

APPLICATION OF GEOLOGY & GEOSTATISTICS AT THE

SCULLY MINE OREBODY, WABUSH, LABRADOR.

A THESIS SUBMITTED FOR THE DEGREE
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BY

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ABSTRACT

The Scully Mine exploits at 20 million tons per annum a Precambrian iron formation at the southern end of the Labrador Trough. Unlike the iron formations to the north, which are unmetamorphosed and leached with relative increase in the iron content, the iron formation at Scully is locally metamorphosed to amphibolite facies and of relatively low iron grade. The orebodies have been highly deformed by at least two periods of folding and faulting to a series of gently plunging close folds of moderate dip, but subsurface interpretation is difficult because structural features have been obscured by the metamorphism. Mineralogically the ore consists of quartz and hematite with much less magnetite, and up to 9.0% Mn. mainly in solid solution in the iron oxides and as pyrolusite.

The ore is beneficiated by gravity and high tension techniques to a concentrate. The main difficulty of the operation since its inception in 1965 has been to maintain concentrate grade, especially for manganese, within the close specifications set by the steel companies. This problem necessitates predictive mining grade control through daily and long term planning procedures which rest in turn on the (original) interpretation of the subsurface geology of the orebody. This research has

sought improvements through detailed statistical and geological methods. Statistical analysis of the test results of the original exploration drillholes has led to definition of eleven distinctive ore units within the 650 feet overall thickness of the ore formation. Furthermore, it is shown that after structural unfolding of the orebody, each unit displays simple internal assay trends. Mineralogical investigations support the subdivision of the orebody, and demonstrate that manganese was co-precipitated with the iron oxides, the recovery of which in the plant is related to their grain size in the ore.

This work leads to recommendations for improvement in techniques for ore reserve estimation and in procedures for mining grade control, by making use of suitable combinations of the bulk grade trends within ore units.

NOTE ON UNITS OF MEASUREMENT

Imperial units have been used throughout the text where these are the actual units of measurement that are used on the mining property or have been used in the publications pertinent to this study. The inclusion of the standard S.I. units would have led to considerable confusion, and inasmuch have been avoided.

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CHAPTER I

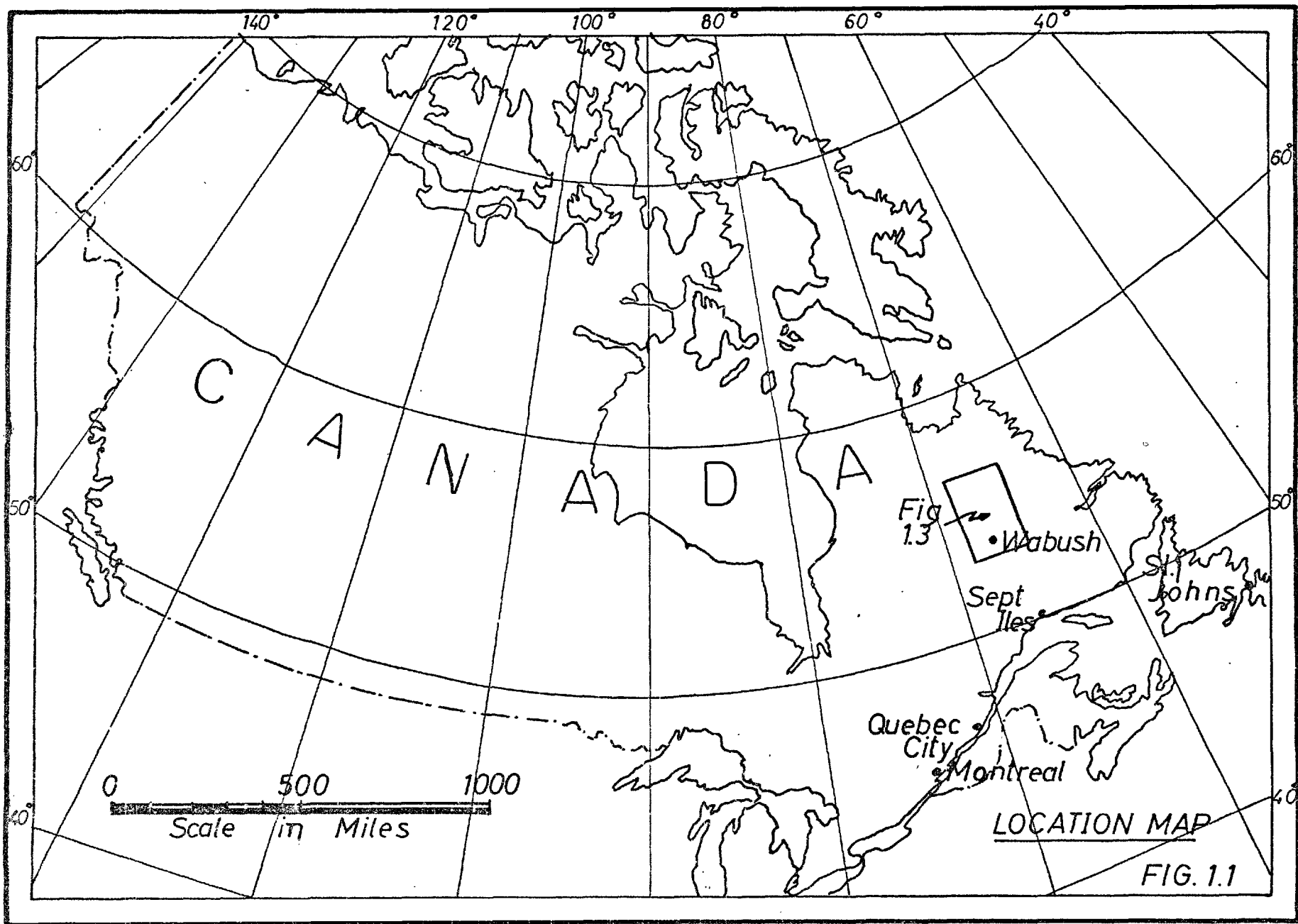
INTRODUCTION

A. GENERAL

The Scully Mine is situated in Wabush, Labrador at latitude $52^{\circ}55'N$ longitude $66^{\circ}50'W$. It is some 640 miles N.E. of Montreal and 650 miles N.W. of St. John's the capital of Newfoundland (see fig. 1.1.). Wabush is connected by rail to Sept Iles which is 186 miles South, on the St. Lawrence, and has regular air services to major points in Quebec and Newfoundland. Labrador City, a township built for personnel working at the Iron Ore Company of Canada's Carol project, is 4 miles to the north. Gagnon, centre of operations of Quebec Cartier Mining Co., is 88 miles South West of Wabush. Pickands Mather & Co. is the managing agent for Wabush Mines and together with seven international steel companies financed the development of Wabush.

The area is drained to the East by the Churchill River and its tributaries and about 30% of the surface of the area is lake covered. The relief is moderate with low lying land at about 2000' above sea level and the mountains to the North and West rising to 1000' above this.

The Labrador climate is notoriously harsh with very hot



summers and exceptionally cold winters. Precipitation is great, with 180 inches falling as snow and about 30 inches of rain in the summer months. The climate makes any outdoor activity unpleasant and difficult.

The area supports only a moderate growth with black spruce, jack pine, balsam and white birch being predominant. The growth is characteristically stunted. The tree line is at about 2,500 feet above which only white arctic moss, small shrubs and lichens are found.

B. EXPLORATION & HISTORY

The first report of iron ore deposits in Labrador was made by Father Louis Babel in 1870 and it was between 1803 and 1895 when A P Low working for the Geological Survey of Canada first recognised the Labrador trough and its economic significance. It was not until the early 1900s with the advent of the aeroplane that commercial companies became interested in the area. In 1929 W F James and J E Gill while looking for gold discovered what they considered to be economic iron ores at Knob Lake. By 1949 400 million tons of high grade iron ore had been proved at Knob Lake and the Iron Ore Company of Canada was formed to develop the area. In the same year exploration began around Carol Lake, to the South of the trough, by the Labrador Mining and Exploration Company, and in 1951 work began on the Mt. Wright, Mt. Reed, Lac Jeanine area. In 1953 Canadian Javelin began their work on the

Julienne peninsular deposit and the iron formation South of Wabush Lake, now known as the Scully Mine.

In 1954 the first iron ore was shipped from the area, over a purpose built railroad, from Knob Lake to Sept Iles and by 1958 Quebec Cartier Mining Co. had started work on the deposit at Lac Jeanine and were producing a concentrate in 1961. Similarly development work was started on the Carol project in 1959 and the Iron Ore Company of Canada was producing a concentrate there by 1962.

Development at Wabush began in 1954 and in 1961 a pilot plant was opened to test the feasibility of concentrating the ore. Production began at the Scully Mine in 1965.

Presently work is being carried out towards the mining of the Mt. Wright deposit and at the time of writing a large part of the over burden had been removed and the townsite building started. Some work has been claimed on the Julienne deposit although its feasibility is in question.

C. GENERAL GEOLOGY

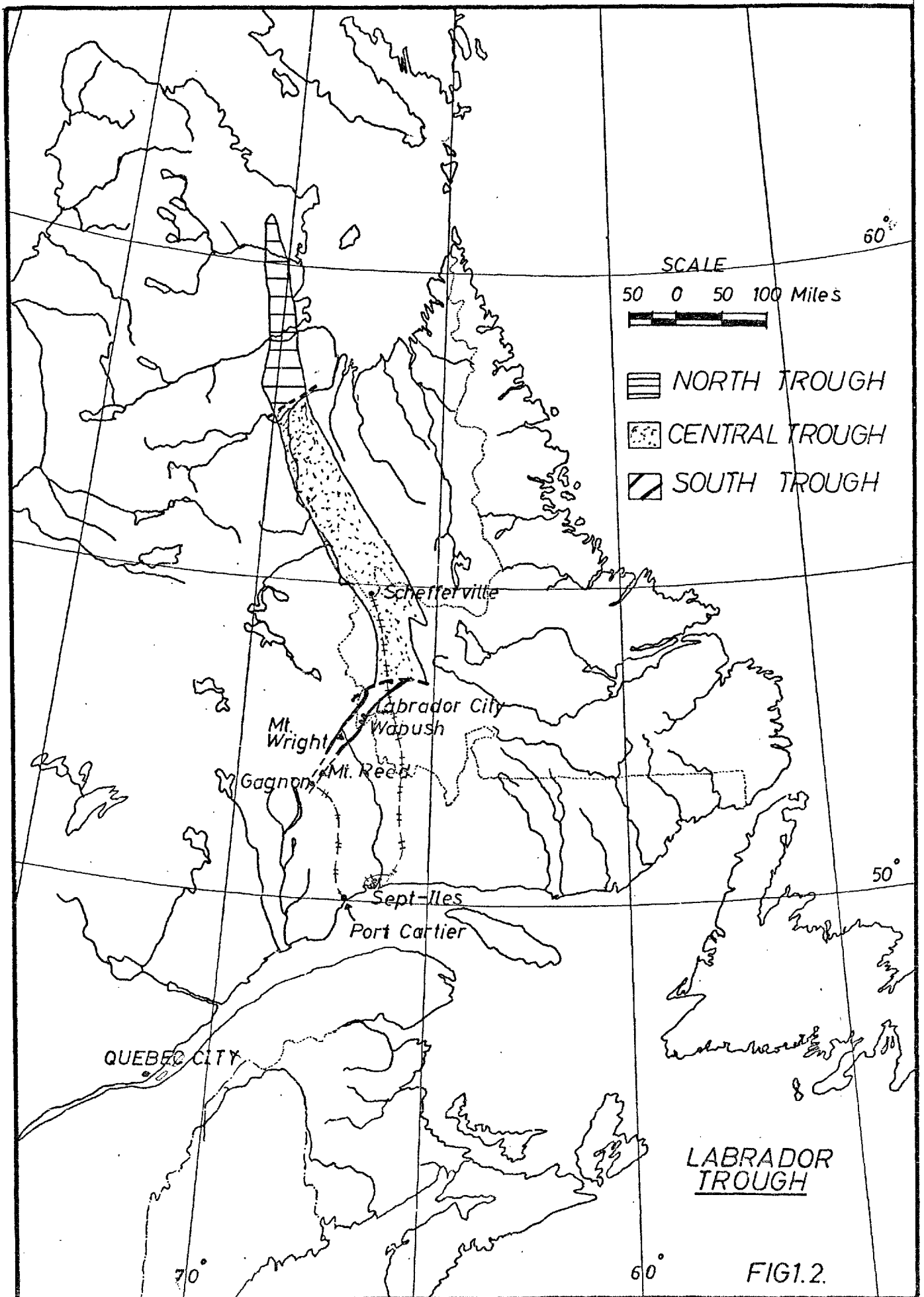
(1) General

The Labrador - Quebec - Ontario - Minnesota belt of banded iron ores has become classical in the discussion of sedimentary iron deposits. These iron formations are part of a succession of shelf type sedimentary rocks associated with minor tuffs and volcanics. The area

under study falls in the southern part of the so called Labrador Trough and this discussion will be confined to this area.

Apart from the work of Dr. G A Gross, who has described the area in a series of Geological Survey of Canada publications, and papers, very little work has been done on the area. Vast regions to the South and West of the Trough have not been mapped in any sort of detail. Only those areas of immediate economic importance have received detailed attention. Much of the knowledge of the general geology of the area is the same as it was in the early 1960's.

The Labrador Trough (see fig. 1.2) is a sinuous to arcuate belt of sedimentary, volcanic and intrusive rocks more than 750 miles long and 60 miles wide. They are Lower Proterozoic in age and lie unconformably on Archean granites, granodiorites and granitic gneisses on the West side of the Trough. The eastern border of the Trough is less easily defined, but in general Trough rocks are in fault contact with a series of gneisses, granites and amphibolites. The Trough rocks are naturally divided into three regions. In the central part of the Trough the rocks have been folded and faulted during one orogenic period, but remain essentially unmetamorphosed. In the northern and southern parts of the Trough the rocks have



been highly deformed and metamorphosed during at least two orogenic periods.

(a) Northern region

Here the rocks have been highly metamorphosed and deformed with iron formations now medium grained magnetite - hematite - quartz assemblage which would be suitable for beneficiation. Orebodies form in monoclines or fold systems where the succession has been repeated. To the present date no iron ores have been worked even at pilot scale.

(b) Central region

Iron formations in this region are not highly metamorphosed and form host rock for hematite - goethite ore deposits. The ore deposits were naturally enriched by the leaching of silica in Mesozoic times and re-mobilisation of iron causing secondary enrichment. The potential reserves of this area are large, and large deposits of unoxidised taconitic ore remain uninvestigated.

(c) Southern region

The iron formations of this region form the hub of the iron ore industry in Eastern Canada. The ores are not unlike the ores of the Northern region, in that they are coarsely crystalline hematite - magnetic - quartz rocks. The deposits lie within the Grenville

orogenic belt and are highly metamorphosed and highly deformed ores that have a history of at least two periods of faulting and folding. In the northern part of the region the rocks appear to have been metamorphosed to epidote - amphibolite facies but further to the South and West the grade becomes higher towards the centre of the orogenic belt. The metamorphism is regional and any effects attributed to minor intrusions are local in scale. Most prominent among the metamorphic changes is the increase in grain size and the iron formations have segregated into a few coarse grained minerals such that mechanical breakdown is simple. There appears to have been little or no diffusion between beds and thus the bulk composition of the original sedimentary units has remained unchanged, except for the loss of the volatiles. It appears that the final stages of re-crystallisation took place after faulting and folding so that not only are the original sedimentary features obscured, but also many structural features such as cleavage and fault zones.

(2) Geology of the Wabsuh Lake Area

(a) Regional Geology

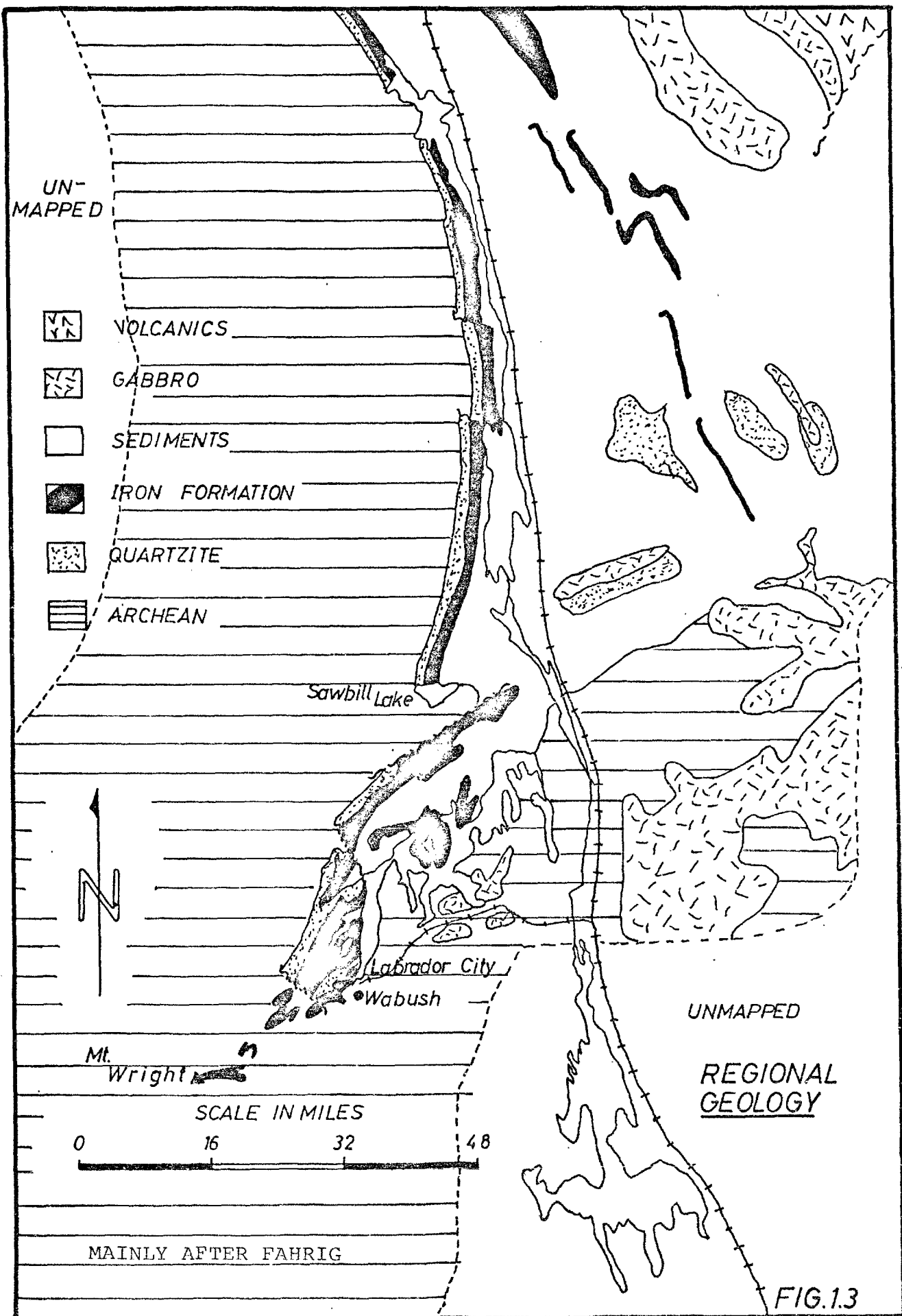
In the southern area of the Labrador Trough the succession of rocks overlying the basement gneisses, consists of former greywacke limestones and sandstones

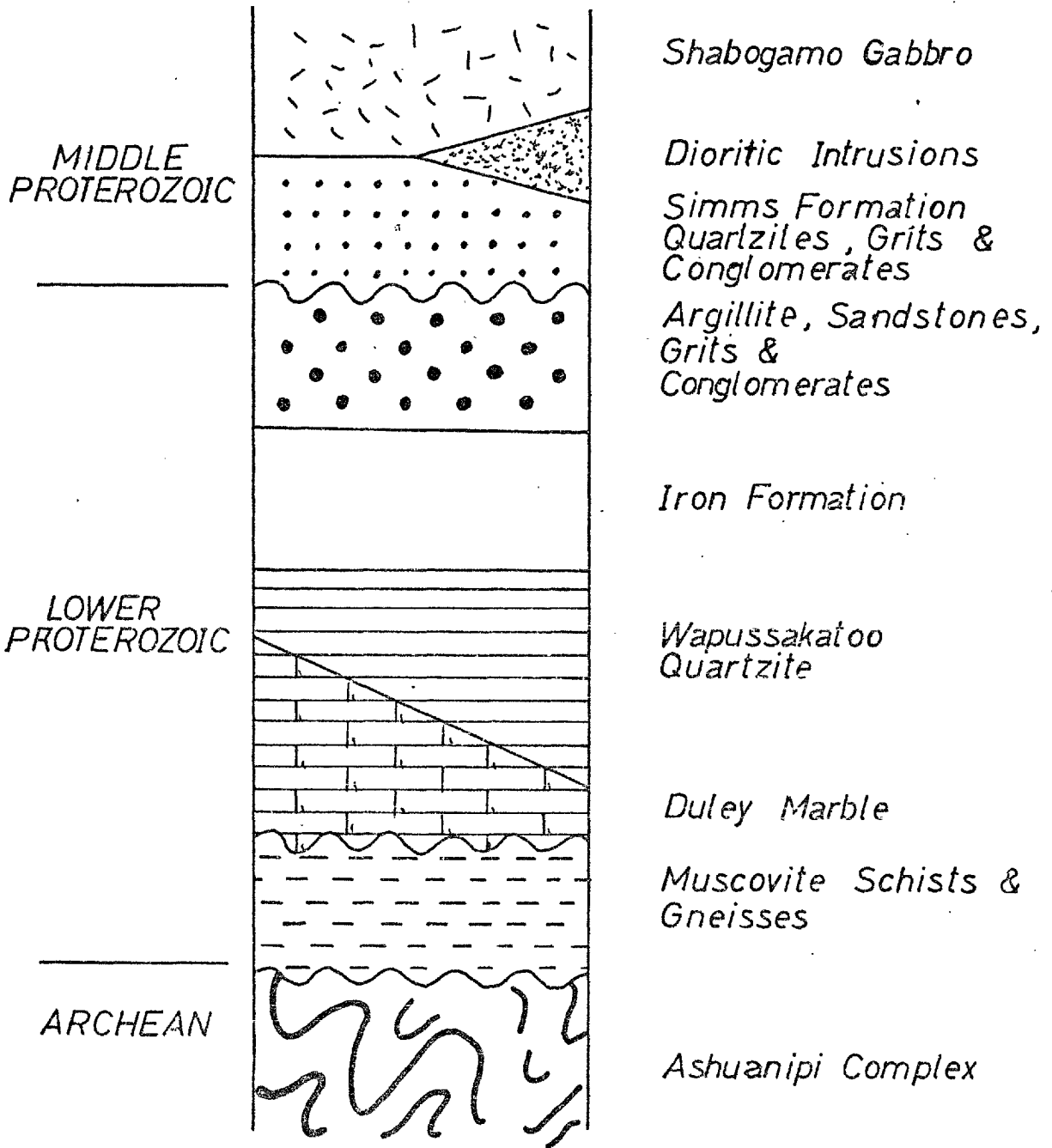
that have been regionally metamorphosed to mica schists, marbles and quartzite. Overlying the quartzites are the iron rich sediments of Lower Proterozoic age. These are predominated by a silicate member at their base and grade up into iron oxide facies. Above the ore horizon is a carbonaceous quartz conglomerate that grades up into graphitic slates. Gabbroic sills and dykes were intruded into the sediments in late proterozoic times. In places the area is overlain by recent sediments. The regional geology is shown in fig. 1.3 and a generalised stratigraphy in fig. 1.4.

(b) Local geology

The area is situated close to the intersection of three major structural provinces (see fig. 1.3). To the North is the Archean Ashuanipi complex, last deformed during the Kenoran orogeny.. To the South of the area and lying unconformably on the Archean are the rocks of the Lower Proterozoic which include the iron formations. To the East and overlying the Lower Proterozoic are altered sediments and volcanics of the Middle Proterozoic which were last deformed during the Hudsonian orogeny.

In plan the deposit at Scully Mine is seen to be a continuation of the iron formation from the North. South of Sawbill Lake, the rocks are considered to be





STRATIGRAPHY
WABUSH LAKE AREA

FIG. 1.4

part of the Southern Trough and it is here that the Grenville front crosses the geosyncline. South of that point the rocks have been regionally metamorphosed as indicated by the mineralogy of the sediments. Metamorphic grade increases in a southerly direction from the front, where greenschist facies is predominant, to the Wabush area where the rocks belong to the amphibolite facies. Further South and West at Mt. Wright the grade has increased to pyroxenite facies with large garnets and kyanite prisms being developed locally in alumina rich schists. The recrystallisation of the iron formation to coarse grained specularite-magnetite-quartz schists, has rendered them amenable to concentration.

D. SCULLY MINE

(1) Geology

At the completion of the exploration programme, the company geologists had divided the iron formation into five units:

Upper member	150'-200'
Middle member	200'-250'
Middle quartzite member	40'-60'
Lower member	180'-260'
Basal silicate member	70'

The footwall rock locally is a white coarsely crystalline

quartzite which lies on a coarse grained muscovite schist. The hangingwall rock is a carbonaceous quartz conglomerate.

Basal Silicate

This is predominantly a banded quartz amphibole rock with minor amounts of chlorite and biotite. The banding is generally schistose but individual layers may be up to four or five inches in thickness. Above the water table the silicate minerals have altered to limonite but still show the original fibrous silicate texture.

Lower Member

Although in many cases indistinguishable from the middle member rocks, the lower member ores grade up from the basal silicate with a gradual increase of the iron oxides relative to the silicate minerals. It had been thought that the unit was appreciably higher in its magnetite content and considerably harder than the other units, but I believe this is because the only good exposure of the ores, until recently, was in the South Pit area where the ores in general appear to be harder and have a higher magnetite content. The ores do show the banding that is classical in Pre-Cambrian iron ores, and this is probably their single distinguishing feature.

Middle Quartzite

This is a narrow, but persistent band of iron poor

formation, in which the iron content is generally less than 15%. The iron that is present is normally in the form of ochrous limonite and goethite, and these together with earthy manganese oxides fill the veins and vughs that run through the unit.

Middle Member

This, it is generally agreed, is the richest of the three ore horizons at Wabush (the quartzite and basal silicate being mined as waste). Prior to this work some of the units that form this member were recognisable in the field, but only with some difficulty. At the base of this unit is an ore with a distinct conglomeratic appearance. This grades up into the more friable homogeneous quartz specularite rock about 50ft thick. Above this is a leaner band with layers of limonite and goethite running parallel to the bedding. The rock above this almost always has a distinctive spotted appearance. The rock is generally homogeneous but for circular to ellipsoid vughs of manganese oxides. These have been the centre of some speculation as regards their origin, the most popular theory being that they are the residuals of garnets or spinels that have now decayed. Indeed below the water table some garnets have been seen in the ore but not in this unit. The ores above this defy an individual description but appear very similar in nature, being the coarse friable quartz hematite rocks

that typify the area. Little differences can be seen in the hand specimen between the low and high manganese ores.

Upper Member

The ores here are generally low in iron oxide and manganese oxide content and like the ore of the lower member have a distinctly banded appearance. Their distinctive feature in fact is their overall low grade, only just making the ore category in some places, in others being mined as waste. It has been suggested that at one time the unit was predominantly a quartz carbonate rock, and that the iron carbonates have altered to limonite giving the ore its brown colour. No evidence however can be put forward to defend this theory.

A plan of the surface geology is given in fig. 1.5.

Intrusive Rocks

The gabbros which intrude the iron formation to the north are not present at Scully Mine but the ore has been intruded by a series of fault controlled dykes in the South Pit. These now are almost entirely chlorite with a core of magnetite but have inherited a distinct fibrous fabric and may have been originally an amphibolite or lamprophyre. The presence of 'veins' of chlorite with an asbestiform texture, in the dykes has also been noted.

SCULLY MINE SURFACE GEOLOGY

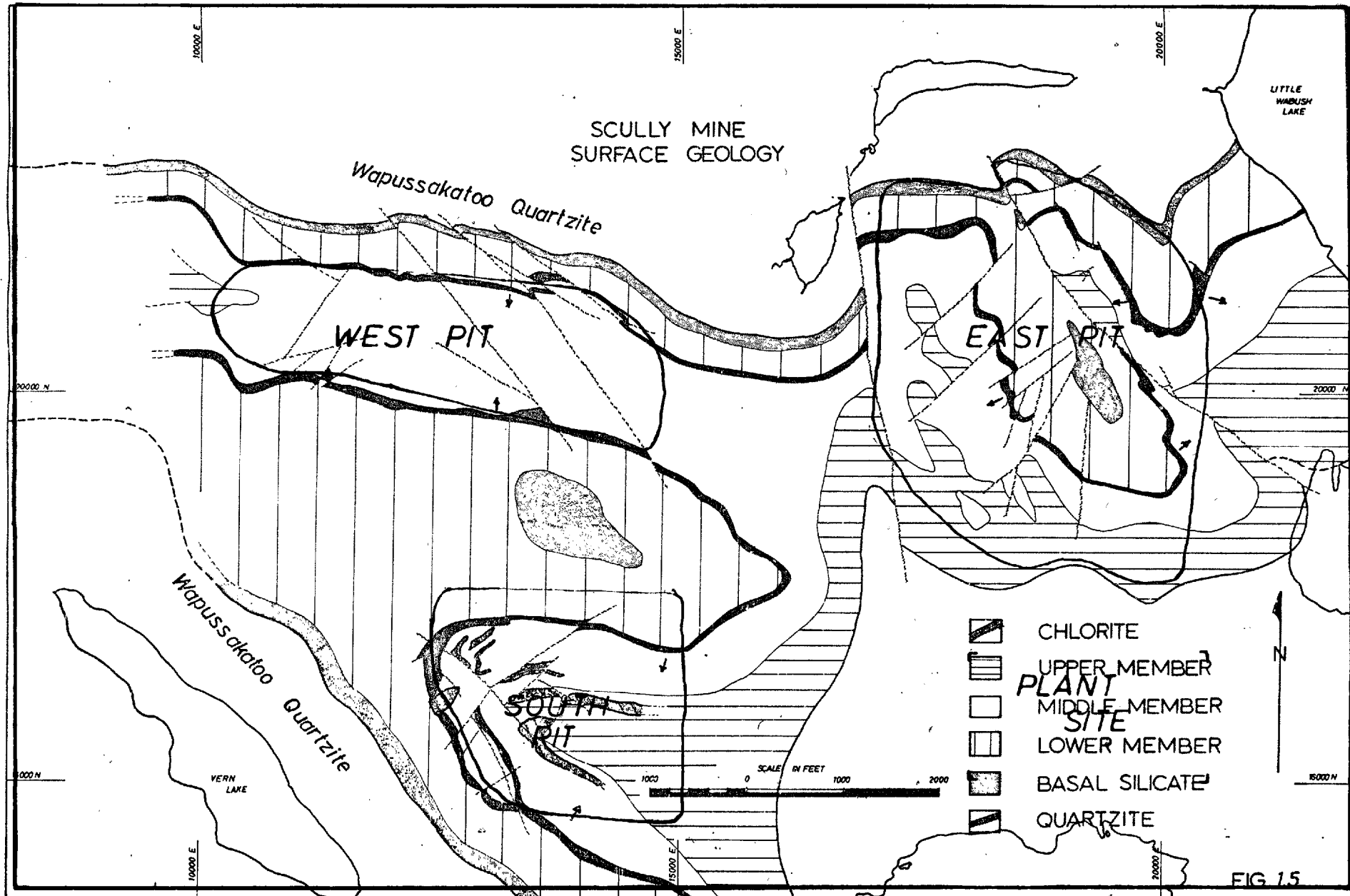


FIG. 1.5

Structure and Metamorphism

The outcrop pattern illustrated in fig. 1.5 is typical of interference patterns type 1 (see Ramsey 1967). The first fold axes run 090° to 070° whilst the axes of the superimposed folds run 150° . Two sets of faults are present, and post date the folding. The first set strikes 030° and are predominant in the East and South Pit areas. These dip at a high angle to the S.E. A later set of faults strikes 150° and dips steeply N.E. These tend to have a higher displacement than the first set and in the case of ore in the East Pit this displacement is of the order of 250ft. The main period of deformation occurred during the Grenville orogeny. The effect of the metamorphism is to remove almost entirely the trace of bedding from the ore horizons and surrounding rocks and impart a strong fabric to the rocks. This can be seen in the way in which the manganiferous vugs in the middle member ores change shape from almost circular to elongate ellipses. In this field there could be scope for some further study. The other effect is to completely re-crystallise the ores into coarse grained quartz hematite magnetite schists. Some work was carried out on the mine to find a grain size distribution for the ores. Polished sections were used and the long axes of mineral grains measured with an on stage micrometer. A number of points emerged from the study. Firstly that there was a great variation in grain size from one area of the mine

to another and in particular that the ores from the South Pit in general appeared to be somewhat finer than the others. Secondly, in general the grains were not flattened to any extent i.e. the ratio, long axis to short axis, was close to unity. This supports the theory that re-crystallisation was going on after the main period of deformation had finished. An exception to this again is the ores of the South Pit where a later period of intrusion and faulting seems to have modified the ores somewhat. Thirdly, that the distribution was gaussian or nearly so with the mode of the distribution in the 35 to 65 mesh sieve opening size (0.42 mm to 0.21 mm), and thus the ores are suitable for autogenous grinding and gravity separation (see fig. 1.6). The structure and grain size have had considerable bearing on the economics of the area.

(2) Mining Operations

The mining department is responsible for moving 20 million tons of material a year; of this 17 million tons are moved to the crusher and 3 million tons to the waste dumps. The operation is carried on with conventional open pit methods and equipment. Bench heights are restricted to 40 feet and bench slopes 1 to 2 with a 20 foot berm at each successive level. Normally operating levels are developed at both ends simultaneously using a loop haulage road system.

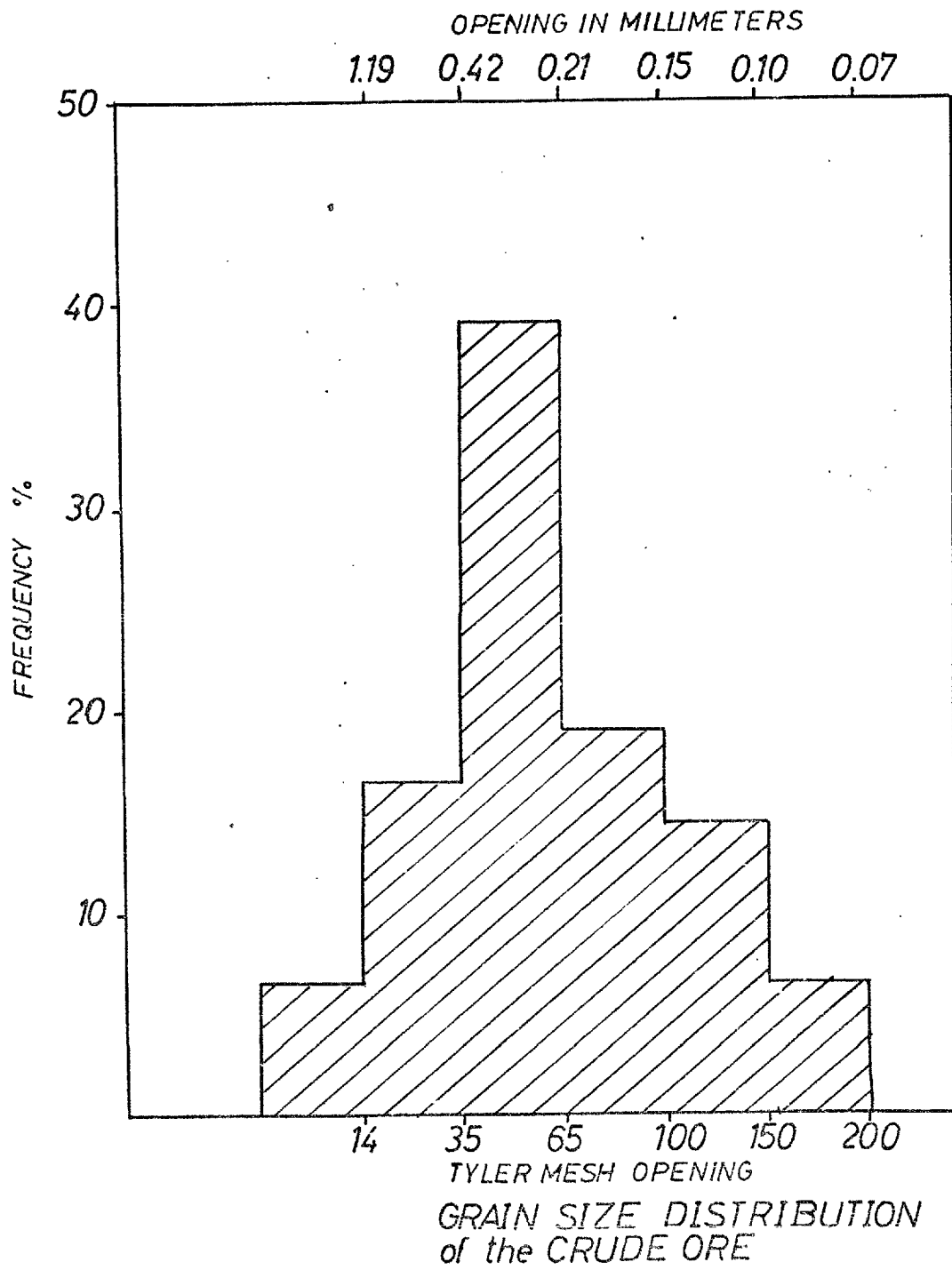


FIG. 1.6

Drilling and Blasting

Drill patterns for 12½" holes vary from 26' x 35' to 28' x 35'. Sub grade drilling is carried to 3 feet. Holes are loaded using packaged slurry and AN-FO delivered by truck in bulk form. The standard charge for a 12½" hole with a 26' x 35' pattern and 3 foot sub-grade drilling is 200 lbs as a toe load, 1300 lbs AN-FO and 13ft of stemming. Two primacord down lines feed two 1lb primers one at the toe and one in the slurry. A typical blast is six rows deep with a burden of 26 feet and spacing of 35 feet and the holes are staggered. The blast is squared off at each end and fired in rows to the open face with a 65 millisecond delay between rows.

Loading and Haulage

Seven 8 cu.yd. electric shovels are used to load ten 100ton electric wheel haulage trucks. The shovels are moved around the operation benches to optimise the crusher feed. Snow is a major problem, not so much from the standpoint of visibility or road conditions, but that it causes crusher plugging. The annual snow fall is around 180 inches and it is common practise to move the snow in advance of loading the ore. Haulage roads are 80 feet wide with a maximum grade of 5%. Haulage distances vary from 2000 feet to 12,000 feet.

(3) Ore Dressing

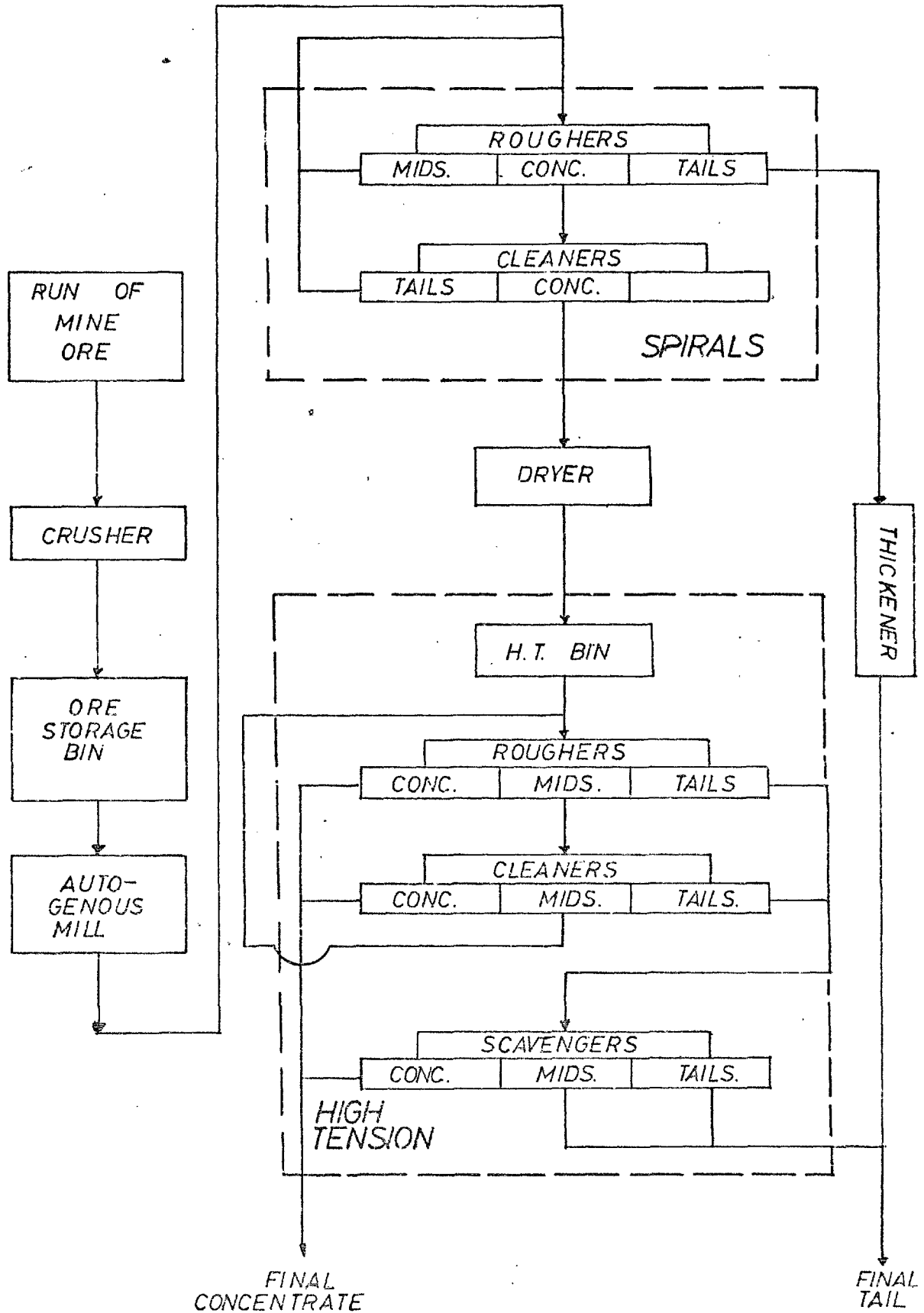
A simplified flow sheet for the operation is given in fig.1.7.

Crushing and Grinding

The crushing installation is comprised of two 54" x 74" gyratory units in parallel. Although only one unit is used at a time, the two crusher arrangement has been a major factor in maintaining a 90% availability. Normal crushing rate is 2,400 tons per hour but may reach 3,200 tons per hour at peak capacity. Maximum lump size from run of mine feed delivered to the storage bins from the crusher, is kept at 15 inches. Vibrating feeders deliver the ore to the mill feed conveyors and water added to the mill for the proper grinding density. The discharge ends of the mills are fitted with trommels and the trommel oversize immediately returned to the mill. The trommel screens have an opening of 2mm and give an effective spiral feed size of about 1mm.

Gravity Separation

Primary separation is by Humphries Spirals. The mills feed 1,152 rougher units and the concentrate from here is scrubbed on the same number of cleaner units. Rejects from the cleaner spirals are returned to the rougher spirals for re-processing without re-grinding. Grade of the spiral concentrate is approximately 6% SiO_2 and 63.5% Fe.



SIMPLIFIED FLOW DIAGRAM

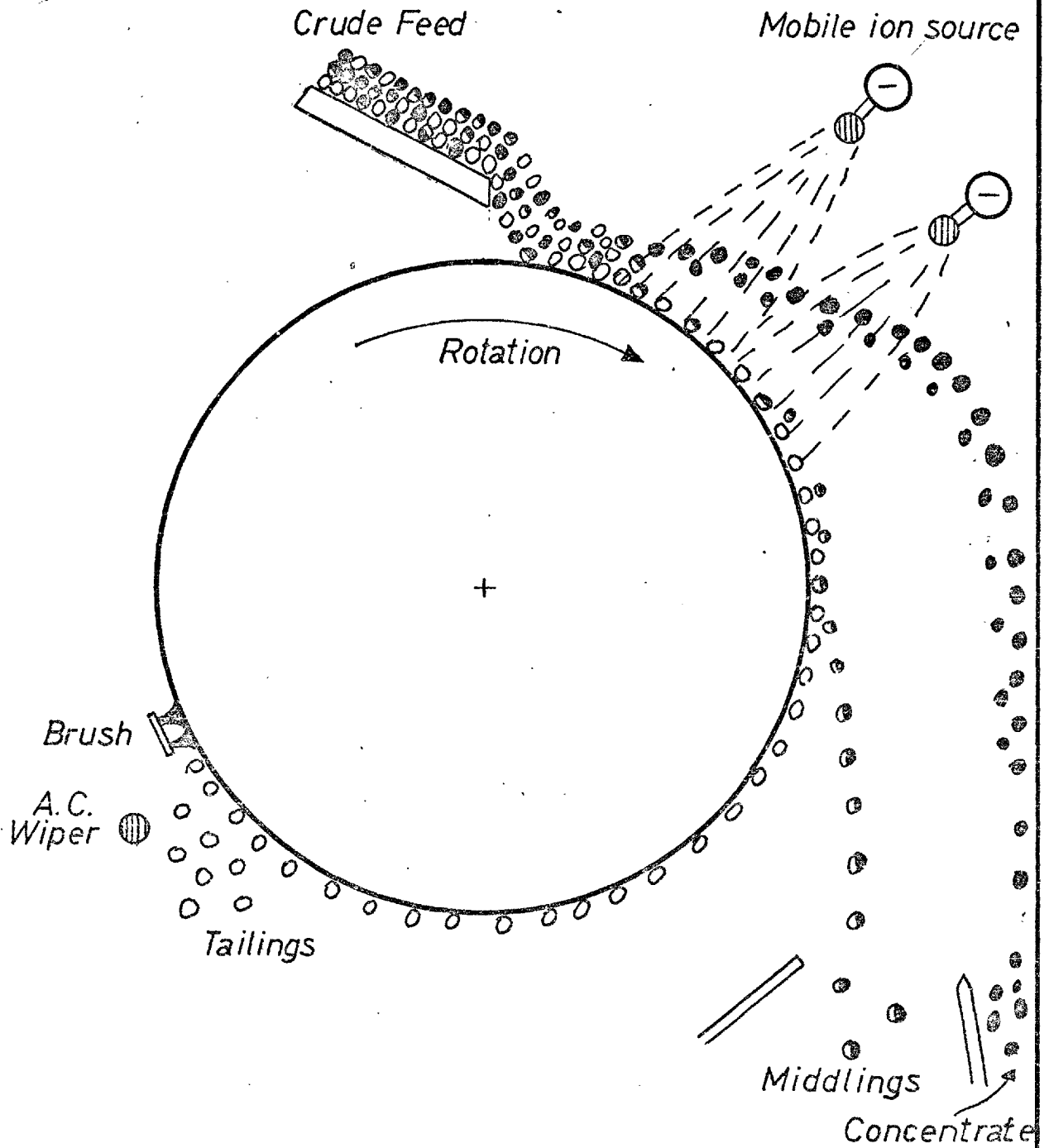
FIG.17

Drying

In order to provide a bone dry feed for the high tension separators, the spiral concentrates are dried through three oil fired fluo-solid reactors. Air at 1800°F is blown through a refractor dome which distributes the air through a 3 ft. bed of concentrate at sufficient velocity to expand and fluidise the bed. For proper distribution of the air the dome is fitted with 280 nozzles known as tuyeres. A classification of fines into the upflowing air stream above the bed meakes it necessary to scrub the discharge gases, using two air cyclones and a wet scrubber on each dryer.

High Tension Separation

The electrostatic separator uses the selective separation of dry mineral particles primarily on the basis of their electrical conductivity. The modifying factors are specific gravity, size and shape, surface condition and purity of the mineral particles. Fig. 1.8 illustrates the process diagrammatically. The charging of particles in the process is achieved by a bombardment of the concentrate, with negative ions from two ionic beam electrodes, placed at a distance from the rotor. The concentrate is saturated with a negative charge. The hematite, being a good conductor, loses its charge to the rotor, immediately it passes beyond the range of the ion beam and adopts the same charge as the rotor; it is repelled. Silica, on the other hand, does



THE HIGH TENSION SEPARATOR

not lose its charge and sticks to the rotor and is carried around. Both roughers and scavengers produce a concentrate, middling and tail. The middling goes back as a re-circulating load and the tail goes to a scavenger unit to produce a concentrate and tail (see fig. 1.7). Final grade is 66.3%Fe., 1.69%Mn., 2.25%SiO₂.

The dry concentrate is shipped in sealed railway cars to Pointe Noire, Quebec where it is pelletised and transported to the steel companies.

E. BASIS FOR THE WORK

The background and basic ideas on the nature of the research that was required at the Scully Mine, in order to improve the operation, were generated during four years employment with Wabush Mines as a mine geologist. During this period I was exposed directly to almost all sides of the operation, not only the mining and mine planning, but also beneficiation and process development. My main responsibilities were within the Mining Department and covered grade control problems, ore reserve estimation, day to day pit mapping and the layout and evaluation of exploration work. Many of the problems that are discussed in the text were discussed openly on the mine but the problems that arise during the day to day running of a large open pit operation over-ride the longer term problems of ore reserve estimation and a solution is never found.

The bulk of the research has been completed during a three year stay at the Royal School of Mines, London. During this time a great deal of use has been made of the facilities in the Mining Geology Department and benefit gained from conversations with many experts in the field of Mining Geology. Extensive use has also been made of Imperial College CDC 6400 computer and the University of London CDC 6600 computer, and a total of over 150,000 seconds central processor time used.

F. ACKNOWLEDGEMENTS

A generous Research Bursary was awarded by the R.T.Z. Educational Trust and this support is gratefully acknowledged. Also I would like to thank Wabush Mines for providing the facilities for field work and for the co-operation of their staff during the two summers I spent in Wabush. Wabush Mines also provided a magnetic tape on which all the exploration data for the ore body has been recorded; without this the work would have been impossible.

The main body of the work was supervised by Mr.C.J.Dixon and Professor G.R.Davis and their suggestions, guidance and constructive criticism is acknowledged with thanks. My fellow research students and the departmental staff provided an atmosphere for work that encouraged me and gave me confidence. Mr.B. Foster, Mrs. K. Irvine,

Mr. E. Hill and Mr. J.A. Gee and his staff, gave technical advice and played no small part in the preparation of this thesis.

Dr. Balderson of the Rutherford High Energy Laboratories gave me a great deal of help with tape conversion programmes.

Finally the support, both moral and financial, of my wife Ingrid is very much appreciated.

CHAPTER II

ORE RESERVE ESTIMATION AND GRADE CONTROL AT SCULLY MINE

A. INTRODUCTION

Estimation work is carried out on both a short and long term basis, up to the entire life of the mine. During the initial planning of the mine it was assumed that for the first 30 years of its life, no ore would be mined below the present water table. Inasmuch all 'live' planning is confined to that outlined in the 30 year plan.

It follows that the objectives of each level of planning will be different. The 30 year plan for the most part will be concerned with the generalities of the ore body and naturally less concerned with detail than, for example, the daily plan (ore grading). However we would expect to find each successive level of planning contained within the previous estimate.

Dall Negro (1969) outlined the system of planning at Wabush as follows:

The 30 year mining estimate is the basic tool for all the mining estimates. It must supply the following information:

- (1) The ore and waste tonnages (known and probable) and consequently the ore to waste ratio for the ore body.
- (2) The economical ore to waste ratio
- (3) The production rate
- (4) The overall chemical and metallurgical characteristics of the ore body. This gives the ranges of values, within which mining must be kept to operate in a well balanced way over the life of the mine.
- (5) The final pit design based on the contacts between the iron formation and the confining rocks.

It is obvious that it is necessary to avoid all errors in estimating the grade and tonnage of ore, however at this level of planning it is not necessary to have detailed knowledge of the contact between ore types.

The 5 year and 1 year mining estimates are contained within the 30 year estimate and are designed to meet the overall tonnage and grade requirements. Geological requirements for this estimate are still low. They should indicate how mining will develop in the pits and outline the requirements for such items as stripping, roads and equipment needs. The five year plan should include five one year plans.

The monthly mine estimate is established exclusively on

the contingent needs of the mine operations. The only link with the longer term estimate is the proportion of ore to be mined in each pit. No information is available to establish these estimates with any precision and therefore its fulfillment depends only on the probability that the mining of such quantities of ore from each pit will supply an average close to the required mill feed. It can happen, that one blast will supply a high proportion of waste or unsuitable ore type. Under these conditions, new solutions have to be found to make daily grade control possible and an emergency state created which is detrimental to the operation.

Daily estimate (grade control) is an attempt to provide the mill, on a daily basis, with a homogeneous and constant feed, within a certain range. At present the weight recovery and concentrate manganese content are of prime importance. This is achieved by the blending of ore from a number of shovels and is only possible if the engineer in charge has a precise knowledge of the ore types in front of each shovel.

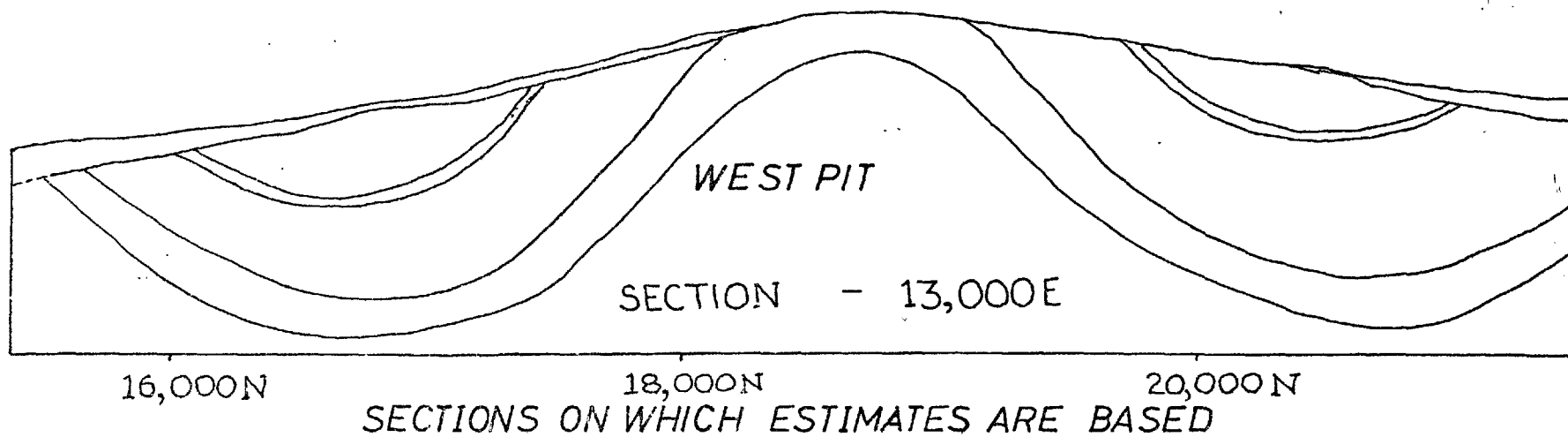
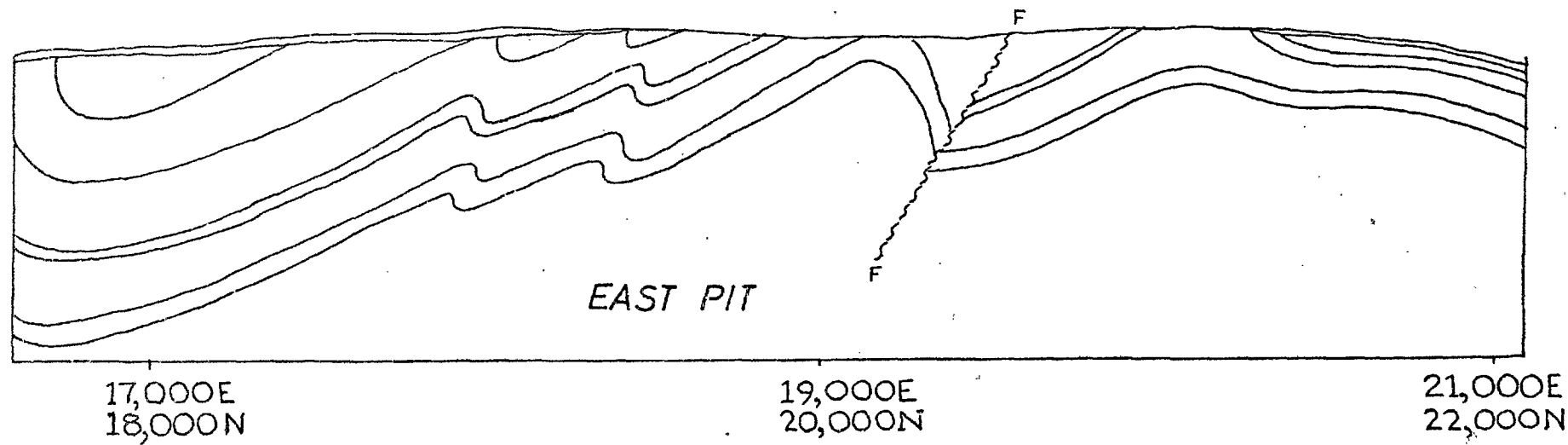
The 1 year, 5 years and 30 years estimates are considered long term. These use diamond drill hole information on sections, as their data source. The monthly and daily grade plans are short term and rely on blast hole data on bench maps for their data. Very little attempt has been made to date, to reconcile these two sources of data with each other.

B. ORE RESERVE ESTIMATE PROCEDURE

In the category of ore reserve estimates are included the 1 year, 5 year and 30 year mining estimates. The 30 years estimate was drawn up in 1964 for the West Pit and revised somewhat for the East Pit in 1968. The other two estimates are based on the 30 year estimate.

Between 1957 and 1964, one hundred and eighteen thousand feet of drilling was completed to cover practically all the areas where mining now takes place. In the East and West Pits a 250' x 250' square pattern was completed and in the South Pit a 250' x 500' pattern drilled, but filled in to a 250' x 250' square pattern in recent years. In addition holes were scattered in the area to the west of the West Pit area to prove the continuation of the ore. It is necessary to point out here that only a few of these holes go through to the footwall rock. Many holes have been terminated at the water table.

The diamond drill samples were grouped into composites of different length and assayed for crude iron, crude manganese and crude magnetics (Davis Tube Method). The samples were also tested on a shaking table, to simulate plant spiral concentrating characteristics, and a weight recovery and concentrate assays for iron, silica and manganese reported. The test procedure was identical



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FIG. 2.1

to that used at present on exploration and blast hole samples, and will be described below. The diamond drill holes, with their assays, were reported on vertical sections drawn as close as possible perpendicular to the strike direction of the ore body (North-South sections for the West and South Pits, East-West for the East Pit). The distance between sections is 250' (except for the South Pit where sections are 500' apart) and in general bore-holes are 250' apart on section, although in some areas of the ore body it is greater.

A general structure of the ore body was first established from field evidence and contacts obtained from diamond drill data. However it is interesting to note a geologist's comment at the time '... it has been difficult to make bedding correlations between adjacent drill holes, on the appearance of the iron formation. Furthermore, correlation has been complicated by the fact that not all of the drill holes go through to the bottom rock, at the base of the productive iron formation'. The situation was eased considerably by the recognition in the East Pit of the Middle quartzite and its subsequent recognition in the other two pits. The resultant structural interpretation is shown in fig. 2.1. In the West and South Pit zones there are two synclines separated by an anticline, with a general strike East-West and two synclines separated by a faulted anticline in the East Pit. Here the fold axes strike North West - South East. Using this basic idea the main contacts were drawn as smooth regular curves.

Assays from each hole were split into the classifications shown in fig. 2.2. and the limits for these 'ore types' drawn in parallel to the general structure of the ore body. The area of influence of an assay in a bore-hole was extended halfway towards the next bore hole or a maximum of 375' and half way to the next section. An example from a section is shown in fig. 2.3.

Preparation of Geological Data for Machine Input

The drill hole information and complete analysis plotted on sections drawn to a scale of 1 inch = 50 feet provides the basic input data required for Ore Reserve Estimate programme. The area of each ore classification was measured within the limits of (i) the area of influence of a bore hole on the section and (ii) the confines of one level. Since the programme is designed to compute the tonnage from the area and the weight factor a list of the areas measured, the appropriate section, drill hole and level, together with the complete analysis was made available.

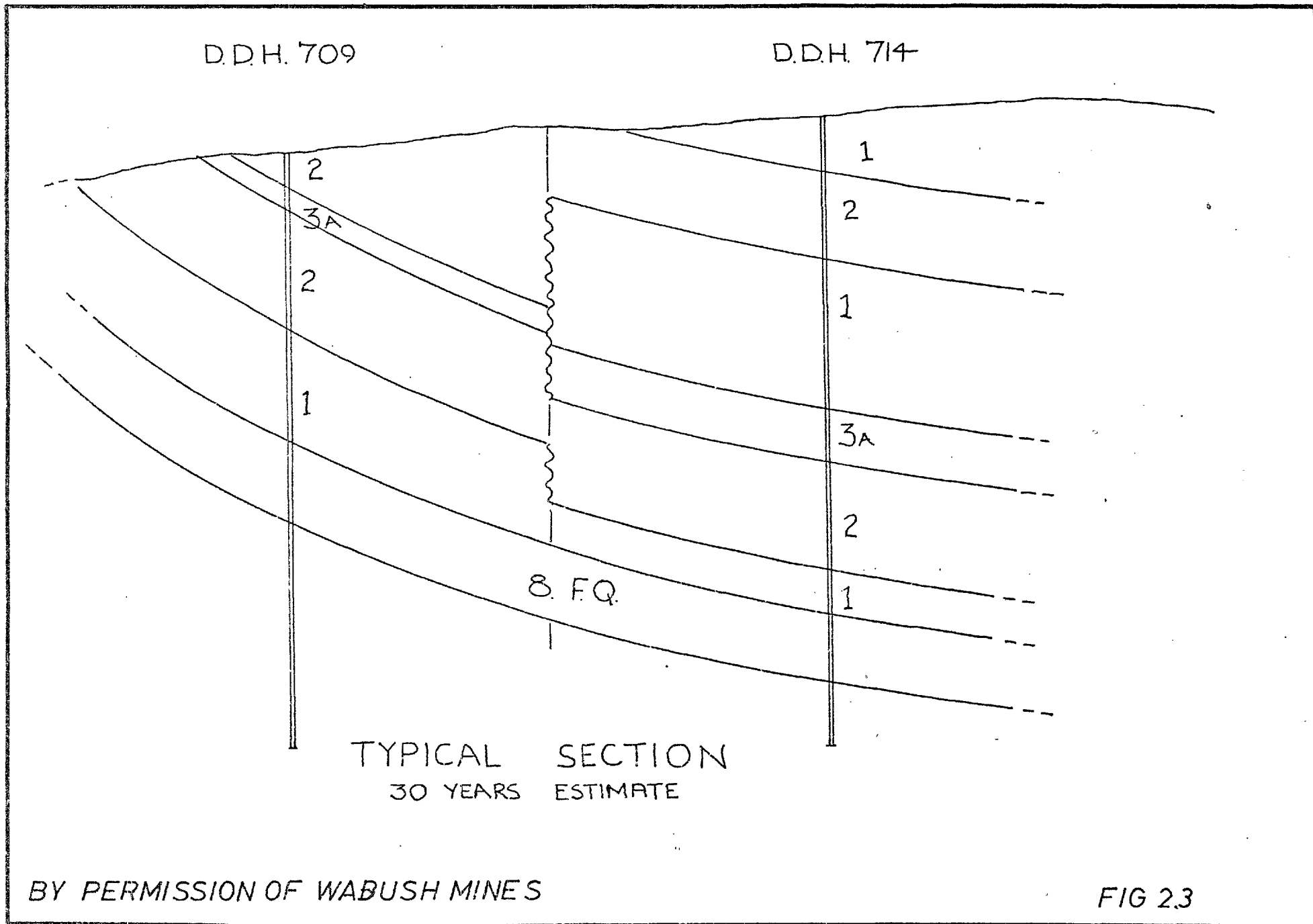
In a large number of cases more than one analysis was available for one area. In such cases each analysis was listed separately together with the area, bore hole section etc., and a weight factor assigned to each listing according to the ratio of each analysis that falls within the area. The procedure for indicating the codes (Ore Classification) to represent the average analysis for the area in question as calculated from the individual analysis and weight factor is

MATERIAL LEGEND

							<u>Min.Fe.Units**</u>
1	Ore	Plus 50% Wt. Rec;	Plus 60% Fe				3,000
2	Ore	40-50%	"	";	"	"	2,400
3	Ore	30-40%	"	";	"	"	1,800
4	Ore	25-30%	"	";	"	"	1,500
5	Ore	20-25%	"	";	"	"	1,200
6	B.G.	Plus 20%	"	";	55-60% Fe;		1,100
7	B.G.	Not in any of the other classifications					
8	-	Middle Quartzite					
S	-	Surface					

NOTE: Minus 4% Mn 1-7
 4-6% Mn 1A-7A
 Plus 6% Mn 1B-7B
 B.G. Below Grade

** Fe units = Wt. Rec. x Conc. Fe.



explained elsewhere in this paper.

The drill hole information as sent by the Mining Department which represents the raw source data, is keypunched onto cards and verified. These cards are then sorted and a listing prepared in card number sequence to check the information for possible errors in coding, keypunching and verifying. After any invalid data has been corrected this data is then stored as a random access file for future processing.

Raw Source Data Reduction

The raw source data stored on permanent files is copied to tapes, sorted according to section, level drill hole and area. The raw source data reduction programme was designed to read this data and based on the area, weight factor and card groupings from the same area, calculate the crude and concentrate tons. This process of grouping cards from the same area with appropriate section, level and drill hole is repeated for all the data. The area and the weight factor determine the tonnage assigned to a particular area. The various assay analysis are then weighted and the weighted averages calculated for the same group of cards belonging to one particular area. This result is then edited and printed as shown in figure 2.4. and the 'SUMMARY CARDS' prepared from this programme has the complete information for long term planning reports. The Summary Reports can be specified

DRILL HOLE ANALYSIS-RAW DATA REDUCTION

SCALE OF SECTION 1 INCH EQUALS 50 FEET DISTANCE BETWEEN SECTIONS 250 FEET TONS/SQ IN = 48076.92 CU YDS/SQ IN = 23148.15

CODE	SECT LEVEL	D.H.	CARDNO	AREA	CRUDE TONS	CR FE	CR MN	MAG REC	WT REC	CONC TONS	CON FE	CON SI	CON MN
1A	21250N 1655	1054	00001	.10	4,807.69	43.65	5.15	.26	63.85	3,070.19	61.84	1.90	5.00
1A	1615		00002	.93	44,711.53	43.65	5.15	.26	63.85	28,552.78	61.84	1.90	5.00

HASH TOTAL 2,99,518.67

FORMS COUNT 05

DETAIL CARD COUNT 101

SUMMARY CARD COUNT 075

DRILL HOLE ANALYSIS - ORE RESERVE ESTIMATE

SUMMARY 1

SCALE OF SECTION 1 INCH EQUALS 50 FEET DISTANCE BETWEEN SECTIONS 250 FEET TONS/SQ IN 48,076.92 CU YDS/SQ IN 23,148.15

CODE	SECT LEVEL	D.H.	AREA	CRUDE TONS	CR FE	CR MN	MAG REC	WT REC	CARD No.	CONC TONS	CON FE	CON SI	CON MN
1A	21250N 1615	1054	.10	4,807.68	43.65	5.15	.26	63.85	00001	3,070.18	61.83	1.89	4.99
1A-5A			.10	4,807.68	43.65	5.15	.26	63.85		3,070.18	61.83	1.89	4.99

HASH TOTAL 2,999,518.67

FORMS COUNT 08

SUMMARY CARD COUNT 075

according to the needs of the Mining Department as mining progresses in different areas of the pit. Hence this system provides adequate flexibility for advanced planning depending upon tonnage and grade requirements.

Inclusion of Ore Classification

This part of the programme was designed and included to save time and effort on the part of Mining Engineers in calculating and coding the ore classification in raw source data. The coding involved calculating the average analysis for the individual areas and the same code to be used for each separate entry for any particular area, the repetitive process was time consuming and prone to error. With the introduction of the matrix table in the Ore Reserve Estimate programme this resulted in a better classification of group codes for differentiating between ore and waste in the present method. The material legend as practiced at Scully Mine as shown in figure 2.2.

The procedure for computing tonnages and weighted assays has been explained earlier. The procedure for including the material code was simplified by building a decision table matrix in the programme and a search through this table after calculating the concentrate iron and concentrate manganese provided the correct code. However it is still necessary to indicate the following material codes:

<u>Material</u>	<u>Code</u>
1. Ferruginous Quartzite	8
2. Surface	S
3. Waste	W
4. Probable ore	P

The material code is then punched onto output 'SUMMARY CARDS' for preparing the summary reports.

Ore Reserve Estimates by Pit, Section & Level

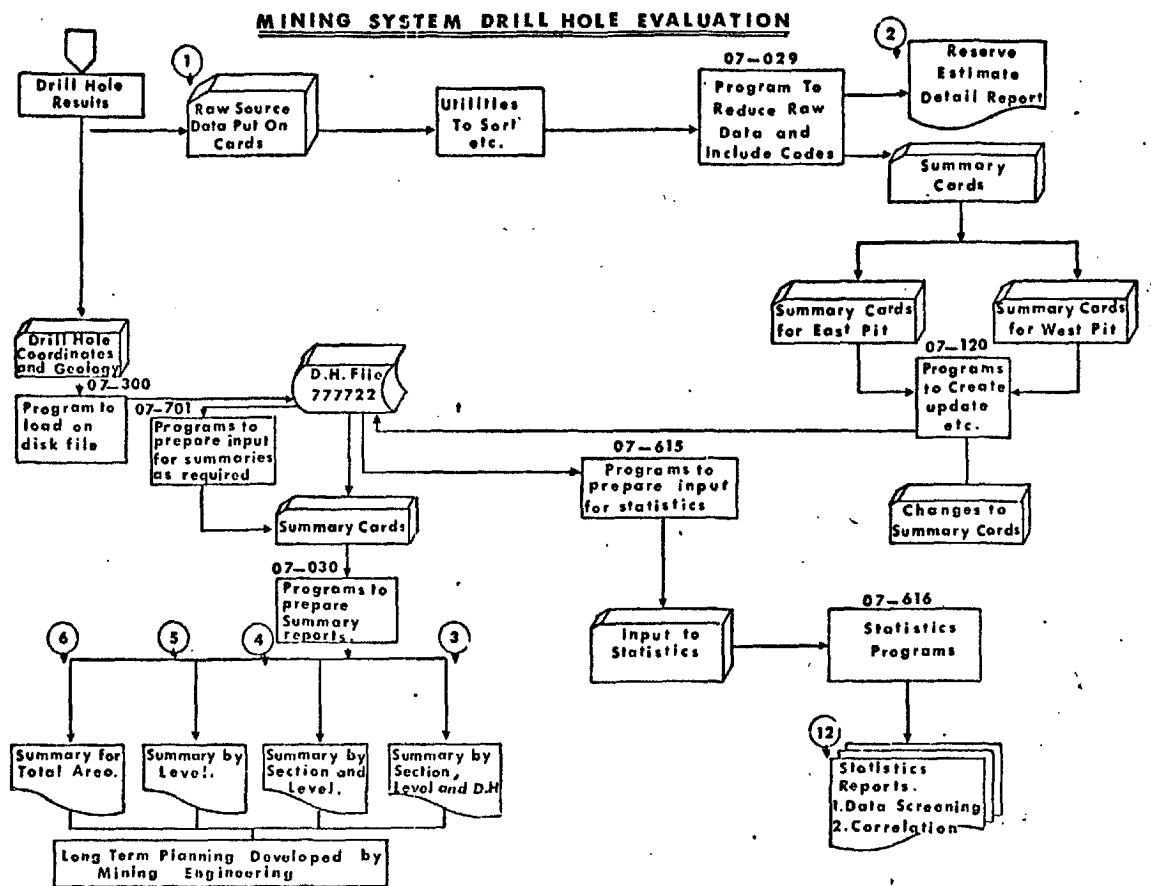
The final part of the programme design and implementation was to prepare the four summaries as follows:

1. Summary by Ore class, drill hole, level and section.
2. Summary by Ore class, levels and sections.
3. Summary by Ore class and levels.
4. Summary by Ore class for the whole pit.

Summary 4 denotes 1-6 material total to give ore, 1-6B as lean ore and lean material; 7, 7B, 8 and W as waste rock and S (in cu yds) as the stripping.

See figures 7 & 8. The other summaries breakdown gives essentially the same information in greater detail.

The 'SUMMARY CARDS' are input to the Ore Reserve Estimate programme to prepare the above mentioned reports. The programming, testing and final preparation of the reports was tackled systematically and the efforts resulted in considerable improvements over the previous method. The testing, sorting and final editing phases were all internally built in the programme phase as individual modules. The



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BLAST HOLE EVALUATION FOR ORE GRADING

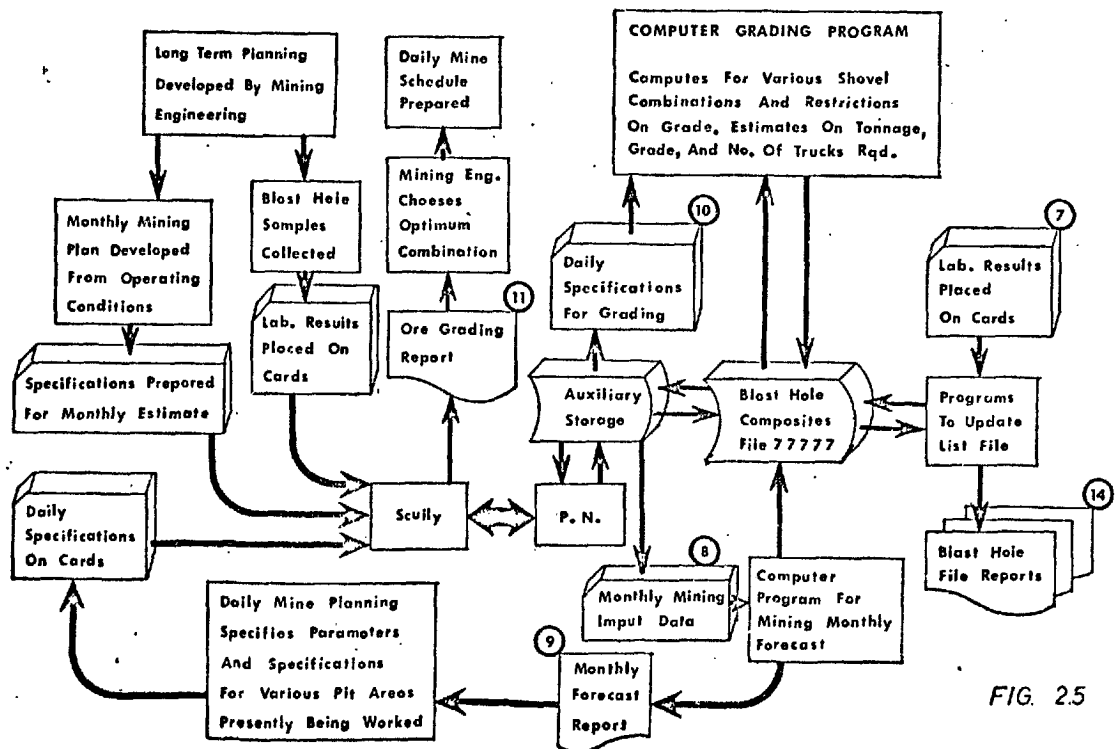
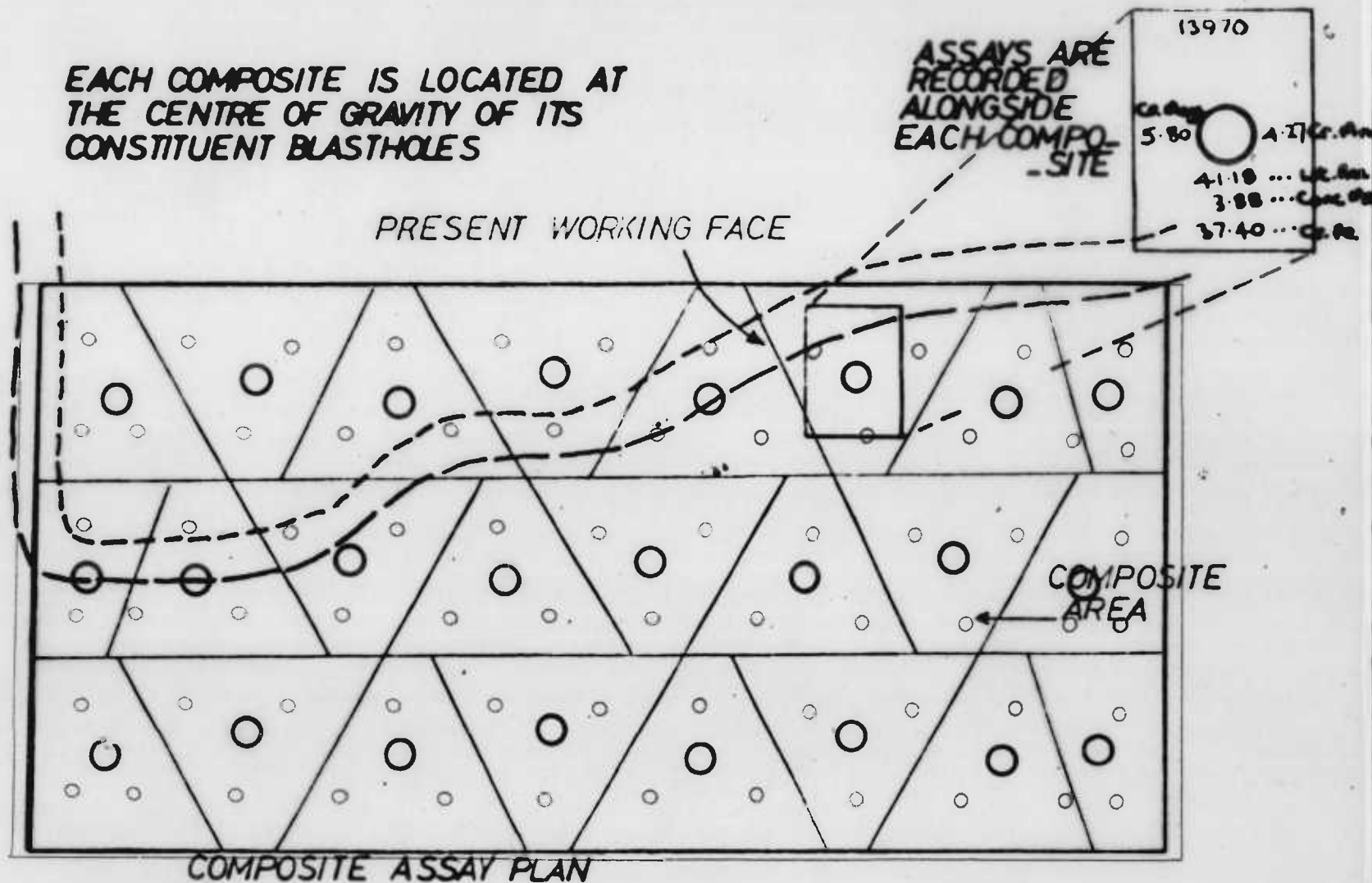


FIG. 25



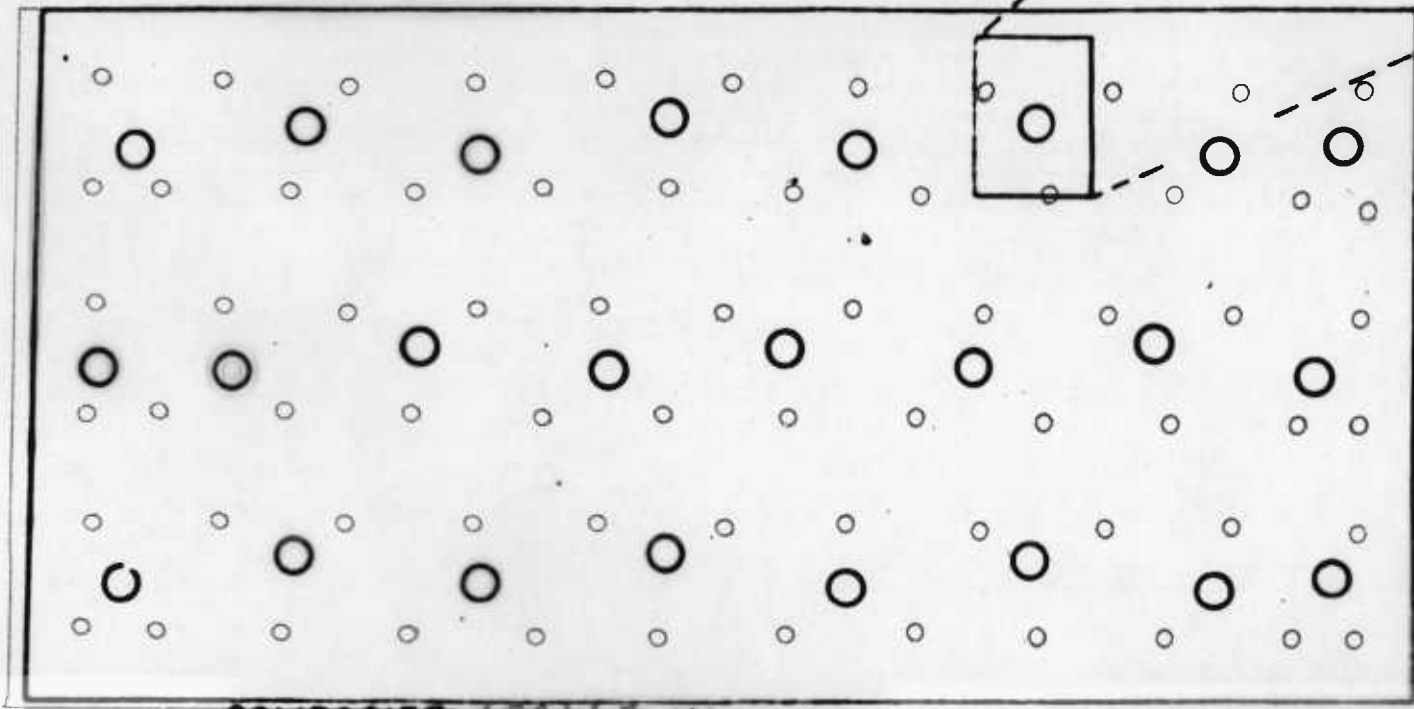
BLAST HOLE LAYOUT

EXISTING MINE GRADING SCHEME

FIG. 2.6

EACH COMPOSITE IS LOCATED AT
THE CENTRE OF GRAVITY OF ITS
CONSTITUENT BLASTHOLES

ASSAYS ARE
RECORDED
ALONGSIDE
EACH COMPO-
-SITE

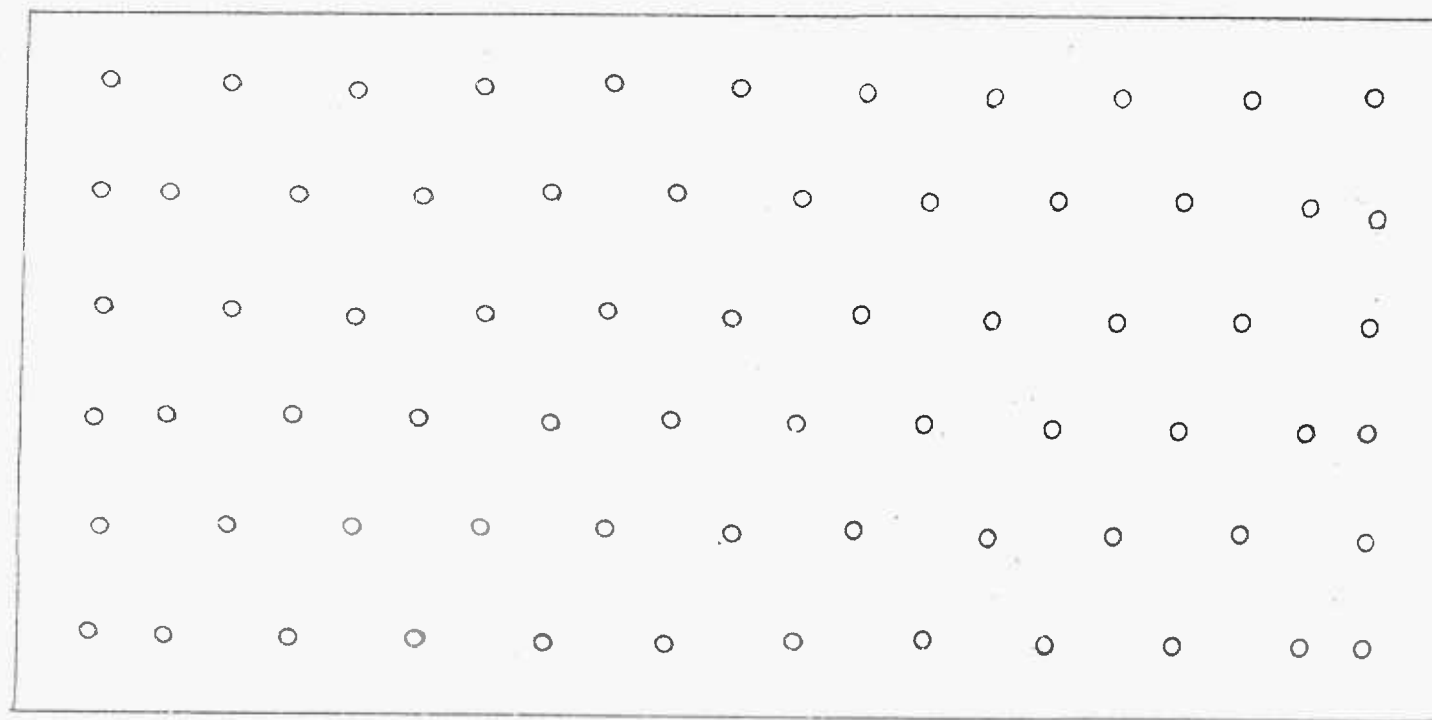


COMPOSITE ASSAY PLAN

BLAST HOLE LAYOUT

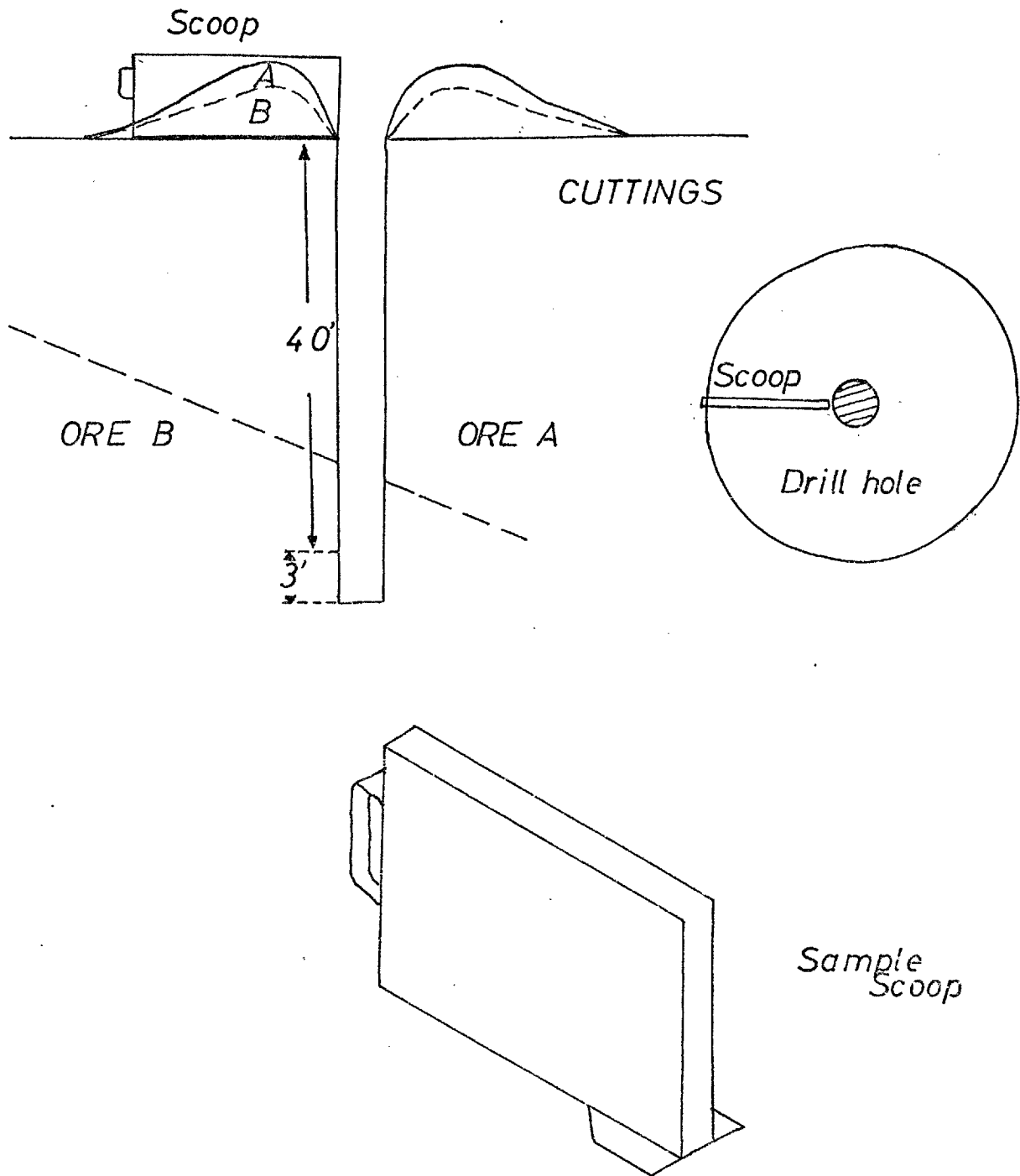
EXISTING MINE GRADING SCHEME

FIG. 2.6



BLAST HOLE LAYOUT
EXISTING MINE GRADING SCHEME

FIG. 2.6



BLAST HOLE SAMPLING

FIG. 2.7

reports are made available periodically without much delay and this provides the Mining Department advance information needed for preparing the long range plans.

Exhaustive checks have been applied to the procedures from drawing of the sections to the final summary preparation to ensure accuracy. With the introduction of an IBM 2770 terminal at Scully Mine the long term planning requirements have been considerably improved. The tonnage available and grade for any particular area under the present scheme of mining is prepared by specifying the area and percentage. This process is then repeated as the reserves get exhausted in that area and a comparison made from what was forecast. This enables the Mining Engineers to make adjustments or alter mine plans to suit their long range plans and mill requirements.

The generalised flow diagram for ore reserve estimation is shown in fig. 2.5a.

C. GRADE CONTROL & MONTHLY MINING ESTIMATE

The short term estimates are prepared from blast hole composite plans, that are established for each bench.

Each composite position on plan (see fig. 2.6.) is the centre of gravity of the polygon obtained by joining up the blast holes which make up the composite. Grouping of blast hole samples to form composites is made after field

examination of the cuttings so that as far as possible only ores of a similar type will be grouped together. Blast hole samples are tagged and combined to form composites in the laboratory prior to testing. Normally three blast hole samples will be combined to form each composite.

Sampling Procedure

Samples are cut from the core of ore cutting that remains after a blasthole has been drilled. When the drill is in position over the hole site, a sampling scoop, designed to cut a thin radial section from the core of ore that will form on drilling, is placed in position two to three inches from the bit, (see fig.2.7). As drilling proceeds cuttings will be blown up the hole and some will settle inside the scoop. On completing the hole the scoop is removed and the cuttings tipped onto a plywood board. For a full bench hole about 120 lbs. of cuttings settle inside the scoop. The sample is successively mixed and coned and quartered until a sample of 10 to 15 pounds remains. The sample is bagged and tagged, with the hole number. The hole position and number is recorded on the appropriate map (see fig. 2.6.) and the sample taken to the laboratory for testing.

In the engineering office the samples are grouped into composites, based on the nature of material in the holes. For a full bench 3 holes will be combined to form one

composite and will represent 8000 tons of ore approximately. Lists showing the hole numbers that made up one composite are sent to the laboratory, the samples combined in proportion to the depth of the hole and a composite number assigned. After test work is complete a copy of the results is sent to the Mine Engineering department and computer services.

Grade Control Procedure

The object of the grade control programme is to provide a blend of ore, suitable for mill feed, at the crusher. This is achieved in two stages. Firstly the grade of ore in front of each shovel is calculated. To do this, the grading engineer estimates the percentages of the composites in front of each shovel that are likely to be mined during the next 24 hours. He must bear in mind, that as far as possible only one ore type should be included per 'set up', and to this end limit stakes may be placed at the operating face to restrict the run of a shovel.

The composite numbers and percentages are transmitted to the computer via a terminal at Scully Mine. The machine is programmed to calculate the tonnage of each composite to be mined and from this the average grade of ore and total tonnage available that day at each shovel.

The second part of the programme is designed to find a

suitable schedule for shovel and truck combinations using the grade computed in the first part of the programme. Using the haulage distance from the crusher to a shovel and back, the maximum tonnage that one truck can haul in a shift is calculated. Then combinations of two, three and four shovels together are calculated varying the number of trucks serving one shovel. In theory it is possible to assign 5 trucks to one shovel but in practice no more than 2 trucks are assigned and the combinations doubled and trebled to use all the available trucks. A number of restrictions are included in the programme to limit the output from the machine, these are as follows:

Crude Fe	-	variable
Weight recovery	-	variable normally 37% to 47%
Concentrate Fe	-	60% to 70%
Concentrate SiO_2	-	0% to 7%
Concentrate Mn	-	variable normally 1.37 to 2%
Trucks per shovel	-	2
Shovels per comb.	-	4

Output is transmitted back to Scully Mine via the terminal, and the grading engineer selects those combinations which best meet the day's requirements and the information given to the operating personnel. A flow diagram showing the procedure for grade control is shown in fig. 2.5b. An example of the machine output is given in fig.2.9.

Monthly Mine Forecast

This is an attempt to forecast grade one month ahead using

BLASTHOLE COMBINATIONS FOR ORE GRADING

SHVL	DIST	NO	PCT	NO	PCT	NO	PCT	NO	PCT	NO	PCT	NO	PCT	NO	PCT
2-0	4000.10978		1.0011105		1.0011106		1.0011108		0.2511107		0.50	0	0.0	0	0.0
		0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0
		0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0

SHVL	WT	WEC	FE	SIL	MN	MAG	CR	FE	CR	MN	WASTE	ORE	CONC
2-0	40.52		66.43	2.24	1.30	2.23	35.66		1.18		0	31643	12821

SHOVEL	WT	REC	FETAIL	MNTAIL	ORE	TAILS
2-0	40.52		14.69	1.11	31643	18823

GRADING COMBINATIONS FOR ALL SHOVELS

SHVL	WT	REC	FE	SIL	MN	MAG	COARSE	DIST	MAX	TONS
	40.52		66.43	2.24	1.30	2.23	0 0	4000	4862.40	
SHVL	TONS	WT	TONS	FE	SIL	MN	MAC			
	CRUDE	REC	CONC	CONC	CONC	CONC	CONC	LOADS	TRUCKS	
2-0	3915.	40.52	1586.	66.43	2.24	1.30	2.23	71.2	2.00	
3-0	1466.	51.65	757.	65.16	2.32	2.99	17.12	26.7	1.00	
TOTL	5381	43.56	2344.	66.02	2.27	1.85	7.05	97.8	3.00	

RESTRICTIONS

HI-SIL-LO	HI-MN-LO	HI-WT-REC-LO	HI-MAG-LO	HI-FE-LO	HI-COARSE-LO
6.00 0.0	2.00	1.37 50.00 37.00	99.99 0.0	70.00 50.00	99.99 0.0

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FIG. 2.9

BLASTHOLE FILE

COMP NO	TOTAL FE	TOTAL MN	HEADS MAGN.	WT REC.	RE	MAGN.	MN	SIL	TOTAL FE	TOTAL MN			AREA	HT.	LEVEL
11142.	36.90	1.10	1.07	41.12	67.40	0.88	1.02	1.81	15.60	1.15	0.0	0.	1.01	45.	1895
11143.	36.30	2.50	2.86	43.10	66.00	3.20	2.00	1.78	13.80	2.88	0.0	0.	0.92	42.	1895.

MONTHLY MINING FORECAST

HOLE NO	PCT	CRUDE	CR	FE	WT	REC	CONC	FE	S102	MN	WASTE
3800	0.30	0.	36.64		0.0		0.	0.0	0.0	0.0	3425.
3803	0.40	3545.	35.74		34.14		1210.	65.80	3.21	1.33	0.
3830	0.50	6050.	26.66		26.62		1611.	60.51	8.79	2.48	0.
3831	0.20	2368.	33.15		30.29		717.	66.70	3.42	0.58	0.
AREA		CRUDE	CR	FE	WT	REC	CONC	FE	S102	MN	WASTE
1975		229207.	35.52		41.50		95125.	63.23	6.50	1.54	15730.
AREA		CRUDE	CR	FE	WT	REC	CONC	FE	S102	MN	WASTE
ALL		229207.	35.52		41.50		95125.	63.23	6.50	1.54	15730

the available blast hole data. Programme details are much the same as for the first part of the grading programme, except that much larger tonnages are used. An example of the machine output is given in fig. 2.10. It was assumed that enough broken and drilled material would be available in the pits one month ahead of production to make a realistic attempt at forecasting grade. In practise very little blast hole data is available and consequently any prediction that may be made is based largely on adjoining blasts and the benches above. This practice, without any understanding of the geology can prove a very dangerous one indeed!

D. TABLE TEST PROCEDURE

Because of its importance to the whole of the ore reserve and grade control programmes, it is necessary to describe the test procedure in some detail. The flow sheet is given in fig. 2.11.

When the samples are received in the Laboratory they are dried in an oven at 100°C. After drying the samples are placed in bread pans and then made up into composites on the data received from the Mine Engineering department. The samples that go to make up a composite are selected and taken to the rolls crusher room, where the samples are riffled down to the weights required (dependant on the depth of the holes, and number of samples per composite).

TABLE TEST FLOW SHEET

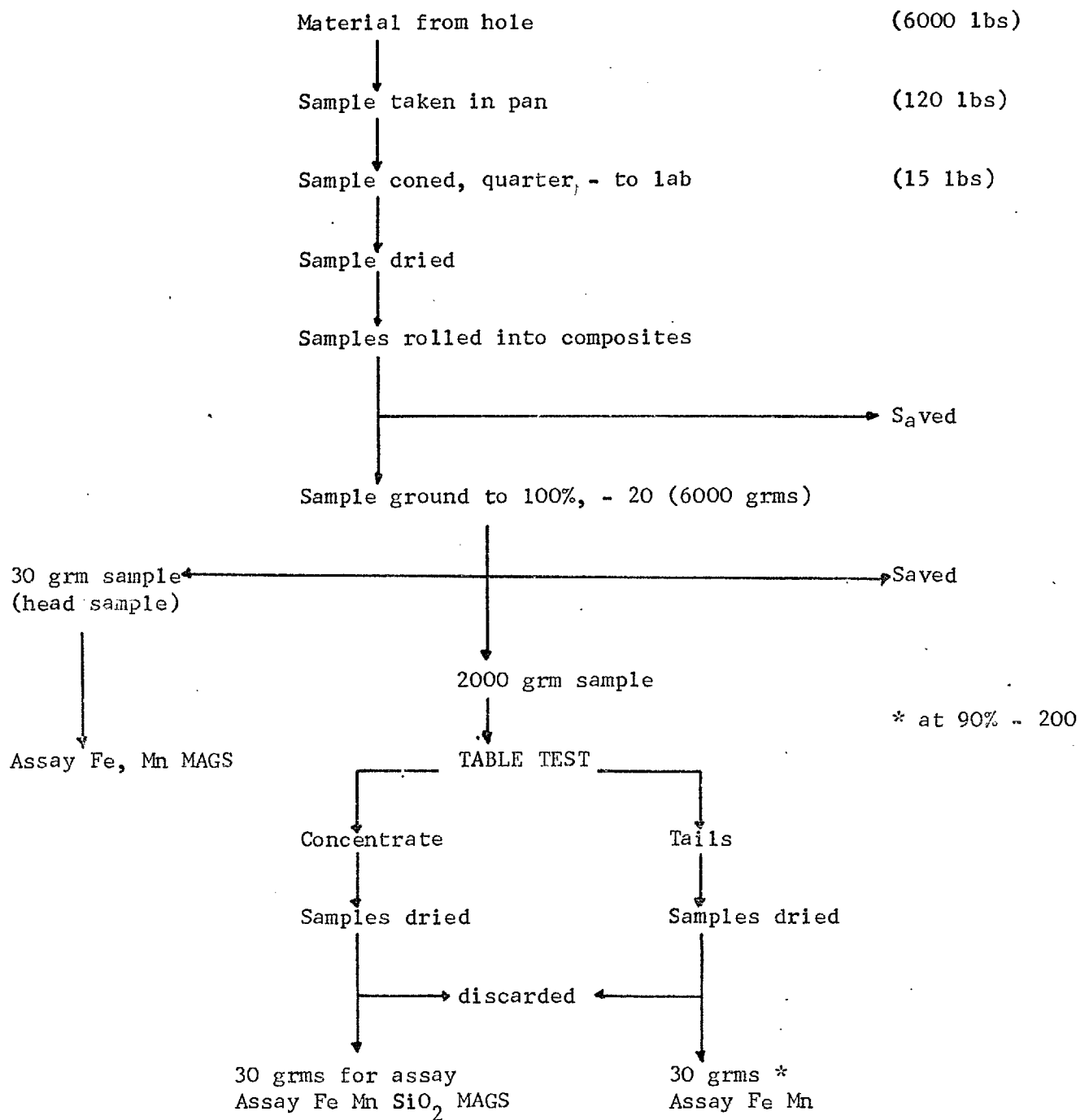


FIG 2.11

The samples are rolled together on a rubber rolling sheet, and then riffled into two equal portions (approx. 6000 grams each). One half is saved whilst the other is screened at 20 mesh. the +20 mesh fraction is crushed to 100% - 20 mesh and combined with the -20 mesh fraction. A composite number is allocated at this point. The composite is now riffled into 3 portions, one of 30 grms for a head assay one of 2000 grms for table test feed whilst the remainder is saved.

The table used is a laboratory scale shaking table, with moving splitters. The dry sample is placed in a hopper above the table and washed down onto the table with a $\frac{1}{4}$ " stream of water. Water is added to the material in the distribution trough to bring the feed to the required dilution. On the first run, a concentrate, tailings and middlings are cut manually. The products are allowed to settle for 20-30 minutes, a flocculent being added to the middlings and tailings to aid settlement. The middling portion is re-run on the table to give a concentrate and tailing only. These are added to the first run concentrate and tailings to give the final products. After filtering and drying the two portions are weighed and a percent weight recovery calculated. If less than 350 grms of concentrate are recorded the samples are disregarded as below grade. The concentrate and tailings portion for the normal sample are rolled and riffled down to 30 grms for assay. Assays are obtained as follows:

Head	Iron (wet)	Mn (XRF)	Calculated Magnetics
Concentrate	Iron (wet)	Mn ₁ SiO ₂ (XRF)	Tube test Magnetics
Tails	Iron (wet)	Mn (XRF)	Tube test Magnetics

A calculated weight recovery is then made as follows:

$$W = \frac{H-T}{C-T} \quad \text{where} \quad W = \text{weight recovery}$$

H = Head assay

T = Tails assay

C = Conc. assay

If the calculated and observed recovery do not agree by more than 3%, the test is re-run.

It will have become obvious that a great deal of importance is attached to the table test data, however no confidence limit for the test results has ever been established.

This problem has been approached in two ways and the results are given below:

(a) Pierre Gy, Theory and Practice of Sampling Broken Ores

In his papers on the sampling of broken ores Gy states that the sampling variance may be calculated as follows:

$$s^2 = \frac{C \cdot d^3}{M}$$

Where s^2 = relative sampling variance = $\frac{\sigma^2}{\bar{x}^2}$

d = size largest fragments in cm

C = Constant for any material

M = Sample weight in grms

To calculate C we apply the following expression.

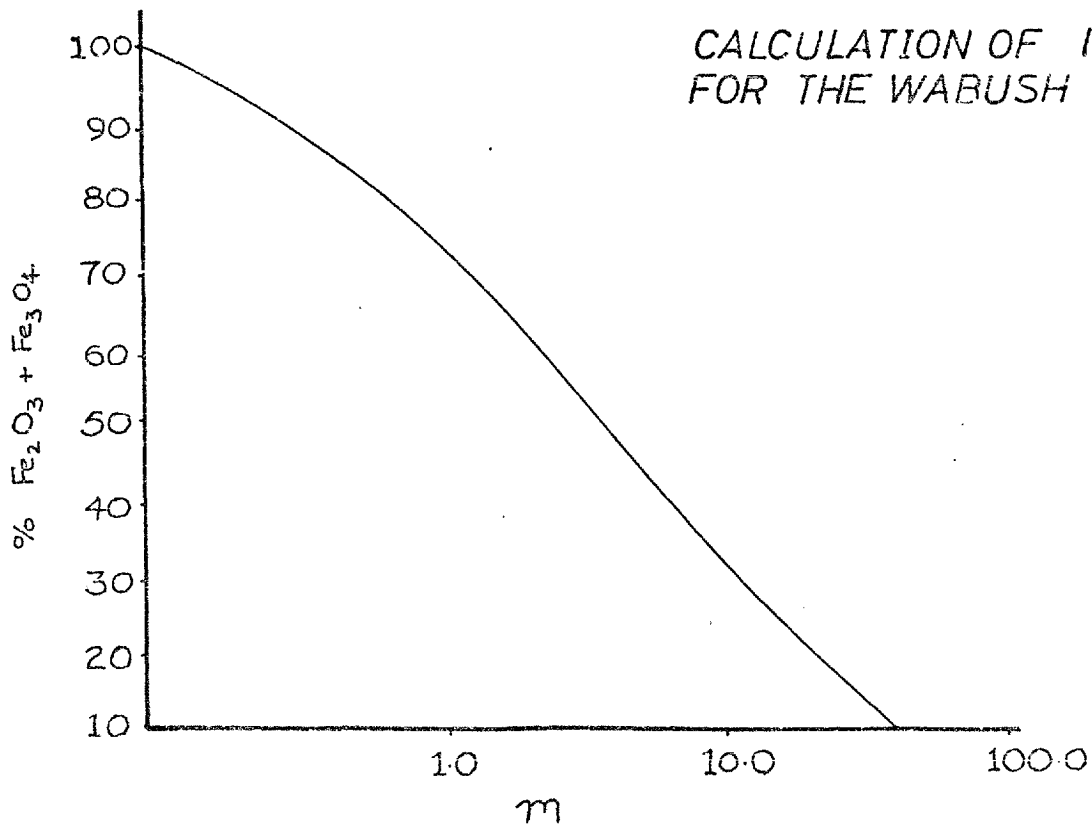
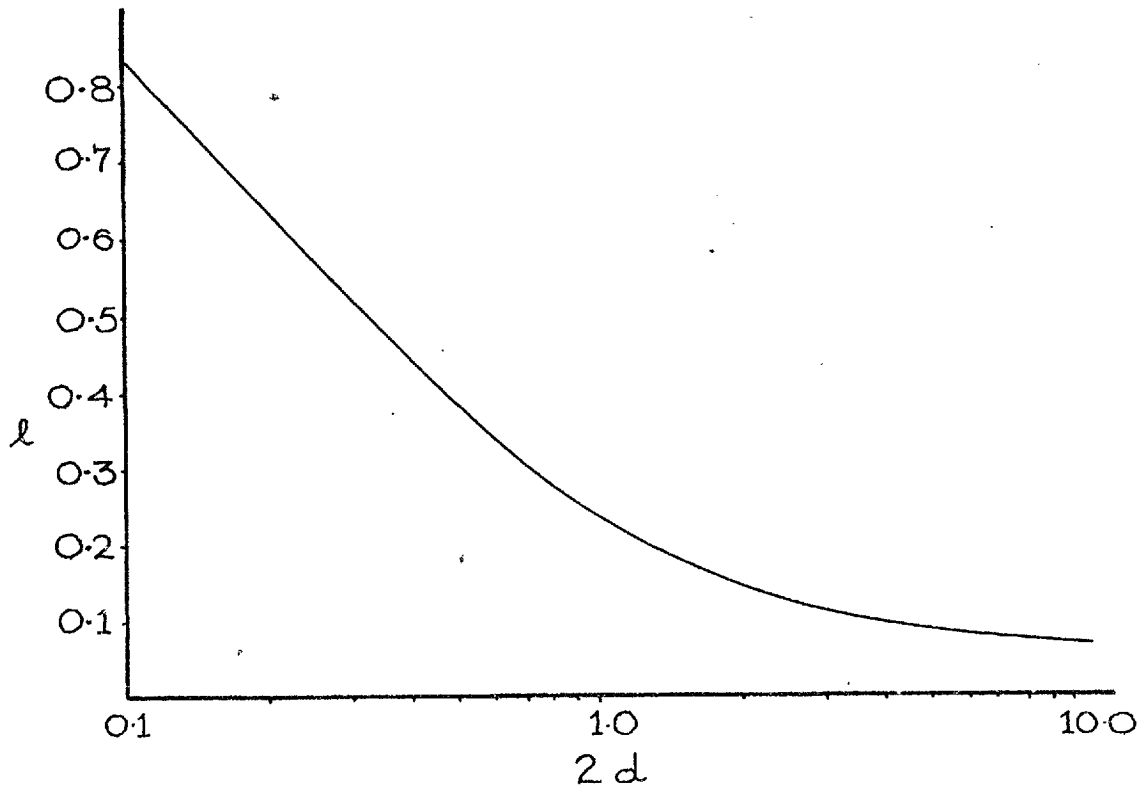


FIG. 212

$$c = 1.k$$

where $l = f(d/L)$ where L = effective liberation size

(see fig. 2.12)

and $k = m.f.g$

where

f = shape factor = 0.5 or 0.2 for flat ore minerals

g = distribution factor = 0.25 or 0.5 for well sized

material

$$m = \frac{1-a}{a} \left((1-a)\delta_o + a\delta_g \right) \quad \text{where 'a' is the}$$

proportion ore mineral SG. δ_o in gangue δ_g

(see fig. 2.12)

Sampling and analytical variance must be calculated each time there is a change in grain size, so that

$$S_T^2 = S_1^2 + S_2^2 + \dots + S_n^2$$

where S_T^2 = Total sampling and analytical variance

and $S_1^2 \dots S_n^2$ = Sampling and analytical variance of individual steps.

The sampling and test procedure at Wabush Mines has already been discussed (see fig. 2.11), and in order to apply the Gy theory, the test procedure has been divided into the following steps:

- Step 1 Blast hole to Composite
- Step 2 Composite to Table Sample
- Step 3 Table sample to concentrate sample
- Step 4 Table sample to tailings sample
- Step 5 Composite to Head Sample

Step 1

$$M = 6000 \text{ grms}$$

$$d = 2.5$$

$$l = 0.1$$

$$f = 0.5$$

$$g = 0.25$$

$$m = 4.2 \text{ (Fe) and } 200 \text{ (Mn)}$$

$$s_1^2 \text{ (Fe)} = 0.000137 \quad s_1^2 \text{ (Mn)} = 0.0065$$

Step 2 All the factors are the same except

$$M = 2000$$

$$d = 0.09$$

$$l = 0.8$$

$$g = 0.25$$

$$s_2^2 \text{ (Fe)} = 0.00000015 \quad s_2^2 \text{ (Mn)} = 0.00007$$

Step 3

$$M = 30$$

$$m = 2.0 \text{ (Fe) and } 200 \text{ (Mn)}$$

All other factors unchanged

$$s_3^2 \text{ (Fe)} = 0.000005 \quad s_3^2 \text{ (Mn)} = .0005$$

Step 4

$$m = 28.5 \text{ (Fe) and } 200 \text{ (Mn)}$$

$$s_4^2 \text{ (Fe)} = .00007 \quad s_4^2 \text{ (Mn) not calculated}$$

Step 5

$$m = 4.2 \text{ (Fe)}$$

$$s_5^2 \text{ (Fe)} = 0.00001$$

Calculation of concentrate Mn error.

$$\begin{aligned} \text{Total Variance} &= S_1^2 + S_2^2 + S_3^2 \text{ (Mn)} \\ &= 0.065 + 0.000067 + 0.0005 \end{aligned}$$

$$S_T^2 \text{ (Mn)} = 0.071$$

$$S = 0.084$$

$$\sigma \text{ (Mn)} = 0.084 \times 1.70 = 0.15$$

Average Conc. Mn assay = 1.7 $\pm$ 0.25 at 95% confidence level

Calculation of possible error on weight recovery.

The possible error expected in weight recovery calculations using the Gy method must be based on the calculated variances for Head Fe., Concentrate Fe., and Tailings Fe. and table 2.1 lists the results of the Gy method for calculating sampling variances for these:

	S^2	S	σ	95% C.I.*
Head Fe.	0.00015	0.012	0.47%	$\pm 0.9\%$
Tails Fe.	0.00007	0.008	0.12%	$\pm 0.24\%$
Conc. Fe.	0.000009	0.003	0.198%	$\pm 0.26\%$

2.1

* 95% confidence limits

Using the expression:

$$W = \frac{H-T}{C-T}$$

where W = weight recovery

H = Head assay

T = Tailings assay

C = Concentrate assay

Using the average assay for head concentrate and tail, and their expected deviations from average, as calculated above,

WEIGHT RECY.				CONC. MANGANESE			
x_1	x_2	$\frac{(x_1+x_2)}{2}$	$\frac{x_1-x_2}{2}$	x_1	x_2	$\frac{x_1+x_2}{2}$	$\frac{x_1-x_2}{2}$
16.25	44.42	30.34	-13.09				
48.66	42.80	45.73	+ 2.93	1.25	1.42	1.34	-0.08
54.73	51.21	52.97	+ 1.76	0.67	0.50	0.59	+0.08
44.34	48.39	46.37	- 2.03	5.84	4.66	5.25	+0.59
15.91	45.60	30.76	-14.09				
55.98	41.40	48.69	+ 7.29	2.21	2.85	2.53	-0.32
33.93	32.58	33.26	+ 0.67	1.55	1.41	1.48	+0.07
46.44	44.02	45.23	+ 1.21	2.31	2.60	2.45	-0.15
48.76	41.86	45.31	+ 3.45	1.46	1.51	1.48	-0.02
26.79	44.08	35.44	- 8.65	1.66	1.65	1.66	+0.01
21.00	24.32	22.66	- 1.66	2.23	1.91	2.07	+0.16
40.34	26.69	35.02	+ 5.36	0.13	0.08	0.10	+0.02
28.93	26.08	27.51	+ 1.42	0.12	0.74	0.38	-0.26
47.75	47.13	47.44	+ 0.31				
48.02	46.48	47.25	+ 0.77				
52.61	52.55	52.58	+ 0.03				
57.39	52.50	54.95	+ 2.44				
56.21	57.51	56.86	- 0.65				
43.70	44.30	44.00	- 0.30				
56.58	51.16	53.87	+ 2.71				
46.96	44.19	45.58	+ 1.38	1.59	0.15	0.87	+0.72
45.45	41.95	43.70	+ 1.75	0.72	0.08	0.40	+0.32
43.88	36.79	40.34	+ 3.54	0.60	0.36	0.48	+0.12

Although it is realised that there are not as many paired samples as is customary for this sort of application, these were all that were available at the time of writing and any

conclusions must bear this in mind.

Tests for significant difference.

The average of the pairs is the best estimate of the true value and the difference between the first sample and the average calculated. Then the statistic

$$d = \frac{\bar{x} \text{ diff}}{Sd/\sqrt{n-1}} \quad \text{where } \bar{x} \text{ difference is the average of the differences and } Sd \text{ is the standard deviation of the differences}$$

is distributed as 't' with (n-1) degrees of freedom

Weight recovery

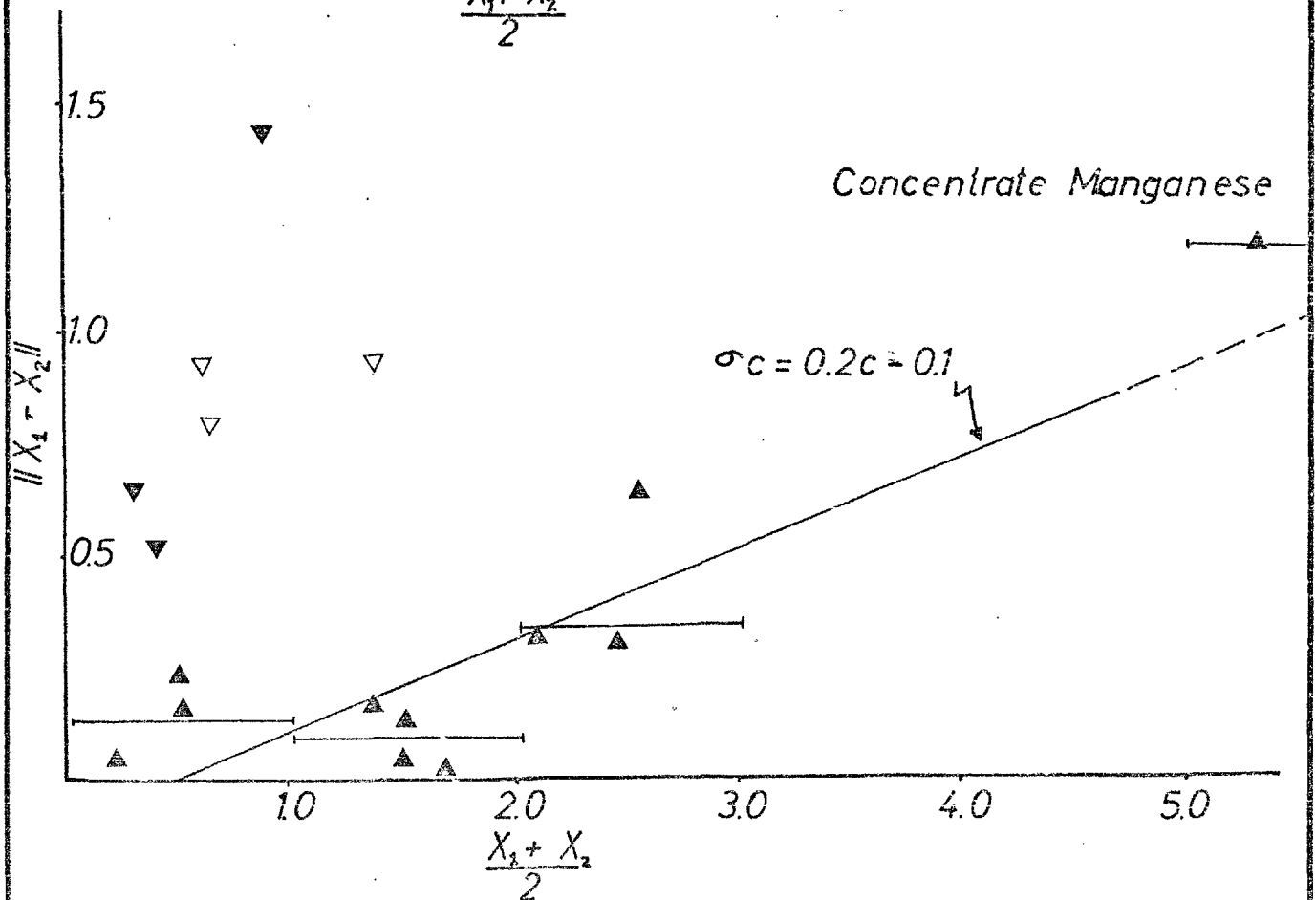
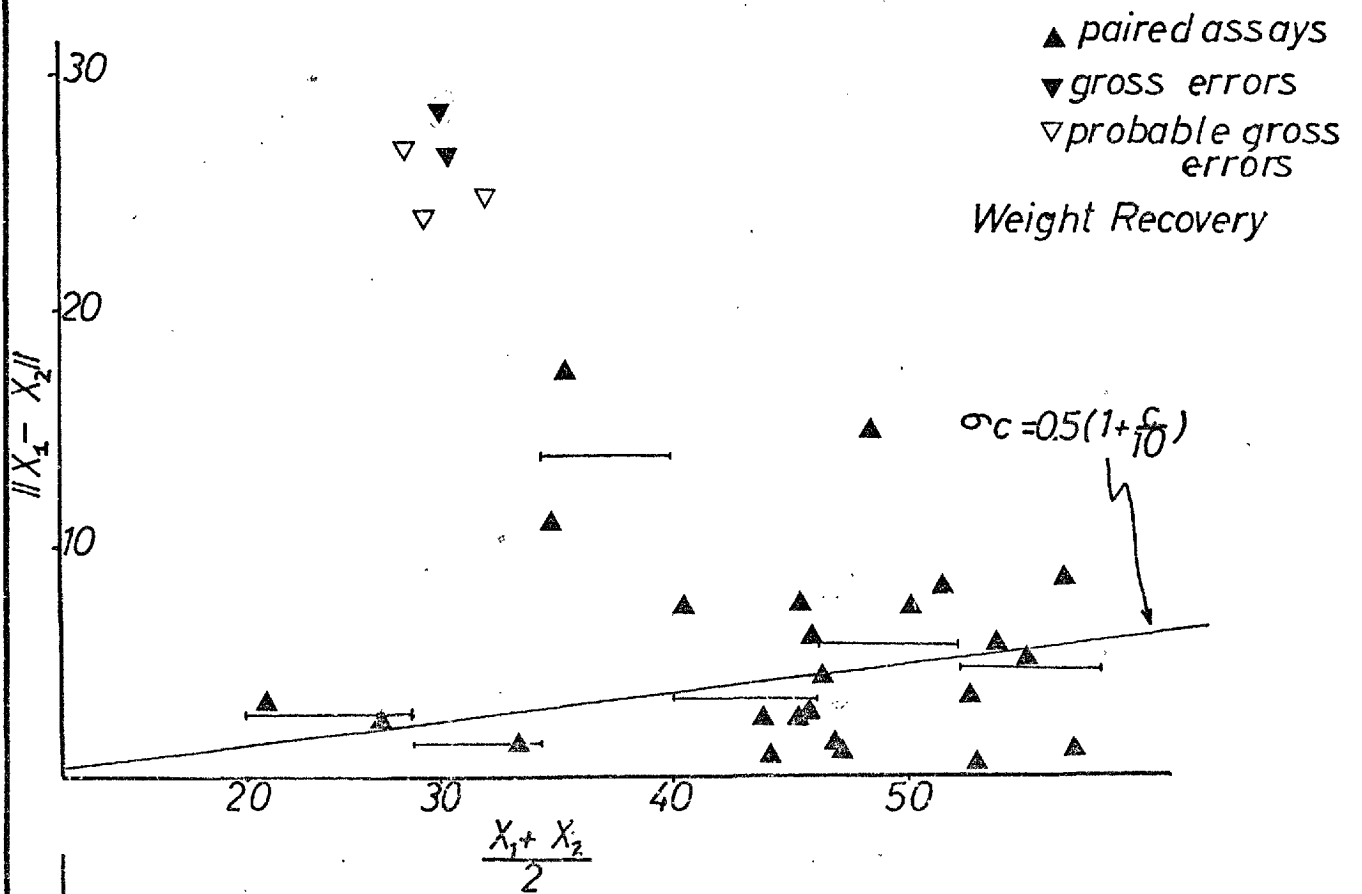
$$\begin{aligned} \bar{x} \text{ diff} &= 0.15 \\ Sd &= 0.72 \\ d &= 0.15 / (0.72/\sqrt{22}) \\ &= 1.0 \end{aligned}$$

$$t_{0.90} (22) = 1.321$$

$$t_{0.95} (22) = 1.721$$

$$t_{0.99} (22) = 2.080$$

as d is lower than t at the 0.90 probability level there is no significant difference between the two sets of samples, with respect to weight recovery.



CALCULATION of STANDARD DEVIATION
from PAIRED ASSAYS

FIG. 2.13

Concentrate manganese

$$\bar{x} \text{ diff} = 0.09$$

$$Sd = 0.28$$

$$d = \frac{0.09}{(0.28/\sqrt{13})}$$

$$= 0.89$$

$$t_{0.90} (13) = 1.350$$

$$t_{0.95} (13) = 1.770$$

$$t_{0.99} (13) = 2.650$$

Similarly to above, there is no significant difference between the two sets of samples with respect to concentrate manganese.

Approximation of standard deviation of the table test is shown in fig. 2.13. The two equations relating standard deviation to concentration are given as:

weight recovery

$$\sigma_C = 0.5(1 + \frac{C}{10})$$

where σ_C = standard deviation at concentration C for average weight recoveries $C = 41.17$

$$\text{then } \sigma_C = 2.5$$

$$\text{Concentrate Mn } \sigma_C = 0.1(2C - 1)$$

$$\text{when } C = 1.7 \quad \sigma_C = 0.24$$

(c) Comparison of two methods.

It is interesting to note that two methods relying on completely different information should give comparable

results. It is also interesting that on average grade ores the assays obtained at best have a 95% confidence limit of $\pm 2.4\%$ on weight recovery and $\pm 0.47\%$ on concentrate manganese. This rather belies the apparent precision with which the grading programme is carried out. Also final grade for any location is the average of several composites and the uncertainty of grade will decrease with n is the number of composites used.

E. DISCUSSION OF PROCEDURES AT WABUSH MINES

The first glance at either the output from the ore reserve estimate programme or the grade control sheet leaves an impression that precision tools have been used to generate such results. Tonnages, even those in the tens of millions, are quoted to the second decimal place and likewise for the projected grades. The apparent accuracy may constitute a major danger and result in wasteful channelling of time and effort. Let us first examine the ore reserve estimation procedures.

At the time when the 30 years estimate was being drawn up, one geologist noted: 'Since surface outcrops showed extreme deformation, and since adjacent drill holes showed considerable variation in the depth of the bottom rocks, it is necessary to work out the geological structure before any valid reserve estimate can be prepared'. The geological

structure that was generated was one of simple symmetrical folds. The 'ore types' that are used in the procedure are drawn up for convenience rather than because they actually exist. Studies have shown that generating grade over a large area, using both diamond drill holes and composites, gives a considerable difference in both weight recovery and concentrate manganese.

This has been done on the areas mined up to the end of 1968 and for 1969 and are tabulated below:

	<u>Wt. Recy.</u>	<u>Conc. Mn</u>
Composites to Dec 31/68	43.38	1.66
D.D.H. to Dec 31/68	47.15	1.84
Production to Dec 31/68	39.16	1.67
Arithmetic Avg. of D.D.H.	43.72	1.66
Composites for 1969	43.65	1.83
D.D.H. for 1969	45.36	2.02
Production 1969	38.86	1.76
Arithmetic Avg. D.D.H.	43.58	1.80

The results are in agreement except for the DDH estimates (ore reserve estimate) and production weight recovery.

The latter may be explained easily in that the actual plant recovery (from Humphrey's spirals) will never reach controlled table test results. However, the DDH estimate difference must lie in the method itself. It seems clear that the interpretation of the West and South Pit areas as simple synclines has had the result of exaggerating the

volumes of ore to which the grades of high grade high manganese ores have been projected. The solution lies in re-interpretation of the sub-surface geology.

The grade control problem is somewhat different. This relies most heavily on the table test data and this and its shortcomings have already been discussed. One man in the Mine Engineering department and one at the Computer Centre are kept almost permanently employed in grade control. The limitations of the method are not fully realised. It has been shown that table test weight recovery and concentrate manganese have confidence limits of $\pm 2.4\%$ and $\pm 0.28\%$ respectively and yet an attempt is made to hold grade well within these limits. The problem is not helped by the fact that longer term planning is not very far sighted. The monthly plan takes no account of grade and it must happen that a blend is not possible with the available ore. To be able to generate grade over an ore type, that is a lithological unit, not a convenient sub-division, would be the best guide in planning and of considerable help in grade control.

CHAPTER III

GEOSTATISTICS OF THE ORE BODY

A. INTRODUCTION

The original approach that had been taken to solve the geology of the Wabush ore body was a classical one, in which the mineralogy and petrology of the ore, as found in bore holes, trenches, surface outcrops and test workings, was described in detail, and then attempts made to correlate between sample points. In retrospect it is only fair to applaud the work that was done, in that many of the major features of the ore body were brought to light. A major break-through in the structural interpretation of the ore body was made in 1964, with the recognition of the middle quartzite. This was seen as a narrow but continuous band of iron poor rock, normally light yellow in colour due to minor amounts of limonite. Even so, the geology as presented in the ore reserve estimates in late 1964 was not detailed enough to allow mining to proceed with confidence on startup in early 1965. Because of the tight limit set by the steel companies on concentrate grade ($\pm 0.15\% \text{ Mn}$ $\pm 0.25\% \text{ SiO}_2$), the operation as a whole soon ran into difficulty. There was a confusion over the location of the high and low manganese zones and poor quality ores began to appear in the middle of the supposedly

rich West Pit area. It was clear that a new approach was necessary. A new start was made of pit mapping, but the similarity of the ores and their general loosely-consolidated, friable nature made this an arduous task. The key, it seemed, was in the many thousands of samples that had been taken during exploration work.

Altogether, there were approximately 6000 sample points, in both the East and West pit areas, all of which had been assayed for crude iron, crude manganese and crude "magnetics". Table tests were run, to give a weight recovery and concentrate assays. These have been loaded onto magnetic tape and have formed the basis of a statistical study of the ore body.

The objectives of the study are simple. It has been demonstrated that the quartzite runs as a continuous unit throughout the entire ore body. It seems natural to assume that other units, albeit somewhat modified by faulting and metamorphism may exist. By dividing the ore body into groups using the available data it may be possible to define some or all of these units in terms of their assays. With the quartzite as the only marker horizon inside the ore body, interpretation was impossible, but the identification of whole suites of rocks, by simple statistical techniques, has given a solution to the problem.

B. STATISTICS OF THE ORE BODY

General Statistics

Using the random number generator facilities on the CDC 6400 at Imperial College, 5 samples of approximately 200 data points each were taken from the East and West pit files. (The fact that the frequency distribution from the random number generator is rectangular has been verified.) These samples have been used to provide the basic statistical information on the ore body.

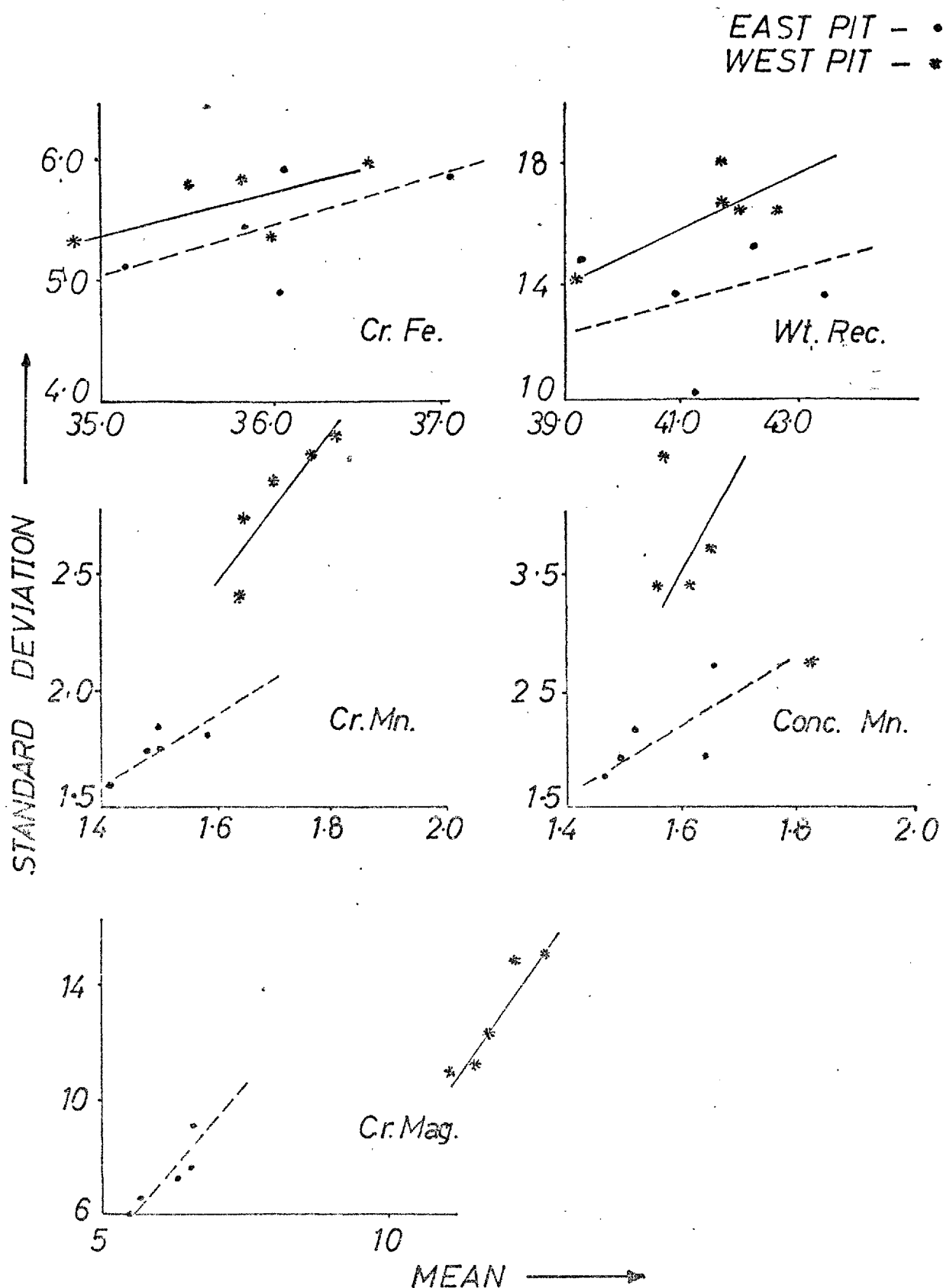
General information about the ore body has been summarised in table 3.1.

<u>Sample</u> <u>No.</u>	<u>Cr.Fe.</u>		<u>Wt.Rec.</u>		<u>Cr.Mn.</u>		<u>Conc.Mn.</u>		<u>Conc.Mag.</u>		
	$\bar{x}$	s	$\bar{x}$	s	$\bar{x}$	s	$\bar{x}$	s	$\bar{x}$	s	
1	36.9	5.8	43.4	13.9	1.47	1.75	1.51	2.18	6.6	9.1	E A S T P I T
2	35.8	5.5	39.2	15.0	1.50	1.89	1.49	1.91	5.7	6.6	
3	36.1	6.0	42.2	15.6	1.58	1.84	1.66	2.76	5.5	6.0	
4	35.2	5.2	40.8	13.9	1.49	1.75	1.64	1.97	6.4	7.1	
5	36.0	4.9	41.5	10.8	1.41	1.59	1.46	1.73	6.5	7.4	
1	35.4	5.7	41.9	16.5	1.73	3.19	1.57	4.85	11.7	11.1	W E S T P I T
2	35.8	5.7	41.8	16.9	1.64	2.73	1.56	3.32	12.2	14.9	
3	36.5	6.0	42.3	16.3	2.04	2.39	1.87	2.54	12.9	11.6	
4	34.8	5.3	39.3	14.8	1.70	2.91	1.67	3.73	10.9	11.0	
5	35.9	5.4	41.5	18.2	1.78	3.20	1.62	3.40	11.7	12.3	
35.8 5.6 41.4 15.2 1.63 2.42 1.60 2.84 9.0 9.0 Average for pit											

Table 3.1 Means and standard deviations for the five East and West Pit samples.

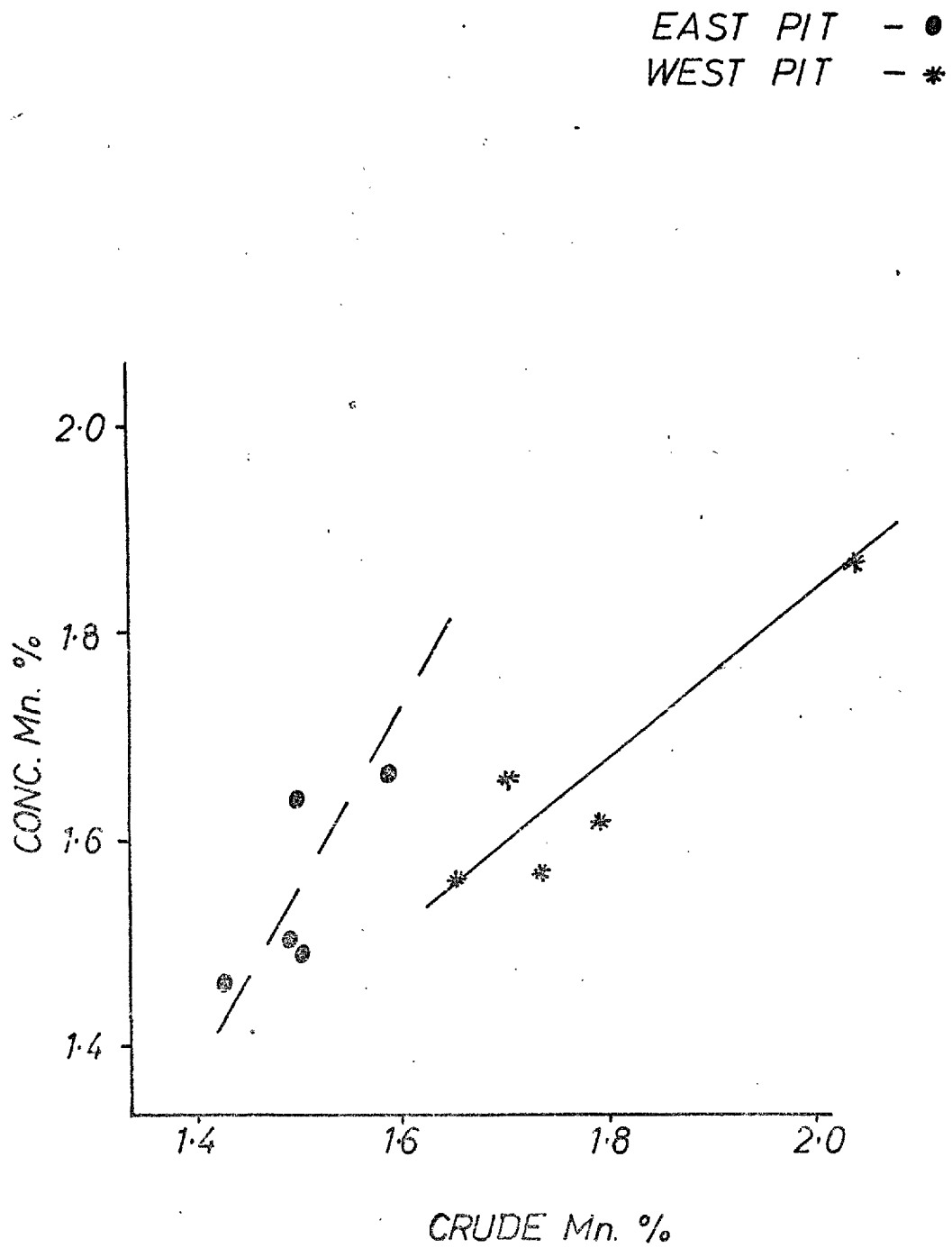
Fig. 3.1 shows a plot of mean against standard deviation for the East and West pit samples, for each of the variables considered. It is interesting to note how the data for the two areas fall into two distinct clusters of points, with the West Pit showing the higher standard deviation. Here we also find a higher mean for crude and concentrate manganese and for crude magnetics. In the West Pit however there is a better coverage of the ore body by drilling and a much larger sample of the lower member ore is present which will not only affect the standard deviations of the samples but also their mean values. Similar reasoning may be applied to fig. 3.2 where the means of the crude manganese samples have been plotted against concentrate manganese samples for both pits. Inclusion of a large sample from the lower member ore will tend to increase the mean crude manganese grade, but will not affect the concentrate manganese mean so much as the manganese is present as earthy pyrolusite and washes to tails in both table tests and on Humphries Spirals.

In order to begin the study in a more systematic way some preliminary work was carried out on the samples from the two pits and this included frequency distribution diagrams and scatter diagrams. Although seven parameters were available only those four which materially affect the operation at Wabush, were selected for further study. These



PLOT of MEAN AGAINST
STANDARD DEVIATION

FIG. 3.1



RELATIONSHIP of CRUDE Mn.
to CONCENTRATE Mn.

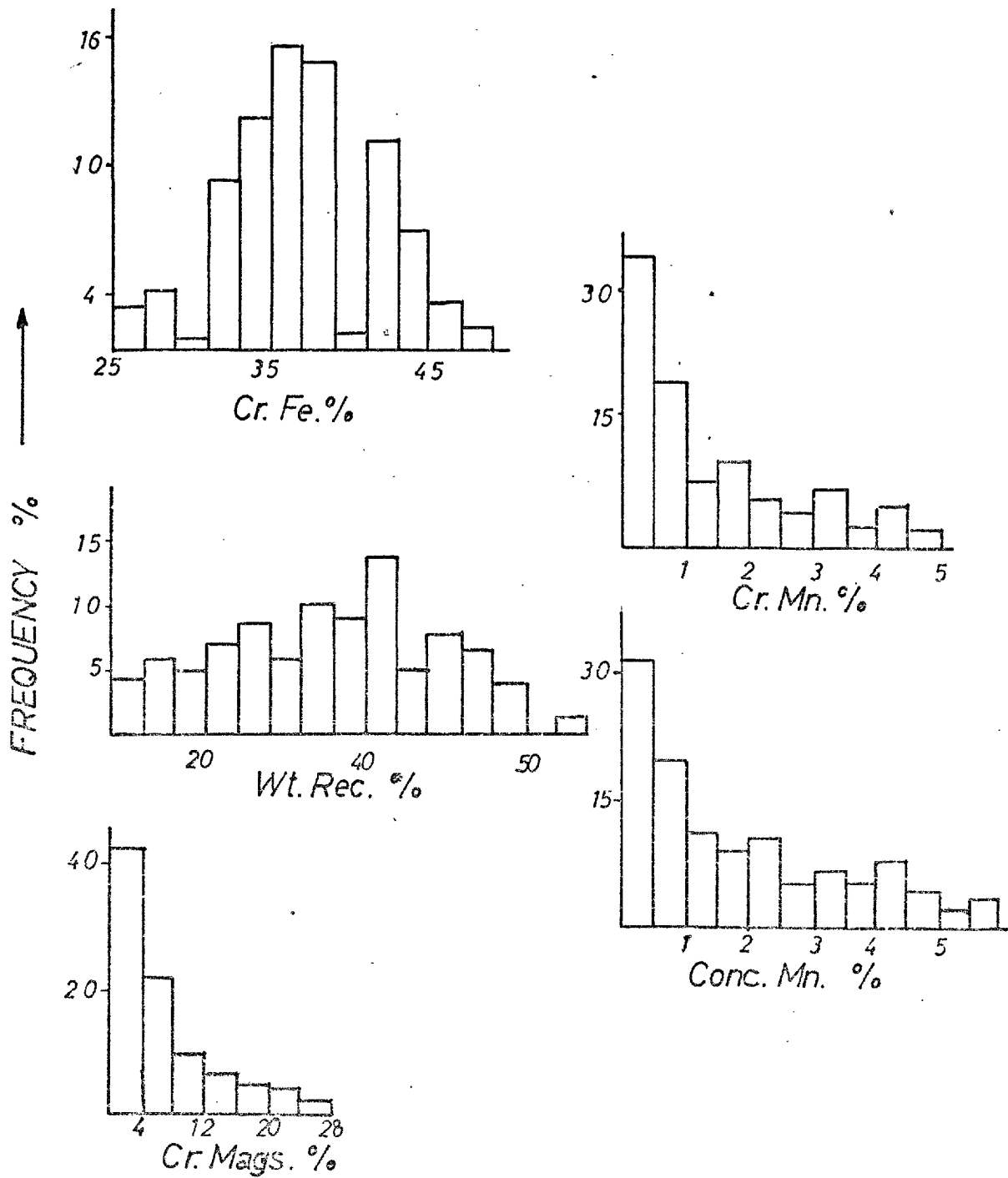
FIG. 3.2

were crude iron (Cr.Fe), crude manganese (Cr.Mn), weight recovery (Wt.Rec.), and concentrate manganese (Conc. Mn.). In some instances the crude magnetics assay was considered where there was reason to believe a relationship existed between the variable and one of the four mentioned above. Since the method by which this assay was obtained (Davis Tube test) is open to considerable criticism, no confidence can be placed in the results of analytical treatment of this data.

Distribution diagrams

Preliminary work was carried out on a 5% sample of the population taken at random from the East and West pit files. The results have been plotted and are presented in figs. 3.3 and 3.4. None of the distributions presented in fig. 3.3 can be fitted to a gaussian model although it may be possible that the crude iron and weight recovery diagrams are multimodal. The crude manganese, concentrate manganese and crude magnetics diagrams show a strong positive skewness. These latter three have been logarithmically transformed in fig. 3.4 (using $\log_{10}$ of the value multiplied by 100 to avoid negative logarithms) and although these now show a more gaussian, less skewed distribution they still appear to be composed of two or more independent populations.

Further work was carried out on the set of samples taken from the East and West pit areas and Fig 3.5 shows the



HISTOGRAMS of
MAIN VARIABLES

FIG 3.3

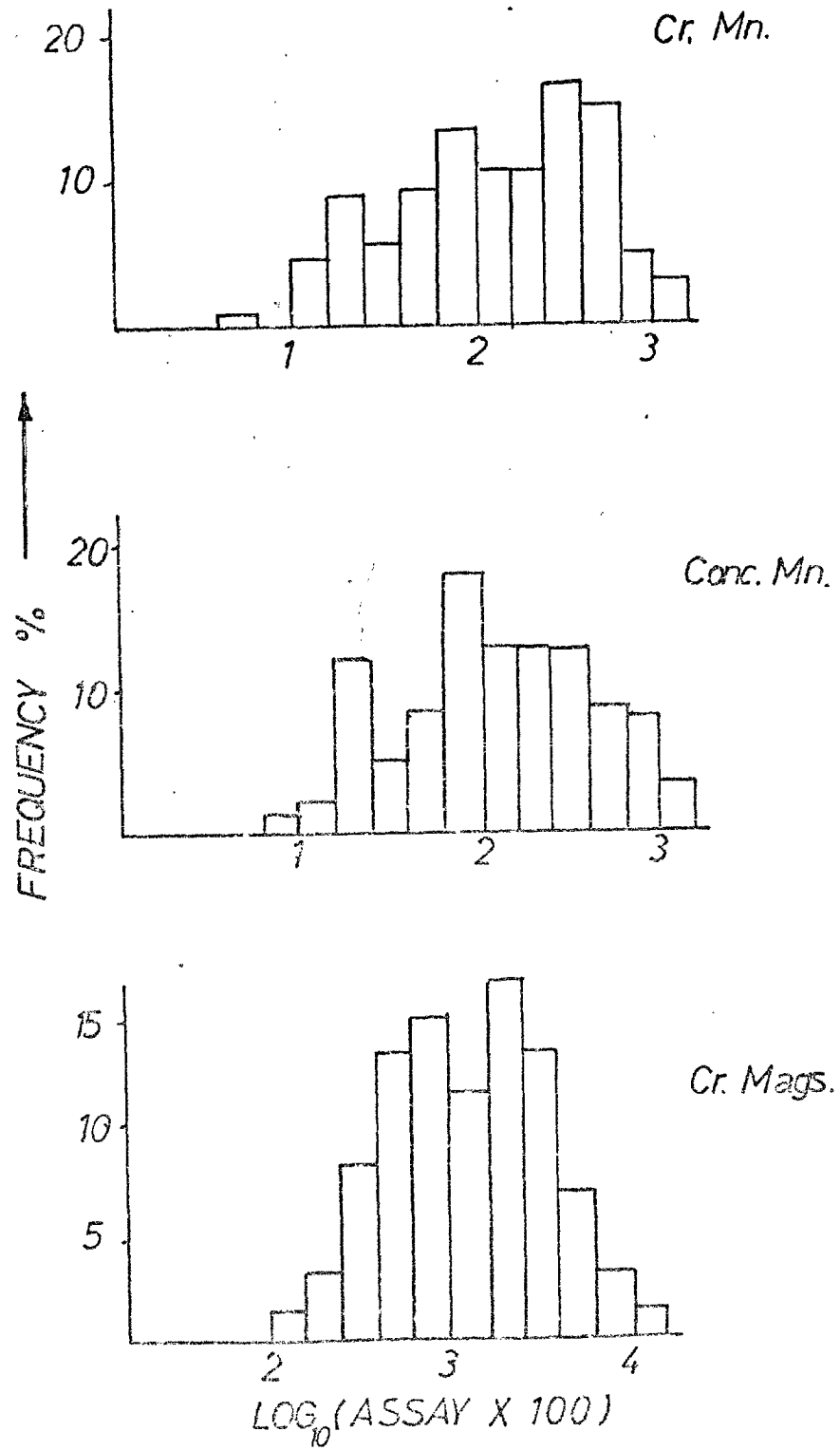


FIG 3.4

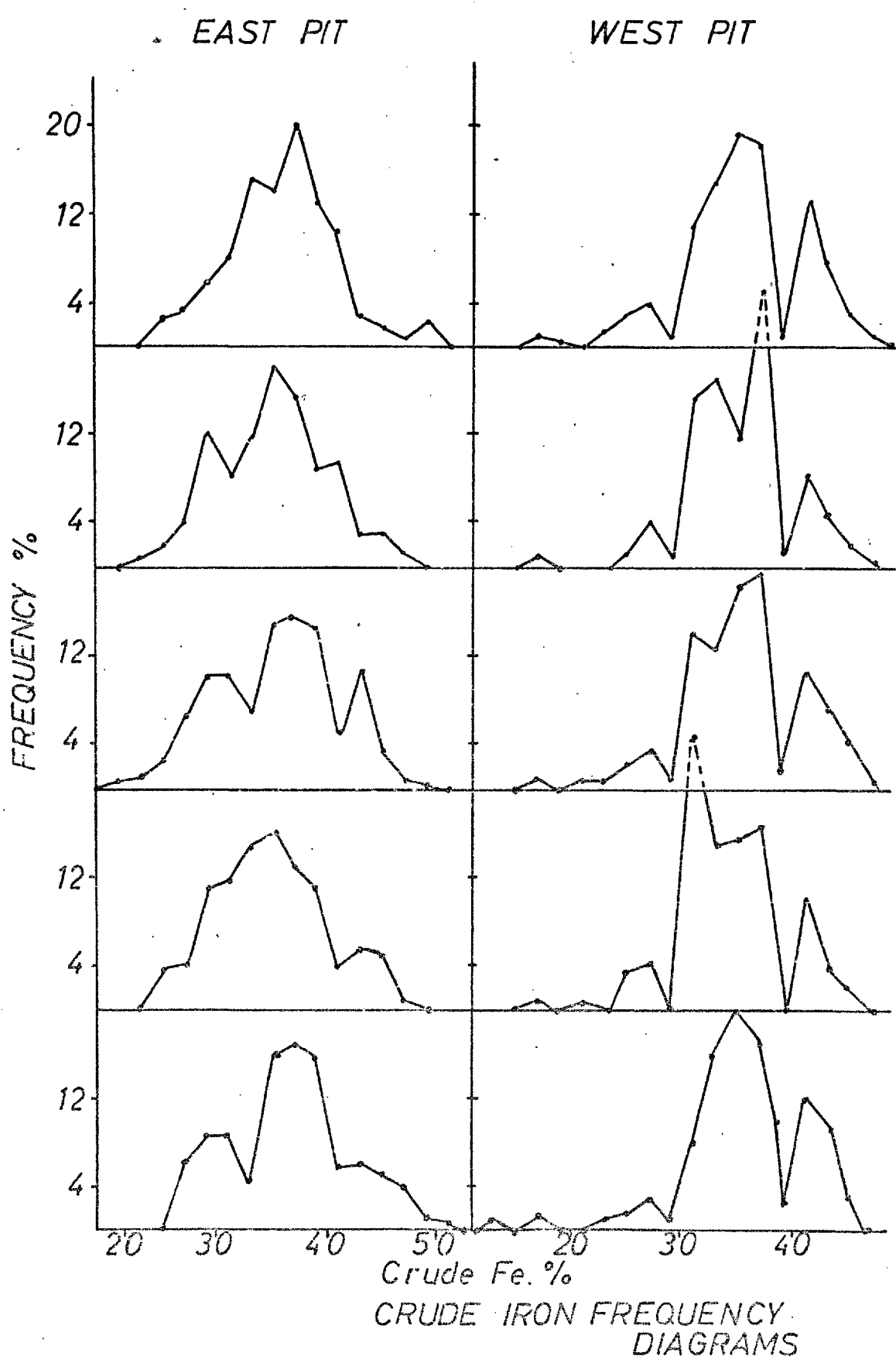
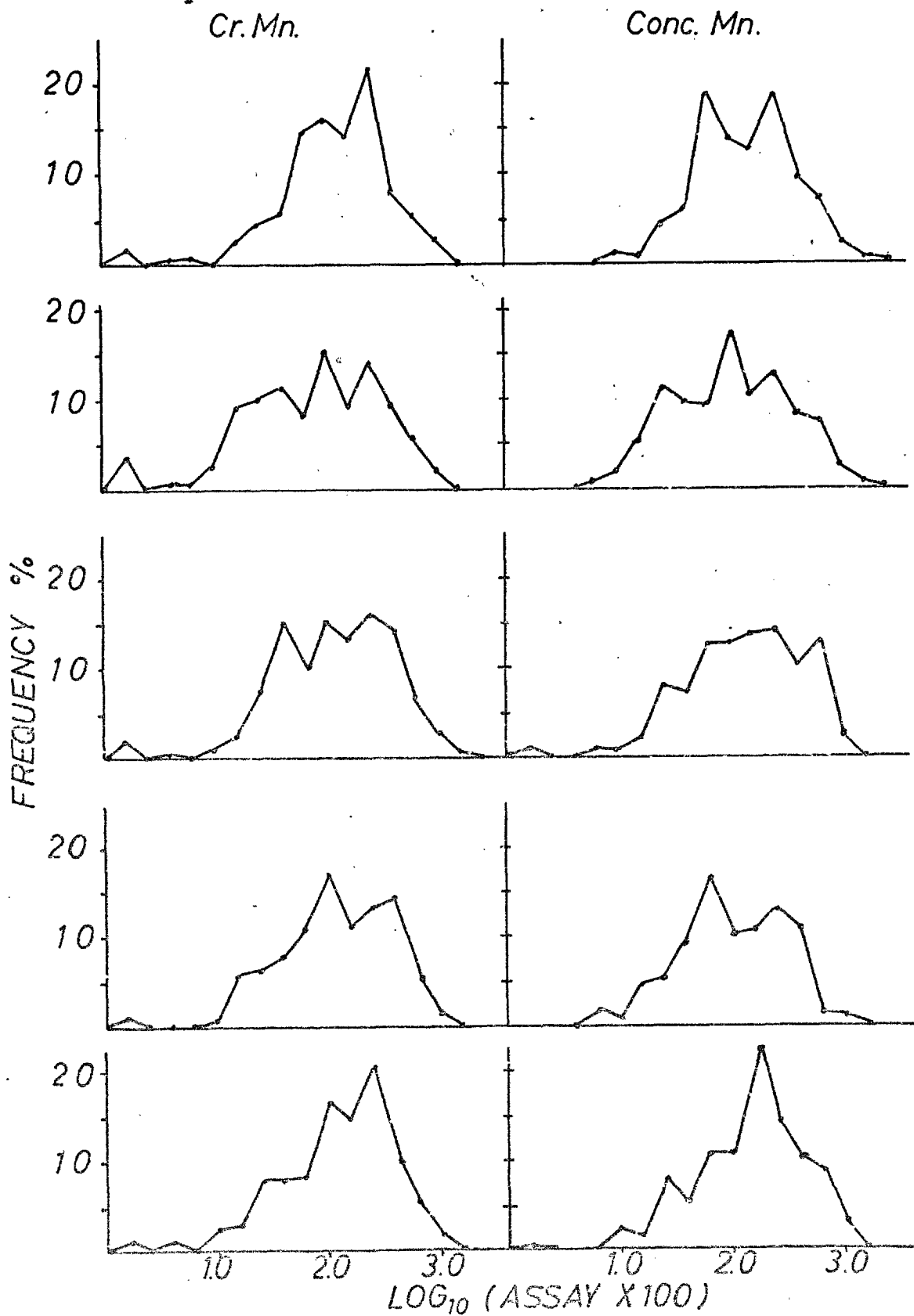


FIG. 3.5

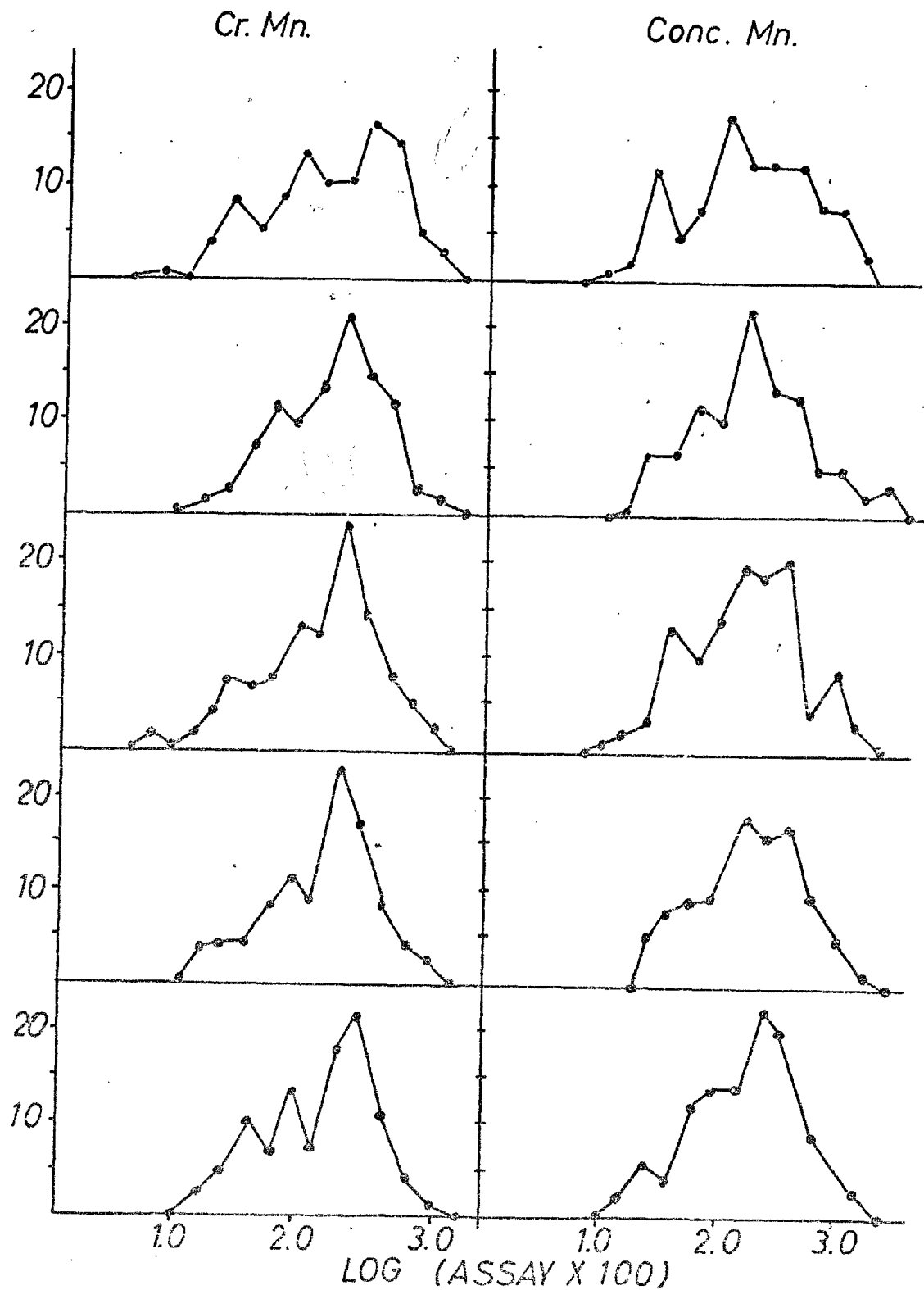
frequency distributions for the crude iron assay for both East and West Pits. One point is immediately clear that, as in the test sample mentioned above, the distribution does not fit a gaussian model. In fact in all of the ten samples, the tri-modality of the distribution is quite obvious. In the West Pit samples the major population is centred around the 34% to 36% Fe group with two subordinate groups centred at 26% to 28% Fe and 40% to 42% Fe group. In the East Pit one major group centres at 36% to 38% Fe and two subordinate groups at 28% to 30% Fe and 42% to 44% Fe; surprisingly an overall 2% shift higher in grade in the East Pit. There is however a much larger percentage of samples in the two higher grade groups in the West Pit. The regularity of the distribution of samples not only for crude iron, but also for crude manganese and concentrate manganese was quite remarkable. The three groups are better defined later in this chapter. Grade frequency distribution for weight recovery has been examined but, although all samples gave a distribution similar to that for the trial sample, they were rather inconsistent, and no attempt has been made to analyse them.

Figs. 3.6 and 3.7 show the frequency diagrams for crude and concentrate manganese for the East and West Pit areas respectively. In fact the diagrams are plots of frequency of $\log_{10} (\text{assay} \times 10^2)$. Here again the polymodality of



EAST PIT FREQUENCY DIAGRAMS

FIG 3.6



LOG (ASSAY X 100)

WEST PIT FREQUENCY DIAGRAMS

FIG. 3.7

the distributions is quite striking and the similarity of distribution from sample to sample greater than could be expected. The centres of the groups have been tabulated below:

East Pit	Crude Mn	1.6, 2.0, 2.4	)	
	Conc Mn	1.4, 1.8, 2.4, 2.8	)	
			)	$\log_{10}$
West Pit	Crude Mn	1.6, 2.0, 2.6	)	(assay x 100)
	Conc Mn	1.4, 1.8, 2.4, 3.0	)	
East Pit	Crude Mn	0.4, 1.0, 2.5	)	
	Conc Mn	.25, .63, 2.5, 6.3	)	% Mn
			)	
West Pit	Crude Mn	0.4, 1.0, 2.5	)	
	Conc Mn	.25, .63, 2.5, 10.0	)	

The three crude manganese populations appear quite consistent and easy to define however there is a general lack of consistency, both in the number of groups and in their centres, between the crude and concentrate manganese. Since nothing has been added to the system, on the contrary material is removed, in moving from the crude assay to concentrate, it seems reasonable to suggest that the mineralogy of the manganese is such that it may behave in more than one way on concentrating. I have made an attempt to define the populations better below.

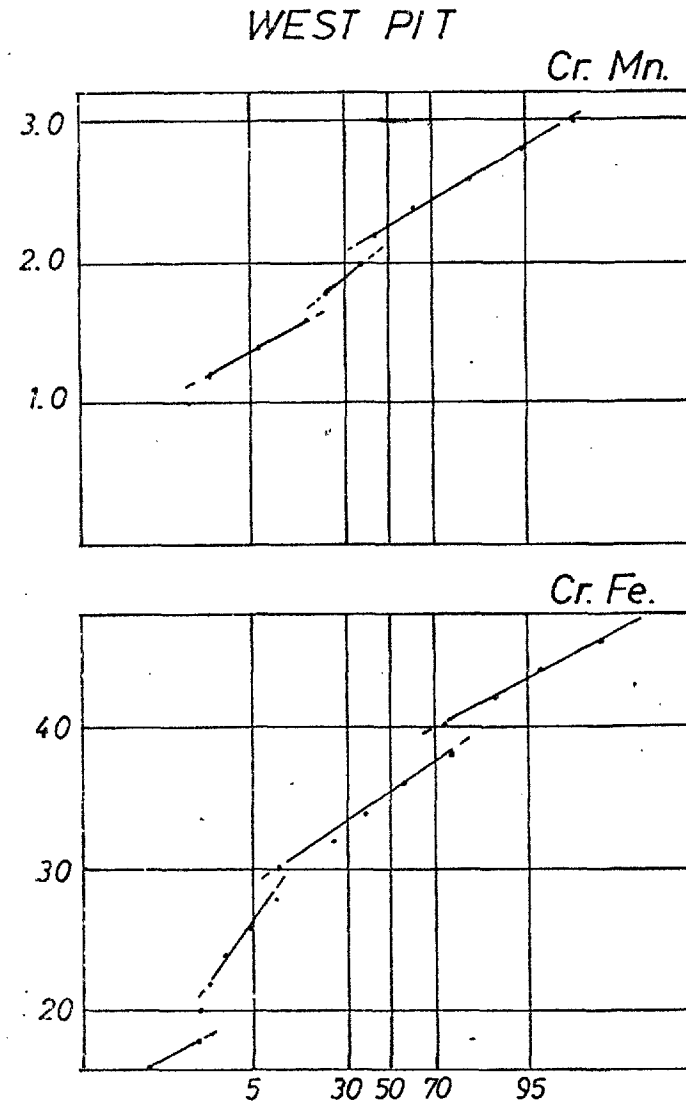
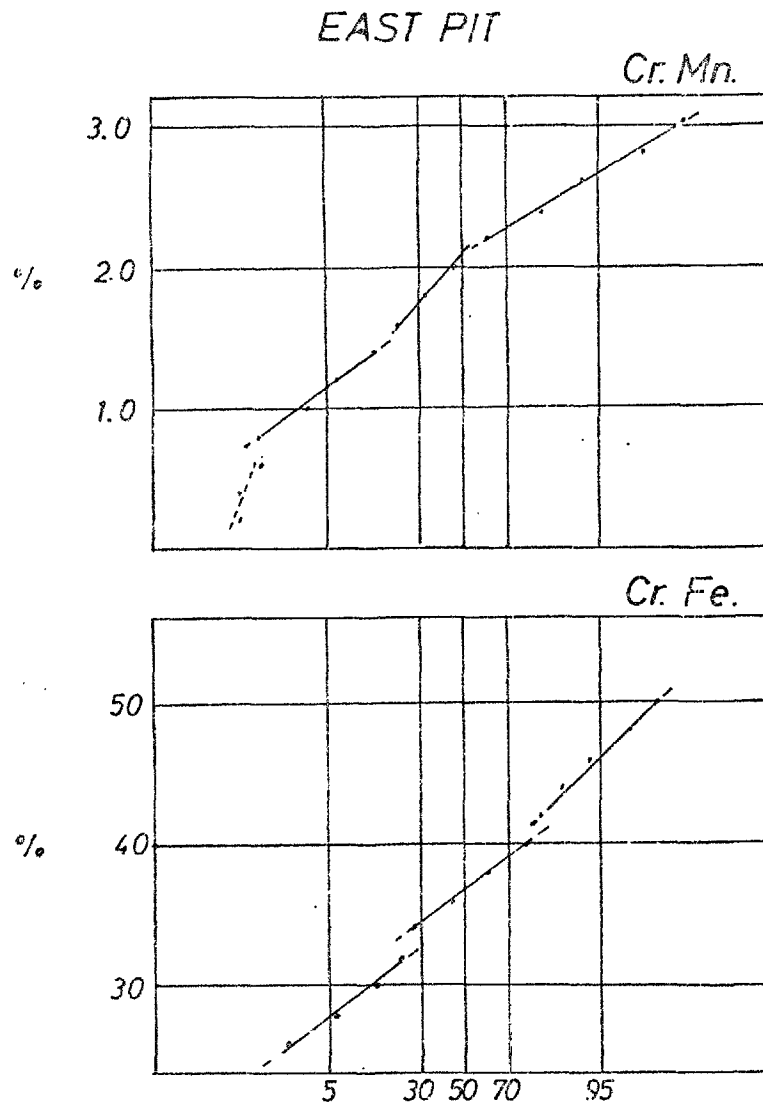
Definition of the populations

Before an attempt was made to define the various populations

in terms of their means and standard deviations, it was necessary to confirm the polymodal nature of the samples. Crude iron, and crude manganese (from both the East and West Pit), have been selected for further study. The complicated nature of the weight recovery distribution has already been discussed and at this stage the concentrate manganese distributions have been ignored because of the multiplicity of interpretation that may be derived from them.

The average distribution for each set of five samples has been calculated and the cumulative frequency plotted on probability diagram, (fig. 3.8). A gaussian distribution should plot as a straight line. Breaks or changes of slope of the line indicate a break in the distribution. For a gaussian distribution the mean will correspond to the 50% probability point on the curve and the 16% and 84% probability points to ~~once~~ the standard deviation on each side of the mean. The plots on fig. 3.8 confirm the presence of three populations in all four cases and possibly a fourth in the West Pit crude iron plot and East Pit crude manganese plot.

To better define each population the mean frequency distribution, for each variable under consideration, has been drawn. Since the distributions are trimodal in each case (where a fourth is present, it has a probability



PROBABILITY DIAGRAMS
for Cr. Fe. and Cr. Mn.

FIG. 3.8

of less than 1% of occurring) then, the upper limit of the population with the lowest mean, and the lower limit of the population with the highest mean, can be drawn in as a mirror image of the other limit. To calculate the limits of the middle population the values of the populations already accounted for have been scaled off and subtracted from the observed value at that point. The population may then be drawn in. This has been done for the variables being discussed and is shown in fig. 3.9. From these drawings the means and standard deviations for each population are as follows:

		$\bar{x}$	s	$\bar{x}$	s	$\bar{x}$	s
<u>West Pit</u>	Crude Iron	28%	1.3	36%	2.0	42%	1.8
<u>East Pit</u>	Crude Iron	29%	1.5	36%	1.3	42%	1.8
<u>West Pit</u>	Crude Mn.	1.6*	0.20	2.0*	0.20	2.6*	0.23
<u>East Pit</u>	Crude Mn.	1.4*	0.20	2.0*	0.20	2.4*	0.27

* Log (crude Mn.x 100)

Table 3.2

At the 95% confidence level we have:

<u>West Pit</u>	Crude Iron	group 1	28%Fe.	$\pm 2.5\%$
		2	36%Fe.	$\pm 3.9\%$
		3	42%Fe.	$\pm 3.2\%$
<u>East Pit</u>	Crude Iron	group 1	29%Fe.	$\pm 2.9\%$
		2	36%Fe.	$\pm 2.5\%$
		3	42%Fe.	$\pm 3.2\%$

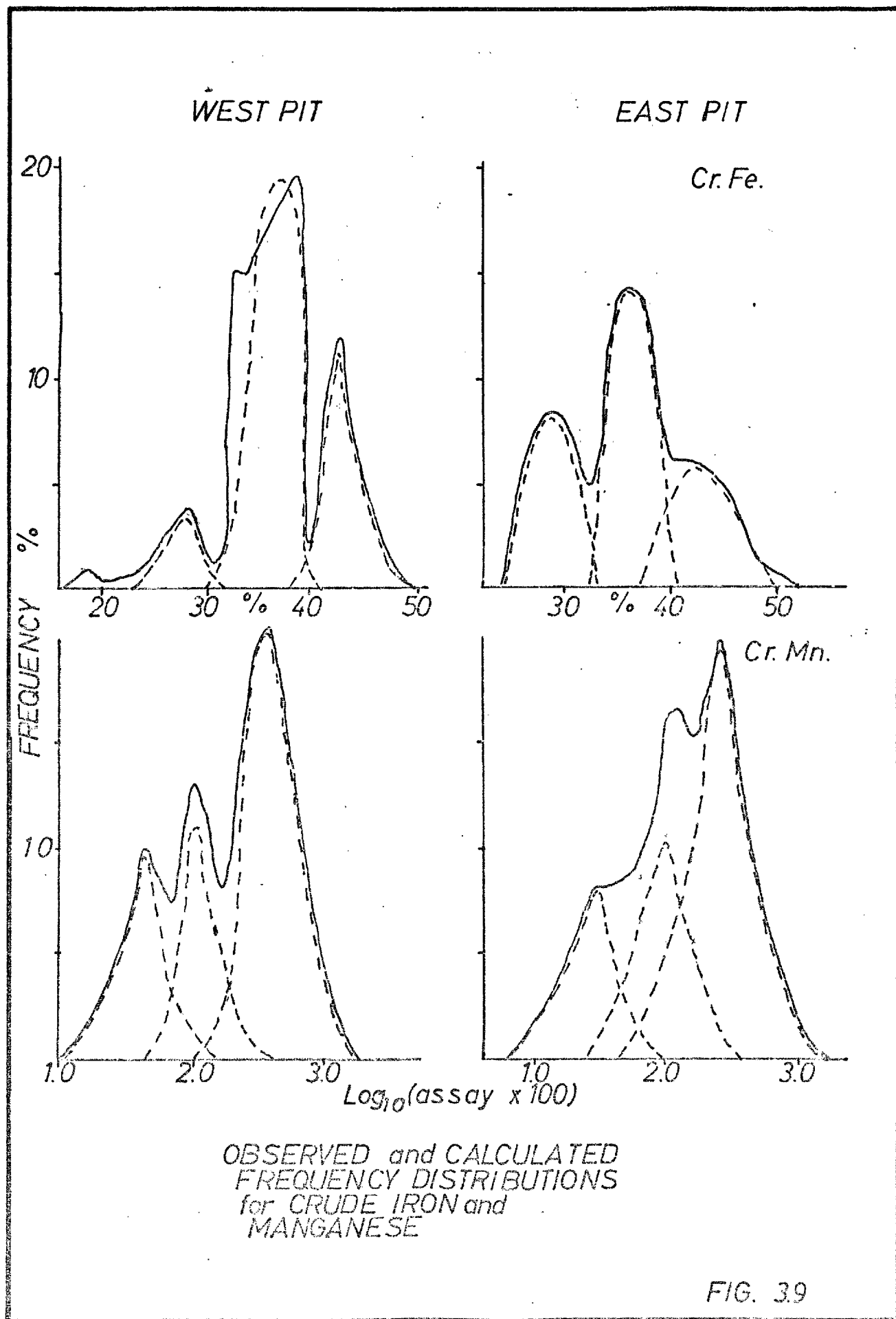


FIG. 39

<u>West Pit</u>	Crude Mn.	group	1	1.6 $\pm$ 0.4	)	$\log_{10}$ (assay x 100)
			2	2.0 $\pm$ 0.4	)	
			3	2.6 $\pm$ 0.35	)	
					)	
<u>East Pit</u>	Crude Mn.	group	1	1.4 $\pm$ 0.4	)	
			2	2.0 $\pm$ 0.4	)	
			3	2.4 $\pm$ 0.53	)	

For the manganese assays in terms of real values upper limit, mean and lower limit for each type is as follows:

		U.L.	mean	L.L.		
<u>West Pit</u>	1	1.0	0.4	0.15	)	% Manganese
	2	2.5	1.0	0.4	)	
	3	9.0	4.0	1.8	)	
<u>East Pit</u>	1	0.6	0.25	0.1	)	
	2	2.5	1.0	0.4	)	
	3	8.0	2.5	0.8	)	

The skewness of the manganese distributions may point to the folly of such a close grading limit for this element on the mine; (present grading limit is $1.69 \pm 0.15\%$ Mn in the final concentrate).

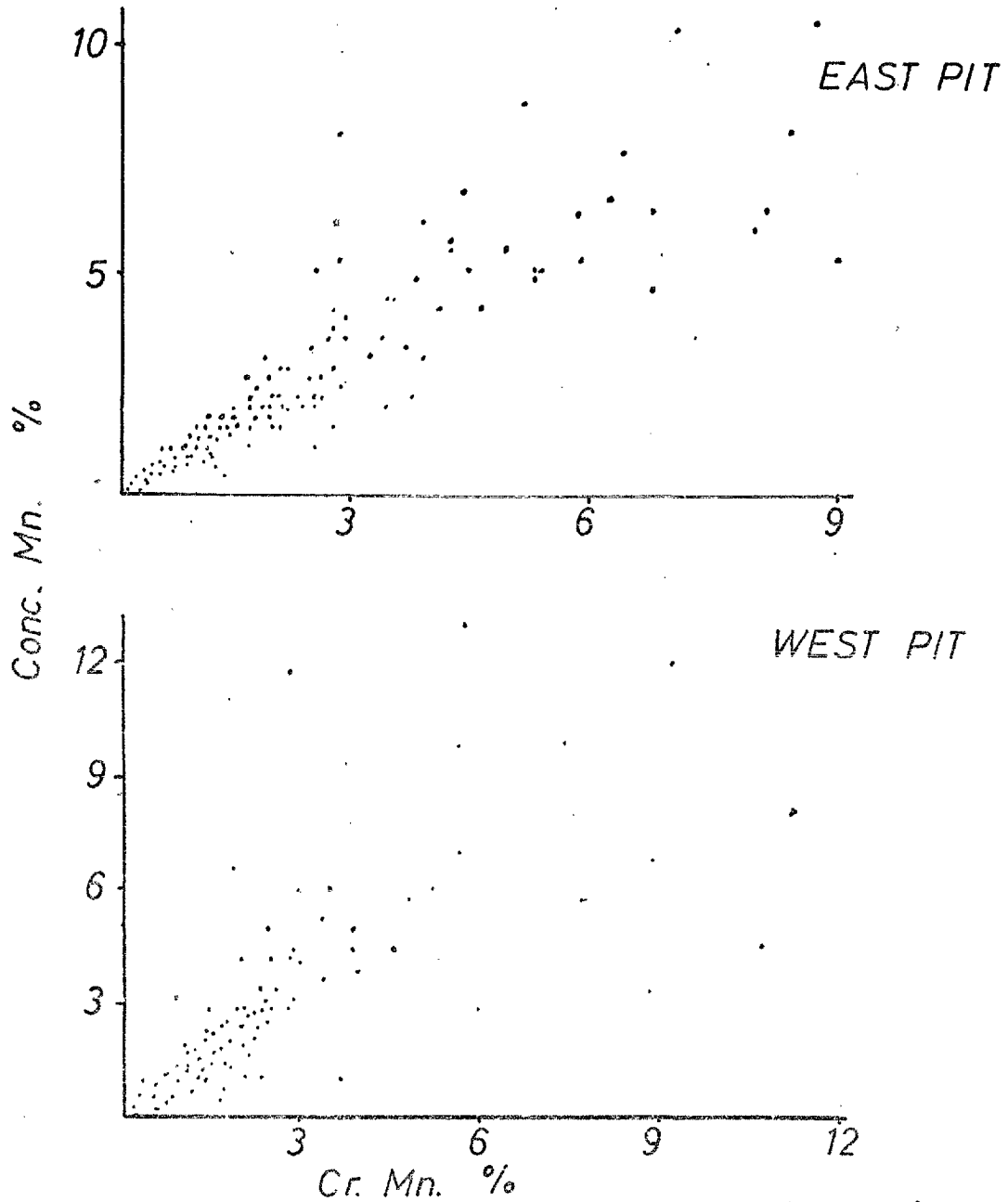
The upper, middle and lower members, as such, have been sampled to see if they fit the parameters calculated above, as an obvious split, however each unit has a very wide standard deviation and different mean from each of the above groups:

	Crude Iron		Crude Mn.	
	$\bar{x}$	s	$\bar{x}$	s
Upper	32.4	4.8	1.4	0.4
Middle	38.6	5.2	1.9	0.45
Lower	35.5	5.0	2.5	0.45

It must be concluded that these groups are smaller units inside the already defined units of upper, middle and lower ores.

Scatter diagrams

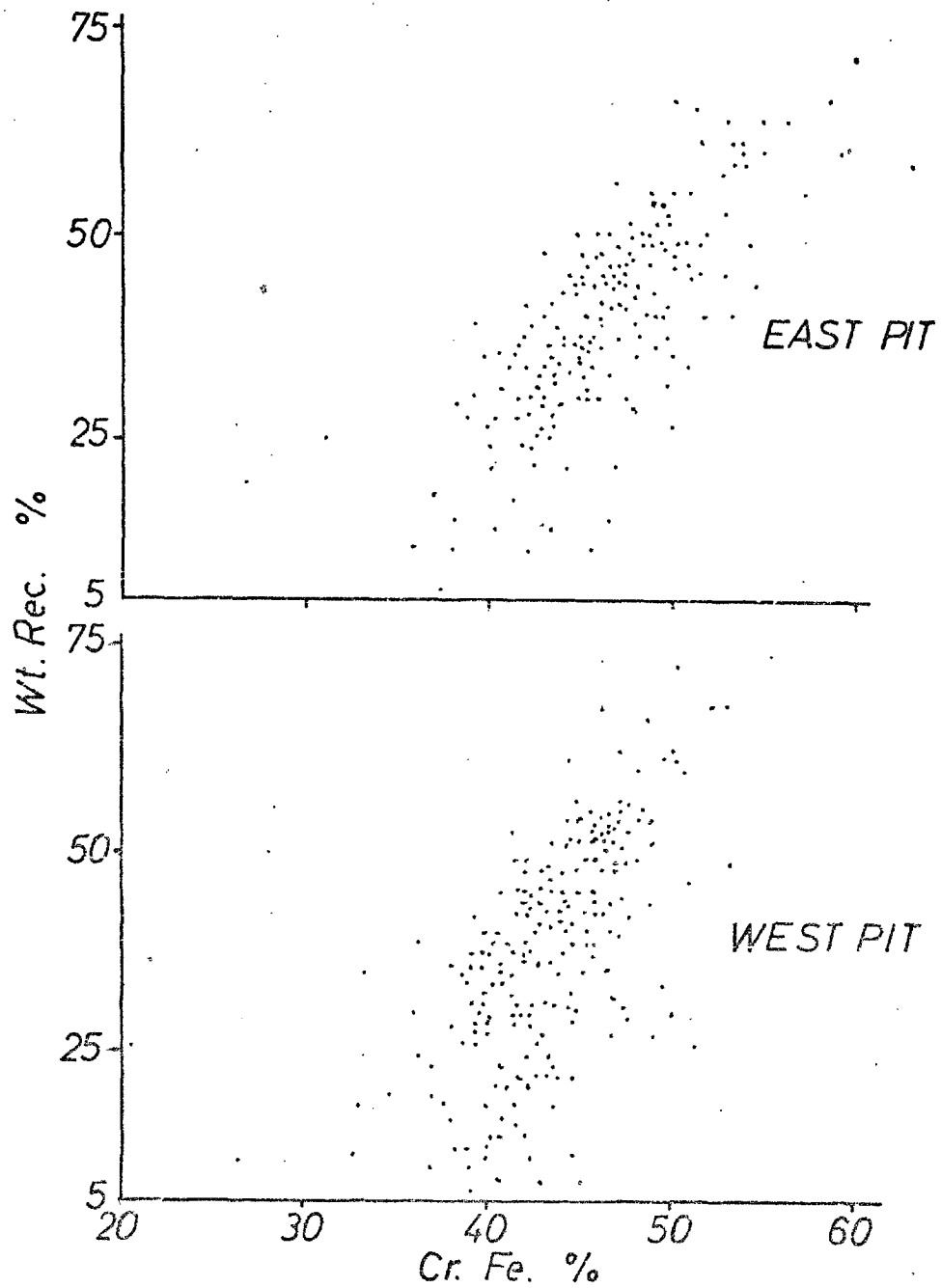
Although it has been possible to identify groups within the iron formation based on the crude iron and crude manganese assay, this is not sufficient to provide a thorough breakdown of the ore statistically. Two further parameters, which are of vital importance in the control of the operation at Wabush, namely weight recovery and concentrate manganese, were next subjected to a statistical interpretation. A scatter diagram is merely a plot of one variable against another, but is a quick, easy method of revealing simple relationships between pairs of data. Figs. 3.10 to 3.16 inclusive are scatter plots of the data available. Fig. 3.10 is an arithmetic plot of crude manganese against concentrate manganese and fig. 3.11 a plot, arithmetically of crude iron against weight recovery. In both cases a



SCATTER PLOT of
CRUDE Mn. AGAINST
CONCENTRATE Mn.

FOR LOG PLOT
SEE FIG. 3.17

FIG. 3.10



SCATTER PLOTS of
CRUDE Fe. AGAINST
WEIGHT RECOVERY

FIG. 3.11

good direct relationship can be seen, more so in fig.3.10, than fig.3.11. The other diagrams show considerable scatter. On this basis it was decided to concentrate on the relationships crude manganese to concentrate manganese and crude iron to weight recovery.

Correlation and regression

Correlation is a statistical technique for measuring the degree of association between variables, whereas regression analysis is a technique for describing the relationship between two variables in terms of an equation relating the response variable (X_2) in terms of the predictor variable (X_1).

(a) Correlation.

The measure of correlation between two variables used in this study is the correlation coefficient (r)

given by

$$r = \frac{\frac{1}{n} \sum (x_{1i} - \bar{x}_1)(x_{2i} - \bar{x}_2)}{s_1 s_2} \dots\dots\dots 3.1$$

The significance of r is tested by measuring its standard error (S_r)

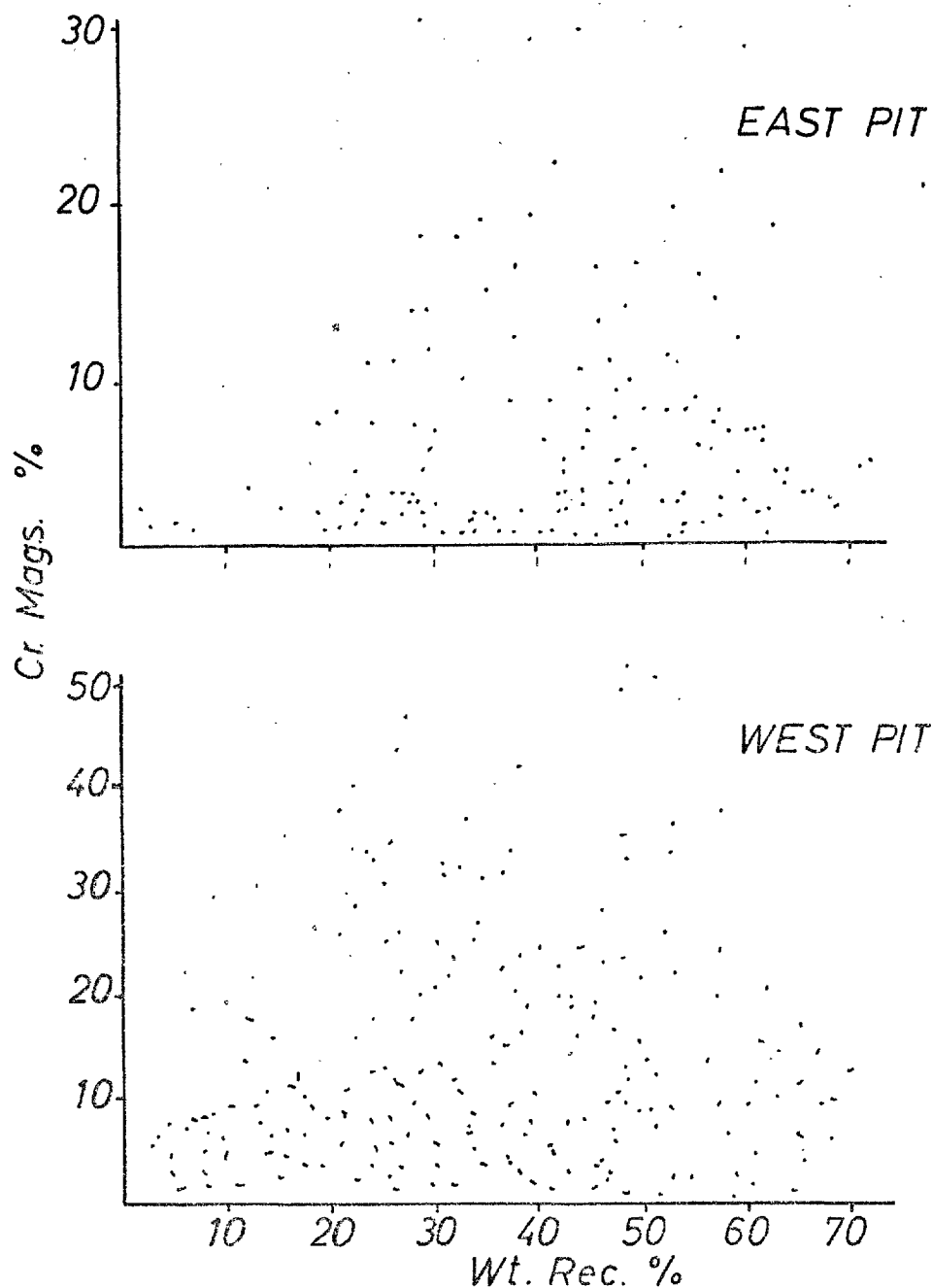
where $S_r = \frac{1-r^2}{\sqrt{n}} \dots\dots\dots 3.2.$

for large n or

$$S_r = \sqrt{\left(\frac{r^2 (n-2)}{1-r^2} \right)} \dots\dots\dots 3.3$$

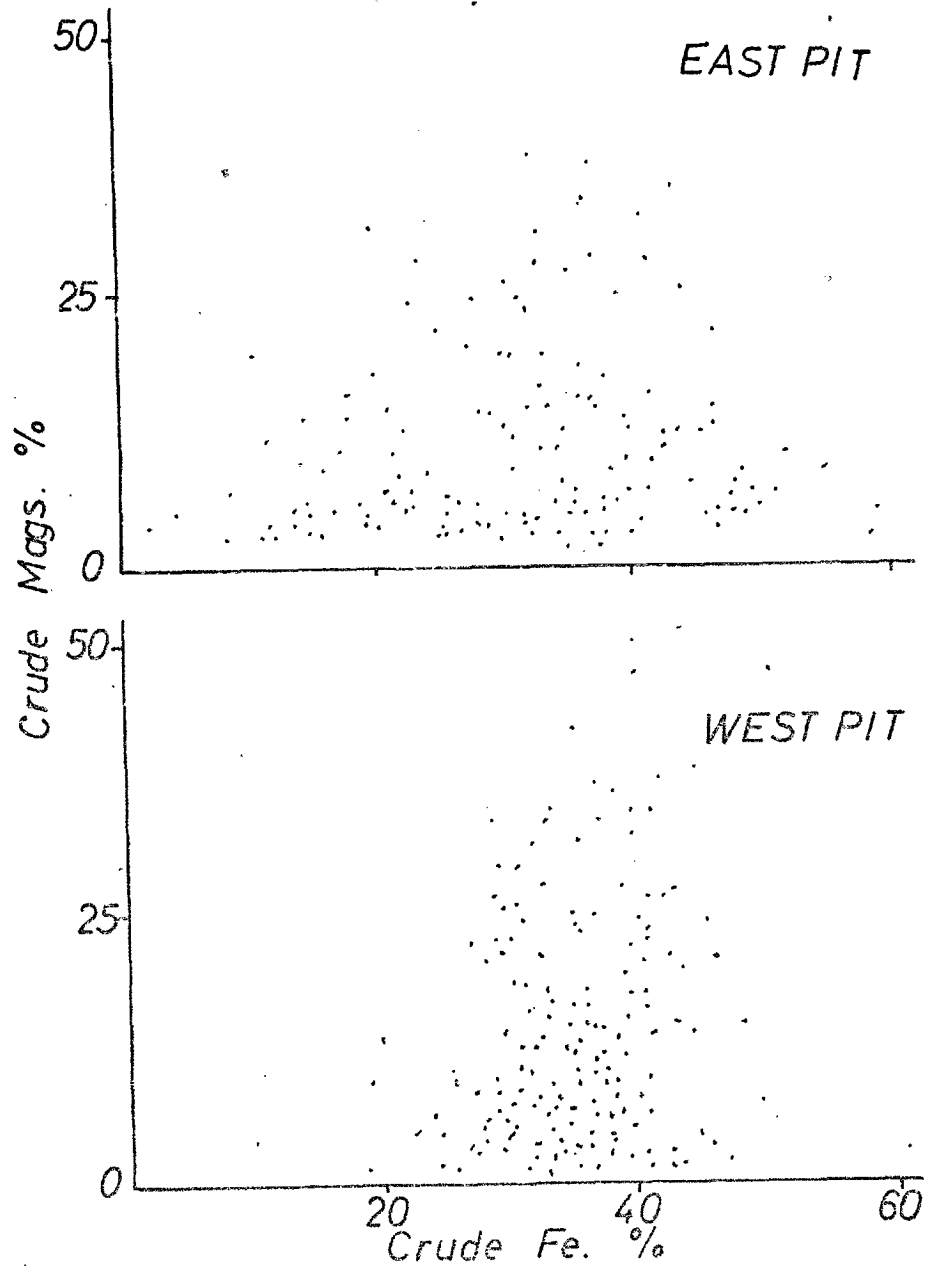
where r is small. This is distributed as 't' with $n-2$ degrees of freedom.

The confidence of the correlation can be measured



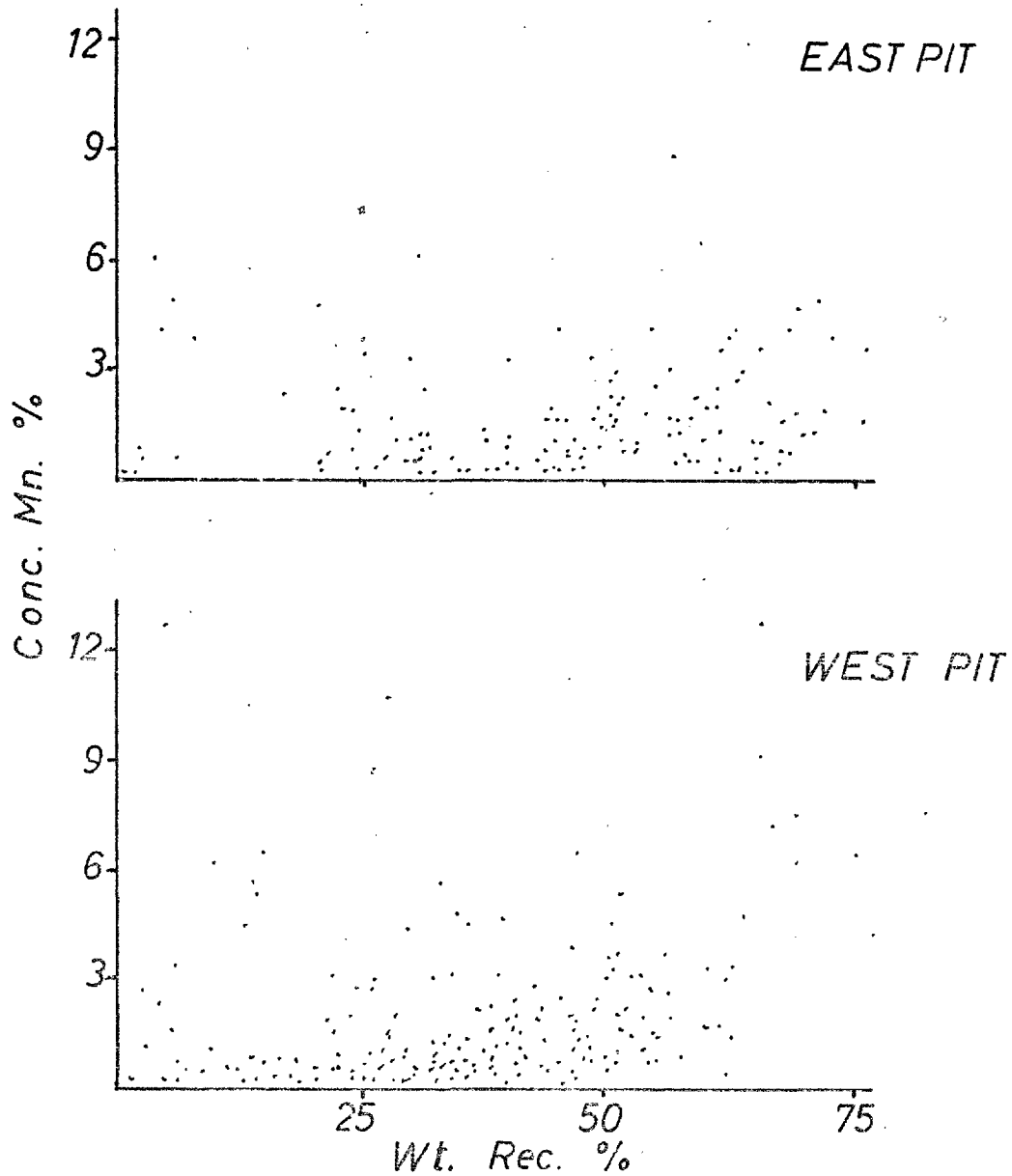
SCATTER PLOT of
WEIGHT RECOVERY
AGAINST CRUDE
MAGNETICS

FIG. 3.12



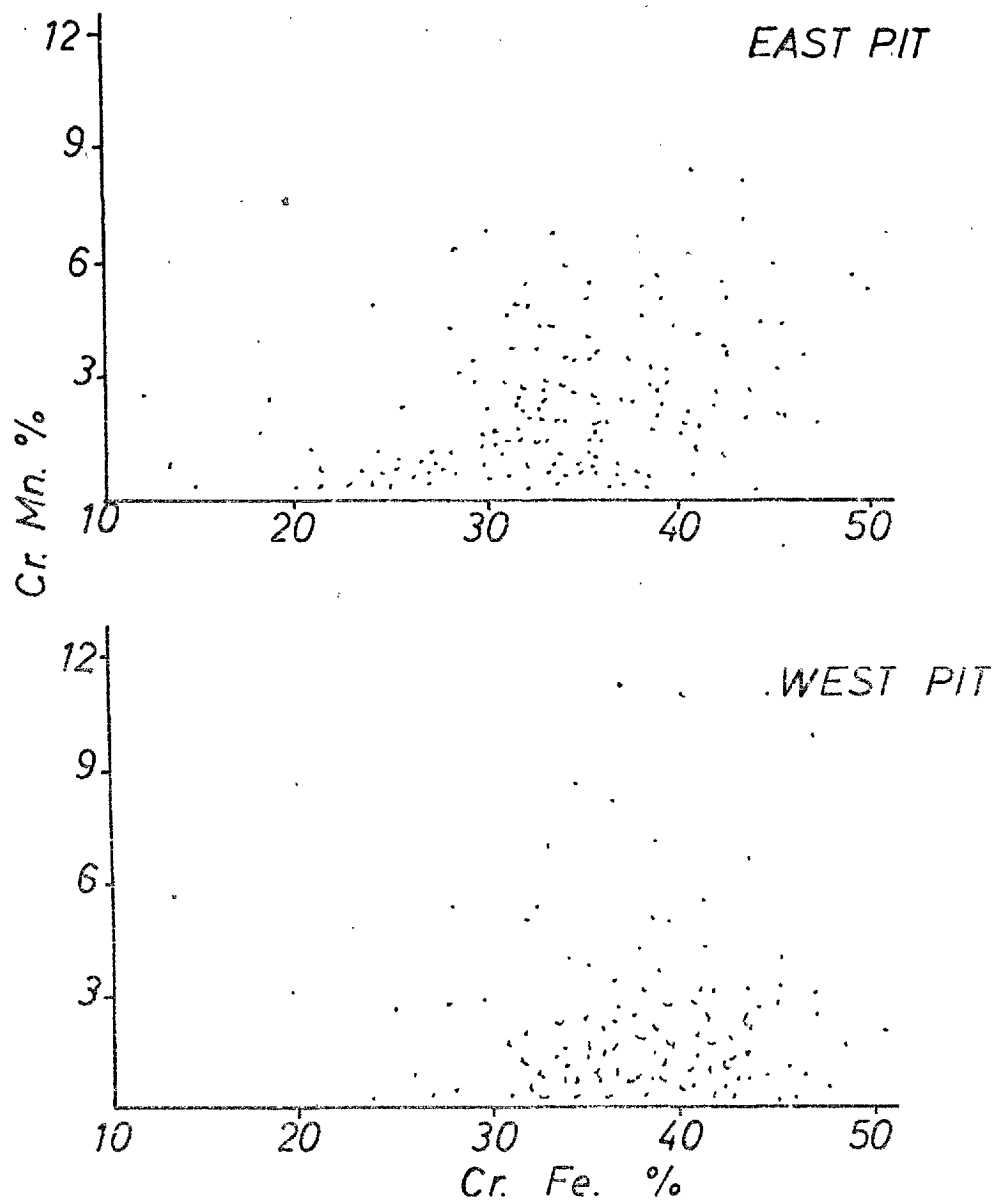
SCATTER PLOTS of
CRUDE IRON against
CRUDE MAGNETICS

FIG. 3.13



SCATTER PLOTS of
WEIGHT RECOVERY against
CONCENTRATE MANGANESE

FIG. 3.14



SCATTER PLOTS of
CRUDE IRON against
CRUDE MANGANESE

FIG. 3.15

using the probability function *.

$$Pf\left(\frac{\text{uncorrelated}}{\text{correlated}} \parallel r^2\right) \approx \sqrt{\left(\frac{2n-1}{\pi}\right) (1-r^2)^{\frac{n-2}{2}}} \dots\dots 3.4$$

which will tend to zero for high correlation.

The correlation coefficient varies between +1 and -1. A figure of zero indicates no correlation whilst +1 indicates perfect positive correlation and -1 perfect negative correlation.

(b) Regression

Where the points in a cluster appear to group around a line then regression is appropriate. Regression fits the best line to a series of points to minimise the sum of the squares of the distances from the line. Unless correlation is perfect ($r = \pm 1$) then it is possible to draw two lines through a cluster of points. One will be the regression of x_1 on x_2 and the other of x_2 on x_1 . Where there is perfect correlation the two lines are coincident. The equation:

$$(x_2 - \bar{x}_2) = \frac{rs_2}{s_1} (x_1 - \bar{x}_1) \dots\dots\dots 3.5$$

provides an estimation of x_2 in terms of x_1 . Under

* Jeffreys, H (1961) Theory of probability
Oxford: Clarendon Press.

normal circumstances it is easy to decide which value should be the predictor variable (x_1) and which the response variable (x_2). One for example may have the more reliable assay procedure, however in the case under study, because we have already been able to define crude manganese and crude iron in terms of their distributions these make for more suitable predictors on which to better define the concentrate manganese and weight recovery data.

(c) Residuals

Having defined a relationship between two variables it is possible to better understand the nature of the cluster in terms of the residuals. After generating an equation say for example relating one variable A to a second variable B so that

$$B = mA + C$$

it is possible to calculate a value of B to correspond to each value of A. The value of the residuals is then given by subtracting the calculated value of B from the real one. Residuals can normally be expected to have a mean of zero and show a gaussian distribution around this point.

(d) The crude to concentrate manganese relationship

Examination of the relationship between crude and concentrate manganese has been approached on a section by section basis. The relevant data is summarised in tables 3.3 and 3.4 for the West Pit and 3.5 and 3.6

Note on tables 3.3 - 3.6

Tables 3.3 - 3.6 have been copied
directly from computer output and
inasmuch:

MN to be read as Mn.

+ - to be read as -

<u>CRUDE MANGANESE</u>				<u>CONC. MANGANESE</u>			<u>REGRESSION</u>			
<u>SECT</u>	<u>AVG</u>	<u>VAR</u>	<u>STD</u>	<u>AVG</u>	<u>VAR</u>	<u>STD</u>	<u>R</u>	<u>SR</u>	<u>PF</u>	<u>N</u>
9750	1.60	.302	.549	1.62	.315	.561	.96	.008	.000000	39
10000	1.98	.355	.596	2.06	.310	.556	.97	.003	.000000	105
10250	1.53	.344	.586	1.57	.353	.594	.97	.004	.000000	66
10500	1.83	.426	.653	1.87	.440	.663	.98	.004	.000000	80
10750	1.81	.412	.642	1.72	.437	.661	.96	.006	.000000	62
11000	1.69	.346	.588	1.67	.341	.584	.94	.009	.000000	63
11250	1.86	.214	.462	1.80	.251	.501	.93	.010	.000000	54
11500	2.23	.242	.492	2.22	.289	.537	.95	.006	.000000	100
11750	1.95	.271	.521	1.96	.261	.511	.97	.004	.000000	68
12000	2.12	.221	.470	2.08	.229	.478	.93	.007	.000000	160
12250	2.26	.197	.444	2.19	.245	.495	.96	.006	.000000	123
12500	2.10	.207	.455	2.08	.228	.477	.97	.004	.000000	96
12750	2.20	.185	.430	2.18	.216	.465	.91	.005	.000000	121
13000	2.22	.154	.393	2.22	.184	.429	.92	.007	.000000	132
13250	2.35	.116	.341	2.37	.147	.384	.92	.009	.000000	84
13500	2.10	.118	.433	2.07	.228	.478	.90	.009	.000000	97
13750	2.28	.181	.425	2.20	.198	.435	.93	.009	.000000	60
14000	2.17	.175	.418	2.19	.141	.376	.88	.011	.000000	78
14250	2.51	.064	.253	2.48	.136	.368	.95	.013	.000000	43
14500	2.17	.169	.411	2.17	.171	.413	.91	.011	.000000	69
15000	2.16	.222	.472	2.16	.188	.434	.88	.018	.000000	50
15500	2.17	.281	.530	2.24	.287	.536	.99	.003	.000000	50
16000	2.21	.245	.495	2.95	.224	.473	.98	.006	.000000	39

TABLE 3.3

SECT = SECTION
 AVG = AVERAGE
 VAR = VARIANCE
 STD = STANDARD DEVIATION
 R = CORRELATION COEFFICIENT
 SR = STANDARD ERROR OF R
 PF = PROBABILITY FUNCTION $Pf\left(\frac{\text{Uncorrelated}}{\text{Correlated}} \mid r^2\right)$
 N = NUMBER OF POINTS

EQUATIONS

SECTION

CRUDE-MN = .939X CONC. MN + .078	9750
CONC. MN = .981X CRUDE MN + .050	
CRUDE-MN = 1.040X CONC. MN + -.162	10000
CONC. MN = .905X CRUDE MN + .268	
CRUDE-MN = .957X CONC. MN + .028	10250
CONC. MN = .983X CRUDE MN + .066	
CRUDE-MN = .965X CONC. MN + .025	10500
CONC. MN = .995X CRUDE MN + .049	
CRUDE-MN = .932X CONC. MN + .206	10750
CONC. MN = .988X CRUDE MN + -.069	
CRUDE-MN = .946X CONC. MN + .109	11000
CONC. MN = .934X CRUDE MN + .092	
CRUDE-MN = .858X CONC. MN + .316	11250
CONC. MN = 1.009X CRUDE MN + -.076	
CRUDE-MN = .870X CONC. MN + .298	11500
CONC. MN = 1.037X CRUDE MN + -.092	
CRUDE-MN = .989X CONC. MN + .012	11750
CONC. MN = .951X CRUDE MN + .105	
CRUDE-MN = .914X CONC. MN + .218	12000
CONC. MN = .946X CRUDE MN + .075	
CRUDE-MN = .861X CONC. MN + .374	12250
CONC. MN = 1.070X CRUDE MN + -.229	
CRUDE-MN = .925X CONC. MN + .175	12500
CONC. MN = 1.017X CRUDE MN + -.055	
CRUDE-MN = .842X CONC. MN + .366	12750
CONC. MN = .984X CRUDE MN + .015	
CRUDE-MN = .843X CONC. MN + .349	13000
CONC. MN = 1.004X CRUDE MN + -.009	

CRUDE-MN	=	.817X CONC. MN + .414	13250
CONC. MN	=	1.036X CRUDE MN + -.065	
CRUDE-MN	=	.815X CONC. MN + .412	13500
CONC. MN	=	.994X CRUDE MN + -.016	
CRUDE-MN	=	.909X CONC. MN + .281	13750
CONC. MN	=	.952X CRUDE MN + .030	
CRUDE-MN	=	.978X CONC. MN + .028	14000
CONC. MN	=	.792X CRUDE MN + .472	
CRUDE-MN	=	.653X CONC. MN + .890	14250
CONC. MN	=	1.382X CRUDE MN + -.988	
CRUDE-MN	=	.906X CONC. MN + .205	14500
CONC. MN	=	.914X CRUDE MN + .186	
CRUDE-MN	=	.957X CONC. MN + .093	15000
CONC. MN	=	.809X CRUDE MN + .412	
CRUDE-MN	=	.979X CONC. MN + -.023	15500
CONC. MN	=	1.001X CRUDE MN + .067	
CRUDE-MN	=	1.026X CONC. MN + -.815	16000
CONC. MN	=	.936X CRUDE MN + .880	

TABLE 3.4

NOTE CRUDE MN = Log₁₀ (ASSAY X100)

<u>CRUDE MANGANESE</u>				<u>EAST PIT CONC. MANGANESE</u>			<u>REGRESSION</u>			
<u>SECT</u>	<u>AVG</u>	<u>VAR</u>	<u>STD</u>	<u>AVG</u>	<u>VAR</u>	<u>STDEV</u>	<u>R</u>	<u>SR</u>	<u>PF</u>	<u>N</u>
18500	1.90	.195	.442	1.99	.252	.502	.94	.010	.000000	42
18750	2.00	.148	.385	2.06	.184	.429	.94	.007	.000000	57
19000	2.05	.169	.411	2.10	.187	.433	.92	.007	.000000	60
19250	2.04	.156	.395	2.09	.187	.433	.92	.008	.000000	64
19500	2.03	.155	.393	2.06	.199	.446	.96	.006	.000000	40
19750	2.17	.140	.374	2.20	.157	.396	.91	.010	.000000	52
20000	1.96	.234	.484	2.10	.205	.453	.89	.010	.000000	63
20250	2.09	.266	.561	2.12	.234	.487	.96	.005	.000000	58
20500	1.93	.204	.452	1.96	.193	.436	.94	.007	.000000	74
20750	1.97	.232	.481	2.01	.210	.458	.90	.010	.000000	82
21000	2.00	.268	.571	2.01	.271	.521	.96	.004	.000000	70
21250	2.08	.233	.472	2.05	.318	.564	.90	.011	.000000	67
21500	1.90	.361	.601	1.99	.290	.538	.97	.005	.000000	53
21750	2.09	.291	.540	2.14	.252	.502	.94	.008	.000000	45
22000	2.13	.179	.424	2.20	.177	.420	.91	.020	.000000	32

TABLE 3.5

LIST OF ABBREVIATIONS IS AS FOR TABLE 3.3

<u>EQUATIONS</u>	<u>SECTION</u>
$\begin{aligned} \text{CRUDE-MN} &= .828\text{X CONC. MN} + .253 \\ \text{CONC. MN} &= 1.068\text{X CRUDE MN} + -.038 \end{aligned}$	18500
$\begin{aligned} \text{CRUDE-MN} &= .844\text{X CONC. MN} + .262 \\ \text{CONC. MN} &= 1.047\text{X CRUDE MN} + -.035 \end{aligned}$	18750
$\begin{aligned} \text{CRUDE-MN} &= .873\text{X CONC. MN} + .216 \\ \text{CONC. MN} &= .969\text{X CRUDE MN} + .113 \end{aligned}$	19000
$\begin{aligned} \text{CRUDE-MN} &= .839\text{X CONC. MN} + .286 \\ \text{CONC. MN} &= 1.009\text{X CRUDE MN} + .033 \end{aligned}$	19250
$\begin{aligned} \text{CRUDE-MN} &= .846\text{X CONC. MN} + .287 \\ \text{CONC. MN} &= 1.089\text{X CRUDE MN} + -.152 \end{aligned}$	19500
$\begin{aligned} \text{CRUDE-MN} &= .859\text{X CONC. MN} + .279 \\ \text{CONC. MN} &= .964\text{X CRUDE MN} + .109 \end{aligned}$	19750
$\begin{aligned} \text{CRUDE-MN} &= .951\text{X CONC. MN} + -.037 \\ \text{CONC. MN} &= .833\text{X CRUDE MN} + .467 \end{aligned}$	20000
$\begin{aligned} \text{CRUDE-MN} &= 1.106\text{X CONC. MN} + -.254 \\ \text{CONC. MN} &= .833\text{X CRUDE MN} + .378 \end{aligned}$	20250
$\begin{aligned} \text{CRUDE-MN} &= .974\text{X CONC. MN} + .020 \\ \text{CONC. MN} &= .907\text{X CRUDE MN} + .210 \end{aligned}$	20500
$\begin{aligned} \text{CRUDE-MN} &= .945\text{X CONC. MN} + .070 \\ \text{CONC. MN} &= .857\text{X CRUDE MN} + .322 \end{aligned}$	20750
$\begin{aligned} \text{CRUDE-MN} &= 1.052\text{X CONC. MN} + -.115 \\ \text{CONC. MN} &= .876\text{X CRUDE MN} + .258 \end{aligned}$	21000

CRUDE-MN	=	.753X CONC. MN +	.536	21250
CONC. MN	=	1.075X CRUDE MN +	-.187	
CRUDE-MN	=	1.084X CONC. MN +	-.256	21500
CONC. MN	=	.868X CRUDE MN +	.340	
CRUDE-MN	=	1.011X CONC. MN +	-.074	21750
CONC. MN	=	.874X CRUDE MN +	.314	
CRUDE-MN	=	.919X CONC. MN +	.109	22000
CONC. MN	=	.901X CRUDE MN +	.280	

TABLE 3.6

NOTE CRUDE MN = Logio (ASSAY X100)

for the East Pit. A number of points emerge from examination of the tables.

- (i) Increase in manganese content both crude and concentrate from West to East in the West Pit, and the lack of any apparent trend in the East Pit.
- (ii) The higher average for concentrate manganese, in both pits
- (iii) The higher variance for concentrate manganese
- (iv) The high value of r (.90) and low value of S_r and the zero value (to 6 decimal places) of the probability function
- (v) The consistency of the relationship as expressed in the regression equations.

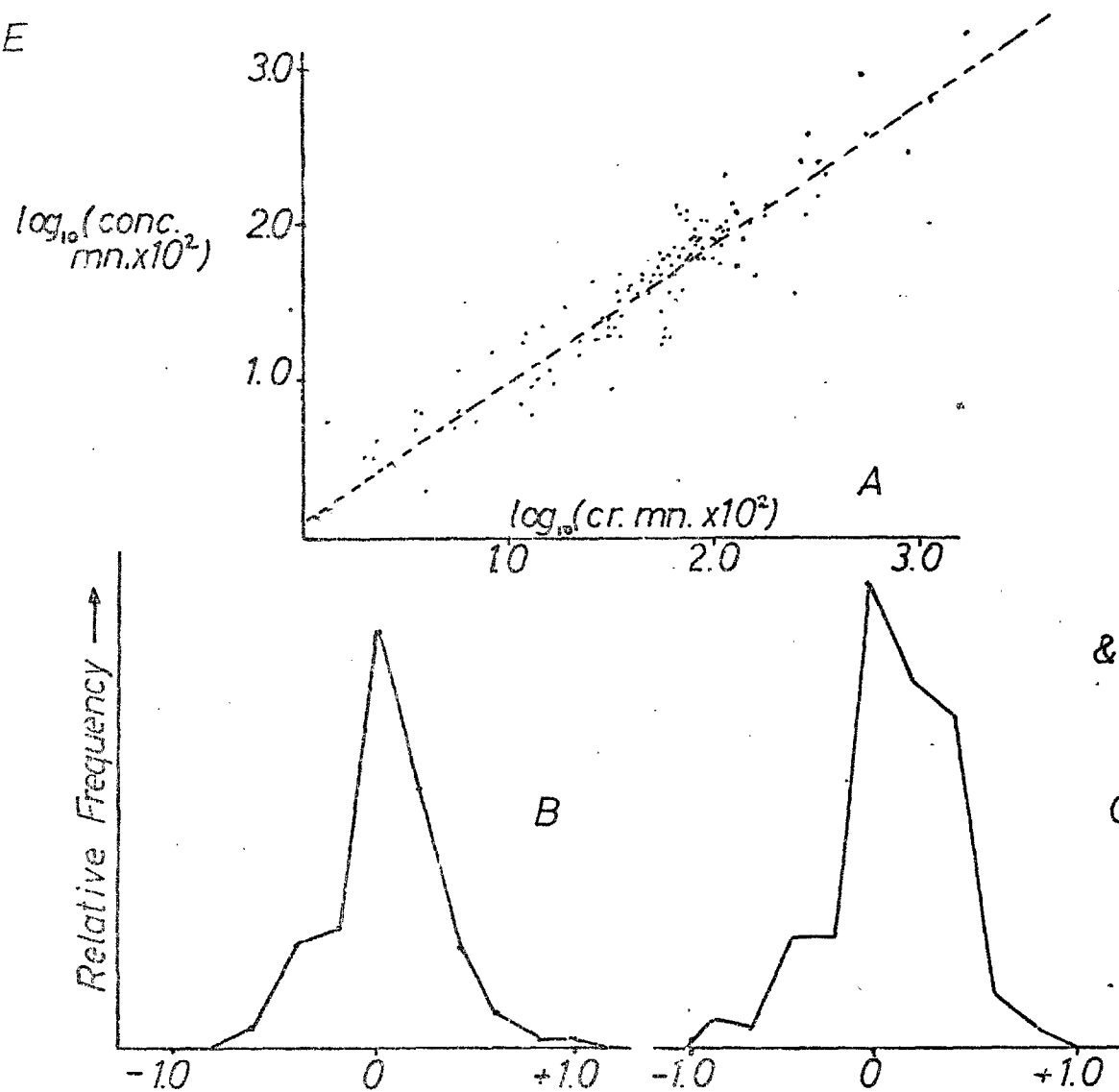
Sections 20750N from the East Pit and 13,250E from the West Pit have been chosen to demonstrate the features of importance that have emerged from the study. They are reasonably typical in that they show those features that are consistent throughout the pits they represent. Scatter diagrams, residual distributions, on the average regression equation for the pit, and on the regression equation for the particular section, for the West and East Pits are given in figs. 3.16 and 3.17 respectively.

As was demonstrated with the scatter diagrams above

Note on figs. 3.16 - 3.17

- A - regression line
- B - distribution of residuals from
unique equation for section
- C - distribution of residuals from
average equation

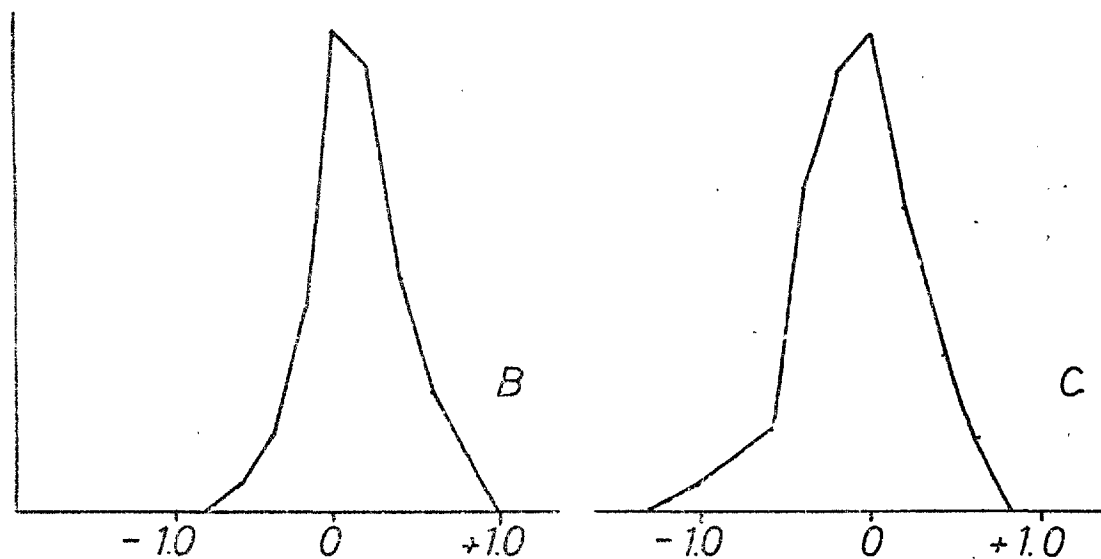
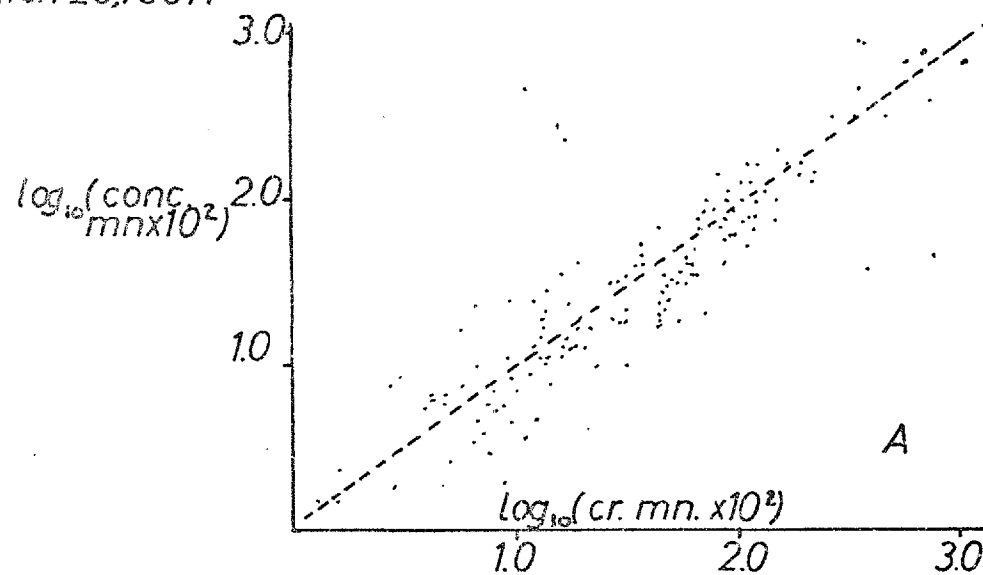
Section 13,250E



TYPICAL WEST PIT
REGRESSION LINE
& RESIDUAL
DISTRIBUTION

FIG 3.16

Section 20,750N



TYPICAL EAST PIT
REGRESSION LINE
& RESIDUAL
DISTRIBUTION

FIG. 3.17

the scatter of points around the regression line is very close. However it is the distribution of the residual values that is of interest. The distribution of residuals around the unique equation for section 13,250E appears distinctly trimodal, with modes of -0.4, 0.0 and +0.4 (standard deviations in each case are in the order of 0.2). In fig. 3.17 we see the data for the East Pit, which has a tendency towards a gaussian distribution, or as for section 20,750 towards slight bi-modality with modes at -0.4 and zero (standard deviations here being somewhat higher at around 0.3).

The situation, then, is one where the manganese oxides behave in one of three ways as they move through the processing plant. Firstly we have the situation where the percentage manganese oxide is lower in the concentrate than it is in the crude, secondly, where there is virtually no change in grade and thirdly where there is a tendency for the grade to increase. This may be an indication that there are three different phases of manganese minerals present and will be further examined in the chapter which deals with the mineralogy of the ores.

(e) The relationship between crude iron and weight recovery

The problems of expressing a relationship between crude iron and weight recovery were somewhat different than those discussed above for the manganese data.

Typical scatter plots of the two sets of data have been shown in fig.2.11 and although in all cases examined, a similar relationship could be observed, no constant regression line can be fitted to the data with any confidence. The problem was approached in a different way.

If all the iron present in the ore were hematite, and if all the hematite could be recovered, then we could express the relationship between crude iron and weight recovery quite simply:

$$\frac{\text{Crude Iron}}{\text{Weight recovery}} = \frac{\text{Fe}}{\text{Fe}_2\text{O}_3} = \frac{1}{1.43}$$

Since however a certain amount of manganese oxide and quartz are always caught up in the concentrate, then 'ideal separation' can be expressed:

$$\% \text{ Weight recovery} = (\text{Crude Iron} \times 1.43) + A$$

where A = weight% of manganese oxide + quartz in the concentrate.

In each case the value of A is known and therefore an ideal separation for each assay point can be calculated. The way in which the actual weight recovery differs from the calculated one will be a measure of how far from ideal conditions for recovery were, and will be expressed most clearly in the residuals generated from this ideal separation line. Fig.3.18 illustrates the residual distribution in the West Pit, as typified

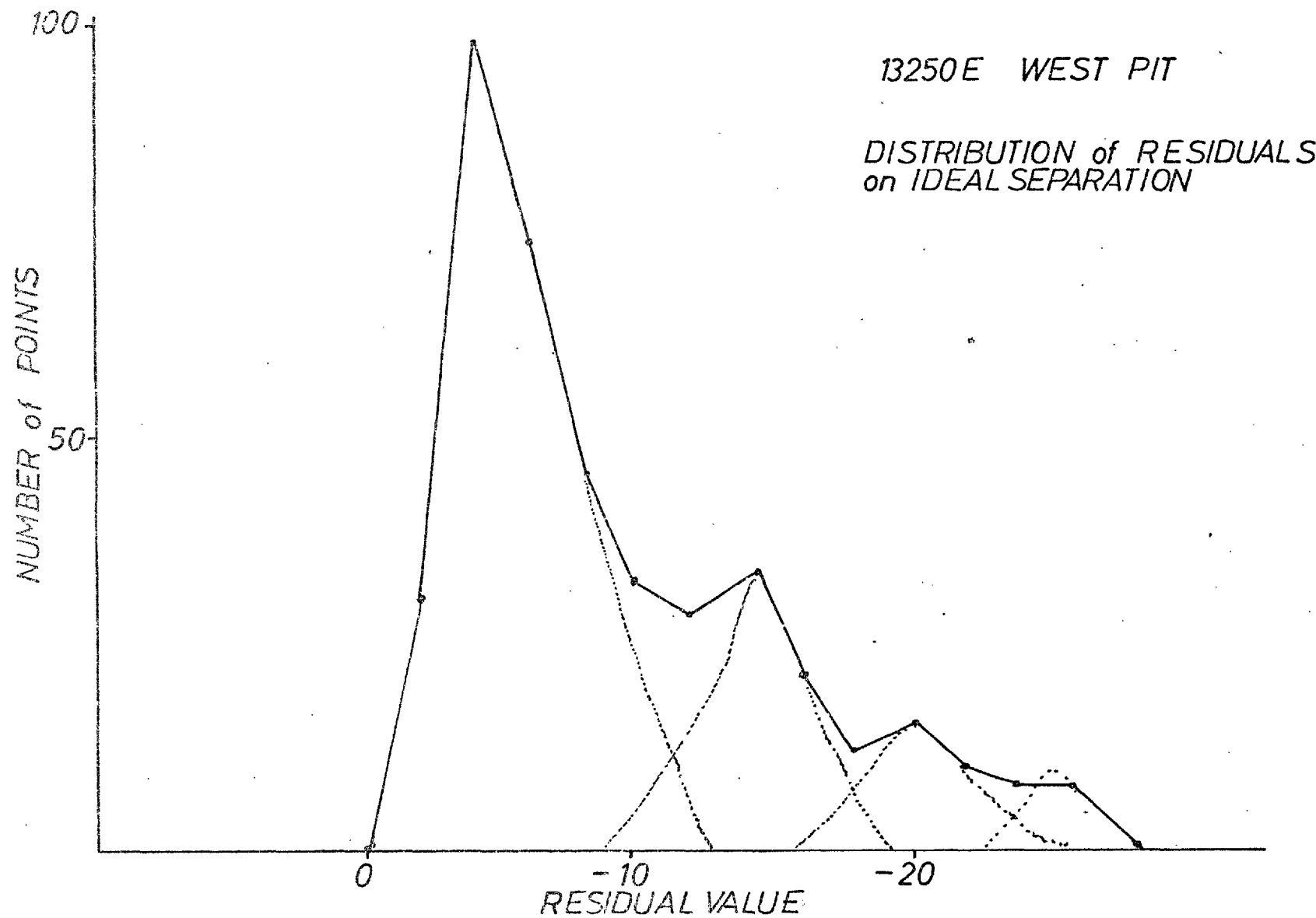


FIG. 3.18

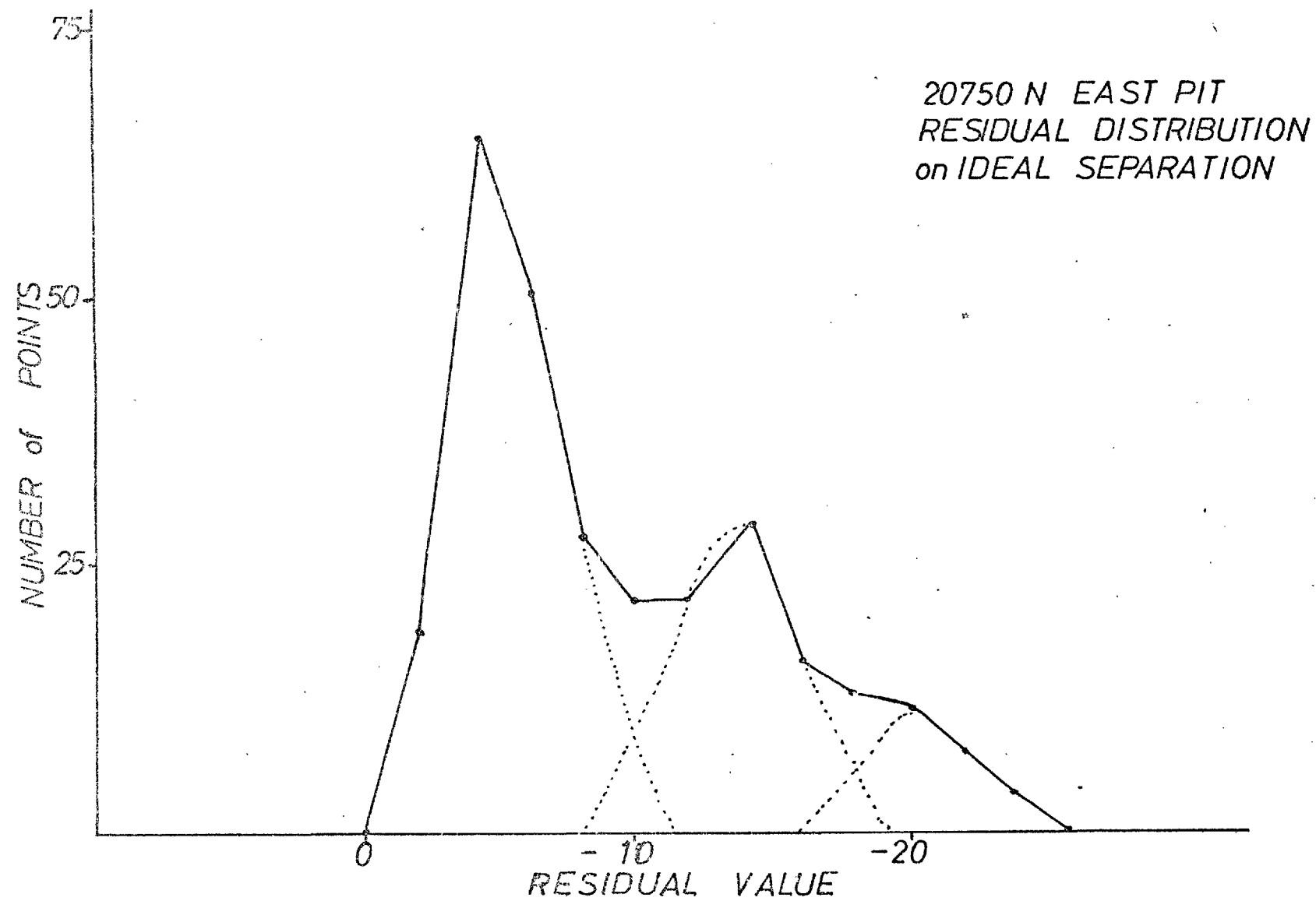


FIG. 3.19

by section 13250E and 3.19 the distribution in the East Pit on section 20,750N.

In the West Pit there is a trimodal distribution with modes at -4%, -14% and -20% with a tendency for a fourth mode at -25% in some sections

In the East Pit the groups are the same with the exception that the -25% group is totally absent.

Variograms

The basic tool for studying the natural dispersion of a regionalised variable is the variogram. This is a mathematical function which expresses the variance of the increments of the regional variable. At a point x the value of the variable will $f(x)$ and at $x+h$, $f(x+h)$. The statistic is given by:

$$\gamma(h) = \frac{1}{2} \frac{\sum (f(x+h) - f(x))^2}{L - h}$$

where $f(x)$ and $f(x+h)$ are the assay values of samples separated by a constant interval h and L is the length of the linear series of assays. In graphing h is plotted along the x -axis and $\gamma(h)$ along the y -axis.

Matheron has stated that the variogram will show the following properties:

- (1) It is an ideal expression of correlation between samples

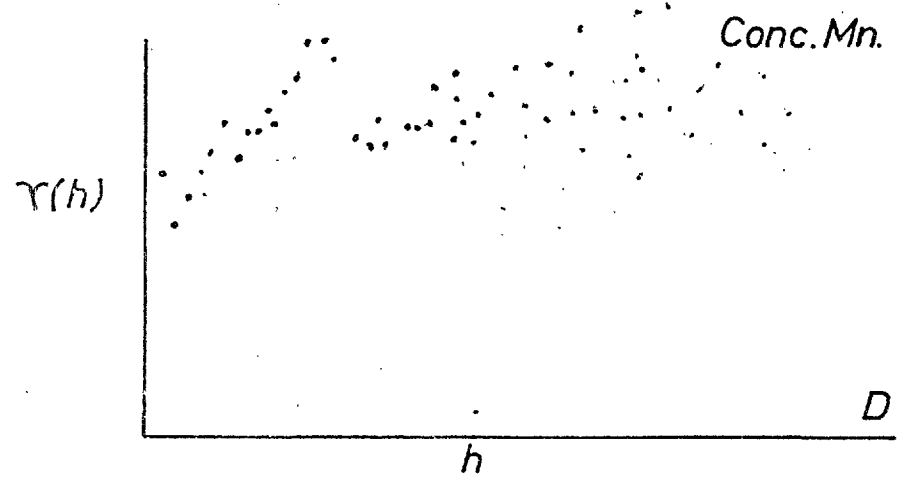
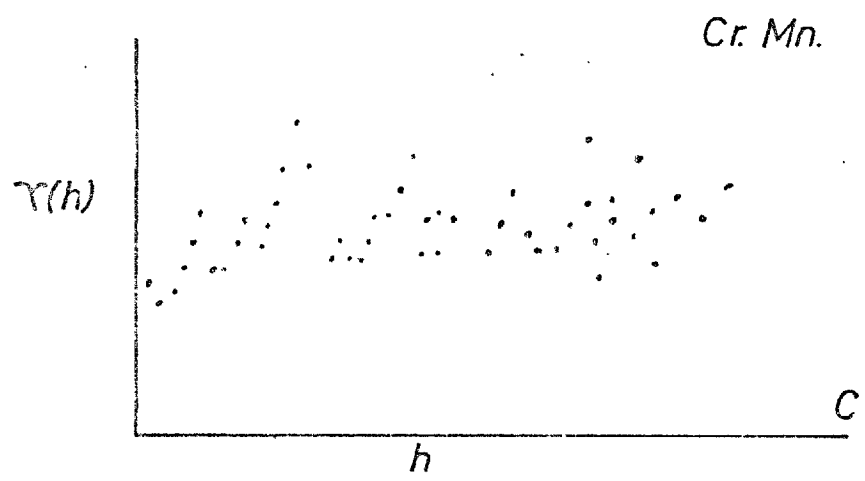
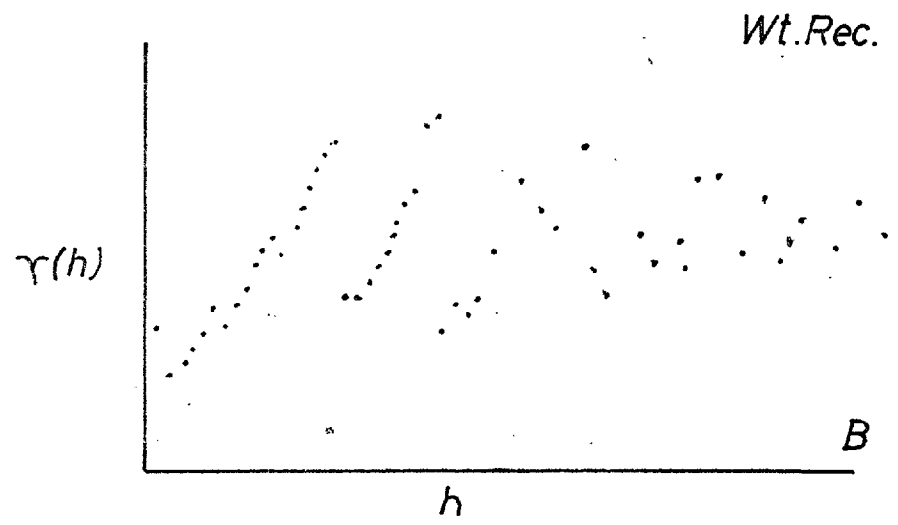
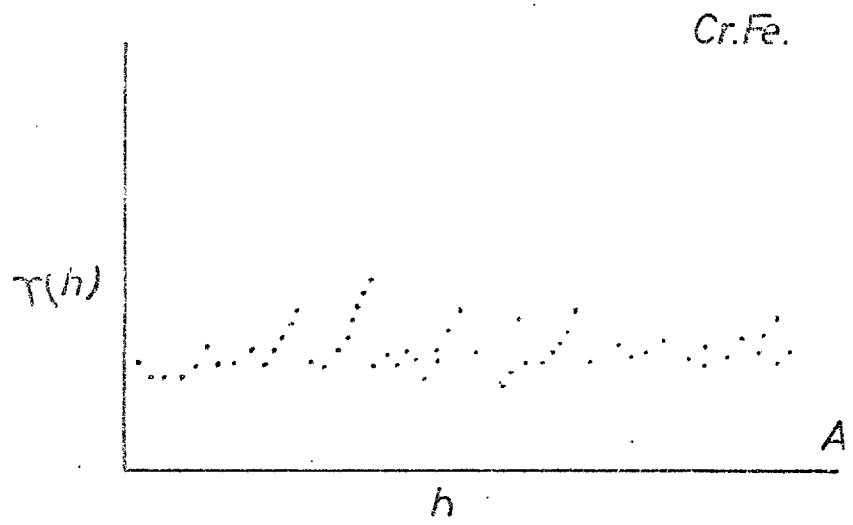
- (2) Anisotropies in the mineralised field will show up in different behaviour of the variogram in different directions.
- (3) Transitional phenomena in continuity over a certain range then randomness over another range, will be shown in the change of slope of the variogram.
- (4) Randomness when the variogram function is independent of the lag.

The variogram is normally used in a linear sense for example along the length of a borehole or drive. In the case in question however a method of examining the ore body in three dimensions with the variogram was used and is discussed in further detail in the appendix.

Typical variograms for the West Pit area are presented in fig.3.20. The variograms were calculated on a 10% sample taken from what is essentially the West Pit working area. Three of the four variograms are weakly transitive although the peculiar stepping effect observed in each case, in especially the concentrate manganese and weight recovery diagrams, is not properly understood. The 'step' interval is between 30 and 50 feet which corresponds approximately to the unit interval in the stratigraphy that is presented below.

Trend surface analysis

An attempt was made to use polynomial regression techniques to model the style of foling present in the sections.



Increment of $h = 5$ feet

3-D VARIOGRAMS
WEST PIT

FIG. 3.20

KWIKR8 FORTRAN IV programme for multiple regression and geological trend analysis was the programme used, but although the contour interval and expected means were taken to their expected limits, the resulting maps up to the 7th order polynomial were not interpretable.

C. CONCLUSIONS

The methods that are used at Scully Mine are inadequate for proper grade control and there is a tendency in the ore reserve estimate programme, to exaggerate the quantities of higher grade ore available for mining. Although the geology that had been outlined to the time when mining started was good in itself it proved inadequate and a method that could delineate ore types was sought.

It has been shown in this chapter that it is possible to subdivide the ore body into units statistically and a detailed investigation of the sections has shown that these units are probably lithological ones in that they can be traced in the correct position relative to the quartzite all over the ore body and when used to draw a structure a synclinorium compatible with the regional structural geology emerges. These units have unique characteristics and although it would be unwise to be dogmatic about the statistics of the ore body being a tool precise enough to define an exact location within

the stratigraphy, their application has been used to define suites of rocks that have identifiable characteristics.

The criteria that have been used to divide the ore body into its units are as follows:

1. Crude Iron)
 2. Crude Manganese)
- } three populations with gaussian
(or log gaussian for manganese) distributions
3. The relationship of crude manganese to concentrate manganese and the residual distributions.
 4. The relationship of crude iron to weight recovery and the residual distribution.

The stratigraphy developed for the ore body on the basis of these parameters is given in table 3.7.

So far however it could be argued that this stratigraphy is an imaginary statistical one and until it is backed up by some supporting data it must be regarded with some reservation. On proving the idea however this division into some eleven ore units of the Scully Mine ore body represents a considerable improvement on the arbitrary ore classification that is presently used in ore reserve estimation and grade control. It could form the basis for the reconstruction of the mining sections and the basis for the reconstruction of the mining sections and the basic tool in a re-evaluation of the ore body. The significance is far wider reaching than this however in that it will provide an invaluable exploration tool. The approximate position in the stratigraphy can now be easily forecast. The position of the high and low manganese areas can be forecast with confidence as can the high and low grade

FORMATION	MEMBER	UNIT	STATISTICAL CHARACTERISTICS				APPROX. THICKNESS
			Cr.Fe. %	Wt.Rec. %	Cr.Mn. %	Conc.Mn. %	
	TOP NOT SEEN						
IRON FORMATION	UPPER	1	25 - 31	22 - 30	.15-1.0	as crude	150' - 200'
	MIDDLE	2	32 - 40	36 - 47	1.8-9.0*	-	50' - 70'
		3	39 - 45	52 - 60	.15-1.0	as crude	50' - 70'
		4	32 - 40	36 - 47	.15-1.0	-	40'
		5	39 - 45	52 - 60	1.8-9.0	-	50' - 70'
	MIDDLE QUARTZITE						50'
	LOWER	6	32 - 40	36 - 47	1.8-9.0	+	20'
		7	32 - 40	26 - 37	0.4-2.5	+	50' - 70'
		8	25 - 31	22 - 30	.15-1.0	as crude	50' - 70'
		9	39 - 45	42 - 50	1.8-9.0	as crude	20'
		10	32 - 40	46 - 57	0.4-2.5	+	30'
		11	25 - 31	31 - 40	.15-1.0	as crude	30'
	BASAL SILICATE						70'
WAPUSSAKATOO QUARTZITE							

* 4.0% maximum observed

+ concentrate manganese greater than crude manganese

- concentrate manganese less than crude manganese

TABLE 3.7 STRATIGRAPHY WABUSH OREBODY

areas, and the approximate grade of the extensions of the ore body can be forecast with some degree of accuracy.

CHAPTER IV

GEOSTATISTICS OF INDIVIDUAL ORE UNITS

A. INTRODUCTION

The division of the ore body at Wabush into a large number of small ore units is of considerable economic importance. With the revised stratigraphy one should be able to project grade for long term planning along the strike of a unit, divide up blasts into their types for a better operation and in future drilling projects more easily interpret the information obtained. In addition to this, a much more satisfactory method of ore reserve estimation could be formulated, where ore types rather than arbitrary blocks are used as the main criteria in the evaluation processes.

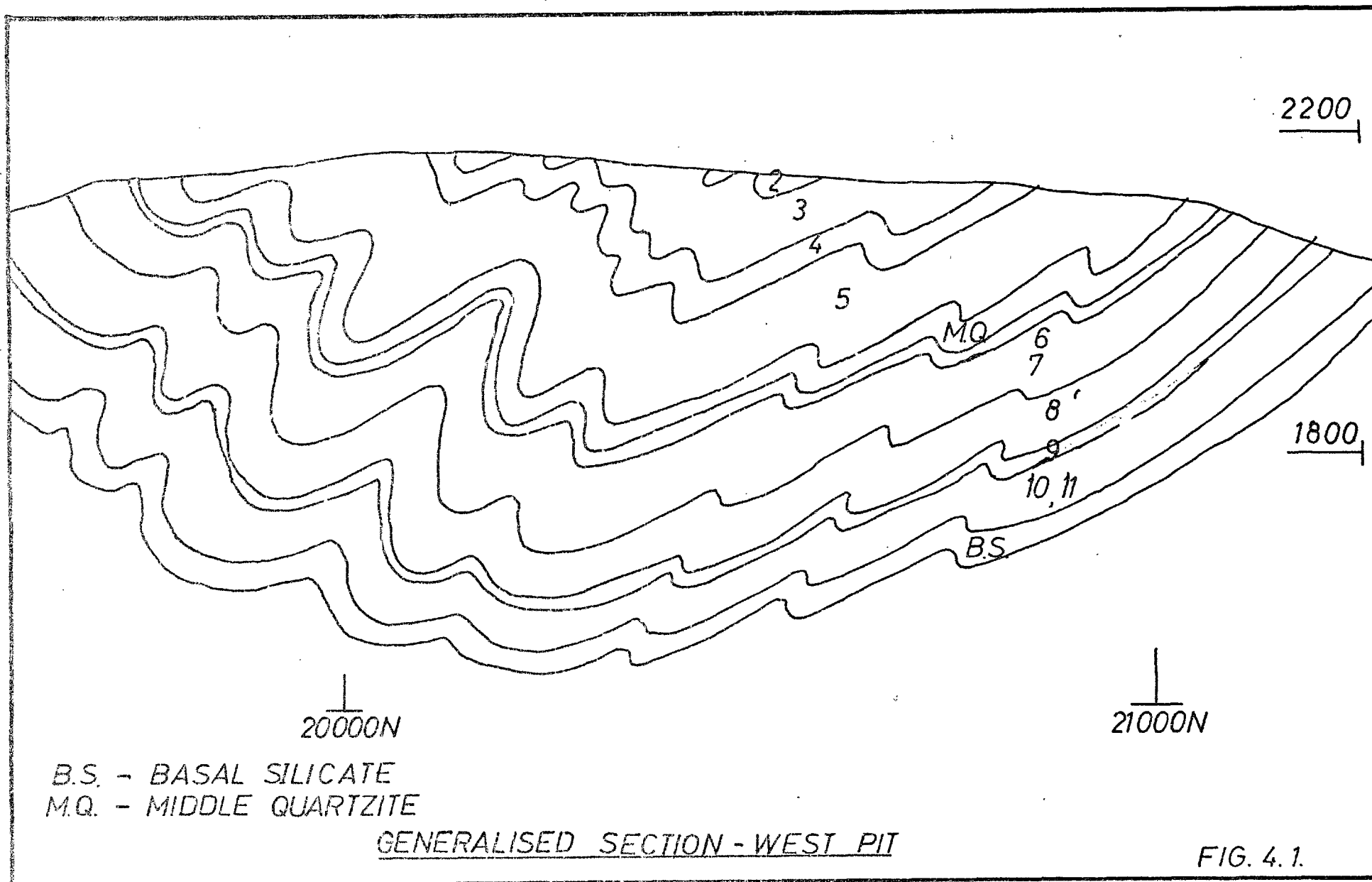
Before any new method can be considered, it is necessary to determine just how good are the divisions. To do this, two adjacent ore types have been examined in some detail to determine whether or not their statistics are unique and if it is possible to project their grade over the whole pit. Selection of the two types was made because they have been well sampled and have been easiest to define, because of their location in the stratigraphy. The units that have been chosen are units 4 and 5 (see table 3.7) immediately above the middle quartzite. Unit 4

is only 30 to 40 feet thick and has a low iron content (32 - 40% Fe) and low manganese content (0.15 - 1.0% Mn).

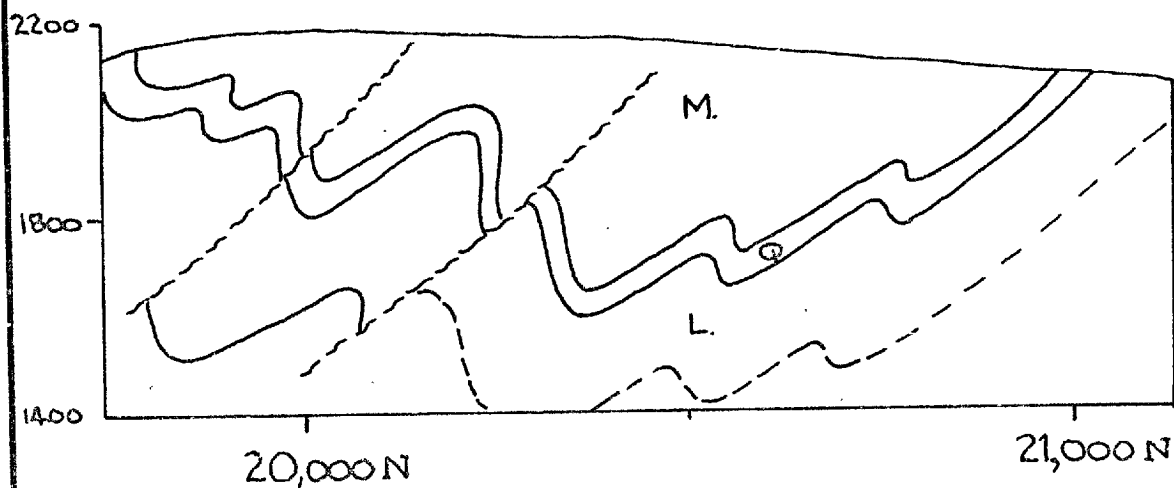
Recovery is poor but with little change in manganese content from crude to concentrate. Unit 5 on the other hand has a high iron content (39 - 45%) and high recovery with manganese in the middle range in both the crude and concentrate. So although the two ore types are stratigraphically adjacent they should provide a contrast in terms of their properties. The area over which the study has been conducted corresponds approximately to the working area of the West Pit, but in fact was determined by the limits to which the sections can be interpreted with the present drilling information, that is from 11,250 E to 14,000 E and 19750 N to 21250 N. However, any results which emerge from this area of study should be equally relevant to the rest of the pit.

B. UNFOLDING THE ORE DEPOSITS

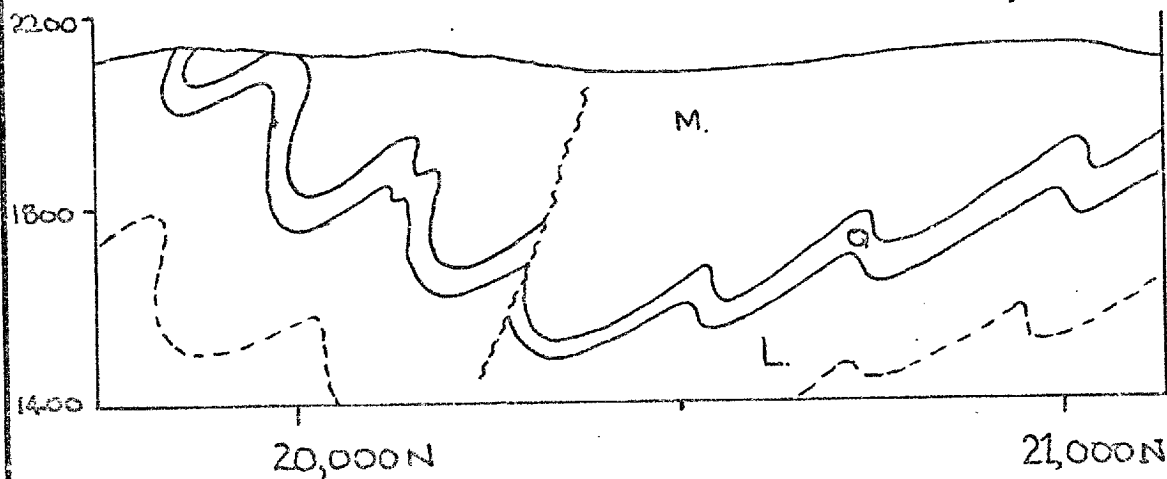
The stratigraphy developed in the last chapter was used to reconstruct the West Pit sections and details of this revision are presented in Chapter VI. A generalised section through the West Pit has been constructed (Fig. 4.1) to illustrate the present discussion. The ore body is folded into a series of close, moderately inclined, gently plunging folds. Two orders of folding are immediately recognisable. The first order folds can be determined by drawing a line through the median surfaces of the present fold system. This is folded and has the form of open upright folds with a wavelength somewhat



13,250 E



12,250 E



- M. Middle member
- Q. Middle quartzite member
- L. Lower member

SIMPLIFIED WEST PIT
SECTIONS

FIG 41a.

greater than 2000' and an amplitude of 400'. The second order folds have a wavelength of about 200' and an amplitude of 100' and are slightly flattened parallel folds (type 1c). It would be considerably easier to examine each ore type if the fold systems were not present; to this end the system has been unfolded in order to re-examine the grade distribution within the ore types studied. It should be noted at this point that although it was necessary to determine the relative positions of the sample points in the unfolded state, it was not necessary to reconstruct the thickness of each unit accurately. The folds are not flattened to any great extent and a precise knowledge of the geometry of the original basin of sedimentation is outside the scope of this work, so it has been assumed that the original unit had a constant thickness.

The mine sections reconstructed represent the result of normal sub-surface interpretations on bore hole data. The results as illustrated in fig. 4.1a are based on a few exposures of folds seen in the working face, and reflect only the overall bulk geometry of the deposit. The absence of any strain markers makes it virtually impossible to better this. It is possible at some later date, the manganese nodules described in Chapter I that occur in unit 3, may be used as strain markers and the details of the folding brought to light.

To reproduce the local fold style Ramsey's 'isogon' method was used. A well documented fold was drawn and dip isogons drawn across it. (see fig.2a). A number of folds were then constructed at different thicknesses, keeping the relationships between dip isogons a constant. These folds were used as a template in the reconstruction of the sections.

The method used in the unfolding exercise is illustrated in fig. 4.2. The nature of the first order folds was determined on each section and then the position of the axial surface on section was drawn in. A plan of the location of the axial traces at 1600 elevation was then constructed and this was used as the base map for the data from the two ore types. The position where a borehole intersects a unit was measured around the fold (distance 'a' in fig. 4.2) and the position of the intersection drawn in plan at that distance from the appropriate axial trace. In some cases a bore hole has intersected a unit at a low angle and more than one assay been taken inside the same unit. Where this occurs the mean position of each assay was noted and plotted separately. Where the borehole intersected the formation at a high angle and more than one assay taken (such that each assay would occupy the same plan position) then the average assay is plotted. The resulting maps form the basis for the trend surface work described below.

LOCATING THE RELATIVE POSITION
OF A SAMPLE AFTER
UNFOLDING

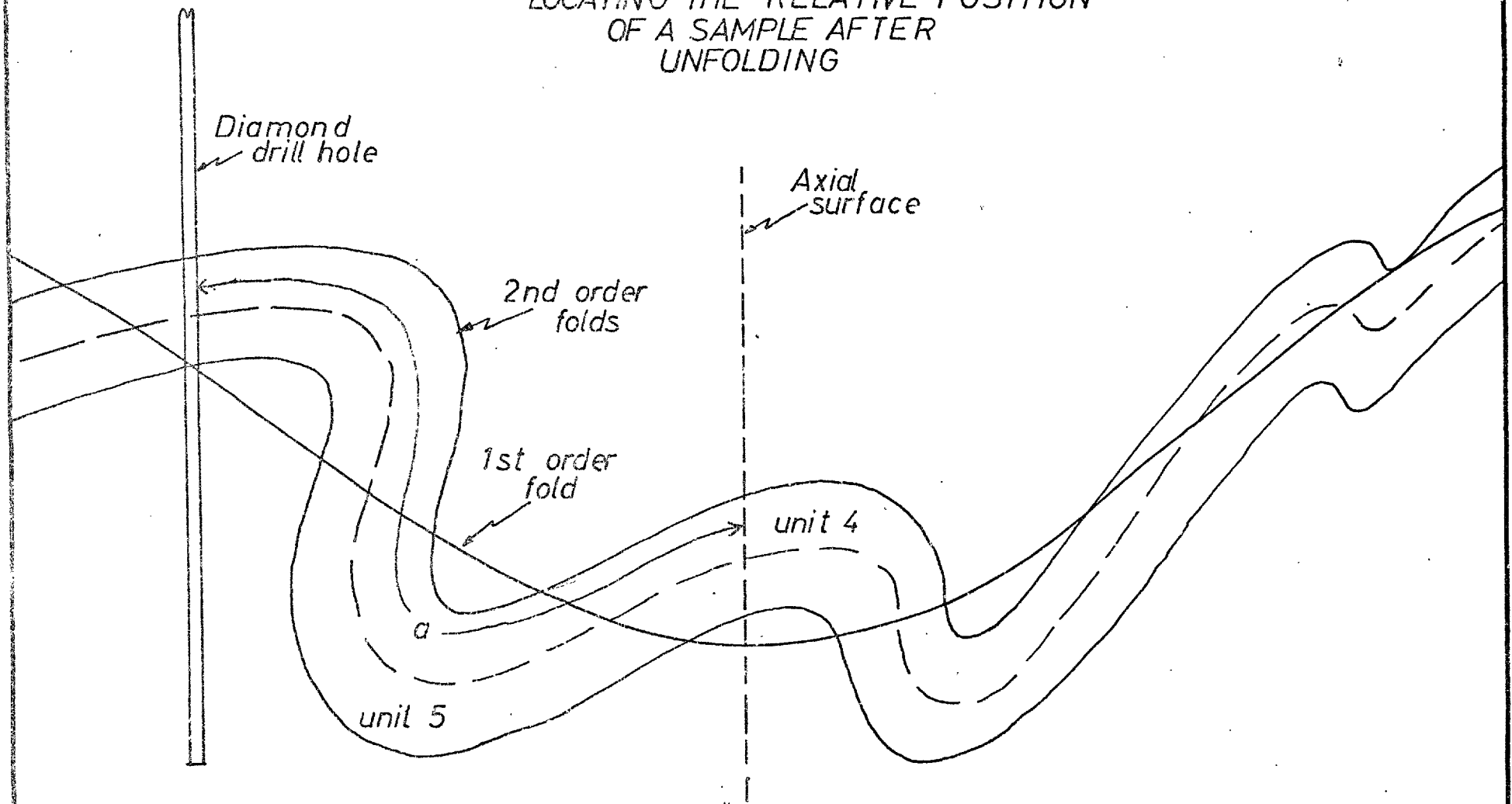
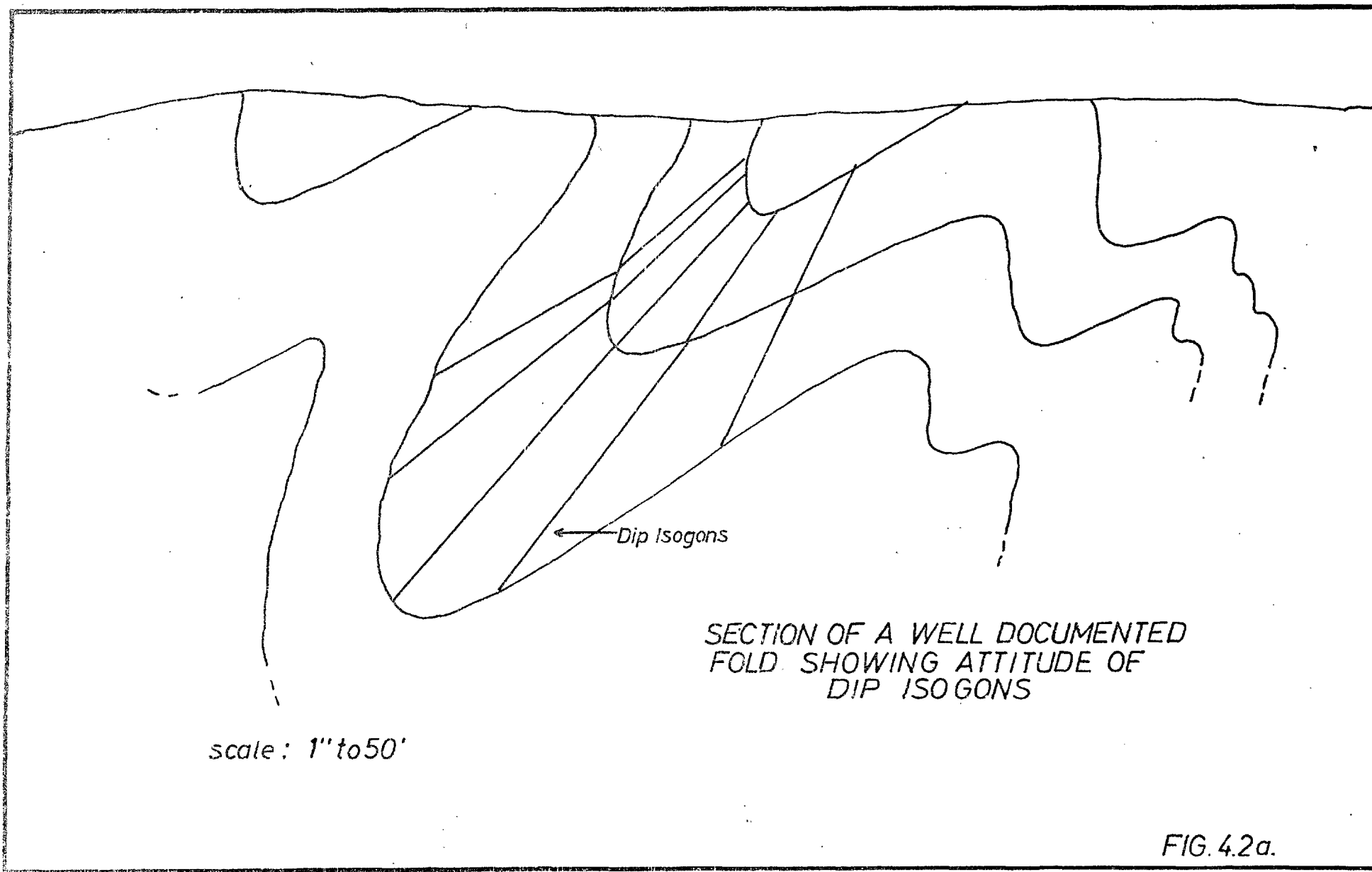


FIG 4. 2.



C. ORE TYPE 4

(1) General Statistics

The estimates of the means and standard deviations for the four variables considered are presented below:

	Mean	Standard Deviation	
Cr.Fe	35.7	1.6	
Cr. Mn	0.7	0.2 *	
Wt. Rec	41.1	2.9	* $\log_{10}$
Conc. Mn	0.6	0.2 *	(assay x 100)

The variation of the grade may be expressed as follows:

$P(32.6 \leq \bar{x} \leq 38.8) = 0.95$	 Cr.Fe;
$P(35.4 \leq \bar{x} \leq 46.8) = 0.95$	 Wt.Rec;
$P(0.2 \leq \bar{x} \leq 1.3) = 0.95$	 Cr.Mn;
$P(0.16 \leq \bar{x} \leq 1.0) = 0.95$	 Conc.Mn;

$\bar{x}$ = mean grade

These follow very closely the predicted expected ranges set up in the stratigraphy in Chapter III.

Comparison of the distributions (Logarithmic in the case of the manganese assays) using the chi-square test to a gaussianly distributed model has given the following results:

	χ^2	
Cr.Fe	7.2)	
Cr.Mn	4.6)	
Wt.Rec	8.4)	with 14 degrees of freedom
Conc.Mn	3.5)	

The calculated value for cumulative χ^2 with 14 degrees of freedom is greater than any of those above and so the frequency distributions for these variables are best described as gaussian (after logarithmic transformation of crude and concentrate manganese assays).

(2) Trend Maps

The data for crude iron, weight recovery and crude and concentrate magnetics were reduced to the horizontal, as described in section B above, and run on a multiple regression trend surface programme. Original data and trend maps for the ore type are given in figs. 4.3 - 4.5. Correlation coefficients and F ratios are given in table 4.1.

	Corr. Coef.	F ratio (3,38)	
Crude iron	0.17	1.06)	1st order $F_{0.95} = 2.8$ $0.50 = 0.8$
Crude Mn.	0.46	3.42)	
Wt. Rec.	0.22	1.20)	
Conc. Mn.	0.45	3.15)	
	Corr. Coef.	F ratio (28,13)	
Crude iron	0.73	0.56)	6th order $F_{0.95} = 2.4$ $0.50 = 1.0$
Crude Mn.	0.65	0.35)	
Wt. Rec.	0.51	0.09)	
Conc. Mn.	0.67	0.38)	

TABLE 4.1.

Correlation coefficients and F ratios for trend surface maps on ore type 4.

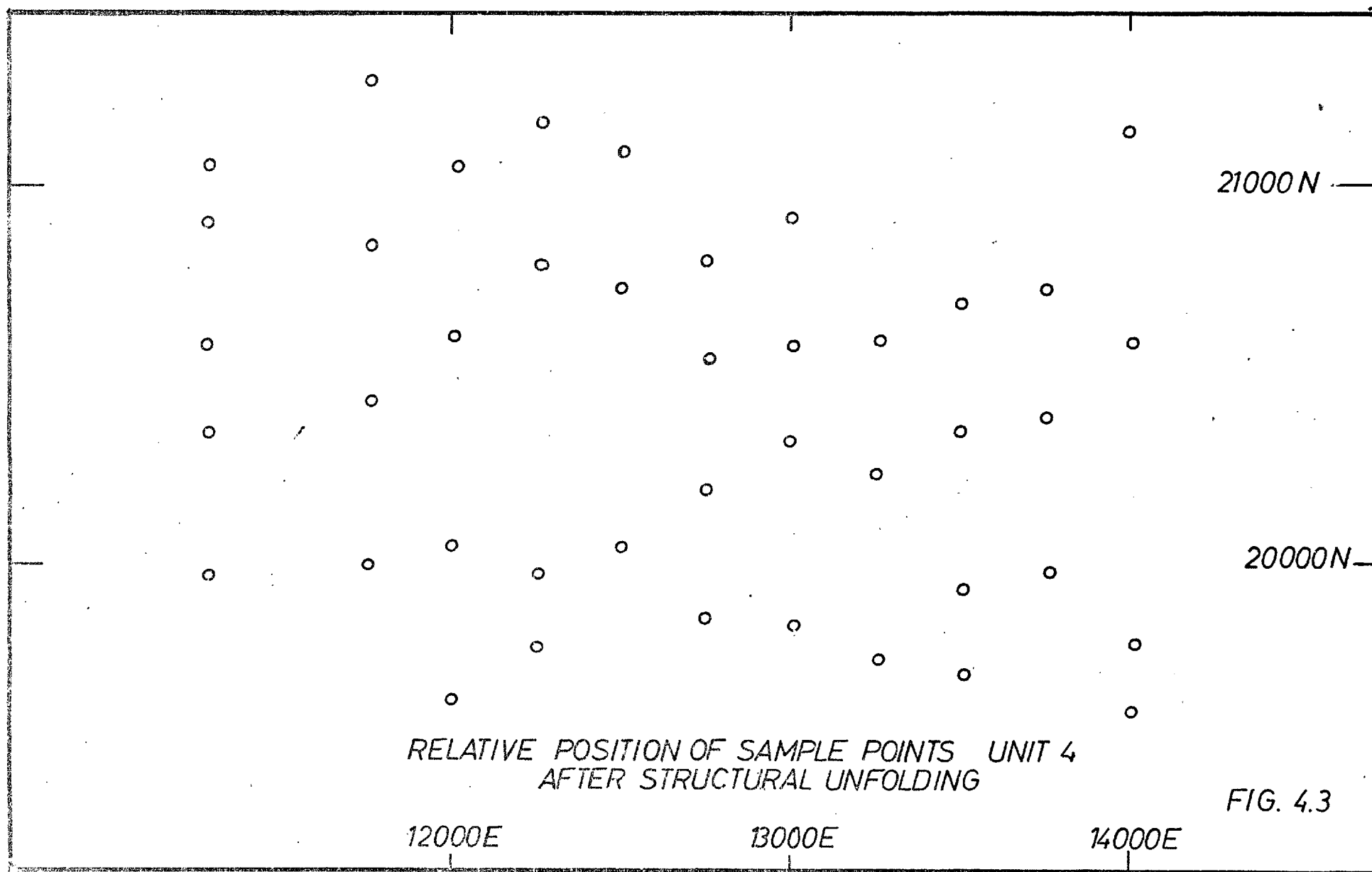
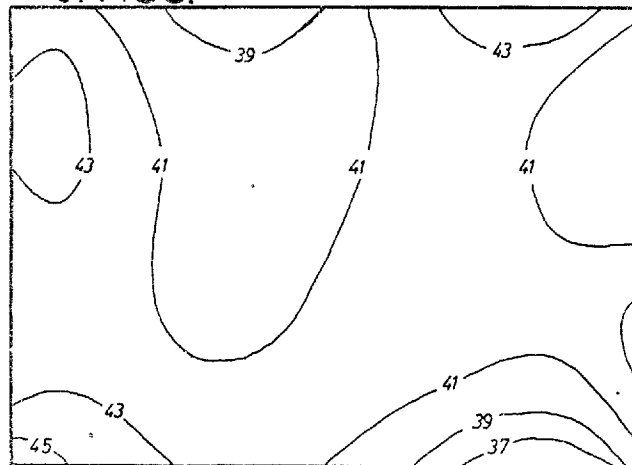
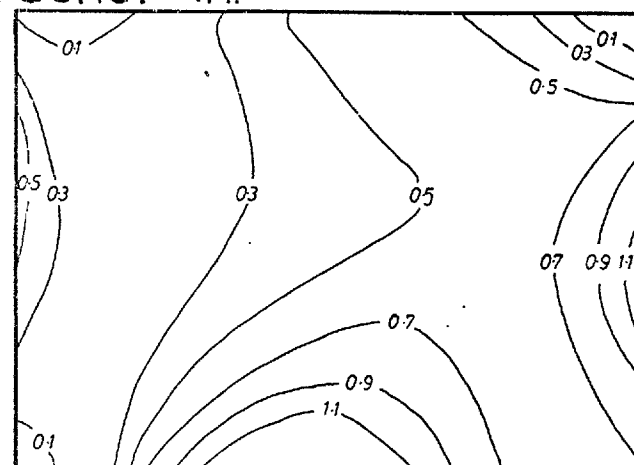


FIG. 4.3

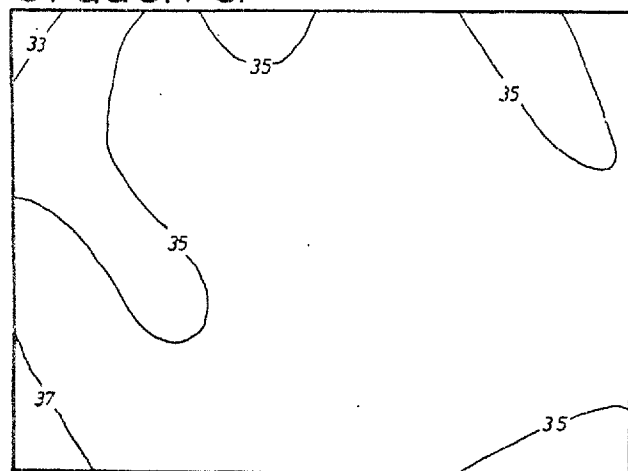
Wt. Rec.



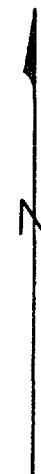
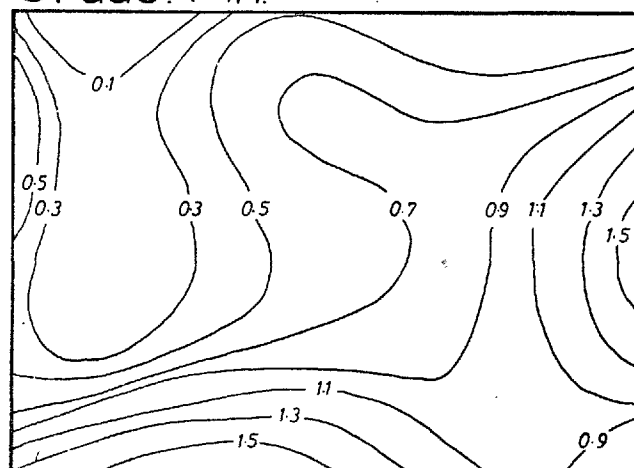
Conc. Mn.



Crude. Fe.



Crude. Mn.



6 TH ORDER TREND SURFACES
VALUES IN PERCENT

UNIT 4

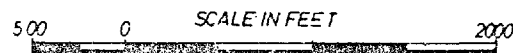
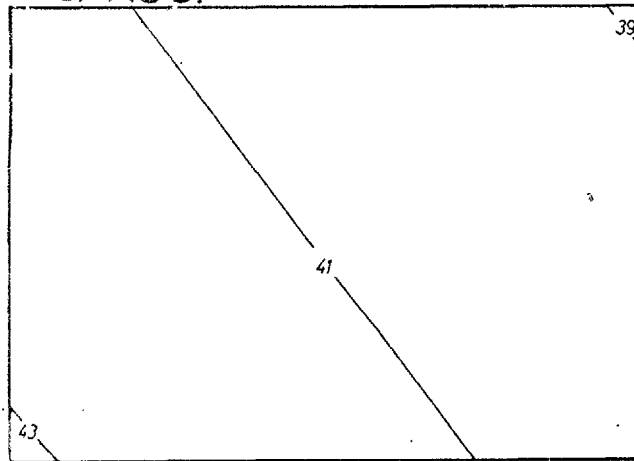
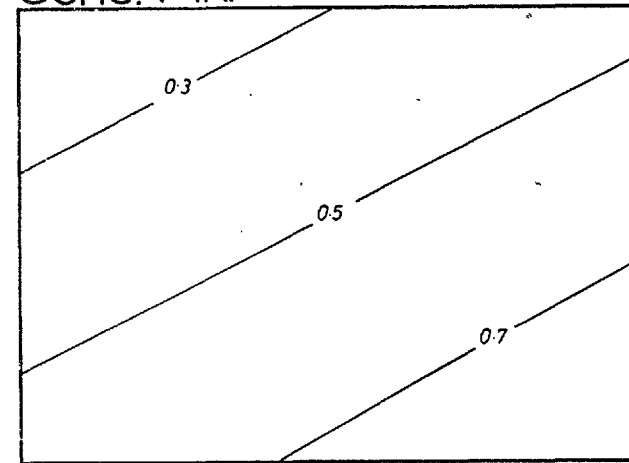


FIG. 4.4

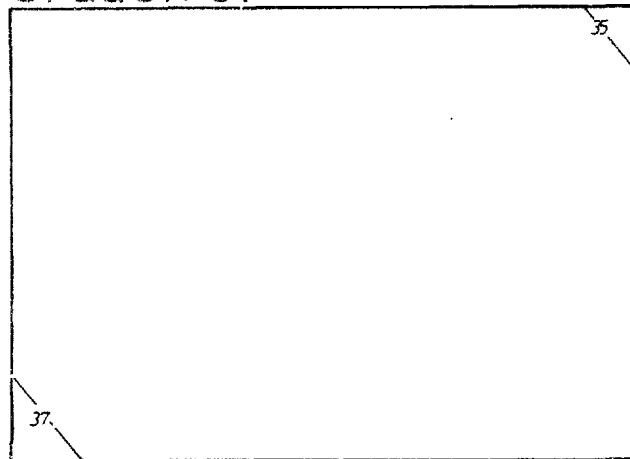
Wt. Rec.



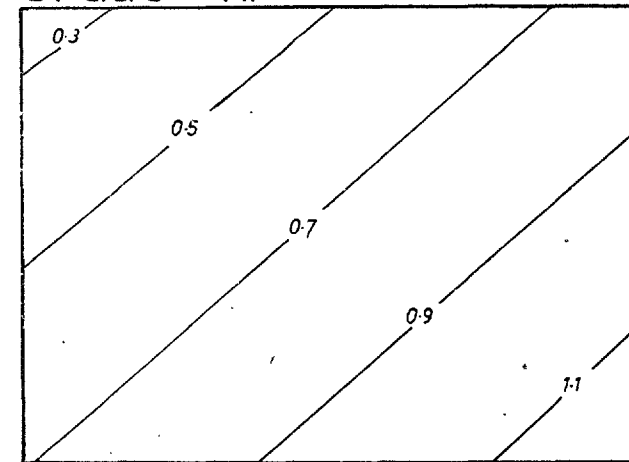
Conc. Mn.



Crude Fe.



Crude Mn.



1ST ORDER TREND SURFACES.
VALUES IN PERCENT

UNIT 4

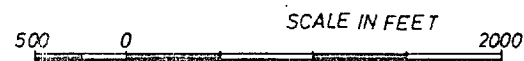


FIG. 4.5

The sixth order polynomial trend surfaces give reasonably high correlation coefficients but the F ratio which tests the hypothesis that the variance of the fitted surface is not significantly different from that of the observed surface is less than the tabulated value of F at the 5% significance level. It is therefore reasonable to conclude that the fitted surfaces are significantly different from the observed ones. The sixth order maps require that the calculation of 30 coefficients is made and with the number of data points available we are approaching the limit of being able to do so. All maps from 1st to 7th order have been made, however only the 1st order maps are of interest and the sixth order are presented along with these.

More interesting however is the goodness of fit obtained with 1st order polynomials on the manganese assays. The crude and concentrate manganese show similar first order trend maps with the manganese grade increasing in a south easterly direction whilst weight recovery map shows grade increasing in a north easterly direction. The crude iron map has almost no trend, but examination of the coefficients of the polynomial indicate a slight increase in grade in an easterly direction. The latter are only significant at the 50 per cent level and have a low correlation coefficient. The significance of the trend maps will be discussed later in the chapter.

D. ORE TYPE 5

(1) General Statistics

The estimate of the means and standard deviations for this ore type are as follows:

	$\bar{x}$	s	*log assay
Crude iron	41.4	2.4	
Weight Rec.	54.7	4.9	
Crude Mn.	0.49 *	0.17 *	
Conc. Mn.	0.46 *	0.15 *	

The estimated grade follow the probability expressions

$$P(36.7 \leq \bar{x} \leq 46.1) = 0.95 \quad \text{..... Crude iron}$$

$$P(45.2 \leq \bar{x} \leq 64.2) = 0.95 \quad \text{..... Weight recovery}$$

$$P(1.5 \leq \bar{x} \leq 7.7) = 0.95 \quad \text{..... Crude manganese}$$

$$P(1.5 \leq \bar{x} \leq 5.7) = 0.95 \quad \text{..... Conc. manganese}$$

As for ore type 4 these values follow very closely the predicted range laid down in Chapter III. The distribution has been tested against a gaussian distribution using the χ^2 test for normality (using the log transformations in the case of the manganese assays) and have given the following results:

	observed χ^2 (15 d.f.)
Crude iron	2.13
Weight recovery	4.14
Crude Mn.	6.91
Conc. Mn.	3.32

The tabulated value for χ^2 with 15(d.f.) at the 5%

critical level is 25.00, is greater than the observed value and therefore the hypothesis that the distributions are gaussianly distributed is upheld.

(2) Trend Maps

The plots of original data and trend maps for the unit are presented in figs. 4.6 to 4.8 respectively.

Correlation coefficients and F ratios are presented in table 4.2.

	Correlation Coef.	F. ratio (3,43)	
Crude Fe	0.17	4.07)	1st order
Crude Mn	0.40	2.7)	
Weight Rec.	0.24	1.29)	
Conc. Mn	0.49	4.7)	
Crude Fe	0.44	0.16)	6th order
Crude Mn	0.84	1.5)	
Weight Rec.	0.59	0.21)	
Conc. Mn	0.81	1.3)	

TABLE 4.2

Both the trend maps and the goodness of fit of the trend maps are very similar to those described for type 4. The first order maps show manganese grade increasing in a S.E. direction ranging from 2.0% in the N.W. corner to 4.0% in the S.E. corner. The weight recovery and crude

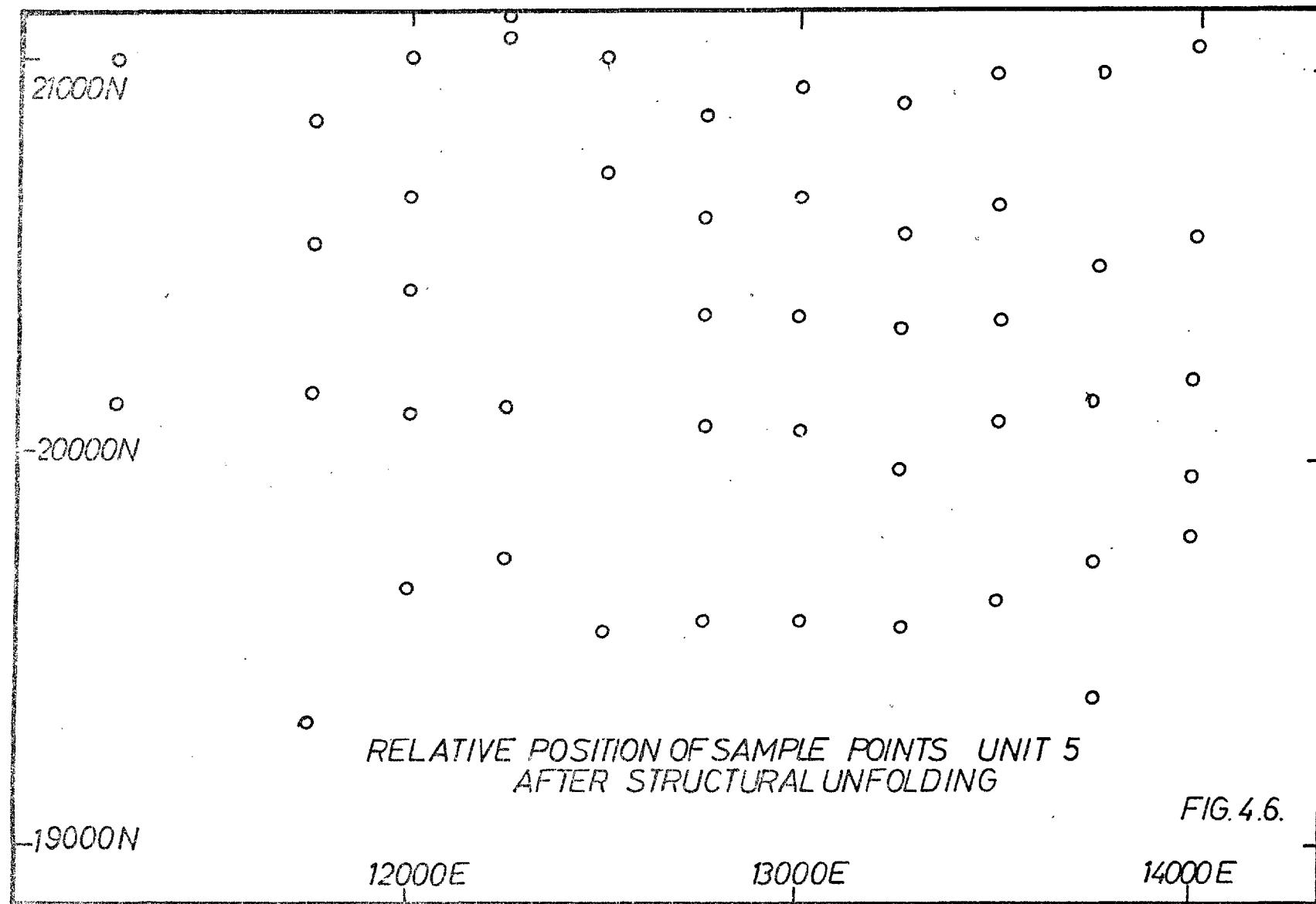
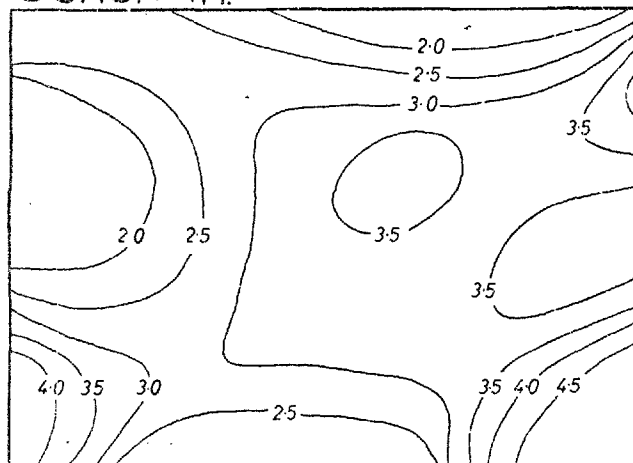
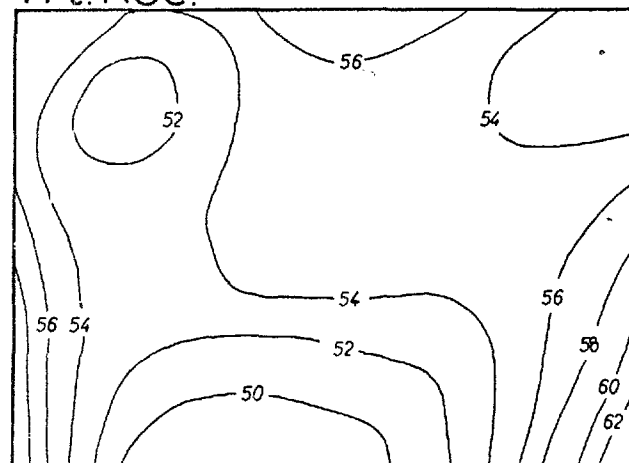


FIG. 4.6.

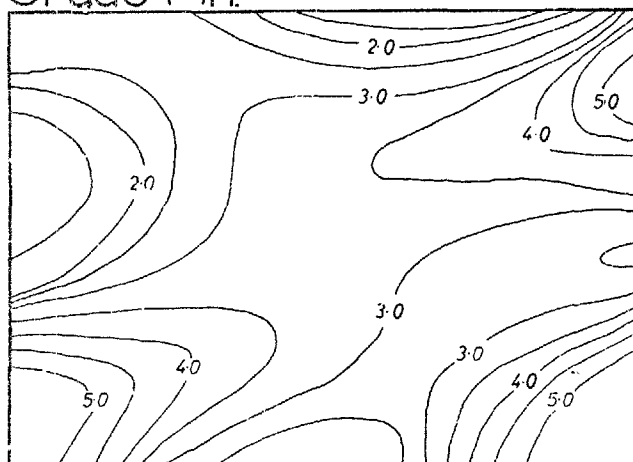
Conc. Mn.



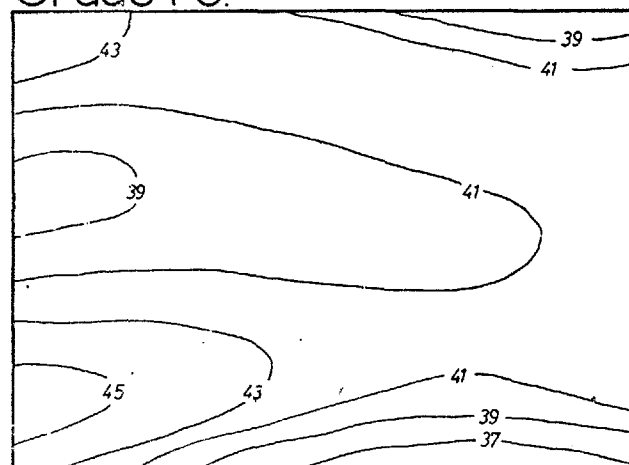
Wt. Rec.



Crude Mn.



Crude Fe.



6TH ORDER TREND SURFACES UNIT 5
VALUES IN PERCENT

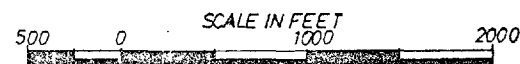
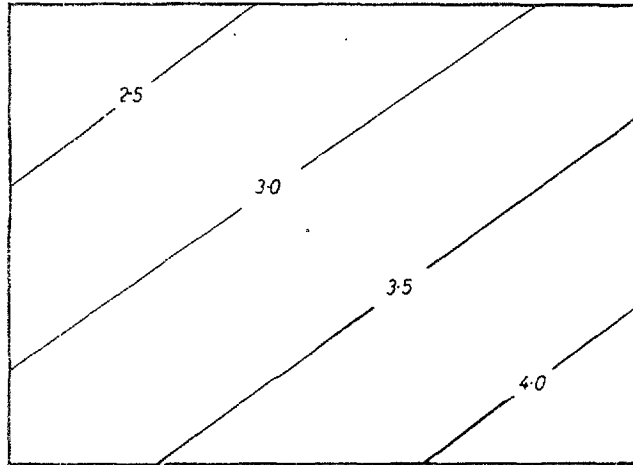
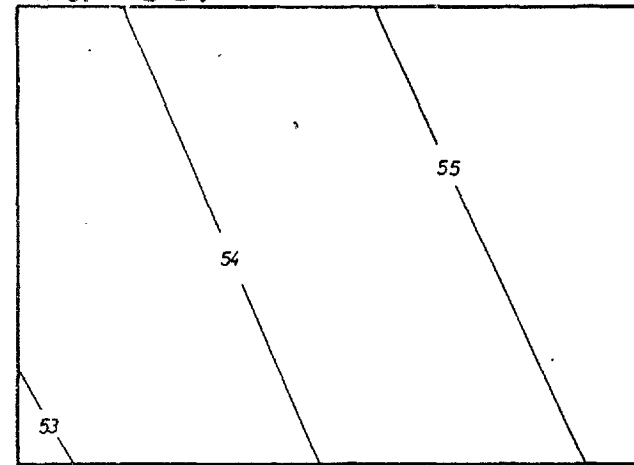


FIG. 4.7

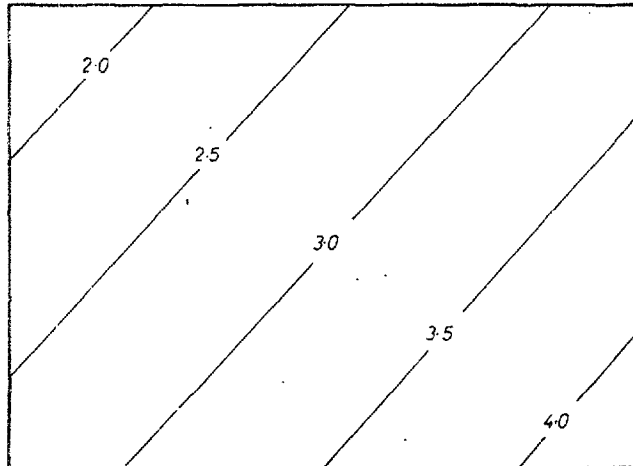
Conc. Mn.



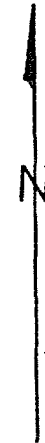
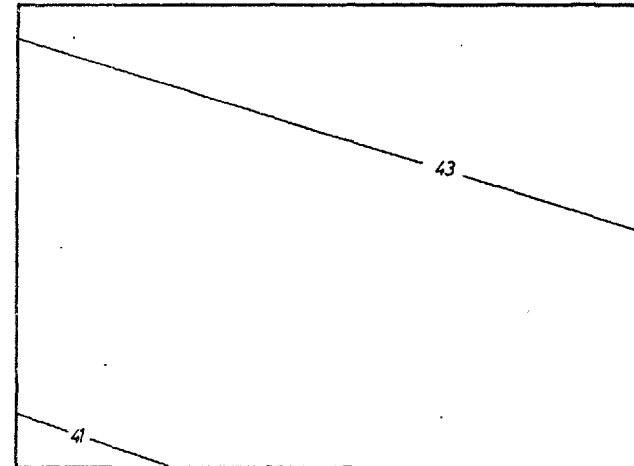
Wt. Rec.



Crude Mn.



Crude Fe.



1ST ORDER TREND SURFACES
VALUES IN PERCENT

UNIT 5

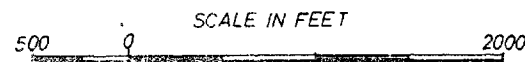


FIG. 4.8

iron maps both show a flatter trend with grade generally increasing in a north easterly direction. Again crude iron and weight recovery 1st order maps are only significant at the 50% level and have a low correlation coefficient. The sixth order maps are not significant.

E. RESIDUALS ON TREND SURFACE MAPS

(1) 1st Order

	Type 4		Type 5	
	$\bar{x}$	s	$\bar{x}$	s
Crude Fe	0.012	0.40	0.01	0.46
Crude Mn	-0.004	0.12	0.004	0.17
Weight Rec.	0.002	0.64	0.0012	0.35
Conc. Mn	-0.007	0.13	0.0008	0.21

TABLE 4.3

Means and standard deviations from 1st order
polynomial surfaces.

Table 4.3 lists the means and standard deviations of the residuals from the 1st order polynomial surfaces generated on the 'flattened' ore types. The means in all cases are very near to zero and the observed residuals are all symmetrically distributed around the mean with low standard deviations.

The first order maps for manganese are significant because of their relative 'goodness of fit' and because of the consistency in trend they have shown in the two ore types. The contour lines in the manganese maps are parallel to the Grenville Front and its associated isograd line. (The biotite isograd runs through the area and is parallel to the contour lines on the manganese trend maps). The iron trend is perpendicular to this direction however is only significant at the 50 per cent level.

Certainly the manganese if not the iron seems to have some regional control and its grade will be dependant on this. The actual grade will then be function of the regional trend -

$$g = f(x) + f'(x)$$

where g = observed grade

$f(x)$ = grade due to regional trend

$f'(x)$ = random variation on the trend

If this is true then the residuals from the first order fit should have the following properties:

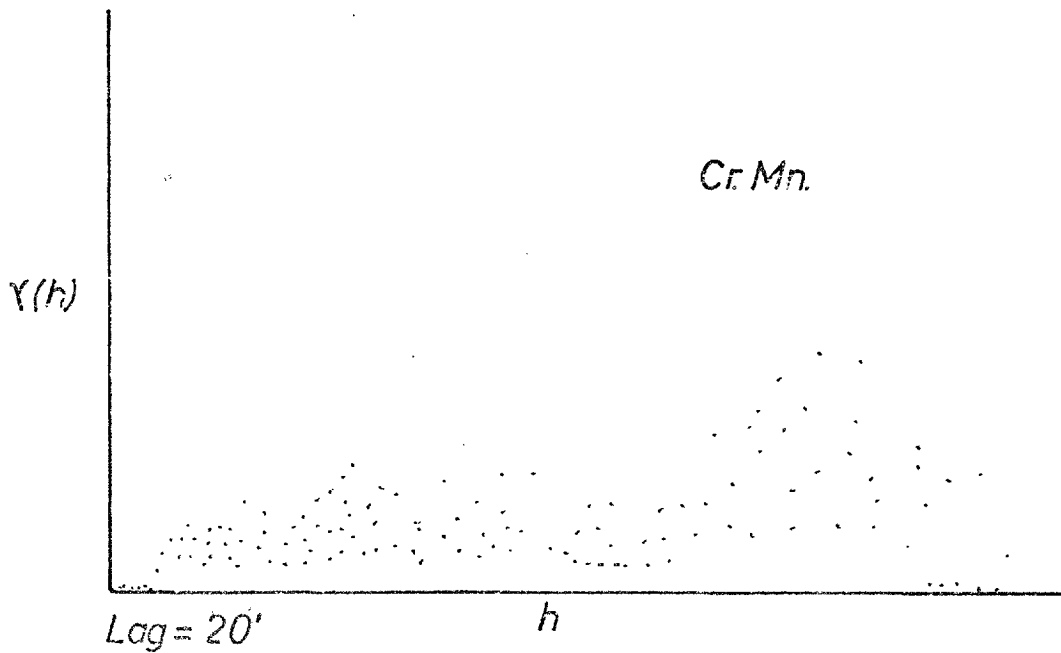
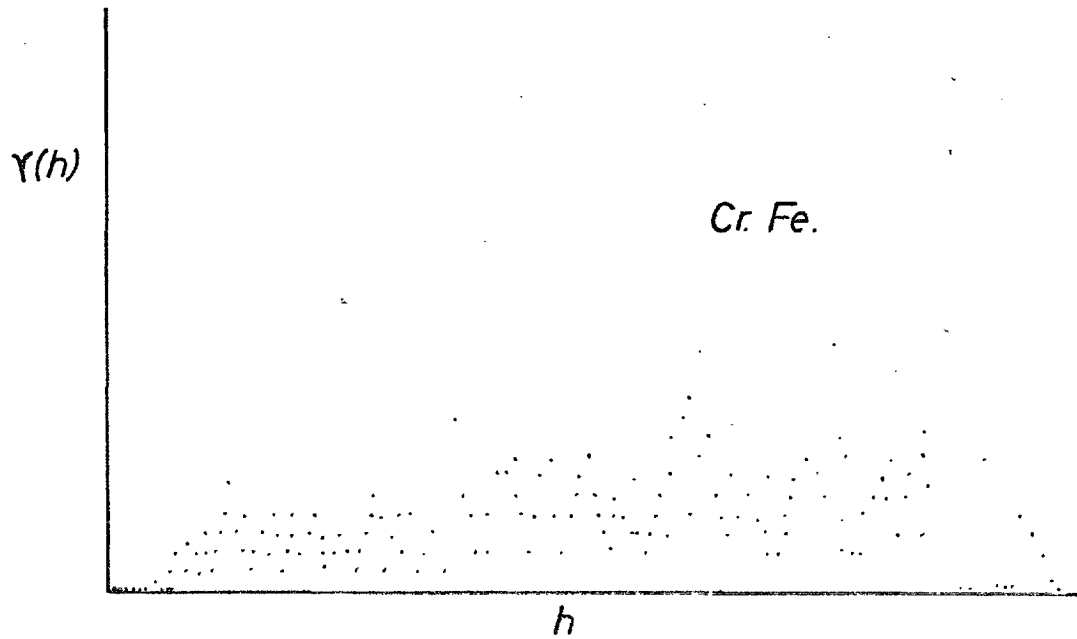
- (a) should be symmetrically distributed about zero
- (b) mean close to zero
- (c) should be regionally random

Frequency distributions of the residuals showed that even if they are not gaussianly distributed they can certainly be described as symmetrically distributed around zero. It could also be noted at this point that the residuals indicate that the first order maps, as far as the manganese assays are concerned, provide a reasonably accurate method of predicting grade.

To test if the residuals are regionally random variograms were run on them in the two dimensions of the surface in which they lie. Two typical examples are shown in fig. 4.9. Only a few point pairs contribute to the value of $\gamma(h)$ at low lag. Therefore the apparent transitive nature of the variograms may be misleading. For all practical purposes they may be treated as showing a considerable degree of randomness.

F. DISCUSSION AND CONCLUSIONS

The approach of examining the grade distribution in the two ore types above the middle quartzite may be regarded as a pilot study on the ore body and has had some interesting results. Of immediate importance is the fact that actual grade for all the variables examined was



TYPICAL VARIOGRAMS ON
RESIDUALS FROM FIRST
ORDER TREND SURFACES

FIG.4.9

excellent support for the arguments expressed in Chapter III when the stratigraphy of the ore body was set up. Yet the relatively large difference, in terms of grade, of the two ore types has confirmed the idea that the ore body is composed of a number of distinctly different units. The manganese distributions are noticeably skew and have been successfully fitted to a log-gaussian model. Inasmuch the grading limit of $1.69 \pm .15$ will be extremely difficult to hold especially on the high side. Here the uncertainty associated with the mean is relatively large. It has also been shown that the grade distribution is predictable and does follow some basic laws that have simple solutions. The trend maps are interesting in their similarity, possibly reflecting an underlying basic geological trend. In this connection it is interesting to note anomalies that occur due to sampling error and mis-interpretation of the sections. Neither the iron oxides nor manganese minerals appear to have any fold or fault controlled trend. The possibility that weight recovery and concentrate manganese can be predicted with an acceptable accuracy from crude iron and crude manganese assays could be investigated further. This study has certainly shown that this possibility is a reality. For example the

correlation between crude iron and weight recovery in unit 5 was high (0.91) with the regression equation $\text{Weight recovery \%} = 1.43 \text{ Crude iron \%} - 4\%$. The residuals from this equation were gaussianly distributed with a standard deviation of less than 1% weight recovery. It has already been shown that present tabling procedures do not offer an answer with a standard deviation around the mean of less than 2.5% weight recovery. A continuation of the study on the remaining ore types is recommended. It was suggested to Wabush Mines that trend surface work may solve some of their grade problems and at the time it was thought that this work was best done on level plans. However, the mathematical technique for displaying the shapes produced by interference of fold systems and faults on each other has not been produced yet. It seems that this suggestion was ill founded. If we remove the geological control, by restoring the units to their unfolded, unfaulted state, then the grade distribution of the units can be examined with comparative ease. A composite plan of the grade distribution within each unit would be the key to the generation of an ore reserve estimate not using arbitrary blocks but based on the geological control naturally present within the ore body. If the grade within each unit is accurately known as well

as its thickness then ore-reserve estimates become a matter of routine, planning easier and grade control considerably simpler.

The method of using trend surface analysis has not been entirely successful. It does suggest however that some other method, for example, hand grade contouring, perhaps assisted by smoothing techniques, may be of considerable help in forecasting grade. However, it should be emphasised that the regional variation within one ore unit is much less important than the variations between units.

CHAPTER V

MINERALOGY OF THE ORES

A. INTRODUCTION

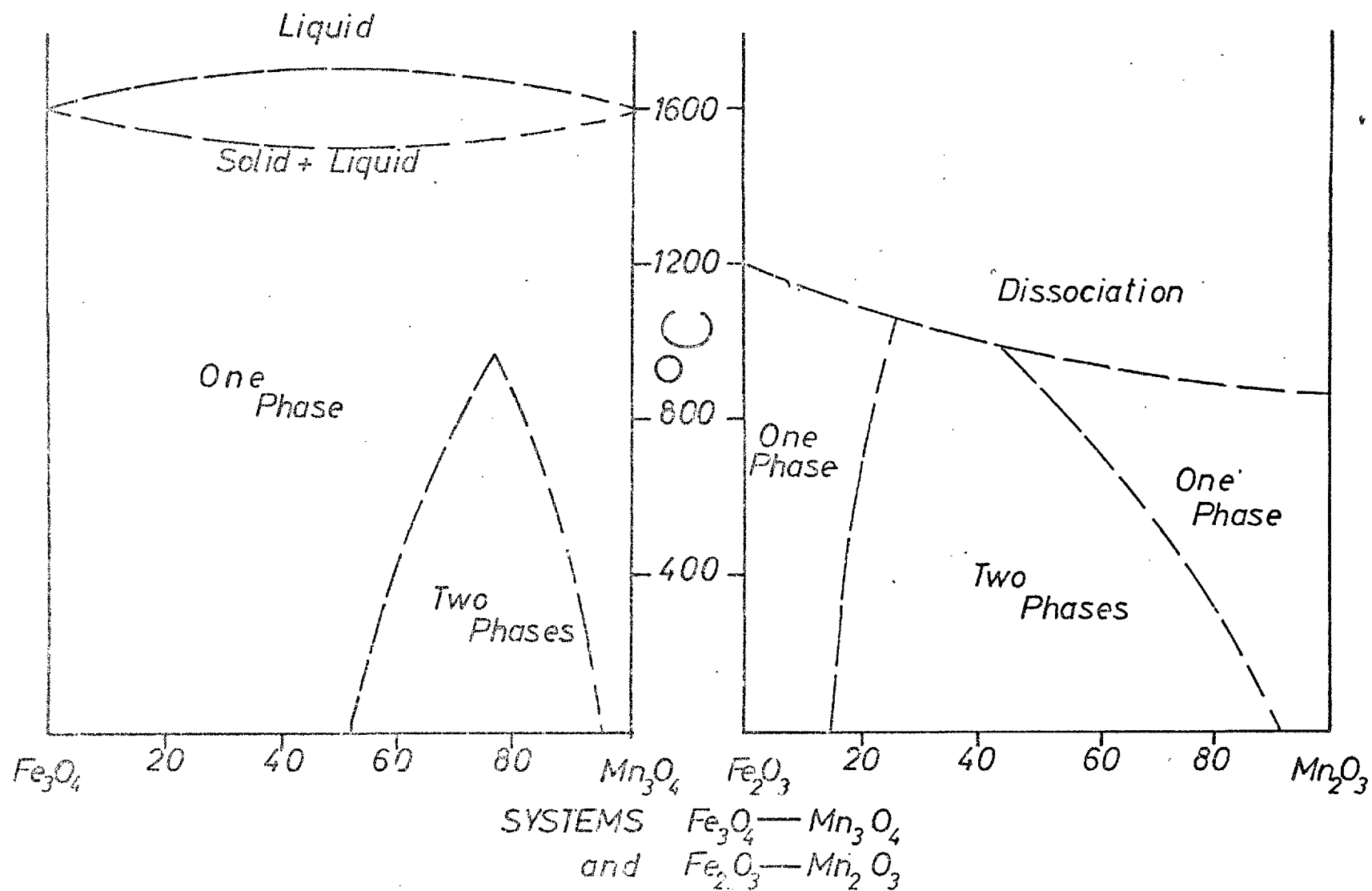
In the previous two chapters it has been shown how a subdivision of the iron formation at Wabush can be made and justified on the basis of the differing chemistry of the ore units. In the section below an attempt has been made to show that these differences not only manifest themselves in the physical and chemical properties of the ores, but also can be traced in their mineralogy. This is hardly surprising since if we have a situation where two ores containing the same amount of manganese yield concentrates of widely differing manganese content, then the segregation may well be explained in terms of a mineralogical difference. A similar argument can be applied to the iron oxides.

The ores are essentially very clean. Several hundred samples have been taken, both in early exploration and up to present time, to search for accessory elements which may be present in the ores. However iron manganese, and silica only are present in any significant amounts, and so the mineralogical search can be confined to the iron and manganese oxides, silicates and quartz. The manganese present in the ores, is present in the normal

concentration range for sandstones. Here the iron to manganese ratio is about 1:25 whereas in sedimentary rocks it varies from about 1:20 to 1:60. The transportation mechanism for manganese is similar to that for iron, except that manganese oxides more slowly and less completely than iron for any set of conditions and that manganese oxides and hydroxides are more soluble than their iron counterparts at any given Eh and pH: manganese will tend to go into solution and stay in solution longer than iron. Apart from these, which are of degree rather than kind, manganese and iron behave very similarly. The fact that in some cases there is virtually no manganese present presumably represents a break in precipitation and sedimentation from a solution from which manganese for some reason was excluded.

Nichol (1958) has studied the systems Fe_2O_3 - Mn_2O_3 and Fe_3O_4 - Mn_3O_4 under different temperature conditions (see fig. 5.1). The solid solution of Mn_2O_3 in Fe_2O_3 is markedly effected by temperature whereas the solubility of Mn_3O_4 in Fe_3O_4 is high under almost all temperature conditions and becomes complete at about 100°C . A magnetite, therefore, containing say 25% Mn, in solution on inversion to hematite would have to exclude most of the manganese as a separated phase.

The ores at Wabush have already been subdivided and the mineralogical study has been conducted to seek mineralogical



After Nichol (1958)

FIG. 5.1

explanations for the differences between ore types.

B. MINERALOGY OF THE DIFFERENT ORE TYPES

The mineralogy of the ores at Wabush will be discussed in reference to the subdivisions made in the previous chapters. The nature of the ores is controlled by the amount of iron and manganese present and also by the way in which the minerals of these elements behave on processing. Eleven different ore types have been described and in the discussion below these have been grouped according to their manganese content and the way in which the iron minerals behave as the ores are processed.

Variations in Manganese Content

(a) Ores with high manganese content (1.8 - 9.0%).

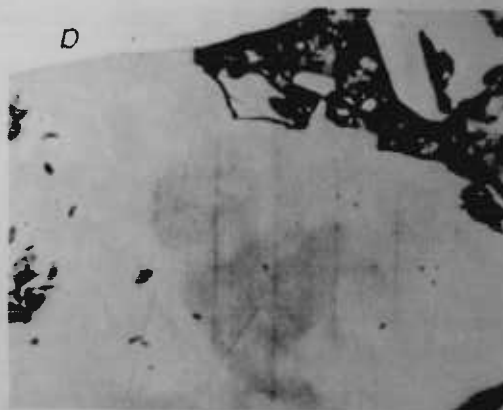
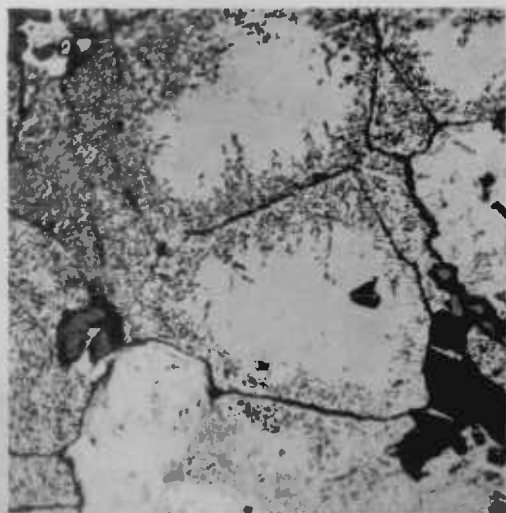
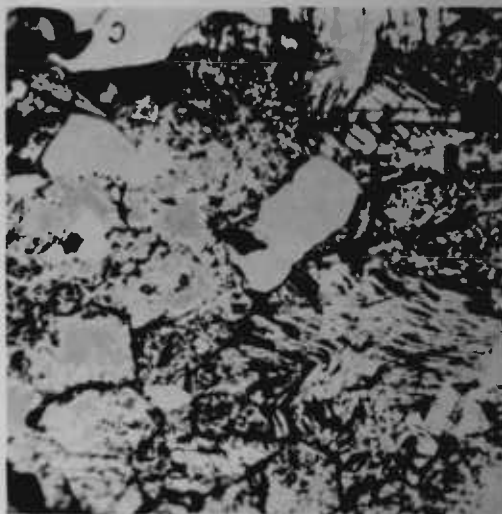
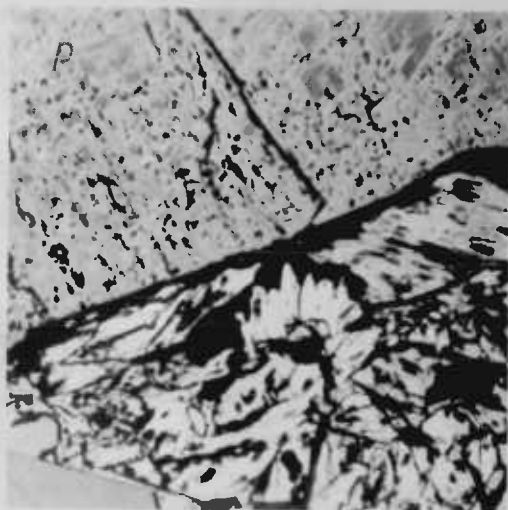
The magnetite in the ores here have all been extensively martitised with the change from magnetite to heamtite originating at the magnetite cleavage planes and finally in some cases engulfing the whole grain.

Plate 5.1 shows a series of micro photographs of the high manganese ores. The magnetites are decidedly pink coloured and the intensity of colour is proportional to the magnetite content. The change from magnetite to heamtite is accomanied by the exclusion of manganese oxides from the grain and this crystalises nearby as pyrolusite sometimes as an earthy variety but more commonly as well crystalised needles. Some

PLATE 5.1.

- (a) Virtually unaltered magnetite grain, very dark in its centre showing a high percentage manganese. Alteration to hematite has just started along cleavage planes.
- (b) Magnetite grains still in the early stages of martitisation but the manganese oxides still not excluded from the lattice.
- (c) Magnetite almost completely altered with only the cores of the original crystals left. Manganese oxides have now been excluded as fine grained lath shaped pyrolusite.
- (d) Martitised magnetite and pyrolusite completely segregated but with the pyrolusite crystallising close to the magnetite.
- (e) Well formed pyrolusite crystals completely segregated from the iron oxides.

NOTE Except where indicated otherwise, photographs on all plates x 110.



further examination of these ores on the electron probe microanalyser is shown in plate 5.2. This shows quite clearly the almost total absence of any iron oxide from the area where pyrolusite has crystallised and the fall in manganese content from magnetite to hematite. In the high manganese ores in general the crude and concentrate manganese assays are very similar. The pyrolusite is finer grained and less dense than the iron oxides, although it is more dense than quartz. The effect of this is to cause the pyrolusite to report equally to concentrate and tailings on processing. In one case (ore types immediately below the quartzite) a slight concentration is noted where the pyrolusite grains are not totally free from the iron oxide crystal (see plate 5.1.(b)).

(b) Ores with manganese in the mid range (0.4 -2.5%)

Plate 5.3. (a) (b) and (c) show the two ore types in the lower member that possess this characteristic. Here all the manganese present appears to be included in magnetite grains. The magnetite grains have been mortitised, but their original manganese content was sufficiently low, that exclusion did not take place on inversion of the magnetite to hematite. The magnetites here contain up to about 3.5% Mn in solid solution and this accounts for all the manganese present in this ore type. Since all the manganese

PLATE 5.2.

- (a) Magnetite in the beginning of alteration to hematite with the exclusion of pyrolusite.
- (b) Iron X-ray back scatter. Note the almost complete absence of iron in the manganese 'vein'.
- (c) Manganese X-ray back scatter. Note the absence of Mn. in the hematite and low Mn. content of magnetite. Mn fills open crystal interfaces.
- (d) Magnetite in the beginning of martitisation.
- (e) Electron image of (d).
- (f) Probe scan for manganese of (d). Note the drop in manganese content as we move from the magnetite to the hematite.

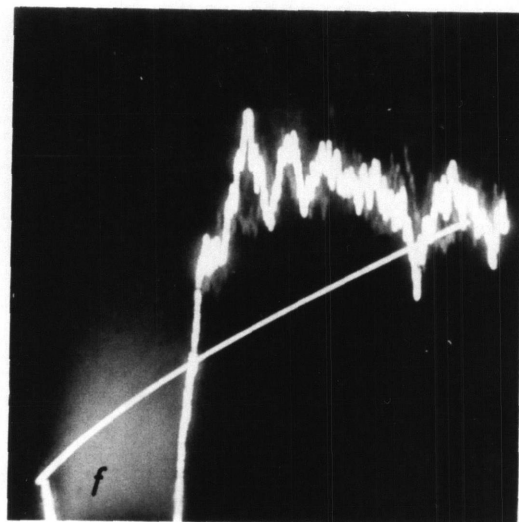
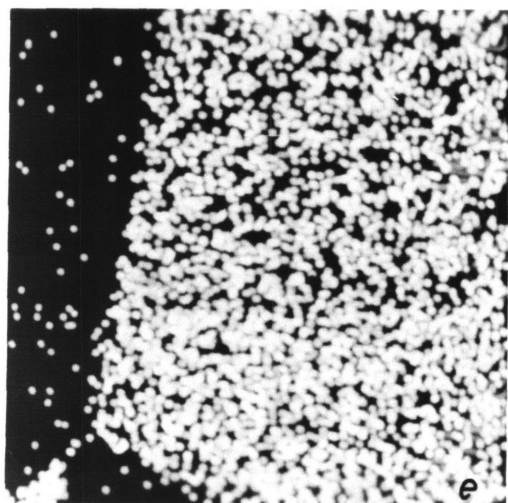
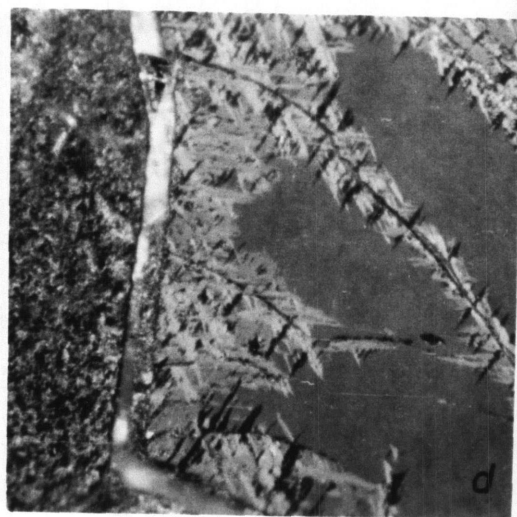
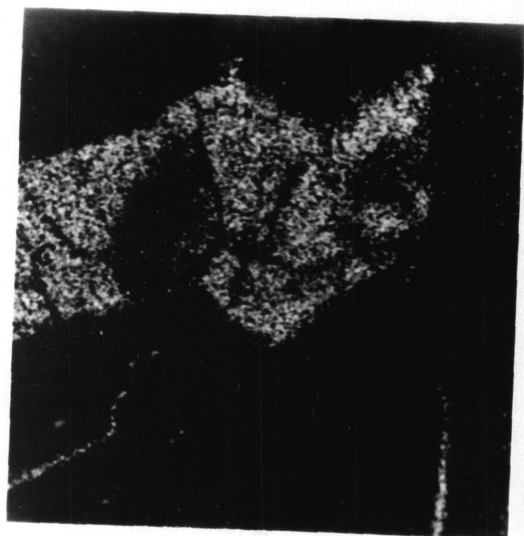
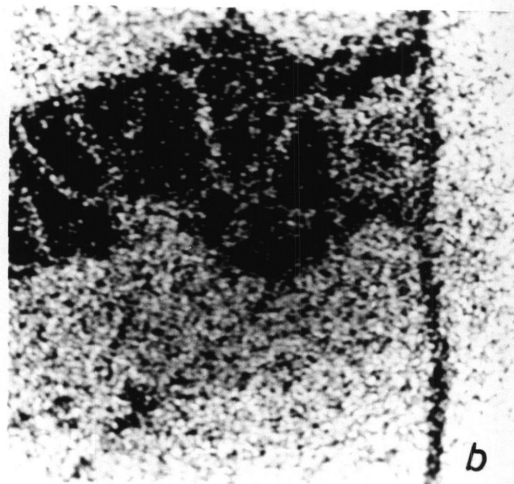
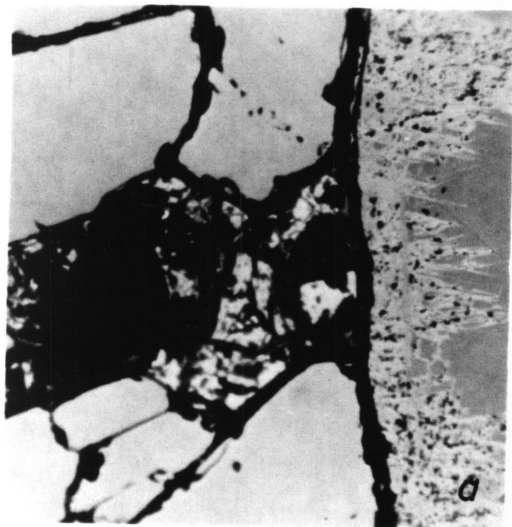
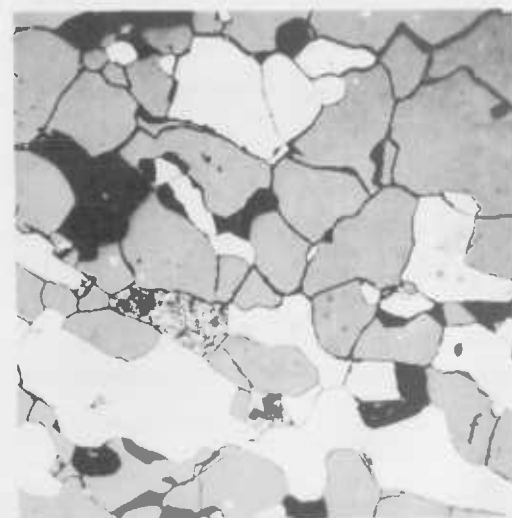
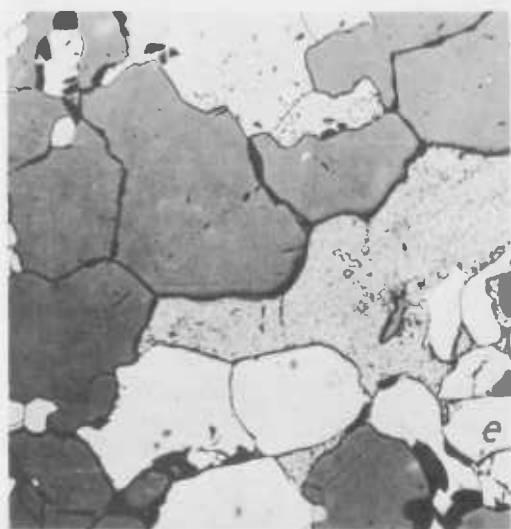
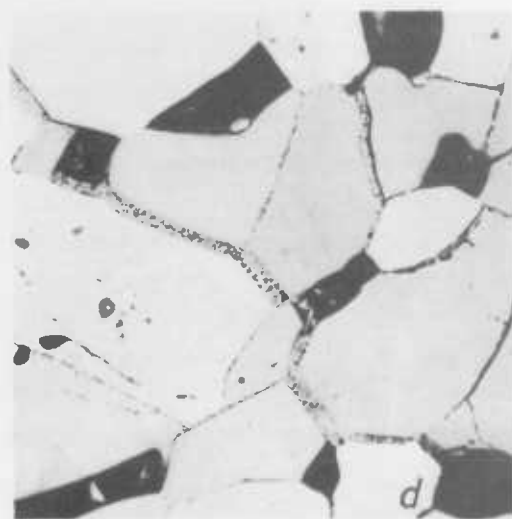
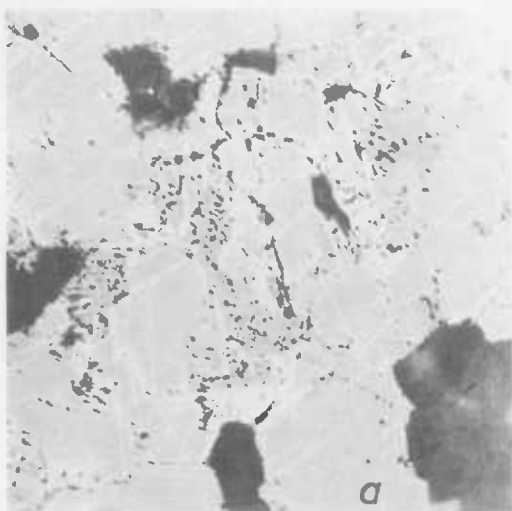


PLATE 5.3.

(a,b,c) Magnetites altering to hematite without the exclusion of manganese oxides.

(d,e,f) Clean manganese free hematite grains. In (e) two phases of hematite are present, one original and one after magnetite with minor amounts of manganese present as exsolution features now removed by polishing.



is present in solid solution in magnetite, one would expect a concentration of manganese during the processing of these ores. One of the ore types shows this characteristic but the other does not and the reason is unexplained.

(c) Ores with low manganese content (0.15 - 1.0%)

Examples of this group of ores is shown in plate 5.3 (d) (e) and (f). The ores contain only hematite as an iron oxide. On polished section some of the hematite shows small pitting and cavities that have the appearances of an exsolution feature, although the mineral exsolved has now been removed by polishing. This is very similar to some features seen in the high manganese ores, where the manganese oxides were in the process of being extruded from magnetites (compare plate 5.3 (e) with 5.1(b)). Since these ores do contain a small amount of manganese it can only be presumed that these cavities were originally filled with manganese oxides, that have been removed by polishing. All the manganese present is present in this form, or in solid solution in the hematite or as a minor cementing agent between hematite grains (plate 5.3 (d)). In as much we would expect table test results on samples taken from these ore types to show an increase in the relative manganese content on processing, and although samples from the diamond

drill holes do not confirm this, blast hole samples do.

Variations in the content of iron oxides

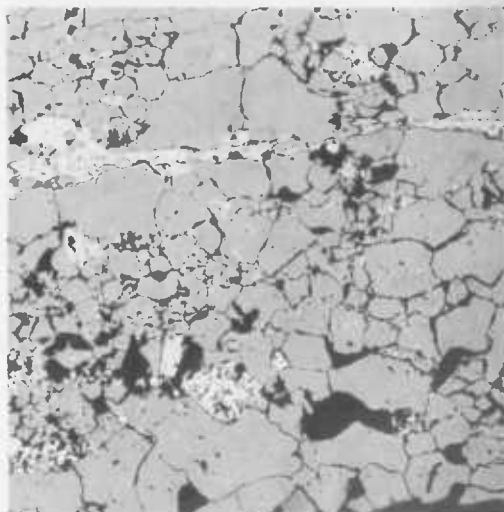
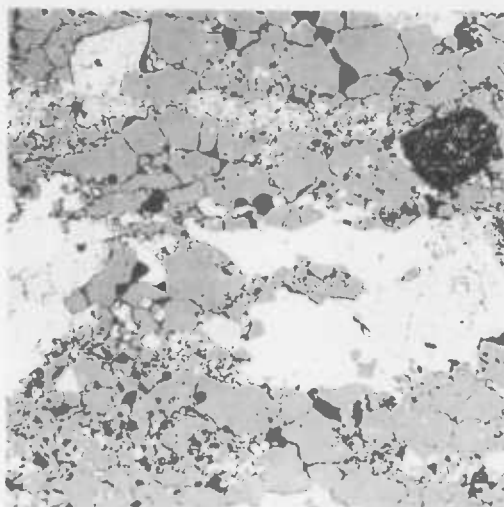
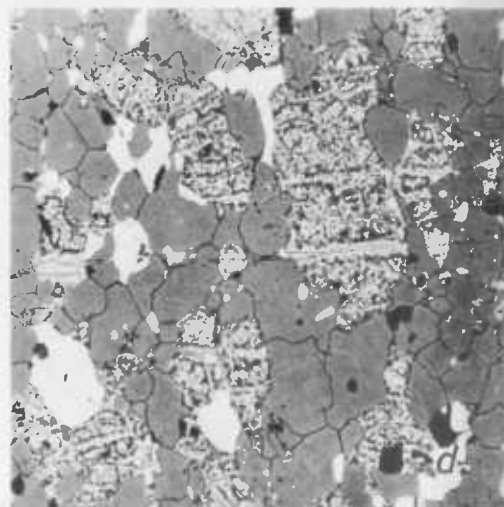
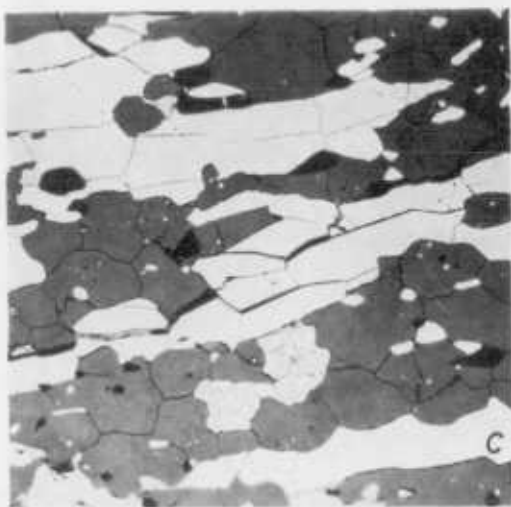
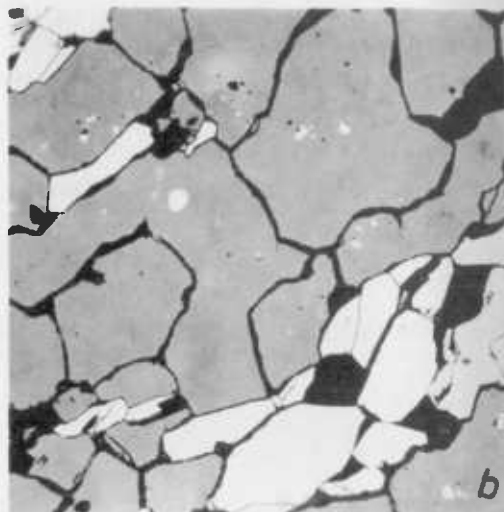
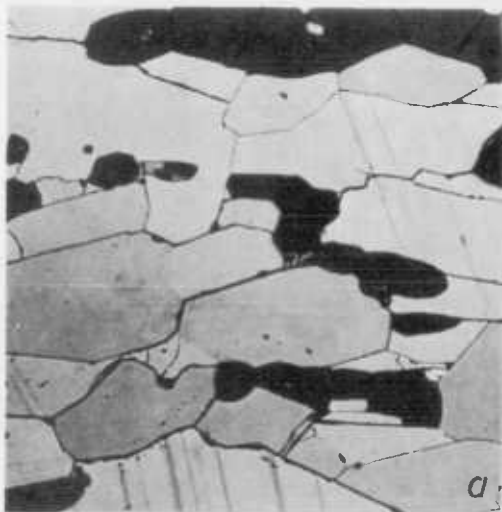
There are two issues which could be considered in this discussion. First of all how much hematite and magnetite and other iron oxides are present in the ores and how far does this reflect the trimodal distribution of iron in the diamond drill hole assays. Secondly, and more pertinent to the operation of the concentrator at Wabush, is the question of what form the iron oxides take and how does this affect their beneficiation. The second point is of more interest and is discussed below.

Plates 5.1.(b) and 5.3(d) show typical coarse grained euhedral textures and one would expect these types of ores to have a good recovery, and indeed these samples were taken from high recovery ores.

Plate 5.4. shows ores with a less ideal grain size and distribution. In the photographs 5.4(b) and (c) although much of the hematite is present in a recoverable form a good deal of the iron is trapped in quartz or at a grain size where spirals will not be very efficient in its recovery. Here separation will be somewhat less than ideal. In photographs 5.4 (d) (e) and (f) recovery will be very poor because of the fine and uneven hematite grain size.

PLATE 5.4.

- (a) Coarse grained aggregate with hematite slightly tabular and showing twinning.
- (b) Hematite is present in two grain sizes, the larger size being slightly tabular but showing some fracturing, the smaller almost entirely locked in quartz grains.
- (c) Hematite showing its platy habit although the quartz and magnetite grains are still equidimensional. Smaller hematite grains are present at quartz grain boundaries and locked in quartz grains.
- (d,e,f) Hematite showing a vast range of grain sizes whilst the quartz remains equidimensional and of a more consistent grain size.



Study of the hard ores

There are some areas in all pits, but in particular in the South Pit where the ore is exceptionally hard.

Although the problem has not been studied in any great detail it appears that the cause is two fold. Firstly that the ores are unleached. Examination of the ores in situ and in section shows that the ores are in very much in an original state and that the small amount of silica that has been removed from some ores to render them friable has not been removed in the harder ores. Plate 5.4(a) and (b) show ores from which silica has not been removed (compare with 5.4(c)). Similarly 5.4(d) shows a close knit aggregation of quartz and hematite in contrast to 5.4(f) in which the system is considerably more open.

However, many of the ores have the appearance of being re-cemented by re-worked hematite and occasionally silica. This is often seen in the field in the East and South Pit areas around fault zones, and to a lesser extent in the West Pit. For about 2ft. either side of a fault the ore will become extremely dense and hard and become progressively softer away from the fault. Where the packing of the faults is dense this provides an easy route for the introduction of reworked material and a whole zone may become indurated. A whole variety of textures are seen in conjunction with the re-introduction of material, and are illustrated in plates 5.5 and 5.6. A particularly good example of

PLATE 5.5.

- (a) Colloidal hematite in filling vein; darker goethite on edge of vein.
- (b) Hematite infilling vein and re-cementing brecciated quartz.
- (c) As (b). Note the banding in vein representing different sequences of hematite veining.
- (d) 'frozen' colloidal hematite. (x 220)
- (e) Hematite vein cutting fine grained quartz goethite ground mass; partially altered quartz crystals now ringed with hematite.
- (f) Brecciated quartz vein.

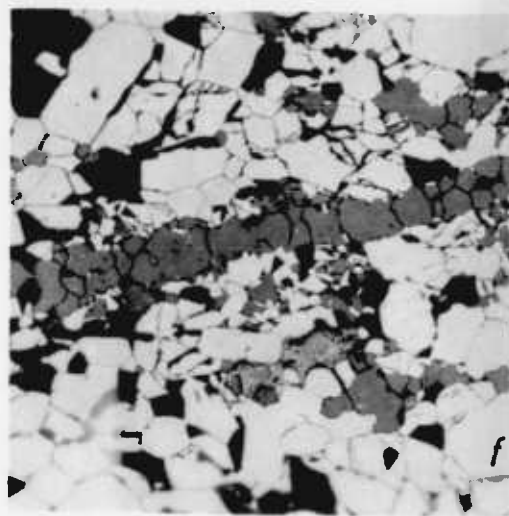
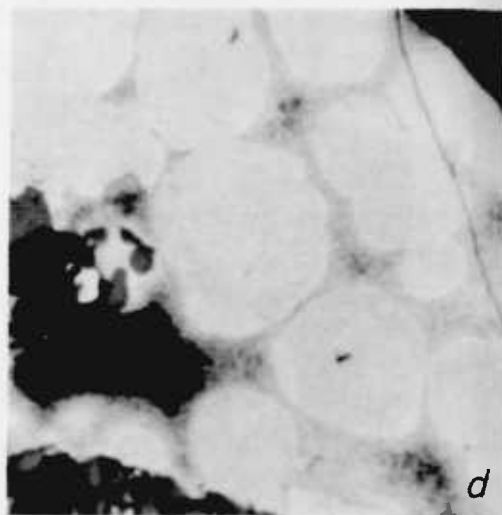
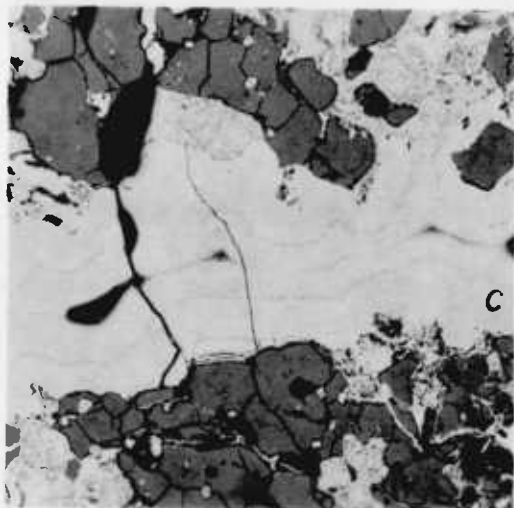
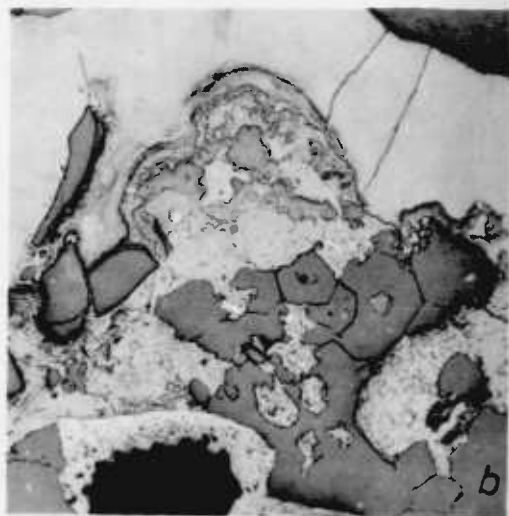
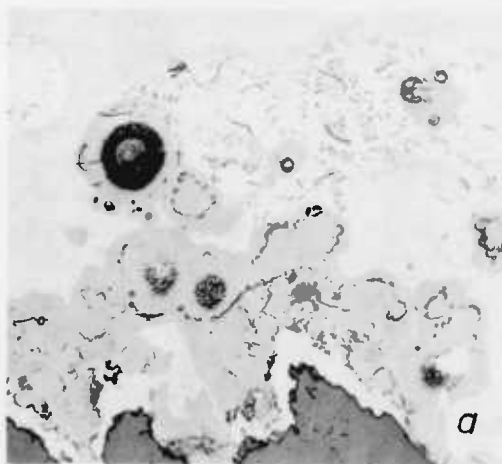
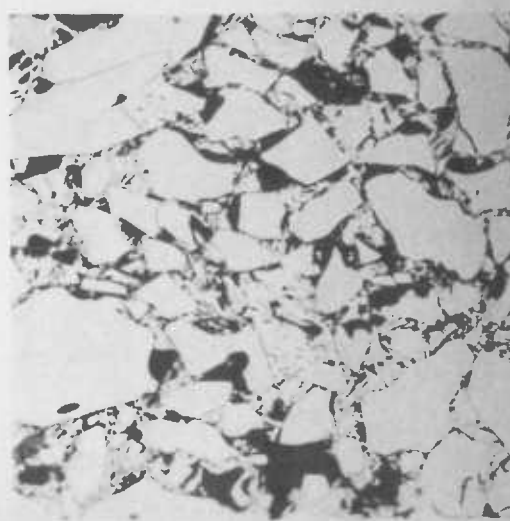
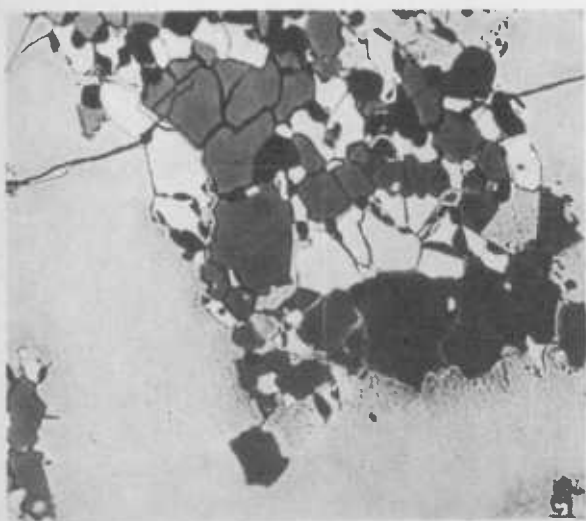
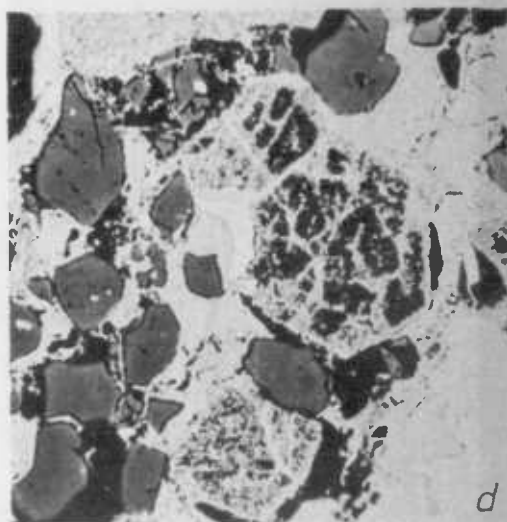
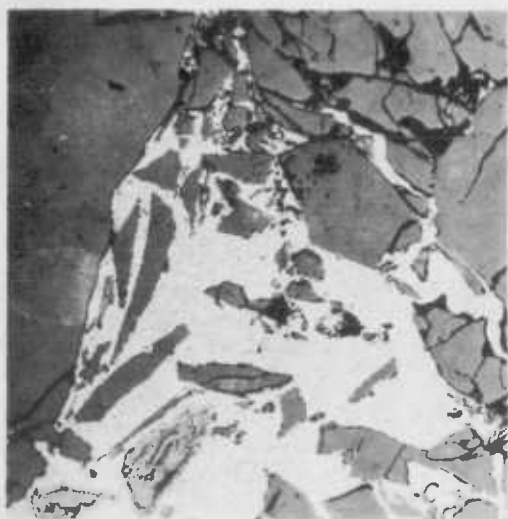
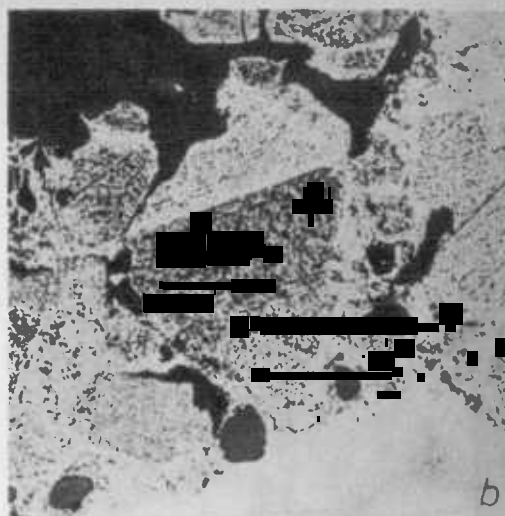
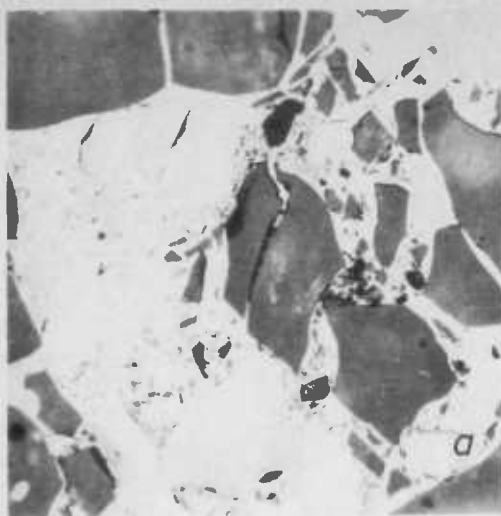


PLATE 5.6.

- (a) Hematite vein re-cementing brecciated quartz
hemative fragments.
- (b) Replacement of magnetite by hematite along
the edge of a vein.
- (c) Brecciated quartz re-cemented by hematite.
- (d) Replacement of quartz by hematite.
- (e) 'Island' of ore surrounded by hematite vein,
with re-cementing along crystal boundaries.
- (f) Brecciated hematite ground mass - re-cemented
by hematite.



recementing by hematite and formation of the dense blue high grade 'steel' ore is shown in plate 5.6(f).

C. CONCLUSIONS

Positive differences in mineralogy and fabric of the ores can be seen between units. The manganese content would appear to be an original sedimentary feature and reflects the amount of manganese co-precipitated with the iron oxides. This is now seen in solid solution in magnetite and as a discrete phase, pyrolusite, which appears in the main to have been excluded from the magnetite lattice on martitisation.

The manganese does not seem to have moved very far from the iron, and this was predictable because of the similar chemistry of the two elements. In some cases there is evidence of solution of manganese and reprecipitation close by as a fine grained phase. The lower manganese ores do not in the main contain manganese oxides as a discrete phase - rather small amounts in solid solution in the iron oxides. We would then expect a concentration on processing along with the iron minerals and although this is not made evident in the diamond drill hole table test results, it is the experience of plant operations.

The coarseness and evenness of crystallisation will effect the overall recovery, and this appears to have been fairly homogeneous in any one ore type.

In solving the hard ore problem two points have to be considered. Some ores are obviously unleached and this causes the ore to be siliceous and hard. How much of this type of material is present is not known. It would be in keeping with the experience of other similar operations to expect an increase in unleached material with depth. There also appears to be an ore whose hardness is due to re-cementation, by material introduced along faults. How much of each ore type is present in the ore body and particularly their effects on milling I cannot tell. Of these the latter may be solved by a detailed investigation of the faults and the fault density, but to estimate the amount of unleached material present poses a difficult problem, which will require intensive on site investigation to find a solution.

CHAPTER VI

CONCLUSION AND RECOMMENDATIONS

A. CONCLUSIONS

The geological setting against which the work in the mining regions of south west Labrador is set, is very poor. At the present time material movement in the area is in excess of 40 million tons per year and with the opening of the Mount Wright deposit this will exceed 60 million tons per year. The economy of the provinces of Newfoundland and Quebec as well as Canada itself is heavily dependant on the iron ore concentrates that are produced within this small region, yet the best background information that is available on the geology of the area is 1:250,000 map that stops just north of the area of economic interest, and this relies to a large extent on three months field work carried out in 1959. Vast tracts of land lie unmapped. Against this background any serious regional geological interpretations in the area are difficult, to say the least. To date very little attempt has been made at correlation of the iron formations that are exposed in the mining areas. These at most are only a few miles apart and the problem of reconciling the interpretation at one mine with the interpretation at another, should not be a difficult one if a serious attempt were made to do this..

Interpretation of the subsurface geology at the Scully Mine has now reached a relatively sophisticated level. The fold and fault style in the pits have been identified, and new mining sections constructed. In Chapter IV I described the structural style of the ore body as a series of close, moderately inclined gently plunging folds type 1c (Ramsey 1967). These folds are cut by two sets of faults striking 030° and 150° respectively. Inside the ore body eleven ore units have been recognised and examination of these has led to the conclusion that the mineralogy of units are quite different, but at the same time relatively simple. Chapter V outlines the mineralogy of the ore types in some detail.

The problems at the Scully Mine to a certain extent reflect the poor background information that the exploration geologists had at their disposal. Most of the problems that have now been identified in the working operation reflect the poor interpretation of the mine geology that was made prior to the original ore reserve estimates and on which the grade control programme was based. Both of these procedures reflect the feeling that inside the iron formation geology was unimportant. Basically the ore reserve estimates as far as weight recovery is concerned appear to be over optimistic and it is difficult to judge, because of its unusual distribution, how the manganese

estimate would vary if a re-estimation of the ore body were carried out. The indications are that the average manganese content will decrease since the logarithmic average is lower than the weighted arithmetic average used. Even so, the problem of holding an even grade on manganese is made no easier, because even if mining could be restricted to within one ore type the grade uncertainty has wide limits, especially on the high side.

The operation as a whole relies very heavily on the table tests carried out on diamond drill hole and blast hole samples. I have discussed this at some length in Chapter II. Using Pierre Gy's method to estimate the sampling variance from blast hole to final concentrate assay we find that the relative sampling variance on crude iron is 0.02 which gives a standard deviation of 0.47 on a head sample of mean of 36%Fe. Thus the expected uncertainty of the mean will follow the expression:

$$P(34.6 \leq \bar{x} \leq 37.4) = 0.95 \quad \text{..... Crude Iron}$$

Similar calculations have been done on concentrate and tailings assays and from these we can see that the expected error in the weight recovery figure is in excess of 2.5%.

A similar exercise on the manganese assays put the uncertainty of its mean at $\pm 0.28\%$ at 95% confidence level. Methods for calculating standard deviations using duplicate sampling techniques give similar results. These are not directly attributable to bad laboratory techniques

but rather to the expected variations under the given sampling procedures. The fact still remains that grade is required to be held well within the uncertainties of the means that are used to establish the grade. The table test results are taken far too seriously. Provided an assay looks right it is used with conviction to the second decimal place and precious man, and computer hours used to do so. Although the uncertainty of a single assay is high this does not imply that the uncertainty of even the monthly estimate let alone the 1 year or 5 year estimate will be high. The table test results as such will not affect these as the averaging of a large number of assays will tend to reduce the uncertainty, and when, as in the total ore reserve estimate the number of table tests is in excess of 12,000 the uncertainty will be very small. Any error which may arise in these long term plans will be inherent to the method itself. There is, I feel, cause for concern on the daily level. Here a few assays are used to assess grade and the resulting uncertainty, high.

The ore body can be divided into eleven ore units as shown in Chapter III. The fact that faulting and folding may cause up to five or six of these units to autocrop on any one bench at any time, does not immediately help the grade control engineer, but the fact that the grade within one of these units can be expressed in simple terms should

provide help in the long term. I have shown in Chapter IV how by unfolding the ore body, grade can be assessed within any one ore unit. Since the thickness of these units is known then the next logical step would be the re-calculation of ore reserves using as data, the grade estimated for each unit taking into account its observed lateral variations and its thickness.

B. RECOMMENDATIONS

The assessment of the quantity of hard ore in the pits is a major problem. There is no doubt in my mind that some of the harder ores are confined to faults and fault zones and are due to re-cementation of the ores by hematite and silica. A detailed study of the faulting will provide many of the answers as to how much of this ore is present and its whereabouts. Much of the hard ore is unleached and this being the case the percent hard ore could be expected to increase with depth. I would suggest that a considerable amount of research into this topic alone would not be wasted money. This could include not only the assessment of the quantity of hard ore, but the relationship between hardness tests and the way in which an ore behaves on milling.

I would also recommend a change in the grade control procedure. At present time a great deal of manpower and computing time is lost on calculations that are based on

inadequate data in an attempt to hold grade. An assay of say 41% weight recovery and 1.0% Manganese may well prove on mining to be 37% weight recovery and 1.4% manganese because of the very nature of the table test. It seems that the more reasonable technique would be to average grade within a blast within an ore type, thereby minimising the uncertainties associated with the assay value. This grade could then be used for all ore being moved from an ore unit from a blast.

The ore reserve estimate procedure could also bear change. New sections showing the revised structure and ore units should be drawn and used as the basis for these estimates. Instead of the 'block-grading' method presently used I would suggest further investigation into the assessment of grade using the grade plans generated for each unit. By superimposing each unit's grade plan, one on top of the other, we would have in effect the expected grade distribution tonnage and overall grade for the pit. Re-constitution of these grade plans to the folded state would provide the basis for the construction of monthly, yearly, five yearly, etc. mining estimates, and at any one time the trend that grade is likely to follow, would be readily available. .

I have expressed my concern in the way in which the grading limits are presently set up. It has been shown that the manganese frequency distribution is strongly positively skewed and can be fitted to a log gaussian model. This

being the case range of values on the high side of the mean to which the uncertainty of the mean may extend, is far greater than on the lower side of the mean.

Although the position of the steel companies is appreciated, to have an upper limit on concentrate manganese grade of 2.0% or thereabout is something which will cause continuous trouble and a considerable amount of time and tonnage lost. The grade control problem is not helped by the fact that at present an attempt is being made to hold grade within the uncertainty of the means of the table test data. A serious review of the grading limits is recommended.

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APPENDIX A

A PROGRAMME TO CALCULATE A VARIOGRAM IN THREE DIMENSIONS

When considering the calculation of the variogram function in three dimensions it is necessary to compare the assay values at all sample points, in the ore body, with all others. Effectively the variogram function is calculated on pairs of points that bear the same distance relationship to each other. To do this on a large ore body is impractical since an impossibly large number of distance relationships can occur in a three dimensional body. Instead fields of points are set up where the distance between pairs is between set values and these values incremented to give the variogram function. A generalised formula for a variogram in three dimensions is given by

$$\gamma(h) = \frac{1}{2S} \iiint_S \left(f(x+h_1, y+h_2, z+h_3) - f(x, y, z) \right)^2 \delta S$$

where S = number of points within a field

$f(x, y, z,)$ = assay value at point co-ordinates x, y, z.

$f(x+h_1, y+h_2, z+h_3,)$ = assay value at point co-ordinates
 $x+h_1, y+h_2, z+h_3.$

Since in any ore body we are dealing with a finite number of sample points, a summation provides a satisfactory approximation of the above intergral.

In the programme used each assay plus an x, y and z co-ordinate are read in as a series of unidimensional arrays, with their subscripted variables to identify each parameter. This way whole sets of assay data at a single sample point can be compiled. If a three dimensional array were used only one assay value at a time could be considered and a much larger storage facility needed. The value of the field size (SIZE) is read in on cards at the end of the Fortran deck. Output is a series of gamma h values for incremented field sizes. To use the programme in one or two dimensions one or two of the co-ordinate values is held constant.

To compute the value of gamma h we compare the assay value at the origin of co-ordinates with all others. First of all the distance between this assay (subscripted 1) is compared with assay 2 (subscripted 2) and the distance between them calculated using Pythagoras' Law. By dividing this distance by the field size, adding 1 and storing the result in an interger variable position (INT) we obtain the field into which the pair fall.

(e.g. if distance = 37.2 and SIZE = 5.0

$$INT = (37.2 / 5.0) + 1 = 8$$

if distance = 4.0 and SIZE = 5.0

$$INT = (4.0 / 5.0) + 1 = 1)$$

The value of INT can now be used as a subscripted variable in calculating the squared differences. These are stored in a variable A(INT) and summed for each value of INT. Thus we have a series of summed squared differences for pairs of assays values sorted by their distance relationship into different fields. This can be then used in the valculation of gamma h by dividing the summed square differences by the number of pairs in each field.

The programme is written in Fortran IV and will run without modification on a CDC 6000 Series computer on the FUN or MNF compilers without modification.

```

PROGRAM MAIN(INPUT,OUTPUT,TAPE,TAPE5=INPUT,TAPE6=OUTPUT,
1TAPE9=TAPE)
C      A PROGRAM TO CALCULATE A VARIOGRAM IN 3 DIMENSIONS
C
C      DISTANCES BETWEEN POINTS AND ALL OTHER POINTS ARE
C      CALCULATED AND THESE DISTANCES USED WITHOUT
C      CONSIDERATION FOR THE VECTOR BETWEEN POINTS AS
C      THE MEANS OF SELECTION FOR THE VARIOGRAM CALCULATION
C
C      DIMENSION SECT(1200),BENCH(1200),ABSC(1200),FE(1200),CRMN(1200),
1WT(1200),CMN(1200),K(2000),A(1500)
C      DIMENSION 1A(20)
C      COMMON N,SIZE
C
C      SET TAPE TO DATA REQUIRED
C
C      DO 45 NA=1,2419
C      READ(9,5)(1A(J),J=1,10)
5      FORMAT(10A8)
45      CONTINUE
C
C      GENERATE A RANDOM NUMBER TABLE
C
C      Y=RANF(31,1)
C      DO 15 J=1,1500
C      A(J)=RANF(0,0)*10.0
15      CONTINUE
C
C      READ INTERVAL VALUE
C
C      READ(6,1001) SIZE
1001      FORMAT(F7.4)
C      READ IN DATA AND TAKE 10 PERCENT SAMPLE
C      USING RANDOM NUMBERS GENERATED
C
C      NDEC=0
C      J=0
C      DO 25 N=1,1200
C      READ(9,1000) USECT,JHOLE,JORD,JFE,JMN,JWT,JCMN,JBENCH
1000      FORMAT(16,A5,15,7X,3I4,8X,14,4X,14,25X)
C      IF(EOF,9)20,3
C
C      COUNT NUMBER OF RECORDS READ
C
C      NDEC=NDEC+1
C
C      SET UP RESTRICTIONS
C
C      IF(USECT.GT.15000) GO TO 1
C      IF(JORD.LT.19750) GO TO 1
C      J=J+1
C      IF(J.EQ.1500) GO TO 4
C
C      TAKE TEN PERCENT SAMPLE
C
C      IF(A(J).LE.10.0) GO TO 2
C      GO TO 1

```

```
2  SECT(N)=JSECT
   AFE=JFF
   FE(N)=AFE/100.0
   AMN=JMN
   CDMN(N)=AMN/100.0
   AWT=JWT
   WT(N)=AWT/100.0
   ACMN=JCMN
   CMN(N)=ACMN/100.0
   BENCH(N)=JBENCH
```

```
C
C      ASSIGN  NORTHING(ARSC) TO EACH  HOLE NUMBER
C
```

```
      ARSC(N)=JORD
25  CONTINUE
4   WRITE(6,1003)
1003 FORMAT(1H1)
    WRITE(6,61)
61  FORMAT(20X,*CRUDE  IRON  */)
    CALL PAIRS(FE,SECT,ARSC,BENCH)
    WRITE(6,1003)
    WRITE(6,62)
62  FORMAT(20X,*CRUDE  MANGANESE*/)
    CALL PAIRS(CDMN,SECT,ARSC,BENCH)
    WRITE(6,1003)
    WRITE(6,63)
63  FORMAT(20X,*WEIGHT  RECOVERY*/)
    CALL PAIRS(WT,SECT,ARSC,BENCH)
    WRITE(6,1003)
    WRITE(6,64)
64  FORMAT(20X,*CONCENTRATE  MANGANESE*/)
    CALL PAIRS(CMN,SECT,ARSC,BENCH)
20  WRITE(6,1011) NREC
1011 FORMAT(20X,*NO.  OF RECORDS  =*,16)
    STOP
    END
```

```

SUBROUTINE PAIRS(AS,X,Y,Z)
DIMENSION AS(1200),X(1200),Y(1200),Z(1200),K(2000),A(2000),
1DIN(2,2000),H(2000)
COMMON N,SIZE
MAX=0
DO 25 J=1,2000
H(J)=0.0
K(J)=0
A(J)=0.0
25 CONTINUE
C
C      CALCULATE DISTANCES BETWEEN POINTS
C
NN=N-1
DO 15 JA=1,NN
JB=JA+1
DO 35 JC=JB,N
D=SQRT(((X(JA)-X(JC))**2)+((Y(JA)-Y(JC))**2)+((Z(JA)-Z(JC))**2))
C
C      ASSIGN D TO CLASS INTERVAL OF 5FT INCREMENTS
C
INT=D/SIZE
INT=INT+1
K(INT)=K(INT)+1
C
C      CALCULATE SQUARED DIFFERENCES 'A' AND SUM INSIDE GROUP INTS
C
A(INT)=A(INT)+((AS (JA)-AS (JC))**2)
C
C      SET MAXIMUM VALUE OF INT
C
IF(INT.GT.MAX) MAX=INT
25 CONTINUE
15 CONTINUE
C
C      CALCULATE GAMMA H
C
DO 45 INT=1,MAX
IF(K(INT).EQ.0) GO TO 101
B=K(INT)
H(INT)=A(INT)/(2.0*B)
GO TO 45
101 H(INT)=0.0
45 CONTINUE
10 WRITE(6,221) MAX
221 FORMAT(1H1,20X,*NUMBER OF CLASS INTERVALS IN VARIOGRAM = *,I4/)
WRITE(6,52)
52 FORMAT(20X,*GAMMA H*/ )
K7=MAX/10
K7=K7*10+1
K1=0
11 DO 85 J=1,K7,10
WRITE(6,53) H(J),H(J+1),H(J+2),H(J+3),H(J+4),H(J+5),H(J+6),
1H(J+7),H(J+8),H(J+9)
53 FORMAT(20X,10(F6.2,4X)/)
85 CONTINUE
RETURN
END

```