of iron grades together with the apparent undervaluation of silica grades results in overvaluation of the ore reserves and overvaluation of the amount of recoverable iron entering the mill and this may well be the explanation of the failure of the mill to attain its concentrate target figures.

Regression analysis of the Craven control sample assay results has yielded linear regression equations capable of correcting the reported grades to true grades.

The precision of the Marampa assaying facilities is low, particularly in the lower grade range but the reproducibility is good and often high.

C. CONVERSION ERRORS IN SAMPLE ASSAYING AT MARAMPA

Marampa grade data is obtained from either wet chemical or X-ray methods of assaying depending upon the iron content of the sample. Tailings samples, which usually contain about 20 per cent Fe, are invariably assayed by wet methods whereas concentrate samples, which usually contain greater than 60 per cent Fe, are assayed by X-ray fluorescence spectrometry. Mill feed or hilltop ore samples are assayed by X-ray methods except where an iron content of less than 30.0 per cent Fe is indicated; such samples may then be re-assayed by wet chemical methods.

Since only iron determinations are made by wet methods, the distribution of manganese, silica and alumina in tailings and low grade ore samples is unknown but silica and alumina

contents of these samples are almost certain to be high and beyond the upper analytical limits of the spectrometer and can be assumed to be at least 40 per cent SiO_2 and 10 per cent Al_2O_3 respectively.

The X-ray fluorescence spectrometer gives assay values in the form of digital printout with four values for each sample, each value corresponding with one of the four main constituents of the ore; iron (as Fe), alumina (Al_2O_3), silica (SiO_2) and manganese (Mn). These digital values must then be converted to actual percentage values by the following methods.

(a) Iron concentrate, manganese and alumina assay values are read directly from calibrated conversion tables prepared when the instrument was commissioned.

(b) Iron feed (iron content 30.0 to 59.0 per cent Fe); subtract one-tenth (0.1) of alumina reading from iron reading and apply remainder to feed iron conversion tables.

(c) Silica feed (greater than 12.6 per cent SiO_2); subtract one-tenth (0.1) of Fe reading from SiO_2 reading and to the remainder add one-fifth (0.2) of the alumina reading and then read from feed SiO_2 conversion table.

(d) Silica concentrate (less than 8.2 per cent SiO_2); divide reading by 4 and subtract the result from the silica reading and then read from concentrate SiO_2 conversion table.

The analytical range and limits of the spectrometer cater mainly for determination of the iron content of the mill feed and concentrate samples. Silica, manganese and alumina all have upper limits of detection which are readily exceeded particularly in the assaying of low grade feed samples. In addition, there is an analytical "shadow" between 8.2 and 12.6 per cent SiO_2 in which assays are reported as being greater than 8.2 per cent SiO_2 or less than 12.6 per cent SiO_2 .

The analytical ranges and detection limits for the Marampa XRF assaying technique are;

(a) Iron: 30.0 to 69.4 per cent Fe in steps of 0.1 Fe per cent.

(b) Manganese: 0 to 3.26 per cent Mn in steps of 0.01 Mn per cent.

(c) Alumina: 0.2 to 9.05 per cent Al_2O_3 in steps of 0.05 and 0.1 Al_2O_3 per cent. Interpolation may be made to 0.01 Al_2O_3 per cent although this is rarely done.

(d) Silica: two silica ranges are used, one each for concentrate and feed grades. The silica concentrate range is from 0.1 to 8.2 per cent SiO_2 in steps of 0.05, 0.1 or 0.15 SiO_2 per cent although interpolation to 0.01 SiO_2 per cent may be made. The silica feed range is from 12.6 to 38.4 per cent SiO_2 in steps of 0.1 SiO_2 per cent.

Conversion of digital printout into actual percentage assay values has been found to be an extremely weak link in the overall assaying procedure and incorrect assay results due to conversion errors are common.

A number of assay results selected by random sampling methods were recalculated from the original digital printout data and compared with the reported assay values. Two types of errors were recorded;

(a) Major errors which may be taken to indicate numerically large and significant differences between reported and recalculated (correct) assay values. In general, recalculated iron assays differing by more than 0.2 ~~Fe~~ per cent from the reported assays are considered to indicate major errors.

(b) Minor errors. These are attributable to interpolation differences and usually only affect the second decimal place in alumina and manganese assay values and are not considered to be important either in grade control or ore reserve estimation.

A total of 159 sample assay results, i.e. 636 determinations, were examined and errors of either type detected in 194 determinations affecting 122 samples. Fortunately, many of the errors are minor errors only and attributable to interpolation variations in alumina conversions, but major errors were identified in about 34 per cent of the samples.

These major errors are attributed to clerical errors in the conversion of the digital data to actual percentage assay values and it is postulated that the number of major errors

committed during such conversion is directly proportional to the number of calculations or manipulations necessary for each conversion.

The error distribution is tabulated below (Table 8.4)

Determination	Fe	Mn	SiO ₂	Al ₂ O ₃	Total
Major errors	16	7	44	12	79
% of samples	10.1	4.4	27.7	7.6	12.4
Minor errors	11	12	20	72	115
% of samples	6.9	7.6	12.6	45.3	18.1

Table 8.4: Distribution of major and minor conversion errors.

Manganese and alumina assay values, as noted earlier, are converted from digital form by reading directly from standard conversion tables and no manipulation of data is necessary. These are the simplest conversions and it would be expected that the least number of conversion errors should occur for these constituents. This is verified by Table 8.4 above. Mill feed sample iron assays require three data manipulations (division of Al₂O₃ reading by 10, subtraction of this value from the Fe reading and comparison of the result with the conversion table) and errors would be expected for this conversion more frequently than for manganese and alumina conversions. This again is verified by Table 8.4. Silica conversions from digital form require three and five manipulations for concentrate and feed samples respectively and errors would be expected to occur more frequently for these conversions than for any other. Recourse to Table 8.4 once again verifies this prediction and it can be seen that more than one quarter of all silica conversions are incorrectly made and when the minor errors are taken into account more than 40 per cent of the silica assay values are inaccurately reported.

A Chi-square test (Chapter II) was applied to the distribution of major errors in order to determine whether or not the observed variation in the number of major errors could have arisen by chance. It was earlier postulated that the number of errors was associated with the number of data manipulations necessary to make the digital data to assay value conversion and if the Chi-square test yields a significant result this postulate is acceptable on the basis that it is the simplest and the most probable explanation.

The Chi-square test proceeds as follows;

Determination	Mn	Al ₂ O ₃	Fe	SiO ₂	Total
Major errors (O)	7	12	16	44	79
Expected errors (E)	19.75	19.75	19.75	19.75	

Table 8.5; Observed and expected major conversion errors

If all conversions required the same number of data manipulations it is expected that the observed number of major errors would not be significantly different from 19.75 which is the average number of errors per determination.

The Null Hypothesis is that there is no significant difference in the number of major errors for the four determinations, i.e. the number of major errors (O) does not differ significantly from the expected number of errors (E = 19.75). The Hypothesis is tested by the equation;

$$\chi^2 = \sum \frac{(O - E)^2}{E}$$

Substituting the values from Table 8.5

$$\chi^2 = 41.76 \text{ with 3 degrees of freedom}$$

The critical (standard table) value for χ^2 with 3 degrees of freedom at the 95 per cent confidence level is $\chi^2_{3,0.05} = 9.35$.

$$\therefore \chi^2_{\text{calculated}} > \chi^2_{\text{critical}}$$

and the Null Hypothesis is confounded. The extremely high value

for the calculated value of Chi-square indicates that the variations in the distribution of the number of major errors is very significantly different from that expected by chance and thus other factors must be operating which influence the number of errors committed in conversion of digital data to percentage assay values.

Inspection of Table 8.4 shows that by far the greatest number of conversion errors are committed when converting silica digital data to assay form and the least number of conversion errors are committed when manganese and alumina conversions are made. The order of increasing number of errors is directly associated with the order of increasing complexity in the conversion of digital data. Manganese and alumina values are read directly from conversion tables; iron values are either read directly from conversion tables (concentrate grades) or are read from tables after three manipulations (feed grades); silica values require three (concentrate grades) or five (feed grades) manipulations and the earlier formed postulate that the order of increasing number of errors is related to the order of conversion complexity appears to be well founded.

These conversion errors appear to be simple clerical errors and, if so, could be completely eliminated by suitable supervision and training of the African laboratory staff, by checking of results and by design of a suitable self-checking procedure. The large proportion of samples (34 per cent) affected by one or more major errors implies that present ore reserve estimations and other calculations based on analysis of assay data should not be accepted at face value.

Typical major errors include the following:

45.1 per cent Fe reported as 54.1 per cent Fe

43.7 " " " " " 40.8 " " "

2.76 per cent SiO₂ reported as 5.1 per cent SiO₂

29.1 " " " " " 13.5 " " "

1.69 per cent Mn reported as 0.69 per cent Mn

1.83 " " " " " 0.74 " " "

0.53 per cent Al_2O_3 reported as 0.2 per cent Al_2O_3
 0.35 " " " " " 1.13 " " "

The number of errors committed in iron value conversions would also suggest that the number of errors is related to the number of manipulations necessary. Feed grade iron conversions require three manipulations and errors occur in 15 of the 131 conversions examined. Concentrate iron conversions are made by reading directly from conversion table (one manipulation) and no errors were committed in 28 conversions examined.

A 2 x 2 contingency table for a Chi-square test on this data is, (Table 8.6).

	Error	Correct	Total
Concentrate	0	28	28
Feed	15	116	131
Total	15	144	159

Table 8.6: Conversion errors for iron values.

A Null Hypothesis is proposed that there is no significant difference between the proportions of errors committed in the two types of conversion (feed and concentrate).

If the variation in the number of errors committed is determined by chance alone, the proportion of feed grade conversion errors will be similar to the proportion of concentrate conversion errors. Table 8.6 (above) is modified to indicate the number of errors which would be expected if the number of errors in each category (feed and concentrate) were determined by chance alone.

	Error	Correct
Concentrate	0 (2.64) ^x	28 (25.36)
Feed	15 (12.36)	116 (118.64)

^x expected number in parentheses

Table 8.7: Observed and expected major error distribution.

The Null Hypothesis is again tested by the equation,

$$\chi^2 = \sum \frac{(O - E)^2}{E}$$

From Table 8.7;

$$\chi^2 = 3.53 \text{ with 1 degree of freedom}$$

This calculated value of Chi-square is less than the critical (standard table value of $\chi^2_{1,0.05} = 5.02$ and the Null Hypothesis is accepted at the 95 per cent confidence level. There is thus no reason to suggest from the data available that the number of errors committed in conversion of iron values is related to the number of manipulations necessary for the conversion. This conclusion apparently contradicts the earlier formed and accepted postulate concerning conversion errors but the data does show a tendency towards acceptance of the postulate which may be completely acceptable for the iron conversions when it is recalled that the sale of the Marampa concentrate is based on the iron content indicated by the concentrate iron assay which is thus the single most important and, presumably, most scrutinised of the assays and is subject to closer supervision during conversion than all other assays.

The fact that no errors were made during the conversion of the 28 concentrate grade iron assays is difficult to interpret since it is not possible to predict the rate of error commission in such conversions. A larger sample may be of more significance in this problem and, although the Null Hypothesis must be accepted on the basis of the available data, a verdict of not proven may be more realistic in this examination of iron conversion errors.

D. A SELF-CHECKING ASSAY CONVERSION PROCEDURE

Incorrect assay results are worthless, costly and can lead to misleading interpretations in the course of ore reserve estimation and grade control and every effort should be made to eliminate assay errors particularly conversion errors which are attributed to simple clerical errors.

Only the converted assay values are permanently recorded

by the Marampa laboratory, the original spectrometer digital printout sheets being discarded, and it is thus not possible to verify the accuracy of reported assay values. If conversion errors are common, and the preceeding investigation supports this theory, then it is imperative that, either the original data be available for reprocessing or that the initial conversions are guaranteed correct.

An assay conversion procedure has been designed to record both the initial digital assay values and the final corrected assay values and which incorporates a self-checking procedure to ensure that assay conversions are correct.

This self-checking assay conversion procedure requires that the spectrometer digital printout data be processed and the various data manipulations checked before the final conversion of the digital data to the actual assay data. The procedure is described with reference to Table 8.8 which is a suggested form of presentation for this procedure.

Columns 1 to 4 inclusive (Table 8.8) record the initial uncorrected digital assay values for iron, alumina, silica and manganese respectively.

Columns 5 to 8 inclusive record the initial manipulations of the raw data, e.g. in column 8 is entered one quarter of the digital silica value as required for conversion of silica concentrate values.

Columns 9 to 11 inclusive record the various manipulations necessary to transform the raw data to a form able to be converted directly from the standard calibration tables.

Columns 12 to 15 inclusive record the final assay values of the samples.

The conversion procedure is as follows:

- (a) Raw data is entered in columns 1 to 4.
- (b) Initial manipulations made (columns 5 to 8).
- (c) Final manipulations made (columns 9 to 11).
- (d) Columns 1 to 11 exclusive of column 4 are separately summed.
- (e) The following equalities are then tested:

	DIGITAL PRINTOUT				MANIPULATION				IRON FEED	SILICA FEED	SILICA CONC	ASSAY VALUES			
COLUMN	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
SAMPLE No	Fe	Al ₂ O ₃	SiO ₂	Mn	Fe/10	Al ₂ O ₃ /10	Al ₂ O ₃ /5	SiO ₂ /4	1 - 6	3+7-5	3 - 8	Fe	Al ₂ O ₃	SiO ₂	Mn
SPC 139	1299	33	333	38	130	3	7	83	1296	210	250	64.8	0.33	4.7	0.10
SPC 154	1213	21	659	29	121	2	4	165	1211	542	494	60.4	0.25	>8.2	0.08
SPC 161	1139	28	763	63	114	3	6	191	1136	655	572	57.8	0.29	14.1	0.17
SPC 171	981	1	1328	29	98	-	-	332	981	1230	996	51.8	-	30.2	0.08
SPR 11	1166	236	761	236	117	24	47	190	1142	691	571	58.1	1.93	14.7	0.62
SPD 21	1181	276	563	123	118	28	55	141	1153	500	422	58.5	2.23	>8.2	0.32
SPG 7900	1128	242	828	66	113	24	48	207	1104	763	621	56.6	1.96	15.9	0.18
SPG 7901	1180	207	625	67	118	21	41	156	1159	548	469	58.8	1.68	12.6	0.18
SPG 7902	1172	159	715	37	117	16	32	179	1156	630	536	58.6	1.30	13.7	0.10
SPG 7903	1151	173	805	46	115	17	35	201	1134	725	604	57.8	1.43	15.3	0.12
SUM	11610	1376	7380		1161	138	275	1845	11472	6494	5535				

CHECKS: $\sum 2 = 10 \sum 6$; $\sum 9 = \sum 1 - \sum 6$; $\sum 10 = \sum 3 + \sum 7 - \sum 5$; $\sum 11 = \sum 3 - \sum 8$

Table 8.8 Specimen assay conversion sheet.

- (i) $\sum 2 = 10 \sum 6$, i.e. sum of column 2 equals 10 times the sum of column 6, and similarly,
- (ii) $\sum 9 = \sum 1 - \sum 6$
- (iii) $\sum 10 = \sum 3 + \sum 7 - \sum 5$
- (iv) $\sum 11 = \sum 3 - \sum 8$

(f) If the equalities are satisfied then the final conversions of the digital data to assay values may be made from the standard tables. If the equalities are not satisfied then the data manipulations must be checked.

This procedure for assay conversion may be somewhat laborious but it affords a method of checking a batch of samples (say 24 samples, corresponding to a spectrometer charge) without the necessity of checking each individual sample conversion and, in the light of the investigations into conversion errors, could be profitably applied in the Marampa assay laboratory.

CHAPTER IX

SUMMARY OF CONCLUSIONS

The Marampa Mine in Sierra Leone produces a hematite concentrate (mean grade about 64.5 per cent Fe) by Humphreys spiral beneficiation of hematite-quartz-muscovite schist (mean grade about 45.9 per cent Fe). The Marampa grade control problem is concerned with the fact that mill feeds with similar or even identical grades may yield concentrates with significantly different grades and the concentrate grade, although directly associated with mill feed grade, cannot be accurately predicted from a knowledge of mill feed grade alone.

Statistical analysis of concentrate shipment grades indicates that the iron content of the concentrate is steadily decreasing and this grade decrease is attributed mainly to a decrease in mean ore grade and hence, mill feed grade. The mean ore grade has decreased from greater than 55 per cent Fe to about 46 per cent Fe over a period of about ten years, the decrease being attributed to a decrease in the effects of tropical leaching, which upgraded the original ore, with increasing depth of mining below the original topographic surface of the orebody.

It is tentatively suggested that the cyclical nature of the grade/time scattergram of the concentrate shipment grades for the period January 1967-June 1968 may be due to non-technical factors, involving management or personnel, related to the intensity or degree of supervision of mining, milling and blending procedures.

Coupled with the decrease in concentrate iron content is a statistically significant increase in the concentrate silica content and in recent years there has been a significant movement of silica grades of concentrate shipments from the low silica classes (<5.0 per cent SiO_2) to the mid-silica classes (5.0 to 6.5 per cent SiO_2) and high silica classes (>6.5 per cent SiO_2).

The Ti, Al_2O_3 , P and Mn contents of the concentrates are acceptably low and stable and can be ignored for the purposes of grade control and ore reserve estimation unless extreme and detrimental variations occur.

Statistical analysis of the grainsize distribution of the concentrate shipments reveals that the midsize fractions (-60 +170 B.S. mesh), which form the bulk (about 60 per cent) of the concentrates, are deficient in hematite and are invariably of sub-standard grade (mean grade about 63.7 per cent Fe). The hematite deficiency in this grainsize range is caused by the inclusion of excessive amounts of liberated quartz, a phenomenon apparently inherent in the spiral method of beneficiation. The detrimental effects of this quartz may be counteracted to a certain extent by careful control of milling procedures.

The highest iron grades of the concentrate screen fractions are either in the +60 or -170 +200 B.S. mesh fractions and iron grades in the -44 +60 B.S. mesh fraction are consistently about 2.0 Fe per cent higher than the overall concentrate grade, and reach as high as 68.3 per cent Fe (97.5 per cent hematite).

The lowest iron grades of the concentrate screen fractions are almost exclusively restricted to the -85 +120 B.S. mesh fractions. Removal of this fraction from the concentrate would effect a grade increase of between 0.4 and 0.5 Fe per cent but at the prohibitive expense of a 20 per cent reduction in concentrate tonnage.

The control and optimum settings of the Humphreys spiral variables (feed rate, pulp density, number of concentrate withdrawal ports, orientation of withdrawal port splitters and others) are probably more critical in the spiral beneficiation of Marampa ore than has hitherto been realized.

The coarse (+60 B.S. mesh) and fine (-170 B.S. mesh) fractions of the concentrate are almost invariably of high grade (>66 per cent Fe).

Statistical analysis of pilot plant spiral test results indicates that feed grade (Fe_f) is the only factor significantly

related to concentrate grade (Fe_c), the two factors being related by the linear regression equation;

$$Fe_c = 58.9 + 0.13 Fe_f$$

This equation, when applied to the original data, yields mean and maximum deviations of 1.1 and 2.3 Fe per cent respectively, indicating that the equation is not of sufficient precision to allow its use as a concentrate grade prediction model.

Analysis of the results of the pilot plant tests also showed that F_2 , the per cent weight of total silica contained in the +44 B.S. mesh fraction, has no significant association with concentrate grade, thereby refuting a belief held on the mine that coarse quartz in the mill feed was responsible for low grade concentrates, although it should be noted that the full scale mill tests indicated that F_2 and Fe_c are inversely related. Concentrate grainsize analyses indicate that the coarse concentrate fractions are invariably of above average grade.

On the basis of eleven pilot plant spiral tests it cannot be stated with certainty that a 5 + 5 + 5 - turn spiral system yields higher grade concentrates than a 5 + 3 - turn system although a tendency does exist for the higher concentrate grades to be associated with the greater number of spiral turns.

Full scale mill tests indicate that the concentrate grade (Fe_c) is significantly associated with mill feed grade (Fe_f) and X_6 , the per cent weight of total ore in the -16 +44 B.S. mesh fractions, the three variables being related in the regression equation;

$$Fe_c = 58.0 + 0.18 Fe_f - 0.13 X_6$$

When applied to the original beneficiation data this equation yields mean and maximum deviations of 0.5 and 1.1 Fe per cent respectively. This prediction model is the most efficient model available from the Marampa beneficiation data but is not of sufficient precision to allow its use as an absolute prediction model and it is concluded that other, unmeasured factors are responsible for the deviations between expected (predicted) and

observed (measured) concentrate grades.

Correlation analysis further indicates that the optimum grainsize range of the Marampa ore for spiral beneficiation is between 100 and 200 microns, about the range -85 +150 B.S. mesh.

Increasing amounts of +60 B.S. mesh ore are associated with decreasing concentrate grades because of the inclusion of non-liberated hematite and it is suggested that reduction of the ore to at least 90 per cent -44 B.S. mesh before spiral beneficiation would materially improve concentrate grade.

It is considered that the formulation of accurate prediction models relating concentrate grade to other measureable features of the ore will require detailed and comprehensive pilot plant and full scale beneficiation tests, the results of which should be studied with the use of objective statistical analysis.

World reserves of high grade iron ore have increased enormously in the last fifteen years and the supply of iron ore exceeds the present demand. In these circumstances the buyer can dictate the price to a certain extent and it is most important that iron ore producers working low to medium grade ores requiring beneficiation, reduce production costs and produce a consistently acceptable product in order to compete successfully and to maximise profits. Microscopic examination of the ores and mill products is necessary in order to determine the physical form in which the valuable minerals occur so that optimum control may be exercised on the beneficiation process.

The Marampa ore consists essentially of hematite, quartz and muscovite occurring in four major ore synclines, designated A, B, C and D. The ore synclines are separated by anticlinal bodies of quartz-muscovite schist waste.

The hematite occurs in two significantly different forms and it is considered that these differences, together with ore grade variations, are the major contributors to the Marampa grade control problem. The two main forms of hematite are;

- (a) Coarse grained, tabular to micaceous hematite with

common in situ brecciation. Hematite of this type is confined to synclines A and D which have undergone much more deformation than synclines B and C.

(b) Fine grained, equidimensional hematite commonly with annealed texture. This form of hematite is confined to synclines B and C.

The hematite of synclines A and D has a very well-developed dimensional and lattice preferred orientation which is lacking in the hematite of synclines B and C. The main features of the hematite which influence the efficiency of spiral beneficiation are the significant shape and size differentials, the annealed texture and the in situ brecciation.

The coarse and micaceous hematite tends to behave in one of two ways when undergoing spiral beneficiation.

(a) On overloaded spirals the micaceous grains tend to settle and bank on the spirals thereby interrupting the streamline flow of pulp down the spirals. Turbulent flow is thus initiated and may cause significant amounts of gangue minerals to enter the concentrate withdrawal ports. This tends to produce sub-standard concentrate grades.

(b) On underloaded spirals the micaceous hematite grains tend to behave similarly to other, less dense micaceous minerals and are swept around the outside of the spirals to report as tailings. This tends to reduce the iron recovery although the concentrate grade may not be affected.

The in situ brecciation of the hematite, caused by the more severe deformation suffered by synclines A and D, yields large numbers of minute sub-grains from initially much larger single grains and, because of their fine grain size, the small sub-grains are lost as tailings. In situ brecciation of hematite is not apparent in synclines B and C.

Secondary deformation twinning of hematite is common and reinforces the inherent strength of the hematite grains and is therefore a beneficial feature.

Electron probe micro-analysis indicates that the Marampa

hematite is essentially pure Fe_2O_3 , containing only very minor amounts of titanium, certainly less than one per cent.

The annealed texture occurring in the hematite of synclines B and C has two main effects on concentrate quality;

(a) Large composite grains composed of a number of smaller annealed sub-grains tend to behave as less dense gangue minerals on the spirals and eventually report as tailings. The annealed grains resist liberation from the composite grains such is the tenacity of the annealed sub-grains.

(b) The annealed grains often include grains of gangue minerals which thus report to the concentrate, tending to decrease the concentrate grade.

In synclines B and C, hematite often occurs as minute euhedral inclusions in quartz. This hematite is completely irrecoverable by spiral beneficiation and its presence introduces errors to the reporting of assay grades which in all cases must be considered as being only an indication of the maximum amount of hematite present and should not be used as a measure of the amount of recoverable hematite.

Calcite was noted in a diamond drill ore specimen taken from an unknown location and depth. The presence of calcite, if widespread, will be of benefit to the Marampa milling operations by tending to neutralize the corrosion caused by the production of sulphuric acid during Aerofall milling.

The manganese minerals in the Marampa ore appear to be concentrated along the margins of synclines B and C. X-ray analysis indicates that the manganese occurs in the form of lithiophorite $((\text{Li}, \text{Al})\text{MnO}_3 \cdot \text{H}_2\text{O})$. The fine grain size of the lithiophorite ensures that most of it reports as tailings when undergoing spiral beneficiation.

Statistical analysis of sample assay values indicates that a crude form of channel sampling, locally known as groove sampling, is an adequate method for sampling the Marampa ore but can yield anomalous values particularly if the ore hardness is related to the hematite content.

Posthole samples (tabular samples with areal dimensions of 12 by 6 inches and thickness of about 2 inches) taken at ten feet intervals (systematic posthole samples) along sample lines 200 feet apart, or taken at random locations along the sample lines (random posthole samples) yield mean ore grades similar to those given by channel sampling but have a much higher assay variance due to the fact that the smaller posthole samples may readily fall completely within an iron-rich or iron-poor band of ore. Posthole sampling at ten feet intervals could be used for purposes of ore reserve estimation but for purposes of grade control a much closer sample spacing would be required. Random posthole sampling is not recommended for either ore reserve estimation or grade control purposes.

Ore grades from channel samples follow a normal (Gaussian) distribution and, for the 1967 channel sampling programme, yield values for the mean and standard deviation of 45.9 per cent Fe and 6.72 respectively.

Analysis of ore grades indicates that the trend of mineralization at Marampa is strongly anisotropic and a rectangular sampling grid is more cost-effective than a square sampling grid. The greatest variation in ore grade occurs across the strike of the orebody.

Correlation analysis of channel sample assay values suggests that the 200 feet separation of the sample lines is too great to allow meaningful interpolation between adjacent sample lines and a sample line spacing of 150 feet or even 100 feet may result in more effective grade control and ore reserve estimation.

An increase in the size of the channels samples, from the present ten feet, to twenty feet may be warranted since the sample size or interval of ore samples (across-strike) should be related to the method of mining including the minimum width to which a single block of ore could be mined.

It is confidently predicted that, with increasing depth of mining below the original topographic surface of the Marampa deposit, the hardness of the ore will increase and the mean grade

of the ore will decrease. These changes in ore characteristics are caused by the waning influence of the leaching which originally upgraded the ore by abstraction of silica. Three major mining problems are expected to arise if the ore grade and ore hardness alter as predicted;

(a) The lower grade mill feed will tend to produce lower grade concentrates.

(b) Mining costs will increase because of the additional ore which will require beneficiation to produce a given amount of concentrate.

(c) The increase in ore hardness will necessitate alteration to mining and milling procedures and blasting and crushing of the ore will assume major proportions. Effective liberation of the hematite will become increasingly difficult, particularly in synclines B and C, and close control of the secondary crushing will become important.

Statistical analysis of the vertical variations in ore grade over the thirty feet height of a single mining bench failed to detect a statistically significant decrease in mean ore grade although a tendency towards such a grade decrease was certainly shown to exist. It is considered that statistically significant differences in mean ore grade between any thirty feet separation may never be detectable but that cumulative ore grade differences between successive thirty feet levels may indeed be very significant.

A significant increase in ore hardness has recently occurred in several parts of the orebody and it is not known, with any certainty, to what depths the leached ore extends below the present open pit. The hard ore encountered during the present mining operations probably represents "outcrops" of the completely unleached ore which is expected at depth.

Analysis of the assay values of auger drillhole samples from syncline A suggests that auger drilling for purposes of ore grade information would often fail to provide any additional useful information than that given by the surface channel sampling, at least over the thirty feet height of a single ore bench.

One of the major problems related to concentrate grade variation is the variations which occur in the grade and the hematite grainsize and grainshape of the ore passing on to the spirals. Fluctuations of these variables causes variations in concentrate quality because the mill operating conditions have been designed to treat an "average" type of ore and these conditions will not, at all times, be the optimum conditions for treating the range of ore types which occur at Marampa.

One approach to achieving uniformity of feed grade and hematite grainsize and grainshape may be to mine and treat ore from areas of reasonably uniform grade within a single ore syncline or from two synclines with similar ore characteristics, e.g. mine and treat ore from only synclines A and/or D or from only synclines B and/or C, but not from all synclines simultaneously. Mining of ore from synclines having similar ore characteristics has the effect of minimising the between-syncline grainsize and grainshape differentials and the mining of blocks of ore of fairly uniform grade minimises the effects of feed grade variation.

A mine plan based on the five-term moving mean of the channel sample assays can be used to delineate blocks of ore with mean grade falling within the following ranges;

- (i) > 50 per cent Fe.
- (ii) 40 to 50 per cent Fe.
- (iii) 30 to 40 per cent Fe.
- (iv) < 30 per cent Fe (discarded as waste).

These grade ranges are arbitrary and could be altered if experience of mining and milling such blocks dictated otherwise.

Blocks of ore with mean grades falling within the above ranges would be separately mined and milled, the mill variables (feed rate, pulp density and others) being regulated so as to treat ore of a particular grade and type. With a change in ore syncline or ore grade the mill variables would be retuned to deal with the different ore type.

Calculations of the channel sample assay variance indicates that mining to such a plan would reduce the variance

of the ore grades.

The 1968 Marampa grade control objectives were;

(a) Mean iron grade of mill feed to be equal to mean iron grade of proved ore reserves.

(b) Mean silica grade of mill feed to fall within the range 23 to 26 per cent SiO_2 .

(c) Mean concentrate grade to be 64.75 per cent Fe and 5.75 per cent SiO_2 , the silica content to be kept within the range 5.0 to 6.5 per cent SiO_2 .

(d) Silica contents of concentrate shipments to be kept within the range 5.5 to 6.0 per cent SiO_2 .

Examination of the 1968 mill feed grade variations, concentrate grade variations and concentrate shipment grade variations indicates that the grade control policy failed to attain these objectives, the 1968 grade variations being;

(a) Mill feed grade variations.

(i) Fe: from less than 40 to greater than 60 per cent Fe.

(ii) SiO_2 : from less than 5 to greater than 38.4 per cent SiO_2 .

The silica content of the feed rarely fell within the optimum range.

(b) Concentrate grade variations.

(i) Fe: from less than 60 to greater than 68 per cent Fe.

(ii) SiO_2 : from 1.0 to greater than 8.2 per cent SiO_2 .

(c) Concentrate shipment grade variations.

(i) Fe: from 63.70 to 65.75 per cent Fe.

(ii) SiO_2 : 4.28 to 7.40 per cent SiO_2 .

Sample assaying is the third stage of sample treatment after sample collection and sample preparation. The reliability of each of the stages is dependant upon the accuracy and reliability of each of the preceeding stages and this is particularly so in the case of assaying where errors in each of the two preceeding stages may be ingerited and may be the cause of assay bias even before assaying takes place.

Statistical analysis of a series of control samples indicates that the Marampa assaying technique (X-ray fluorescence

spectrometry) yields reasonably consistent results, i.e. the reproducibility of the technique is high, but the precision of the technique is low and the iron content of samples is apparently overvalued, particularly in the range 30 to 50 per cent Fe. The overvaluation of hilltop ore samples and mill feed samples is statistically significant and has serious economic implications and may be a partial explanation of the occasional failure of the mill to attain its target figures of concentrate grade and tonnage. Regression analysis of the control sample assay results has yielded linear regression equations capable of correcting the reported grades to true grades.

The analytical range and limits of the Marampa X-ray fluorescence spectrometer are designed to cater mainly for the determination of the iron content of the mill feed and concentrate samples and silica, alumina and manganese all have upper limits of detection which are readily exceeded particularly in the assaying of low grade samples.

The Marampa spectrometer gives assay values in the form of digital printout data which must then be arithmetically manipulated to give the actual percentage assay values. Conversion of the digital data to the assay values is a weak link in the overall assaying procedure and incorrect assay results due to conversion errors are common.

The four assay variables (Fe, Mn, SiO_2 , and Al_2O_3) require different arithmetic manipulations to convert from digital to percentage assay values and statistical analysis of a number of conversions indicates that the number of conversion errors is significantly associated with the number of manipulations necessary to make the assay conversions. The SiO_2 conversions require three (concentrate grade conversions) or five (feed grade conversions) data manipulations and errors occur for this conversion much more frequently than for Mn or Al_2O_3 conversions which require only that the assay value be read directly from calibrated conversion tables.

Incorrect assay values are worthless and costly and may give misleading results in the course of ore reserve estimation

or grade control and a self-checking assay conversion procedure has been designed for use at the Marampa laboratory. This procedure requires that the digital data be processed and the various data manipulations checked before the final conversions to assay values are made.

The results of this investigation of the Marampa grade control problem indicates that the problem is intimately involved with the different response of the different types of hematite to Humphreys spiral beneficiation and that the problem therefore is mainly a milling or mineral-dressing one. It is considered that detailed and comprehensive pilot plant and full scale mill tests will be required before the potential concentrate quality may be predicted from a knowledge of the in situ characteristics of the ore.

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