UNCERTAINTY IN OPEN-PIT MINE DESIGN

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Abstract

This thesis features a systematic approach to the problem of ore reserve estimation and open-pit mine design, with extensive use of the computer and its graphical aids. Its ultimate objective is to select, from a series of computer-generated open-cast surfaces, the most valuable pit, using the Net Present Value (NPV) Method.

The ore grades are appraised through geostatistics and therefore have an error of estimation associated with them. The uncertainty in the estimation of the ore grades is incorporated into the NPV.

The approach suggested provides mine management with a range of NPV's, reflecting the uncertainty in the estimation of some key variables of the problem of economic evaluation.

The dissertation covers:

a) a review of the literature in respect of computer applications to ore reserve estimation, open-pit design and economic evaluation

b) creation of a geological data base, mapping and graphical representation of the geological data

c) classical statistical and geostatistical analysis

d) ore reserve evaluation through block kriging and creation of a mineral inventory

e) computer-aided open-pit design and establishment of working pit sequences

f) economic appraisal of the optional open-pit designs.

The research work is focused on Mutuca, an open-cast iron ore mine located in Brazil, which have been producing high-grade hematite for more than 25 years.

To my wife, Lúcia, and
my daughters, Silvia and Marina.

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List of abbreviations

APCOM - International Symposium of "Computer Applications to the Mineral Industry".

BESR - Break-even economic stripping ratio

CAPES - Coordenação de Aperfeiçoamento do Pessoal de Nivel Superior

EAC - Equivalent Annual Cost Method

IC - Imperial College

ICCC - Inperial College Computer Centre

IMM - The Institution of Mining and Metallurgy, London.

IRR - Internal Rate of Return Method

LODR - Lump Ore for Direct Reduction

MBR - Mineraçõcs Brasileiras Reunidas S.A. - MBR

MRE - Department of Mineral Resources Engineering

NPV - Net Present Value Method

OPD - Open-pit design

RSM - Royal School of Mines

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1. Preliminaries

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1.1.2 Creation of a computerized open-pit design system

1.1.3 Generation of a series of optional pit shells

1.1.4 Graphical representation of the optional pits

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1.3.2 Methods selected to be used in this study

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1.1 Summary and objectives

This research work is aimed at establishing a methodological approach to the problems related to reserve estimation, open-pit mine design and evaluation of a typical iron ore deposit, with the extensive use of the computer and its graphical aids. The scheme in figure 1.1 features the main points covered by this thesis.

Geological data base
Mapping of the geological data
Statistical analysis
Geostatistical analysis
Geological interpretation through kriging
Estimation of grades by way of kriging
Creation of the mineral inventory
Topographic data base
Input data for pit shell generation
Open-pit design and evaluation
Working pit sequences
Economic data base
Economic analysis

Figure 1.1 Main aspects of this dissertation

The work is based on Mutuca Mine in Brazil, which produces high iron content lump ore for direct reduction ($-1\frac{1}{4}$" ; $+\frac{1}{4}$"). Mutuca's beneficiation plant is fed with a "direct shipping" ore, therefore it is not a "concentration" installation, only crushing, screening, washing and cycloning processes being used. Mutuca is owned by Minerações Brasileiras Reunidas S.A. - MBR, and has been operated since 1961.

MBR has decided on a development plan to increase the capacity of this mine from 3.2 Mtpy to 6.5 Mtpy. Three factors have necessitated this decision:

- The existing beneficiation plant, apart from having some components with many years of use, lies on top of known reserves, sterilizing their exploitation;
- The market for direct reduction of iron ore has improved;
- Recent drilling has increased the estimated reserve from 70 to 100 Mt, that is to say, by 40%.

Before the additional reserve was discovered, the elevation 1080m had been chosen as the lowest bench for the open-cast mine, and that pit was referred to simply as "pit 1080". In this study, a series of open-pits were designed and evaluated, the best option being selected by way of a simplified economic analysis.

As an existing mine which has been mined for 25 years, Mutuca presented some unique real-life problems to which "tailor-made" solutions had to be devised, yet the general guidelines for open-pit mine design and evaluation had to be respected. Although targeted at Mutuca, this study might be of interest, as a piece of applied research, to other people investigating similar areas.

The main achievements and conclusions of the present study are summarized in the following topics.

1.1.1 Preliminary geological interpretation through kriging

The possibility of using geostatistics and indicative or categorical kriging as a means of obtaining a preliminary geological interpretation was discussed(see chapter 7).

The drilling campaign was in progress, at Mutuca Mine, when the author decided to make a trial run with the kriging program, using as input data two categories only: ore (1) or waste (0). "Ore", in that case, was "hematite", as it is defined by MBR (see chapter 3). The resulting kriged estimates divided the blocks of the total reserve into ore and waste, therefore producing a first idea on the location and extension of the additional reserve, while the conventional geological re-interpretation was not available.

By giving the position of the ore blocks in a relatively short period of time, this application of geostatistics allowed for the immediate estimation of the block grades through kriging, and an overall qualitative and quantitative re-assessment of the ore reserve was obtained quickly(see chapter 7).

Later on, when the conventional geological interpretation had been concluded, the two results were compared. The preliminary analysis was used as a reference until the complete

re-assessment of the geological reserve was completed, the latter being used from then on.

1.1.2 Creation of a computerized open-pit design system

A computer-assisted open-pit design system was developed based on a program originally developed by MBR (see chapter 9), incorporating the following facilities:

- Slopes are defined through three possible ways: a) blockwise, as a function of type of material; b) blockwise, as a function of rock engineering parameters; or c) sectorwise, dividing the mine into sectors, i.e., angles in the horizontal plane, a slope being assigned to each sector.
- Slopes are flattened to account for the in-pit road, as a function of road width, number of road laps and maximum terrain elevation, in each sector.
- A base polygon, in the lowest bench, and the bench height are input data.
- The algorithm of pit shell generation goes upwards from the base polygon following the slopes given, and expands horizontally according to the shape of the ore body in each bench.
- Different pit shapes are produced by changing other design parameters, like minimum/maximum internal angle for the bench polygons, permitted number of iterations for horizontal expansion, and cord length.
- With the pit shell as the lower boundary and the topographic map as the higher one, the blocks lying within the pit limits are counted, an evaluation being produced in terms of volumes and tonnages of ore and waste.

1.1.3 Generation of a series of optional pit shells

Starting from the previous lowest bench elevation, 1080m, five different pits were designed: Pits 1080, 1020, 1000, 980 and 960, the last being the deepest one.

From each lowest level, two different set of slopes were used for pit shell generation, corresponding to alternative road layouts: a) switchbacks on the low side of the pit (East wall); and b) road spiralling around the pit. Ten different optional pits resulted for evaluation and analysis.

1.1.4 Graphical representation of the optional pits

A complete dissection of the various pits generated was made with the aid of graphical representation, for visual comparison between the options available. The pit boundaries, ore and waste blocks were plotted with different colours in bench maps, cross and longitudinal sections.

It was observed that the computer's graphical aids constitute a valuable tool for mining applications.

1.1.5 Selection of the layout for the in-pit road

It was concluded that it is better to have some switchbacks on the low side of the pit (East wall) than to spiral the road around the pit, for the stripping requirements are reduced roughly by 10 million tons. Besides that, the risk of a slump blocking the road is smaller, because the slope on the East wall is only 43 degrees, whereas it is 52 degrees on the West wall.

Once the road layout had been established, the five optional pits with spiralling roads were eliminated, and the selection process was focused on the other five pits, as described in the following items.

1.1.6 Incremental analysis of the pits - preliminary economic appraisal

The concept of an economic break-even waste to ore ratio, as described by Miller (1982) was adopted for preliminary selection of the ultimate pit. Starting from the shallowest pit, 1080, the tonnages of ore and waste were analysed on an incremental basis, the waste to ore ratio being the key variable (see table 9.11).

The marginal profit obtainable with each ton of ore is approximately 5.0 times the cost of stripping each ton of waste (see chapter 11). The deeper the pit, the greater the ore tonnage, but the waste tonnage increases at a faster rate, resulting in an incremental ore to waste ratio which grows exponentially. Starting from pit 1080, the above mentioned limit ratio is equalled between pits 1020 and 1000, and exceeded by far when the mine's minimum elevation is changed from 980 to 960 (see figure 9.17).

Pit 1000 was selected as the envelope likely to contain the "best" pit, by this criterion.

1.1.7 Establishment of working pit sequences

Pits 1020, 1000 and 980, the ones with intermediate depths, were studied in more detail, a sequence of working pits being established for them. A constant slope much flatter than the ultimate pit slopes was adopted for designing the sequence of pushbacks (see item 10.3.1)

The mine exploitation is divided in three stages: from top benches to level 1200m; from 1200m to 1140m; and from 1140m down to the minimum bench elevation. An even distribution of ore between the three phases was achieved, with total stripping requirements going up as the ultimate pit depth was increased.

Following that, a simplified pit scheduling exercise was made, and the visual inspection of bench maps, cross and longitudinal sections of the working pits demonstrated the mineability of the three sequential stages.

1.1.8 Economic analysis of the optional pits

A straightforward economic analysis of the optional pits was made, as follows (see chapter 11): in terms of cash flow, taking into account revenues and direct costs only - no fixed costs or investments were considered; and in terms of discounting procedure, the Net Present Value (NPV) Method was used. The impact of variations in key components of the cash flow was assessed by means of a sensitivity analysis.

The principal results are:

- it is worth exploiting the additional reserve that has been discovered recently.
- all pits have NPV's of a comparable order of magnitude, pits 1020 and 1000 being the best ones for a very narrow margin.
- the NPV method, as compared with the maximum total profit and break-even economic stripping ratio, is the most conservative criterion, as far as selecting deeper pits is concerned, because the discounting factor penalizes time-distant ore.

1.1.9 Feasibility of the additional reserve

Once established the feasibility of the new geological reserve, and as there is no major difference, on economic grounds, between the NPV of the optional pits, the conveniences of choosing each of them, under different conditions of ore prices and mining costs, are appraised by way of a sensitivity analysis.

1.2 Introduction

This research work has approached two inter-related problems : a) a methodological topic on what are the techniques for open-pit design and evaluation which are currently recognised as the best ones; and b) the practical need of assessing the feasibility of a geological reserve that has been incorporated to the known reserves of an existing mine by way of a recent drilling campaign.

In this context, a series of questions needed to be answered, as follows:

- What are the best techniques available for open-pit mine design and economic appraisal?
- What is the state-of-the-art in computerized systems for open-pit mine design and economic assessment?
- Is there a computer program that incorporates all the best techniques for open-pit design and evaluation?
- If such a computer program existed, would it be available for use?
- If it were available, would it be capable of handling the specific problems of Mutuca Mine?
- What areas, in the bibliography searched, would deserve additional research, or indicate a possibility for new applications in the context of this thesis?
- If capable of meeting the needs of Mutuca Mine, would it be possible to make modifications in that computer program for further research, if needed?
- What is the resulting quality of Mutuca Mine's ore with the addition of the geological reserve which has been discovered recently?
- Is it worth mining that additional ore reserve?
- Once it is interesting to explore that reserve, down to which elevation is the mine economically feasible?
- Given the mine design is to be changed, what should be the best layout for the in-pit road, as far as stripping needs are concerned?
- How should the mining operations be phased, in order to secure a balanced ore to stripping ratio, in terms of working pit sequences?

The objective the author had in mind, when he undertook this research project, was to use the computer and its graphical aids to answer some of the previous questions, particularly those related to methodology. This would not be in an "automated" fashion but through a hybrid solution, with the machine having not too many degrees of freedom and

acting merely as a means of expediting the work that would otherwise be made by hand.

In practical terms, there was the possibility of demonstrating the utility of a dynamic system for mine design and evaluation, capable of reacting to the results of the latest exploration, especially if the "on going" nature of the drilling process is considered.

The prospect of developing a tool which might be of assistance to solve a number of mining-related questions has been extremely rewarding, from the author's standpoint. Computer applications, by their nature, can be progressively refined or, if needed, completely modified. Therefore, the Mutuca Mine problems can be handled in more detail, or perhaps the requirements of other open-cast mines can be met, if the seed represented by the system introduced herein is cultivated in the future.

1.3 Literature review

This review is aimed mainly at three questions :

- What are the techniques for open-pit mine design and evaluation which are currently recognised as the best? ;

- Which methods could be selected, from the best ones, for approaching the specific problems of Mutuca Mine? ; and

- What areas, in the bibliography searched, would deserve additional research, or indicate a possibility for new applications, in the context of this thesis?

The discussion of these topics is presented in the following paragraphs, and each chapter contains specific comments on the related literature.

1.3.1 Current open-cast design and evaluation methods

In general : Both Kim(1978) and Crawford(1979) reviewed advances in open-pit limit design. The former author believes that "an attempt to select the optimal open-pit limit should be made under certain circumstances." The first one he mentions is : "the pit is nearing its final stage in mine life and accurate cost as well as slope information can be utilized,(...)".

The present situation of Mutuca Mine corresponds exactly to the mentioned circumstance.

"Floating cone" : it is one of the earliest principles used in open-pit design, and was mentioned by many authors, like Caldwell (1961) and Pana (1973). A conceptual cone is positioned with its tip pointing down to a block of ore, the use of regular blocks being

implied in this context. Waste blocks lying within the cone boundaries have their stripping costs assigned to this ore block. In the same manner, ore blocks encompassed by the cone contribute to the cone's total revenue. The blocks of ore associated to cones with resulting positive net profits are considered to be economically feasible.

The main shortcoming of this method is the fact that adjacent ore blocks with uneconomical prospects might prove to be feasible, if sharing the stripping costs of waste blocks lying in the intersection of their inverted cones.

Processes to optimize pit design : the paper by Lerchs and Grossmann (1965) has become a classic, and deserves special comment. The two-dimensional pit design algorithm handles each vertical section of the ore body separately, in such a manner that resulting pit sections do not necessarily fit into a harmonic shape, the walls and the bottom of the pit having to be "smoothed out". This requires a great ammount of effort, and the final pit may be far from optimum. For the three-dimensional pit design, the Dynamic Programming approach becomes impractical, a graph algorithm being applicable, instead.

One of the principal weaknesses of this method is the use of the maximum total profit as the economic criteria, ignoring the time value of money.

Geometry of the ore body : some authors, like Plewman (1979) assume the ore body can be adequately represented by a geometric solid with a regular shape, like a cone, cylinder or prism.

The shape of Mutuca's ore body was identified in detail with the aid of drilling, and is essentially irregular, as most ore bodies are. It is best represented by its actual features.

Pit sequence planning and scheduling : in recent years, there has been a tendency to focus on the pit sequence planning, instead of on the ultimate pit design (for additional references, see chapter 9). In this respect, Couzens (1979) lists some interesting principles :

-(...)"we must keep our objectives clearly defined while realizing that we are dealing with estimates of grade, projections of geology and guesses about economics. We must be open to change."

-(...)"we must remember that we are dealing with volumes of earth that must be moved in sequence; geometry is as important to a planner as is arithmetic."

-(...)"we must remember that we are dealing with time, that volumes must be moved in time to realize our production goals, and that the productive use of time will determine efficiency and cost effectiveness."

The relevance of pit sequencing and scheduling is stressed even more by Mathieson(1982) : "(...)far too much importance is often placed on the rigorous definition of ultimate pit limits per se. Given the immense uncertainties surrounding any prediction of future commodity price trends, costs and stable pit slope angles(particularly at the project feasibility stage) such analytical preoccupation and expense is of little real benefit in the overall context of generating a good long term mine plan. After all, profits far in the future have practically no impact on a property's net present value anyway."

"If a pit has been properly phased in its development plan such that each successive sequence is less profitable than the one previous, then the operation will have maximized annual profitability. It will either continue in production or shut down, as the economic climate and the specific profitability of the next phase to be stripped dictates at that time."

(...)

"The use of sophisticated so-called 'optimal' scheduling systems on these deposits would (...) constitute a waste of resources."

In accordance with the aforesaid trend, working pit sequences were established for the most promising pits of Mutuca Mine. No "optimal" procedure was adopted for the scheduling exercise.

1.3.2 Methods selected to be used in this study

Methodology : the approach adopted in the present study was the generation of a series of optional pit shells for subsequent economic evaluation. No "automatic" method for pit design was applied, a hybrid process being used instead, with the computer and its graphical aids merely acting as a means of expediting the work which otherwise would be made by hand.

Computer-assisted open-pit design system : an OPD (Open Pit Design) system was created, its main application being the production of a series of optional pits, obtained with different lowest elevations and base polygons.

Variable slopes and pit geometry : the mine benches were represented by way of points, i.e., polygons with any irregular shapes, as opposed to the representation with the aid of blocks. As a consequence of this, the pit has variable pit slopes taking their actual values, and not being approximated by the block dimensions.

Graphical aids : the resulting pits generated as described above were plotted in four different ways, namely: pit contour maps; cross-sections featuring ore and waste blocks in

different colours; longitudinal sections represented in the same fashion; and bench maps displaying, besides ore and waste blocks, the pit boundary.

If run through an interactive graphics terminal, the system displays the pit plan map on the screen. If a polychromic output device is available, ore and waste blocks are differentiated by colours, but each category is represented by a different symbol, so that they could be identified in black and white hardcopies as well.

The geological re-interpretation of Mutuca Mine, once available, was transferred to the computer with the aid of a digitiser. Points of the ore body boundary, in each bench, were digitised, the resulting polygon being subsequently converted into an area with blocks of 25m×12.5m×10m as elementary units.

Summation of the ore reserve : the ore reserve tonnage figures are approximate only. The bench height actually adopted for mining is 10m. However, 20m was used in the model, in order to generate a smaller number of points and to reduce handling, processing and plotting costs. Therefore, in each other bench, unit blocks are used to convert irregular shapes into volumes. Intermediate benches have their volumes approximated by averaging the volumes of ore and waste of the adjacent levels. The accuracy in the area calculation could be improved in the future (see item 9.3.3).

In-pit road : despite its importance virtually no OPD system generates the road(Kim, 1978). The present work does not generate the in-pit road, although the slopes, which are rock engineering parameters that have been used in Mutuca for many years now, are flattened to account for the haulage road (see item 1.1.2). An additional difficulty for the road layout to be generated automatically is that the location of the existing exit to the mine must be respected.

Interactive facilities : the system offers the flexibility of manually introducing modifications in the pit design, incorporating a "manumatic" characteristic, as described by Kim(1978). If any modification is needed in a certain bench, the computerized pit shell generation process can be stopped after that particular bench is processed. The desired polygon can then be given as input data for the computer to resume its task, and the shapes of upper benches will take into account the modification, the slopes being respected.

Economic assessment : the economic appraisal of the optional pits was made through the Net Present Value Method, as described by Fleischer (1969).

Variations in ore prices and mining costs were analysed through three hypothesis, a pessimistic, a realistic and an optimistic one. The resulting NPV's were ranked in each case, and the pits selected and analysed accordingly.

Working pit sequences : pushbacks were established, in order to study the mineability of the ore body in succesive stages. After that, a brief pit scheduling exercise was made.

Further improvements : the pit selected using the present version of the OPD system is by no means a definitive word in the designing process. The details of the in-pit road, for instance, were not considered in the study. The system was not meant to solve this problem, anyhow, the analysis being of a different order of magnitude. The computer-generated pit shells can be tentatively improved by the mining engineers, and the resulting pit shapes, sections and tonnages rapidly obtained. The possibility of making manual changes in the pit configuration can be of assistance in the process of laying-out the haulage road.

A number of applications is potentially feasible, and the practical interest in the present research work will tell, in the future, whether or not it is worth making the suggested upgradings of the system (see chapters 9, 10 and 11).

1.3.3 Areas deserving additional research

The author observed some predominant trends in the literature, as follows :

- The most common use of kriging is for the estimation of grades. Before this step is taken, though, the location of ore and waste blocks is needed, the conventional geological interpretation normally suplying this information. However, when the drilling process is in progress, and the geologist's analysis is not yet ready, a preliminary idea of the spatial position of the ore body can be generated through categorical kriging, provided the geological data is split into two categories, ore and waste, as a function of cut-off limits. The results of the suggested procedure might be of assistance for the geologist(see chapter 7).
- Results obtained with the aid of "automatic" or "optimal" computerized algorithms for open-pit design and scheduling are not always appreciated, in practical terms(see chapters 9 and 10).

More specificaly, in the application of the OPD system to Mutuca Mine, the following observations were made:

a) the algorithm for pit shell generation expands the pit boundary horizontally according to the shape of the ore body in each bench, but after the first lower benches are processed, no more expansions are made, because waste blocks only are reached, in general(see, for example, from figure 9.24 to 9.27).

b) had the algorithm not included some peripheral ore blocks, particularly in the initial lower benches, and probably less waste would have resulted in upper benches, for most pits. The program for pit shell generation could be adapted to offer the flexibility of controlling the number of horizontal expansions, each sector receiving a permitted range for the number of iterations (see item 9.3.3).

c) on the other hand, the kriging error produced by the geostatistical estimation of grades presents interesting characteristics: 1) in the horizontal plane, it increases with greater distances from the centre of the ore body, that is to say, with greater distances from where the geological information is more concentrated(see figure 8.7, for example); and 2) in the vertical plane, it tends to increase with depth, or, in other words, it grows when the superficial information (trenches, adits) is no more within the searching radii, and drilling data only is available, information from drill holes being more scarce in the vicinity of the ore body limits (see figure 8.4, for example).

d) a possible way of penalizing the decreasing level of information of blocks nearing the ore body boundaries would be through the use of the kriging error. Theoretically, the error of estimation is normally distributed around the mean, i.e., a range of possible values is centered in the kriged estimate. Given a limit grade, the probability of observing values below that limit can be calculated for each block, as a function of its kriged estimate and kriging error, through a well-known statistical procedure called "standardization" (see chapter 8). The next step would be the manipulation of a "probabilistic cut-off target", so that the resulting dimensions of the ore body would be under control, because a certain number of blocks, originally considered as ore, would be discarded as waste, as a result of its "prob-values". The proposed method would not only penalize low grade blocks, but those with high kriging errors as well, that is to say, both **quality of ore** and **quality of information** would be taken into account. The probable resulting effect, in terms of open-pit design, would be less ore blocks being picked up in lower benches and in the ore body's periphery, but this

would be compensated by a smaller number of waste blocks lying within the pit boundaries in upper benches, therefore varying the waste to ore ratio.

e) the incorporation of the procedure recommended previously to the OPD system (see item d), in addition to the possibility of controlling the pit horizontal expansions, sector by sector and bench by bench (see item b), could provide the mining engineer with a powerful tool for experimental refinement of the preliminary computer-generated pit shell. The principal advantage of the suggested method would be the possibility of introducing "manual" changes by way of an "automatic" procedure, therefore creating a system essentially "manumatic".

1.3.4 Geostatistics and mining software offered by consulting companies or alike

Some companies and research institutions offering geostatitics and mining software are listed below, together with a brief description of the programs or systems available. These packages are a representative sample of state-of-the-art computer applications in the mineral industry.

BEHRE DOLBEAR & Company, Inc. Minerals Industry Consultants.

New York, NY, USA.

Geostatistics.

CANMET-Canada Centre for Mineral and Energy Technology.

Ottawa, Canada.

DISCODAT, data handling, statistics; Domain analysis programs, definition of domains; Computer manual for seepage analysis; Rotational sheer sliding; Financial computer models; Pit slope manual.

CENTRE de Géostatistique et de Morphologie Mathématique.

Fontainebleau, France.

GEOSMINE(geostatistics); GEOSREC(disjunctive kriging); MULTIPIT(parametrization of the final contour); GEOSVISU(graphical outputs); GJUST(interactive variogram model adjustment); GEOSIMU(conditional simulations of regionalized variables in 3-D); BLUEPACK 3-D(kriging in 3-D); OPD(open-pit design).

COMPUSERVE.

Denver, Colorado, USA.

MEDSYSTEM(R)- Mineral evaluation and design system.

CONTROL Data Corporation.

Minneapolis, Minnesota, USA.

MINEVAL-Mineral Evaluation System.

ENERGY Data Services.

Englewood, Colorado, USA.

COMPUTASC-Exploration, evaluation, mine planning, economics.

GEOSTAT Systems International.

Montreal, Canada.

Geostatistics.

GEOSTOKOS Ltd.

London, England.

Ore reserve estimation, geostatistics.

HANNA Mining Company.

Cleveland, Ohio, USA.

LGMP, Lerchs-Grossmann mine planning system.

INTERNATIONAL Geosystems Corporation.

Vancouver, BC, Canada.

GEOLOG, GEOFORM, GEOCOLEK, GEOSYSTEMS. Geological data base; graphical representation of geological information.

MINERAL Industries Computing Ltd.

London, England.

DATAMINE, data handling, statistics, geostatistics, seam modelling and evaluation, open-pit design.

PROSPECT Mining Services(Control Data).

Transvaal, South Africa.

Mineral evaluation software, geostatistical analysis.

RTZ Consultants.

Bristol, England.

GEOSTAT, geostatistics; OPD, open-pit design; MSS, mine scheduling system.

SNC Group.

Montreal, Canada.

LAMBDA-Deposit Evaluation System.

STEFFEN Robertson & Kirsten.

South Africa.

DMIPS, open-pit mine planning; GEOPAK, geostatistics; STEPS, mine planning.

WRIGHT Engineers Ltd.

England.

Computerized open-pit design.

Many of the above software options for OPD include the in-pit road in the open-pit design.

2. Description of Mutuca Mine

2.1 Brief history
2.2 Geographical situation
2.3 Geological aspects
2.4 Previous reserve estimates
2.5 Mine personnel
2.6 Mining equipment
2.7 Beneficiation plant
2.8 Production
2.9 Transport
2.10 Environmental protection
2.11 Unit consumption
2.12 Product quality
2.13 Market
2.13.1 Lump Ore for Direct Reduction(LODR)
2.13.2 Sinter feed fines
2.13.3 Hematitinha
2.14 Development plan

2.1 Brief history

Mutuca is an open-pit iron ore mine owned by Minerações Brasileiras Reunidas S.A.- MBR, Brazil's second largest iron ore producer and exporter. MBR was established as a subsidiary of CAEMI. Its first activities date back to 1940.

Companhia Auxiliar de Empresas de Mineração - CAEMI is a leading Brazilian industrial group whose chief interests are mining and metallurgy. In addition, CAEMI has been organizing and managing a wide variety of projects in other fields, both in association with foreign investors and on its own.

Mutuca Mine has been operated since 1961. The output of its first plant was in the order of 480 thousand tons per year. Since then, facilities have been upgraded and expanded up to the present capacity of about three million tons per year (3 Mtpy).

The existing beneficiation plant, apart from having some components with many years of use, blocks the exploitation of part of Mutuca's ore body (see photograph 2.1). Therefore, a new installation has been designed.

In 1983 MBR completed the construction at Mutuca of the first stage of a belt conveyor system to carry waste from the mine to a deposition site (see photograph 2.2). Earthwork for the new beneficiation plant was also completed (see photograph 2.3). According to Mutuca development plan, the installation of a semi-mobile primary crushing system will begin shortly.

2.2 Geographical situation

MBR iron ore deposits, Mutuca being one of them, are located in the south central highlands of the State of Minas Gerais, Brazil, near the northwest corner of the region known as "Quadrilátero Ferrifero" (Iron ore quadrangle). Figures 2.1 and 2.2 show the location of Mutuca Mine.

Mutuca is situated in a region that has a mild climate, with temperatures ranging from 5 to 36 Celsius degrees. The average annual rainfall is around 1300mm.

The elevation of the mine bench being mined at present is 1240m above sea level.

Note : This chapter was based on a booklet which had been prepared by MBR in 1984, therefore any changes that occurred later on are not reflected here, but only in subsequent chapters of this thesis.



Photograph 2.1 Mutuca Mine general view



Photograph 2.2 Waste conveyor system



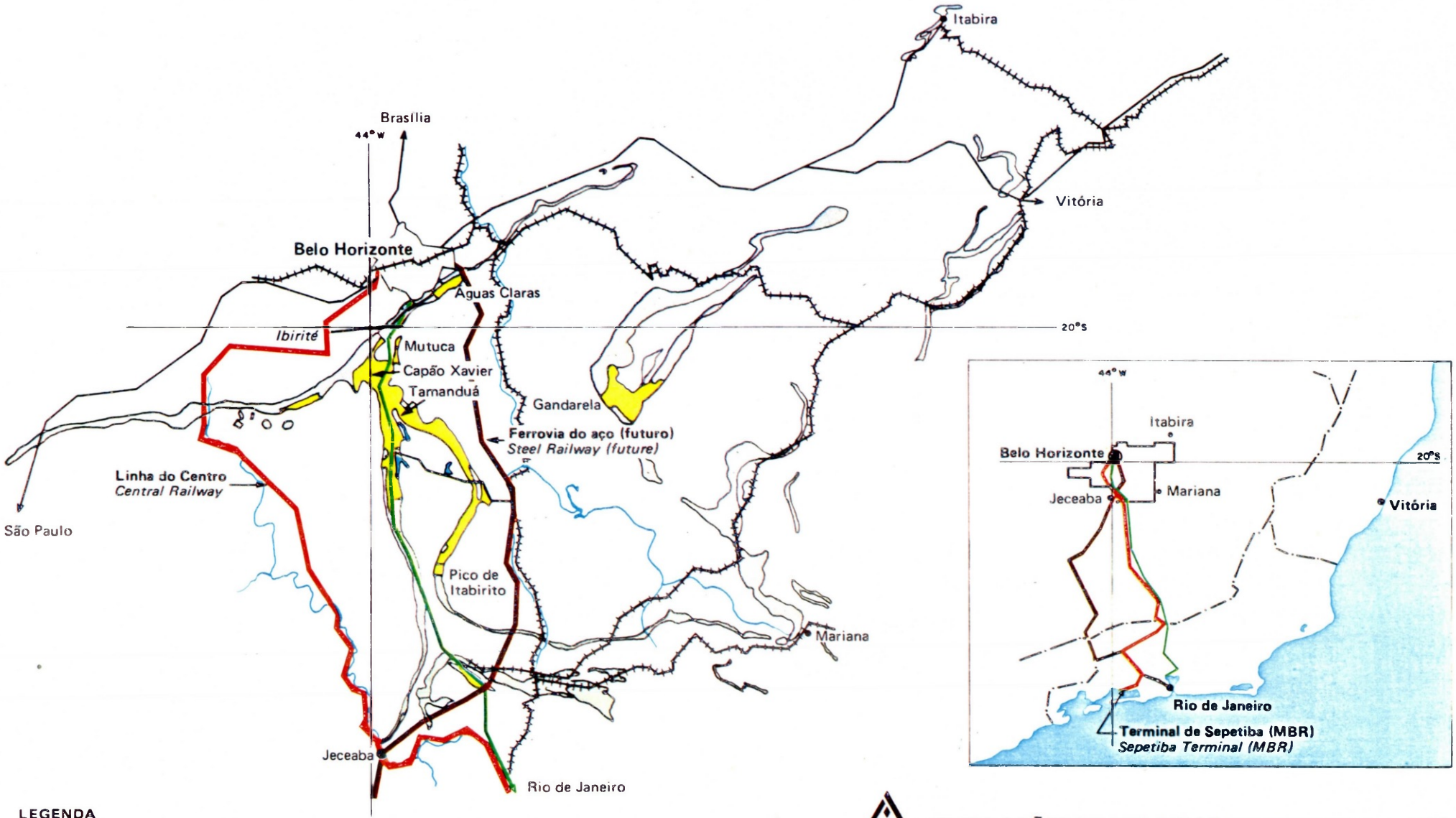
Photograph 2.3 New beneficiation plant site



Photograph 2.4 Mutuca Mine beneficiation plant



Figure 2.1 General location map



LEGENDA
LEGEND

- MBR
- OUTROS/*OTHERS*
- FERROVIA DO AÇO/*STEEL RAILWAY*
- LINHA DO CENTRO/*CENTRAL RAILWAY*
- BR-040 RIO-BELO HORIZONTE/*BR-040 RIO-BELO HORIZONTE HIGHWAY*

MINERAÇÕES BRASILEIRAS REUNIDAS S.A. – MBR

QUADRILÁTERO FERRÍFERO
IRON ORE QUADRANGLE

N

1984

2.3 Geological aspects

The Mutuca deposit lies about 15 kilometres by road south of the centre of the City of Belo Horizonte, the capital of the State of Minas Gerais. Geologically, the deposit is located in the northeast end of the north-south striking Moeda Syncline, a major structural unit about 40 kilometres long that extends between the Curral mountain ridge on the north and the major easterly trending structures on the south side of the Bação Complex. A major thrust fault has displaced the section of the syncline that contains the Mutuca orebody more than five kilometres to the north. As a result, to the west, the iron formation (Cauê Itabirite) is in contact with older rocks of the Rio das Velhas Series (Nova Lima Formation) instead of being in the normal stratigraphical position. To the east, the iron formation is in conformable contact with weathered phyllites of the Batatal Formation.

The Mutuca orebody is intensely folded and strikes N to N 30 degrees W. Dips are 70 degrees SW for most of its north end. Most of the south end dips 60 degrees NE at surface and 70 degrees SW at lower levels. The Mutuca orebody has a general plunge to the southeast and is truncated to the south by a dike of probable diabasic composition.

Ore minerals include hematite with minor amounts of magnetite, goethite, martite and maghemite. Gangue minerals consist mainly of quartz and clay minerals.

Figures 2.3 and 2.4 show geological aspects of Mutuca. For additional details on Mutuca's geology, see Villaça(1980) and Pires(1979, pages 58 through 67). On the broad definition of hematite, see Blyth and de Freitas(1984, pages 83, 85 and 129).

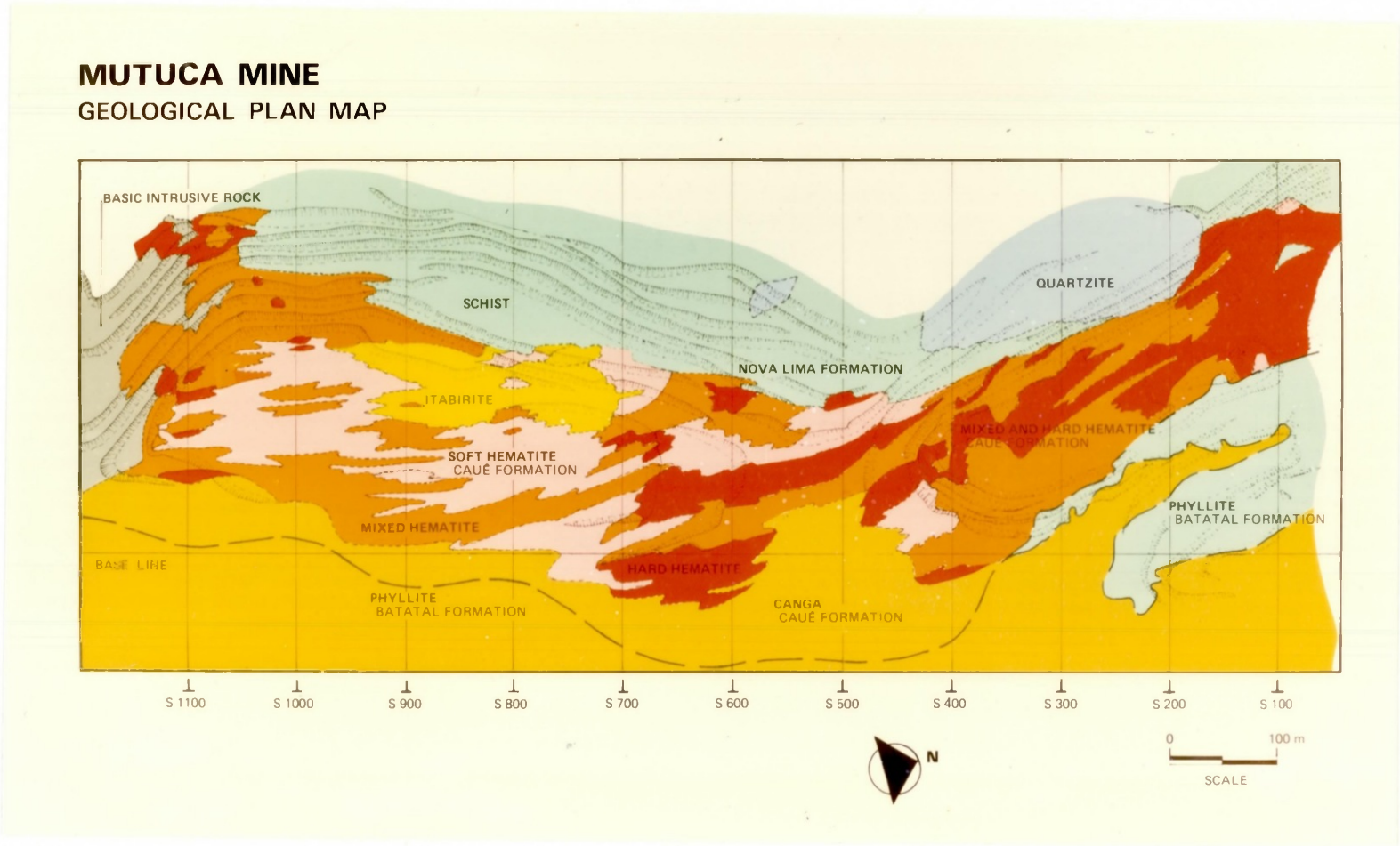


Figure 2.3 Mutuca Mine geological plan map

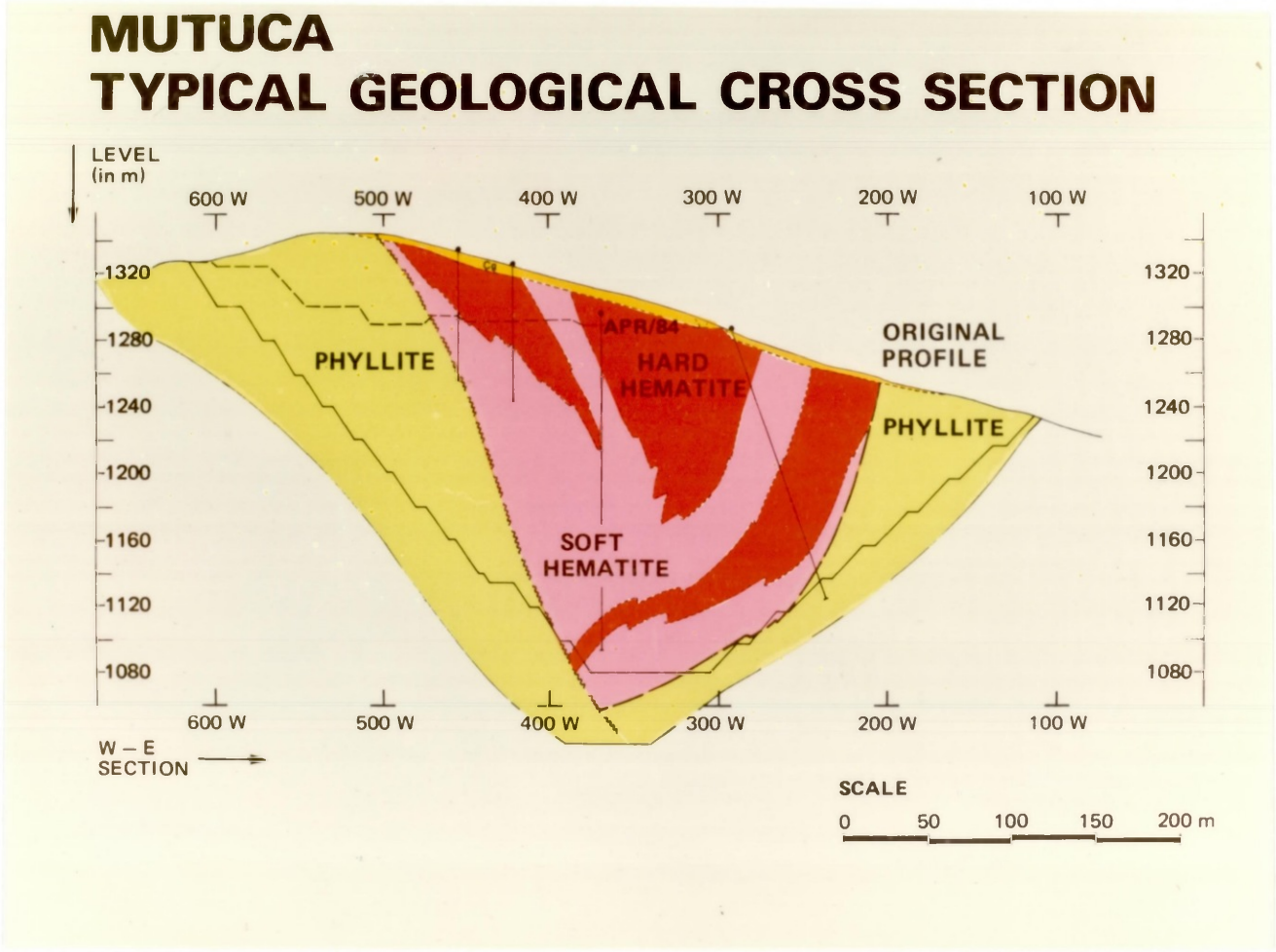


Figure 2.4 Mutuca typical geological cross section

2.4 Previous reserve estimates

The estimated mineable reserve of Mutuca Mine was around 77 million tons in 1981, as shown in table 2.1.

The following notes refer to table 2.1 :

a) Reserve computed by averaging the areas of different materials for each pair of adjacent vertical sections and multiplying them by the interval distance, therefore obtaining the volumes. Tonnages calculated taking into account specific gravities for each material.

b) Distance between cross-sections: 100 metres

c) Maximum pit dimensions : width, 500m ; length, 1200m

d) Lowest pit elevation: 1080 metres above sea level

e) Mine bench being mined at the time: ~1260 metres above sea level

f) Overall pit slopes: west side, 43 degrees; east side, 36 degrees

g) Specific gravities (metric tons/cubic metres):

Hematite (average)..... 3.5

Itabirite..... 3.0

Canga 3.0

Phyllite 1.75

h) Stripping ratio: 0.56 ton of waste/ 1.0 ton of ore

TOTAL MINEABLE RESERVE SECTION BY SECTION (10^3 METRIC TONS)

			HEMATITE - SIZE DISTRIBUTION AFTER -2 INCH CRUSHING										
CROSS SECTION		100	200	300	400	500	600	700	800	900	1 000	1 100	TOTAL
SIZE	TONNAGE	-	1 322.19	3 706.30	9 771.06	11 618.46	13 269.26	11 202.73	6 541.97	7 799.18	6 020.67	5 421.74	76 673.56
- 2 INCH	%	-	19.7	11.9	25.7	20.4	14.4	17.4	8.1	12.5	12.9	13.7	16.1
	Fe	-	68.7	68.5	68.1	68.1	67.6	68.4	68.1	67.7	67.9	67.8	68.0
	P	-	.040	.042	.059	.075	.054	.047	.038	.042	.033	.031	.052
+ 1 INCH	Al_2O_3	-	.73	1.05	.63	.97	1.25	.72	.72	.84	1.28	1.34	.92
	SiO_2	-	.62	.64	.81	.74	1.00	.80	.79	.76	.74	.78	.80
- 1 INCH	%	-	30.2	33.2	34.3	31.6	27.5	26.2	28.9	37.4	31.7	32.0	30.9
	Fe	-	68.8	68.5	68.1	67.6	67.6	68.7	68.1	68.0	68.1	68.1	68.0
	P	-	.033	.044	.052	.064	.052	.040	.028	.032	.037	.037	.044
+ 1/4 INCH	Al_2O_3	-	.72	.93	.72	1.23	1.35	.85	1.11	.97	1.05	1.15	1.04
	SiO_2	-	.58	.77	.75	.81	.96	.90	.75	.84	.87	.71	.83
- 1/4 INCH	%	-	40.5	42.7	27.1	30.3	34.3	34.0	37.1	34.7	35.0	34.2	33.7
	Fe	-	68.4	68.5	68.0	67.2	67.2	67.8	67.9	67.1	67.7	67.7	67.6
	P	-	.033	.037	.041	.055	.057	.031	.026	.033	.031	.026	.039
+ 100 MESH	Al_2O_3	-	1.11	.68	.83	1.46	1.37	.75	.83	.87	1.15	1.38	1.06
	SiO_2	-	.67	.95	.79	1.47	1.41	1.38	1.20	1.64	1.12	.87	1.24
- 100 MESH	%	-	9.6	12.2	12.9	17.7	23.8	22.4	25.9	15.4	20.4	20.1	19.3
	Fe	-	67.8	66.6	65.9	63.6	63.4	65.7	63.8	65.2	65.3	65.5	64.7
	P	-	.034	.052	.054	.091	.092	.050	.072	.056	.048	.048	.066
+ SLIMES	Al_2O_3	-	1.37	.74	1.81	2.68	2.84	1.27	2.71	2.58	2.37	2.27	2.26
	SiO_2	-	.87	1.68	1.41	2.46	2.67	2.15	2.27	2.51	1.89	1.56	2.19
W A S T E	CANGA	83.88	-	-	-	89.12	55.95	176.03	310.52	130.02	81.60	120.32	1 047.44
	ITABIRITE	-	-	-	58.70	-	230.80	82.90	153.56	1 044.37	-	-	1 570.33
	PHYLLITE	273.93	4 210.95	6 417.34	5 281.37	4 990.26	4 110.97	5 509.09	3 548.51	1 782.11	2 179.85	1 779.25	40 083.63
RATIO WASTE (t) / ORE (t)		-	3.18	1.73	.55	.44	.33	.51	.61	.38	.38	.35	.56

Table 2.1 Mutuca mineable reserve as estimated in 1981

In 1984, three years after the estimate contained in table 2.1 had been made, the estimated reserve was around 70 million tons of high iron content hematite, averaging 67% Fe, 0.05% P, 1.2% SiO_2 and 1.2% Al_2O_3 (in situ ore). The waste/ore ratio was 0.50 ton/ton, and the estimated size distribution was as displayed in table 2.2.

Table 2.2 Size distribution of Mutuca mineable reserve in 1984

Size fraction	Distribution (%)	Estimated reserve (in million tons)
-2";+¼"	47	33
-¼";+100 Mesh	33	23
-100 Mesh	20	14

2.5 Mine personnel

Mutuca Mine workforce is 430 people. The mine operates in two production shifts and one maintenance shift. Its work schedule is a follows:

300 days scheduled per year;

16 hours scheduled per day; and

4800 hours scheduled per year.

2.6 Mining equipment

Mutuca is an open-pit mine worked in 10-meter-high benches. Mine ore extraction and haulage is performed by the main equipment listed in table 2.3.

Table 2.3 Mutuca mining equipment

Type	Capacity	Quantity
drills	8 in	1
	6½ in	2
	3 in	2
shovels	6 cu yd	1
	3 cu yd	4
trucks	50 t	5
	35 t	14
front-end-loaders	5½ cu yd	3
	2½ cu yd	2
bulldozers crawler	300 hp	4
bulldozers wheel	400 hp	2
motorgraders	150 hp	2
irrigation tanks	40000 l	2

Source: Rabello; 1985; MBR, Mutuca Mine; personal communication.

2.7 Beneficiation plant

Mutuca has at present two plants. The first crushes, wet-screens, washes and classifies up to 2.3 million tons of iron ore per year (see photograph 2.4). The second, with a capacity of around 0.7 million tons per year, is used to re-screen stockpiled products to new specifications. Figure 2.5 shows the flowsheet of Mutuca Mine. The ore treatment equipment is listed in table 2.4.

Table 2.4 Ore treatment equipment

Type	Quantity
Primary jaw crushers	2
Primary screens	1
Secondary screens	2
Secondary cone crusher	1
Tertiary crusher scalping screens	2
Tertiary cone crusher	2
Tertiary screens	2
Spiral hydroclassifiers	2
Thickener	1
26" cyclones	2

2.8 Production

In 1984, the total production for the Mutuca Mine facilities - crushing and sizing - was around 3 million tons per year. Approximately ⅓ of that production was Lump Ore for Direct Reduction(LODR) and the remainder was split between sinter feed fines and "hematitinha", a special type of ore fed into domestic pig iron furnaces using charcoal.

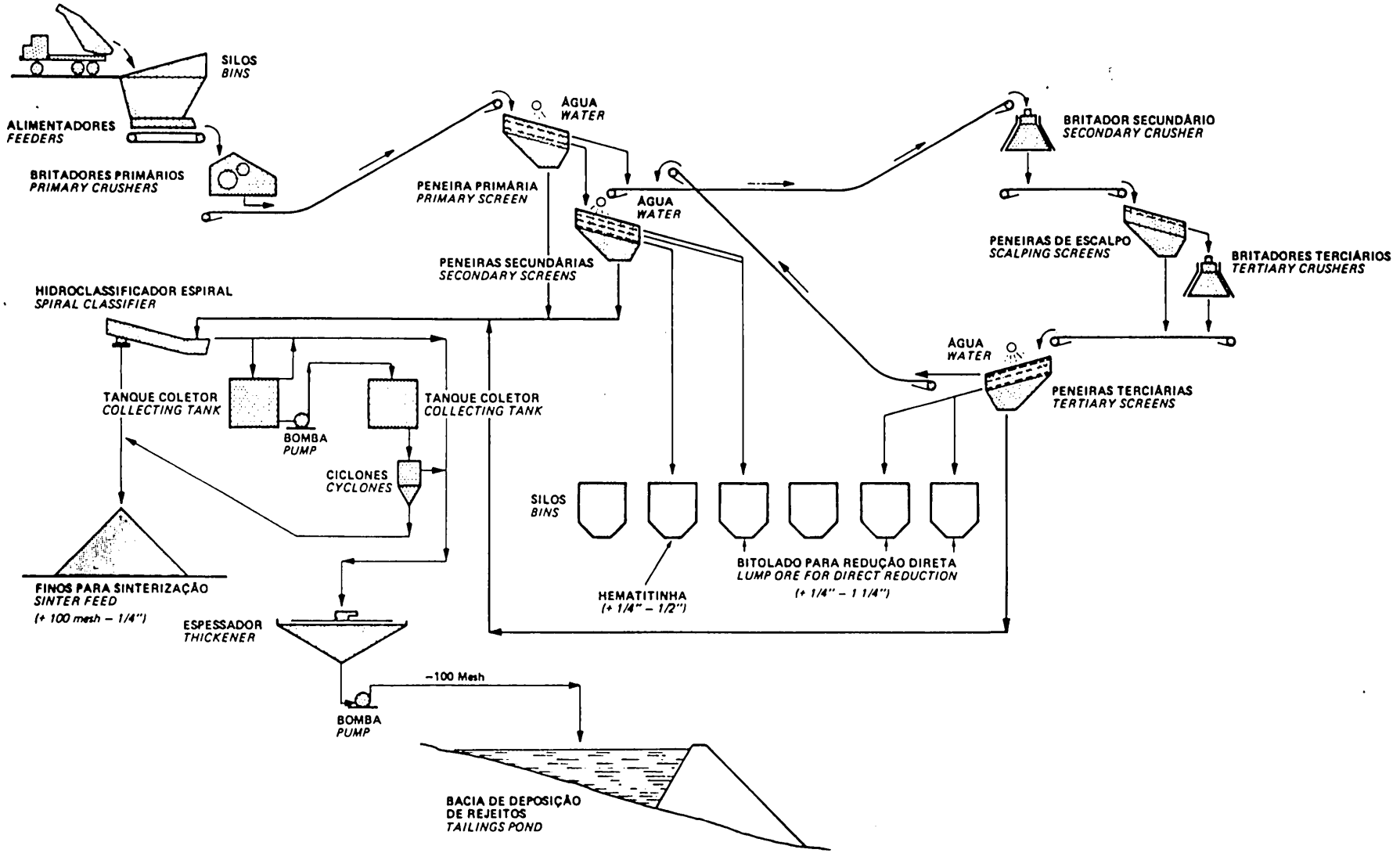


Figure 2.5 Mutuca Mine flowsheet

MINERAÇÕES BRASILEIRAS REUNIDAS S.A. – MBR

MINA DA MUTUCA – FLUXOGRAMA
MUTUCA MINE – FLOWSHEET

1984

2.9 Transport

Off-the-road trucks are used for in-pit haulage in Mutuca Mine. After its beneficiation, most of Mutuca ore is routed to the near by Águas Claras Mine by means of regular trucks via the existing road network.

From Águas Claras Mine, the ore is transported through the "Central Railway" to the "Sepetiba Terminal". Central Railway is the main line of the Brazilian state-owned "Rede Ferroviária Federal S.A. - RFFSA", and Sepetiba Terminal is an iron ore shiploading facility owned by MBR (see figure 2.1) with capacity to receive ships of up to 300,000dwt(dead weight tons).

2.10 Environmental protection

MBR's environmental engineering division monitors and controls erosion, dust, soil sliding, vibration, noise and water/air pollution, and keeps equipment on standby to cope with accidental oil spillage from vessels at the marine terminal.

Slope stability criteria have dictated a disposal plan conceived to meet strict environmental protection requirements at Mutuca Mine waste deposit, including a complete drainage system, slope covering and an ecological dam. Steps have been taken since the beginning of the earthwork to protect the environment around the future Mutuca beneficiation plant site.

2.11 Unit consumption

In 1984, the average consumption of explosives was 0.07 kg/t of extracted material, including ore and waste.

In the same period of time, the electric power required per fed ton was 1.75 kWh. This figure is likely to change whenever the new beneficiation plant comes into operation.

At that time, the fuel consumption was 0.36 litres per hauled ton. A reduction in this rate is expected after the installation of the new semi-mobile crushing system.

The average productivity of trucks is shown in table 2.5 .

Table 2.5 Productivity of 32.5-ton-trucks in 1981

Type of material	Distance (metres)	Travel time (min.;sec.)	Productivity (t/h)
Ore	1000	14"30'	145
Waste	1600	17"30'	120

The transport distances within the mine shall not exceed 1000 metres with the future utilization of the semi-mobile crushing system.

2.12 Product quality

Tables 2.6 and 2.7 show chemical, physical and metallurgical characteristics of Mutuca products, as estimated in 1984.

Table 2.6 Typical chemical analysis of Mutuca products in 1984 (in %)

Product	Size	Fe	Al_2O_3	SiO_2	P	Mn
Lump Ore for Direct Reduction	-1¼";+¼"	>67	<1.0	<1.0	<0.05	<0.20
Sinter feed fines	-¼";+100Mesh	>67	<1.2	<1.0	<0.05	<0.40
Hematitinha	-½";+¼"	>67	<1.4	<1.0	<0.05	<0.40

Table 2.7 Typical physical and metallurgical characteristics of Mutuca Lump Ore for Direct Reduction

Test	Average
Porosity	16.0
Crepitation (- 5 mm)	5.5
Tumble index (+ 6.35 mm)	87.0
Abrasion index (- 595 micra)	7.2
Metallization (Midrex Linder Test)	92 - 96
Reducibility (JIS)	53 - 60

2.13 Market

2.13.1 Lump ore for direct reduction

Mutuca Lump Ore for Direct reduction (−1¼";+¼") has been used by steel mills employing the direct reduction method worldwide, as shown in table 2.8.

Table 2.8 Consumers of Mutuca Lump Ore for Direct Reduction

Mill	Country	Process
Acindar	Argentina	Midrex
Dalmine	Argentina	Midrex
Georgetown	USA	Midrex
Hylsa	Mexico	HYL
Iscott	Trinidad	Midrex
Nordferro	West Germany	Midrex
Sabic (Hadeed)	Saudi Arabia	Midrex
Sidbec	Canada	Midrex

2.13.2 Sinter feed fines

MBR's sinter feed fines (−¼" ; +100 Mesh), with an iron content of about 68%, is suited for the production of sinter feed for blast furnaces. The Far East is one of its principal consumers.

2.13.3 Hematitinha

Hematitinha (−1½"; +¼") is a special type of ore fed into domestic pig iron furnaces using charcoal.

2.14 Development plan

Given the importance of this mine, the company plans to intensify mining operations to a level of 6.5 million tons per year.

The development plan, which is represented in figures 2.6 and 2.7, as well as in photograph 2.5, consists of the following :

- Acquisition of mining equipment to carry out open-pit mining at the desired level;
- Construction of an ore treatment plant at a different site, to enable exploiting part of the mineral reserve presently blocked by the existing plant;
- Construction of a long-distance aerial ropeway, which will transfer output to a new loadout station;
- Construction of a storage yard, with a train loadout station located at Olhos d'Água, approximately 2 kilometres from Mutuca Mine; and
- Construction of a railroad loop at Olhos d'Água.

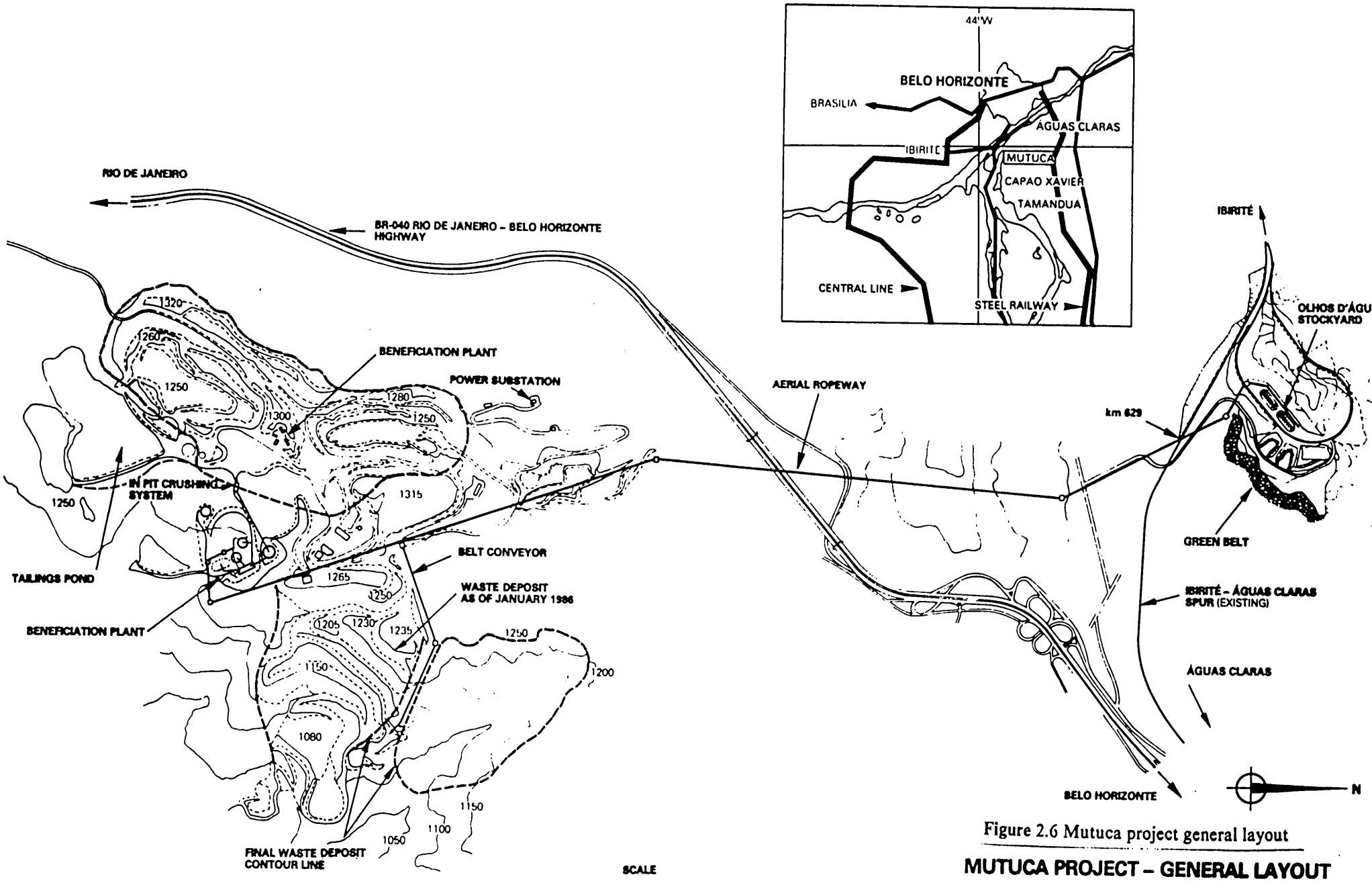
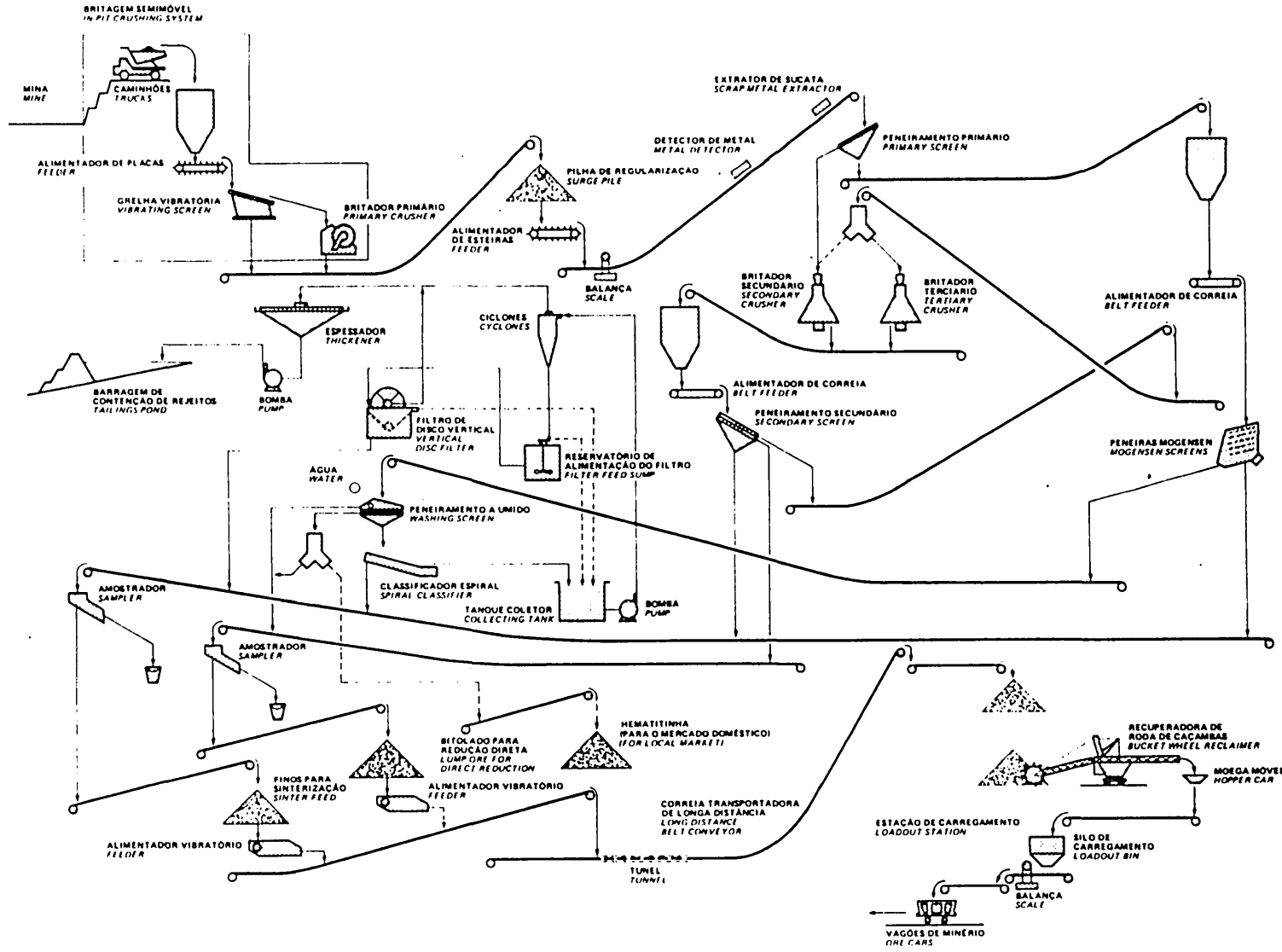


Figure 2.6 Mutuca project general layout

MUTUCA PROJECT – GENERAL LAYOUT

1986



MINERAÇÕES BRASILEIRAS REUNIDAS S.A. – MBR

PROJETO MUTUCA – FLUXOGRAMA
MUTUCA PROJECT – FLOWSHEET

1984

Figure 2.7 Mutuca project flowsheet



Photograph 2.5 Mutuca Mine development plan (artist's conception)

3. Geological sampling, core logging and assaying

3.1 Drilling Method

3.2 Core logging

3.3 Definition of sampling intervals

3.4 Sample preparation

3.5 Physical and chemical analyses

3.6 Data recording

3.7 Geological classification

3.8 Description of ore, rock types and codes

3.9 Description of the ore types and their codes

3.10 Remarks on the geological map and sections

3.1 Drilling method

In early stages percussion drilling was used in Mutuca, despite its limitations. The material recovered by this method can not be analysed other than for identification of some crude ore chemical characteristics. As no information on the nature of the size fractions is obtained, MBR discontinued percussion drilling, adopting the rotary drilling method instead.

The use of diamond drill holes allows the geological exploration team to recover bore hole cores for detailed inspection. As a consequence, the geological structure of the ore can be studied and each size fraction can be individually analysed as for chemical, physical and metallurgical characteristics.

Both Minerações Brasileiras Reunidas S.A.-MBR and its contractors currently use the diameter HX (86mm - 96mm) for rotary drilling in iron ore. In case operational problems arise, or if a high consumption of crowns is noticed, or the loss of drilling mud is detected, the diameter is reduced to NX (66mm - 86mm). Diameters smaller than NX are not used by MBR for pre-production exploration.

The total core recovery is normally high, for diamond drill holes being bored in both friable and compact iron ore. As for drilling in either mixed or fractured iron ore, the recovery might go down, sometimes to 20% or even less.

3.2 Core logging

After completion of the bore hole, its core is geologically described through measurement of the layers; identification of faults, fractures, joints and shearing and schistosity planes; description of structure, mineralogy, contaminating elements, gangue constituents, granulometry, and degree of weathering; and measurement of the total core recovery for each interval length.

Figures 3.1 and 3.2 are two examples of Mutuca bore hole logs.

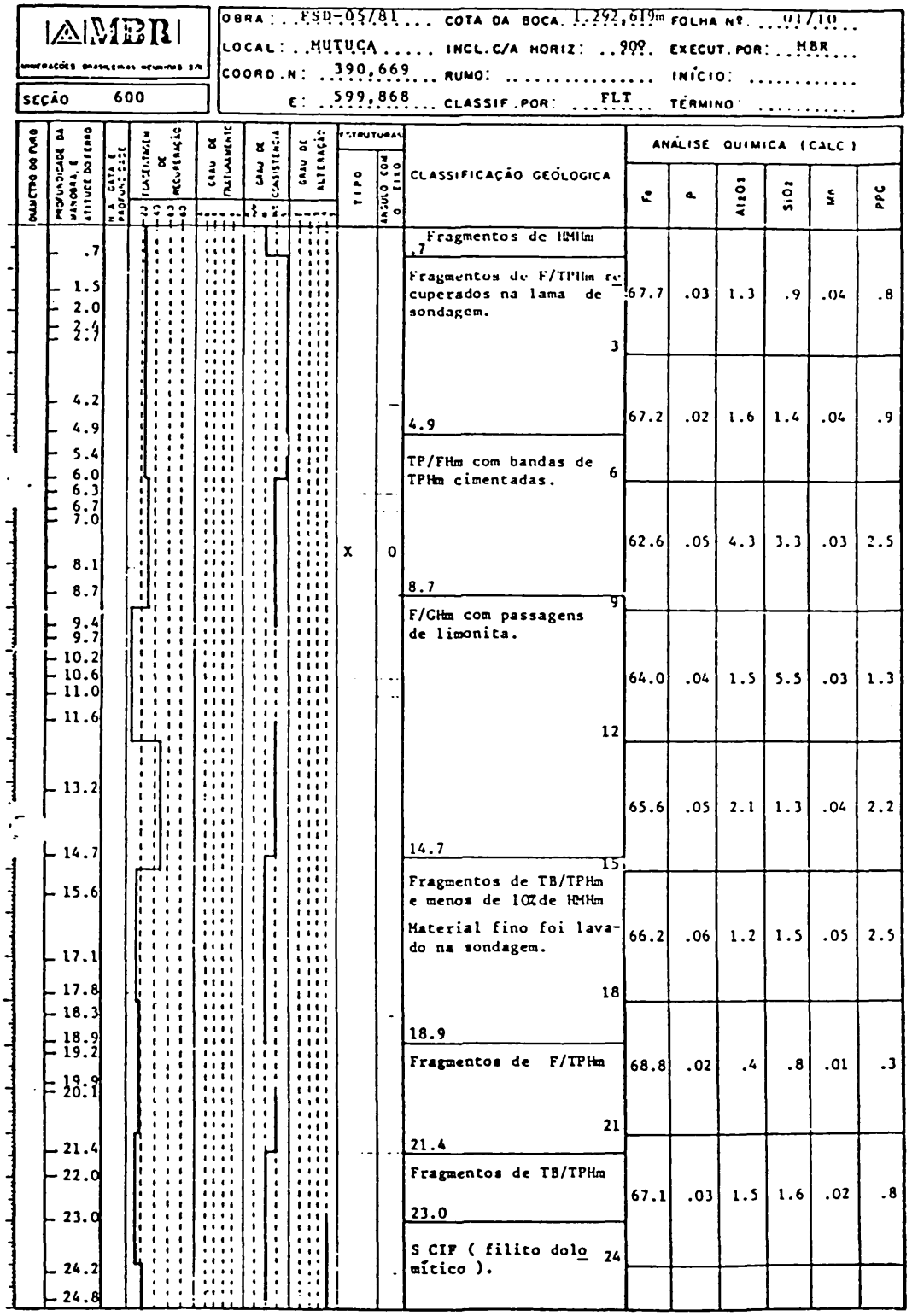


Figure 3.1 Mutuca bore hole log(example 1)

FURO DE SONDA Nº 7/76 FOLHA Nº 1 DE 2 FOLHAS

JAZIDA MUTUCA SEÇÃO Nº 900

LOCALIZ. 70.690.059 N 12.468.250 E ELEVAÇÃO 1.300.57

AZIMUTE 60º W 57.30 m W da L. B.

INICIO OUTUBRO/76 TERMINO DEZEMBRO/76 REVESTIMENTO —

EXECUÇÃO M B R GEOLOGIA OSCAR TESSARI

MATERIAL	METROS	RECUPERAÇÃO %	DIÂMETRO	ANALISE QUIMICA (SÊCO) % Fe	P	SiO_2	Al_2O_3	Mn
Cg Rica (TPHm/HMHm)	0.0	33	HX	64.1	0.099	1.70	2.38	0.090
TPHm ≅ 15% TPHm merg. 40º a 5.4 m 5.3 - 5.4 m - HMHm	4.5	19	HX	62.1	0.030	1.33	6.40	0.090
SIb argiloso rico 30.8 - 32.3 - GHm/TPHm 32.3 - 35.5 - não recuperado.	19.2	20	HX	63.8	0.017	4.18	2.64	0.113
TPHm ≅ 10% TBHm 41.5 - 47.1 - TPHm cont.argila vermelha. 48.3 - 48.5 / 48.8 - 49.3 } HMHm merg. 65º a 49.0 m fratura/merg.40º a 48.9 m	41.5	32	HX	67.5	0.020	0.94	1.55	0.131
TPHm ≅ 10% TBHm 64.5 - 65.3 / 75.5 - 76.0 } Intrusiva amarela. 67.5 - 67.9 / 70.7 - 72.0 / 72.7 - 73.0 / 76.0 - 77.0 } HMHm/TBHm merg. 70º a 72.8 m	60.2	55	HX	67.2	0.022	0.93	1.93	0.251
GHm 80.5 - 85.2 / 87.2 - 91.1 / 92.2 - 93.7 / 97.0 - 100.0 } não recuperado Granulometria duvidosa devido à baixa recuperação.	80.5	07	HX	67.3	0.012	1.57	1.30	0.205
GHm 114.0 - 115.6 - TPHm/GHm	100.0	33	HX	68.4	0.017	0.94	0.57	0.209
TPHm/GHm ≅ 20% TBHm 120.5 - 120.8 - HMHm	118.7	85	NX	68.6	0.010	1.01	0.46	0.146

Figure 3.2 Mutuca bore hole log(example 2)

3.3 Definition of sampling intervals

The next step is the division of the core into segments for material collection. MBR used sampling intervals 3m long in Mutuca in early geological exploration campaigns . A different procedure, based on criteria of geological uniformity, had been used by MBR in the seventies. At that time, 25m or even longer segments were used. A few years ago, MBR resumed the use of 3m sampling intervals, following the decision to apply geostatistical analysis in the ore reserve evaluation of Águas Claras Mine, one of its mineral properties. The possibility of using longer sampling intervals in Águas Claras Mine is currently under study, as the semi-variogram analysis of that mine has been completed.

3.4 Sample preparation

In the past, the whole recovered core in each interval was used for determination of granulometry, that is to say, all the material used to be crushed and screened. Nowadays, the core is split into two halves by means of an electric saw, one half being stored for future reference, the rest being crushed to −2" and sent for assaying.

3.5 Physical and chemical analyses

Following the sample preparation, the material is wet screened to 1", ¼" and 100 Mesh. The resulting four size fractions are used for the purpose of geological classification, as will be explained later in the text.

A representative sample is extracted from each size fraction, for wet chemical analyses. Normally, Fe, P, SiO_2, Al_2O_3, Mn and LOI (loss on ignition) are analysed. The results of these assays are compared with the correspondent geological description made by the geologists, checking for obvious errors.

3.6 Data recording

Chemical and physical analyses are recorded onto a standard form. Chemical analyses of each size fraction are then multiplied by their weight proportion and the weighted average is computed for each mineral (Fe, P, Al_2O_3 and SiO_2), in order to obtain the ultimate crude ore chemical analysis. This procedure is followed for each sampling interval, that is to say, for each length of 3m (or more) of bore hole core.

3.7 Geological classification

MBR adopts a geological classification based on both chemical analysis and granulometry. Tables 3.1 and 3.2 show the limits for classification of materials into Hematite, Rich Itabirite and Rich Argillaceous Iron Formation.

Table 3.1 Classification by chemical analysis

Limits(%)	Classification	Code
64.<=Fe;P<0.12	Hematite	HEMAT
45.<=Fe<64.;Al_2O_3<4.	Rich Itabirite	RIB
45.<=Fe<64.;Al_2O_3=>4.	Rich Argillaceous Iron Formation	RAIF

Table 3.2 Classification by granulometry

Limits(%)	Classification	Code
50.<= +1"	Hard	H
35.<= +1" <50.	Hard/Mixed	HM
+1" <35. ; 50.<= +¼"	Mixed	M
35.<= +¼" <50.	Mixed/Soft	MS
+¼" <35.	Soft, Very Soft	SVS

Other rock types and formations are classified as indicated in table 3.3.

Table 3.3 Classification by granulometry and grades

Limits(%)	Classification	Code
64.<=Fe ; 50.<= +¼"	Debris Ore	DO
64.<=Fe ; 0.12<=P	Phosphatic Hematite	PHEM
Fe<45. ; Al_2O_3<4.	Itabirite	IB
Fe<45. ; 4.<=Al_2O_3	Argillaceous Iron Formation	AIF

In addition to the rock types described above, other materials are represented on Mutuca geological cross-sections, as follows:

Carbonate Iron Formation(CIF);
Gangue;
Phyllite;
Quartzite;
Basic Intrusive Rock(RBI); and
Overburden.

3.8 Description of ore, rock types and codes

Listed and described below are the abbreviations or codes applied to mappable units of iron ore, iron formation, and associated rock types in the "Quadrilátero Ferrífero"(Iron Ore Quadrangle), Minas Gerais, Brazil.

HMHm

- Hard, blue, tough, massive or thick banded hematite. Bands are five centimeters or more thick and commonly one-half meter to three meters in thickness. Indistinct bedding in finely etched laminations is commonly apparent on the weathered surfaces. Freshly broken faces show an even, fine to very fine grain size and have an irregular to semi-conchoidal fracture.

  In mining cross-section, HMHm is indicated as HO, HO/MO, etc., depending on results of screening tests in the laboratory.

  Note: Hematite is formed from itabirites and ferruginous carbonates by metasomatism and supergene enrichment and has, by definition, more than 64% iron and less than 0.12% phosphorus.

TBHm

- Banded, hard to medium-hard hematite with bands two to five centimeters thick and having a schistose or blocky fracture.

  In mining cross-sections, stratigraphic intervals of TBHm or interbedded TBHm and finer structural ore types are classified and indicated as HO/MO, MO, etc.

TPHm

- Thin platy hematite with beds one to two centimeters thick. Breakage occurs preferentially along bedding planes. Visible silica or clay minerals may occur between the bands, but totalling less than three percent in weight.

  In mining cross-sections, stratigraphic intervals of TPHm are classified as MO/SO, or SO depending on results of laboratory tests. Where closely interbedded units of TPHm or HMHm are present, an MO or even HO MO type may result.

FHm

- Soft, black, flakey hematite with bands of hematite less than one centimeter thick. The structure is micaceous and somewhat like graphite in appearance. Specularite is the principal mineral and grain size varies from fine to coarse. Bedding structure may or may not be visible in undisturbed exposures.

FHm intervals are classified as SO or VSO for mining, unless interbedded with the coarser structured ores to yield a screen product of 35% +¼" in which case the interval is classified as MO/SO.

GHm

- Soft, black, fine to medium-grained granular hematite much like FHm in texture, except for the general absence of micaceous hematite. GHm intervals are classified as SO or VSO for mining unless interbedded with coarser structured ores.

Ib

- Itabirite, a metamorphosed iron formation composed of alternating laminae of hematite and granular quartz. Grades are commomnly about 35% - 45% iron and, by definition, must be less than 64% iron. Where the iron content of the itabirite is more than 45%, the prefix "r" is applied to denote rich itabirite. In addition, the same prefixes which are used to differentiate the various types of hematite apply to the itabirites as well.

AIF

- Argillaceous iron formation consists of alternated bands of hematite and dull earthy goethite.

Individual bands are about two centimeters or less in thickness. Sub-surface exposures show the earthy goethite bands, as seen at surface, to grade downward into partially decomposed material that contains increasing amounts of clay. Argillaceous iron formation may occur interbedded with and gradational into itabirite or carbonate formation and, by definition, contains less than 64% iron and more than 4% alumina. Where the iron content of the argillaceous iron formation is more than 45% the prefix "r" is applied to denote rich argillaceous iron formation. The same prefixes which are used to designate the various types of hematite are also applied to the argillaceous iron formation.

CIF

- Carbonate iron formation is a reddish-brown to yellowish-brown facies of iron formation in which the individual beds vary in thickness from less than one millimiter to ten centimeters. The hematite is black and brittle and appears to be partially limonitized in the surface exposures.

  On a fresh break the hematite has a satiny appearance. Bands of goethite which occur near the surface between hematite bands have replaced a carbonate mineral, possibly dolomite, ankerite or siderite.

DO

- Debris, defined here as a mass of loose to partially cemented hematite fragments, mainly larger than five centimeters in diameter, mixed with various amounts of hematite gravel and iron-rich soil. The hematite fragments may be angular to well rounded. This category includes talus and float ores.

HmCg

- Hematite canga, in which the plates and fragments of hematite, making up more than 70% of the rock, show little or no orientation and are cemented by approximately 10% goethite. This unit resembles a ferruginous conglomerate and always occurs at or near the surface.

StCg

- Structural canga containing bands or plates of hematite, itabirite or goethite having a preferred orientation and cemented together by approximately 10% goethite. This type of canga generally reflects the composition of the underlying bedrock and accordingly StHmCg and StIbCg can be used to indicate parent material.

GoeCg

- Goethite canga in which the plates and fragments of hematite, itabirite or other rock types are cemented together by 50% or more goethite.

LimCg

- Limonite canga in which the plates and fragments of hematite, goethite, itabirite and other rock types are cemented together by 50% or more limonite. This unit generally overlies argillaceous iron formation.

LatCg

- Earthly limonitic canga with a relatively smooth surface which commonly contains substantial amounts of clay and partially decomposed rock fragments. Normally this type of canga overlies lean iron formation.

Phy

- Phyllite. This unit, stratigraphically underlying the itabirite, the Batatal formation, is normally a light-to-dark grey, very fine grained, usually schistose rock. The phyllite generally forms smooth slopes and valleys between itabirite and quartzite ridges.

Qtz

- Quartzite, which commonly forms prominent ridges on both sides of the iron formation.

RBI

- Basic intrusive rock probably of diabasic composition. Parts of this unit have been metamorphosed to phyllite or schist.

3.9 Description of the ore types and their codes

The classification of metallurgical ore types and economic products adopted by MBR is based on crushing and screening of representative samples obtained from large-diameter core holes, adits and mine benches. This classification is not to be confused with the geological mapping terms presented previously, which are based on visual classification. The criteria adopted for classifying the material on grounds of friability were described in items 3.4, 3.5 and table 3.2. Table 3.4 contains a detailed description of the ore types and its codes.

Table 3.4 Description of the ore types
on grounds of friability

Code	Ore description
H	Hard, massive, compact or thick-bedded slabby
HM	Hard, thick-bedded, with schistosity or thin bedded, blocky
M,MS	Medium hard to friable, thin bedded and schistose or thin platey with some soft ore
SVS	Soft friable, flakey or granular and schistose thin platey hematite
DO	Float or debris pebbles, cobbles, etc.

3.10 Remarks on the geological map and sections

The following comments are related to the Mutuca Mine geological plan map and geological cross-sections.

- Most channel samples are projected in sections, giving sometimes the false impression that they are located below the present mine level.

- Schists and phyllites were grouped in the same lithological unit as both are produced from pelitic rocks under conditions of low grade metamorphism.

- Only drill holes projected over 3.0 meters in sections are dashed, otherwise they are represented by full lines.

- The 1966 percussion drill holes, in addition to the drill holes DH-44, DH-45, DDH 1/77 and DDH 2/77 of the Mutuca Mine, either had not been chemically assayed or had had extremely low recovery therefore were not represented in the sections.

- "Ib" is a generic term used in areas of insufficient geological information to represent VSIb (Very Soft Itabirite), SIb (Soft Itabirite), MIb (Medium Itabirite), and/or HIb (Hard Itabirite). In the same manner, "AIF" (Argillaceous Iron Formation) stands for VSAIF, SAIF, MAIF and/or HAIF; and "CIF" (Carbonate Iron Formation) represents VSCIF, SCIF, MCIF, and/or HCIF.

- The prefix "P" stands for phosphorus and is applied to any metallurgical ore type (phosphatic hematite) to denote ore bands or lenses having P>=0.12%.

- "Cg" is a generic term used on sections to represent the various canga types (Hm Cg, St Hm Cg, St Ib Ch, St Cg, Goe Cg, Lim Cg, Lat Cg) shown separately on geological maps.

- "Overburden" is a generic term applied here to represent all kinds of capping material, except canga, that would have to be stripped in order to extract the ore.

4. Computerized geological data base

4.1 Summary and conclusions

4.2 Introduction

4.3 Suggested approach for Mutuca Mine

4.4 Input data

4.5 Computerized geological data base

4.6 Suggestions and recommendations

4.1 Summary and conclusions

This chapter starts with an introduction on computerized geological data bases, in general, and proceeds with a brief description of the suggested approach for Mutuca mine. Subsequently, the input data is listed and the resulting computerized geological data base is presented. Finally, suggestions and recommendations for future improvements have been made.

The main conclusions of this chapter are as follows :

- the geological data base is the fundamental information for any ore reserve assessment exercise, therefore it is essential to keep it up-to-date with the latest geological campaigns and ready to be promptly used.
- the computer is a very useful tool for quick and easy data entry; data checking, storage and retrieval; and for additional processing, such as graphical representation(see chapter 5) and statistical analysis(see chapter 6), which are invaluable means of describing, mapping and checking the geological data.
- in the present case, where a fairly large number of sample assays had been generated from previous exploration campaigns and a new geological campaign was proceeding, the computer was the best way of handling the data.
- in addition to the above, the geostatistical analysis and block kriging exercises that were to be undertaken(see chapters 7 and 8) determined that the creation of a computerized geological data base was mandatory.

4.2 Introduction

This chapter, although the shortest of this thesis, describes perhaps the most laborious part of the whole research work, i.e., producing, collecting and organizing the geological data. Computer users have a saying, about the relevance of thoroughly checking the input data "GIGO - garbage in, garbage out"...

Barnes(1980, from page 32 to 40) gives a comprehensive description on the management of sample data, from recording and checking input data up to compositing intervals, as transcribed below.

"The mineral sample is the fundamental unit upon which the mineral appraisal is built. Values derived from the sample are the most important quantitative measurements made in the total evaluation process. The sample value or values, together with the description of its precise location in three dimensions, are basic elements of any mineral

appraisal."

"Mineral sampling, regardless of type, is a costly process. It is important that all useful information be recorded during the sampling program if the investigator is to get full value for money spent."

"The following information is the minimum data that should be recorded for mineral samples in order to ensure maximum usefulness of the sample."

1) Sample identifier
   a) Project, mine, deposit, etc.
   b) Type and size of sample
   c) Sample number
2) Coordinates of samples in 3-D space (X, Y, Z)
   a) Northing
   b) Easting
   c) Elevation
3) Assay or measured values for sample
   a) One or several fields as may be required, i.e., gold, copper, moly, etc.
4) Other sample modifiers or classifiers :
   a) Geological environment, i.e.,
      - Host rock type.
      - Structural relationship; vein, etc.
      - Leached zone, enriched zone, etc.
   b) Physical environment.
      - Sample along drift or crosscut, etc.

"This type of information is often necessary in order to separate various sample populations and thus preventing population mixing that often obscures significant assay value relationships."

On recording of drill hole sample data, specifically, the following information is added to the above(see Barnes, 1980, page 34):

- bearing and inclination of the hole at collar;
- sample length on a "from-to" basis; and
- sample recovery.

Godwin(1977) presented the "GEOLOG System"(GLS), which is a comprehensive computer-based management scheme for collection and analysis of all types of exploration-development data such as :

1) geological data from drill holes, trenches, mine workings and detailed surface traverses;

2) survey data related to the above; and

3) analytical data from drill holes.

The outputs of that system include printed logs called GEOLIST and coloured graphical plots called GRAFLOG Plan, Strip and X-Section. Facilities such as data entry in the field through minicomputer floppy diskettes and interactive programs for analysis and display of the data, in the office, are available in the GEOLOG System.

Lefeuvre(1984) presented GDM, a geological data base management system for geostatistical applications, which is capable of producing drill-hole logs through the computer plotter, incorporating a graphical representation of the different rock types, as well as drill-hole sections showing geologic patterns and grades.

The following topic describes the procedure adopted for handling Mutuca's geological data.

4.3 Suggested approach for Mutuca Mine

The geological data contained in the bore hole logs(see chapter 3) are assigned to points which have their exact location specified by an X, Y, Z coordinates system. If a drill hole is to be taken as an example, the location of its collar is given by easting, northing and elevation, and its inclination is obtained through two angles: **azimuth**, which is taken clockwise from North in a horizontal plane; and **dip**, measured from a horizontal line, in a vertical plane containing the longitudinal axis of the hole. Along the bore hole, the location of each sample is usually given on a "from-to" basis.

Computer programs were developed to convert the above information into the coordinates of the geometric centre of each individual sample, being it originated from a drill hole, a channel or an adit. For a program on "bore hole desurvey calculation", see Howson and Sides(1986).

Due to the particular orientation of Mutuca's system of coordinates and the geological classification adopted by MBR, as well as the required compositing of the (−2";+1") and (−1";+¼") size fractions into the (−2";+¼") one, tailor-made programs were required.

4.4 Input data

Most of Mutuca's geological data have been entered into a computer file and debugged by MBR technical staff, in Brazil. The file has been recorded into a magnetic tape and sent to the researcher in England. Only a few of the more recent drill holes have been entered into the computer at Imperial College, with direct participation of the student.

The data shown in figures 3.1 and 3.2 are typical examples of the geological information recorded into the computer data file. Figures 4.1 and 4.2 show examples of computerized geological information.

Tables 4.1 and 4.2 summarize the geological information which was available for use in June, 1985, before the results of the more recent geological campaign were also incorporated to the data file. Both the geostatistical analysis and the block kriging results were produced with the data of 12 additional diamond drill holes(DDH's), which were available in August, 1985 (DDH's 1/85 to 12/85).

```
MUT DDH 01/84    100 546.619  100.000 1239.7 278.01 60.
  0.0    6.0   57.0 1310
   -.100    -.10     -.1    -.1 -.100   -.10   -.10   -.100   -.10
  20.690   20.69    52.5   67.5   .075   1.10   1.62    .046    .61
   1.300   13.62    34.5   66.7   .067    .86   2.47    .191    .58
    .490    5.13    13.0   51.1   .227   8.43  10.60   1.700   4.60
  22.480   39.44   100.0   65.1   .092   1.97   3.08    .311   1.12
                           67.4   .052   1.11   1.78    .068    .68
  6.0    9.0   67.0 1310
   -.100    -.10     -.1    -.1 -.100   -.10   -.10   -.100   -.10
  12.910   12.91    66.8   67.8   .068    .98   1.51    .047    .67
    .033    5.57    28.8   67.1   .079    .85   1.94    .096    .63
    .005     .84     4.4   56.7   .123   4.45   8.54    .536   4.59
  12.948   19.33   100.0   67.1   .074   1.09   1.94    .082    .83
                           67.8   .076    .91   1.43    .058    .63
  9.0   12.0   56.0 1610
   -.100    -.10     -.1    -.1 -.100   -.10   -.10   -.100   -.10
   3.520    3.52    11.2   68.2   .052    .89   1.33    .058    .49
   8.910   17.67    56.4   68.6   .024    .55   1.11    .116    .30
   5.125   10.16    32.4   57.1   .267   4.84   4.36   3.580   5.15
  17.555   31.35   100.0   64.8   .106   1.98   2.19   1.232   1.89
                           59.4   .196   4.17   3.54   2.610   3.85
 12.0   18.0 100.0 2031
   -.100    -.10     -.1    -.1 -.100   -.10   -.10   -.100   -.10
   -.100    -.10    33.3    -.1 -.100   -.10   -.10   -.100   -.10
   -.100    -.10    33.3    -.1 -.100   -.10   -.10   -.100   -.10
   -.100    -.10    33.3    -.1 -.100   -.10   -.10   -.100   -.10
   -.300    -.30   100.0    -.1 -.100   -.10   -.10   -.100   -.10
                           55.4   .443   5.83   4.38   3.680   5.41
```

Figure 4.1 Computer data file - example 1

```
MUT  GAL414            400.    201.380   424.599  1248.800   270.000      .000
   .0    3.0 100.0 1110
   -.100   50.6    -.1   -.10    -.1    -.1
   -.100   19.9     .1   -.10    -.1    -.1
   -.100   26.7    -.1   -.10    -.1    -.1
   -.100    2.8    -.1   -.10    -.1    -.1
   -.100  100.0   64.2    .09    -.1    1.5
  3.0    6.0 100.0 1110
   -.100   58.1    -.1   -.10    -.1    -.1
   -.100   26.4    -.1   -.10    -.1    -.1
   -.100   14.1    -.1   -.10    -.1    -.1
   -.100    1.4    -.1   -.10    -.1    -.1
   -.100  100.0   63.5    .07    -.1    1.4
  6.0    9.0 100.0 1210
   -.100   43.1    -.1   -.10    -.1    -.1
   -.100   38.4    -.1   -.10    -.1    -.1
   -.100   16.5    -.1   -.10    -.1    -.1
   -.100    2.0    -.1   -.10    -.1    -.1
   -.100  100.0   66.2    .07    -.1     .9
  9.0   12.0 100.0 1210
   -.100   42.7    -.1   -.10    -.1    -.1
   -.100   34.8    -.1   -.10    -.1    -.1
   -.100   20.4    -.1   -.10    -.1    -.1
   -.100    2.1    -.1   -.10    -.1    -.1
   -.100  100.0   64.7    .09    -.1     .6
```

Figure 4.2 Computer data file - example 2

Table 4.1 - Mutuca Mine geological exploration

Type of information	Period of time: Up to 1981		From 1981 to 1984		Planned exploration		Total	
	No.	m	No.	m	No.	m	No.	m
Bore holes	47	3822	20	2668	15	3000(*)	82	9490(*)
Adits	3	370	-	-	-	-	3	370
Channels	-	-	234	1407	-	-	234	1407

(*) Estimated

4.5 Computerized geological data base

Figure 4.3 is an example of a computerized bore hole log, and it is almost self-explanatory. A few symbols have been used according to the geological classification adopted by MBR (see chapter 3). Figure 4.4 is the corresponding example for an adit, and so is figure 4.5.

4.6 Suggestions and recommendations

In addition to the data from diamond drill holes, channels and adits, which have already been computerized, all relevant geological information available should be entered into the computerized data base, that is to say, up-to-date mine surface data; data from the early percussion drill holes, despite their inadequacy for producing block kriging estimates of grades in the various size fractions(see chapter 3); any known or implied boundaries of the ore body, in the form of coordinates (EW, NS, Elevation) of points, with the corresponding codes for ore

Table 4.2 - Mutuca Mine drill holes

DRILL HOLE/YEAR	Coordinates (m) NS SECTION	EW	NS	ELEVATION	AZIMUTH	DIP	TOTAL LENGTH(m)	No. OF SAMPLES	AVERAGE LENGTH(m)	
MUT DDH 01 85	387.	527.000	387.000	1274.500	98.000	55.000				
MUT DDH 04 85	1000.	398.000	1000.000	1248.700	278.010	70.000				
MUT DDH 06 85	200.	498.000	200.000	1239.800	.000	90.000				
MUT DDH 07 85	400.	310.000	400.000	1259.600	.000	90.000				
MUT DDH 01 84	100.	546.619	100.000	1239.700	278.000	60.000				
MUT DDH 02 84	-100.	519.000	-100.000	1248.200	278.000	70.000				
MUT DDH 03 84	00.	527.971	.000	1240.700	278.000	65.000				
MUT DDH 04 84	100.	546.619	100.000	1239.700	98.000	70.000				
MUT DDH 05 84	300.	406.190	300.000	1241.700	278.000	60.000				
MUT DDH 06 84	500.	480.320	500.000	1284.200	98.000	55.000				
MUT DDH 07 84	200.	498.800	200.000	1239.800	278.000	70.000				
MUT DDH 01 81	700.	293.453	698.721	1282.212	.000	90.000	213.0	71	3.0	
MUT DDH 02 81	700.	397.011	699.586	1274.563	.000	90.000	117.0	39	3.0	
MUT DDH 03 81	900.	333.996	899.921	1270.101	.000	90.000	150.0	50	3.0	
MUT DDH 04 81	900.	289.841	899.986	1269.872	.000	90.000	45.0	15	3.0	
MUT DDH 05 81	600.	390.669	599.868	1292.619	.000	90.000	227.0	76	3.0	(1)
MUT DDH 06 81	900.	457.721	899.633	1275.492	.000	90.000	54.0	18	3.0	
MUT DDH 07 81	700.	243.861	699.879	1280.767	.000	90.000	63.0	21	3.0	
MUT DDH 08 81	1100.	314.460	1100.000	1276.472	.000	90.000	66.0	22	3.0	
MUT DDH 09 81	600.	391.709	599.894	1292.509	270.000	60.000	69.0	23	3.0	
MUT DDH 10 81	100.	580.16	100.000	1257.551	.000	60.000	42.0	14	3.0	
MUT DDH 01 80	1100.	410.188	1099.694	1283.280	.000	90.000	54.0	6	9.0	(1)
MUT DDH 02 80	1100.	358.873	1099.547	1290.240	.000	90.000	128.4	9	14.3	(1)
MUT DDH 01 77	700.	418.279	698.854	1297.128	270.000	70.000	67.3	2	33.7	(1)
MUT DDH 02 77	1000.	457.092	998.572	1298.094	270.000	60.000	63.0	4	15.8	(1)
MUT DDH 03 77	400.	352.188	413.013	1283.425	270.000	60.000	23.3	2	11.7	(1)
MUT DDH 01 76	1000.	433.053	998.874	1311.139	90.000	58.000	162.8	11	14.8	(1)
MUT DDH 02 76	600.	294.905	598.852	1284.687	90.000	70.000	170.0	15	11.3	(1)
MUT DDH 03 76	800.	368.785	798.864	1303.499	90.000	70.000	172.7	12	14.4	(1)
MUT DDH 04 76	200.	471.966	197.899	1286.779	.000	90.000	60.0	5	12.0	(1)
MUT DDH 05 76	500.	341.930	498.860	1302.480	90.000	60.000	185.8	10	18.6	(1)
MUT DDH 06 76	300.	428.911	298.871	1286.149	.000	90.000	107.3	5	21.5	(1)
MUT DDH 07 76	900.	357.300	898.845	1300.570	270.000	60.000	173.5	11	15.8	(1)
MUT DDH 01 74	1100.	430.000	1098.000	1311.109	270.000	60.000	63.1	21	3.0	(1)
MUT DDH 03 74 (2)	800.	390.279	799.532	1306.744	270.000	75.000	119.2	16	7.5	(1)
MUT DDH 04 74	700.	353.485	697.917	1289.092	.000	90.000	200.0	23	8.7	(1)
MUT DDH 05 74	600.	372.729	599.047	1296.308	.000	90.000	201.6	23	8.8	(1)
MUT DDH 06 74	400.	379.602	398.762	1291.649	90.000	75.000	181.3	24	7.6	(1)
						TOTAL	3179.3	547	5.8	

Notes : (1) Variable sample lengths.
(2) Drill hole 02/74 does not exist.

MINE	CATEGORY	NUMBER	YEAR/LETTER	NS	EW	ELEVATION	AZIMUTH	DIP	SECTION	PAGE
MJT	DDH	5	84	300.000	406.190	1241.700	278.010	60.000	300.000	1

	NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)
	299.686	408.418	1237.803	.00	9.00	9.00	100.

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	-.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	1.46	8.70	68.40	.03	1.04	.61
-2IN +1/4 IN	1.46	8.70	68.40	.03	1.04	.61
-1/4IN +100MESH	7.20	42.70	68.00	.03	1.03	.61
-100 MESH	8.20	48.60	62.30	.16	2.42	1.68
CRUDE	16.86	100.00	65.30	.10	1.71	1.13

SVS HEMAT

	NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)
	299.164	412.131	1231.308	9.00	15.00	6.00	100.

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	-.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	1.17	7.50	68.70	.03	.63	.48
-2IN +1/4 IN	1.17	7.50	68.70	.03	.63	.48
-1/4IN +100MESH	7.73	49.60	68.60	.03	.68	.51
-100 MESH	6.70	42.90	64.30	.17	1.91	1.28
CRUDE	15.60	100.00	66.80	.09	1.20	.84

SVS HEMAT

	NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)
	298.850	414.360	1227.411	15.00	18.00	3.00	100.

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	-.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	.28	2.60	66.50	.09	1.47	.92
-2IN +1/4 IN	.28	2.60	66.50	.09	1.47	.92
-1/4IN +100MESH	2.81	26.30	62.80	.17	2.20	1.20
-100 MESH	7.58	71.10	54.00	.30	6.00	3.50
CRUDE	10.67	100.00	56.60	.26	4.88	2.83

SVS RAIF

	NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)
	298.537	416.588	1223.513	18.00	24.00	6.00	92.

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	-.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	1.18	7.30	68.00	.06	.65	.66
-2IN +1/4 IN	1.18	7.30	68.00	.06	.65	.66
-1/4IN +100MESH	6.59	40.90	68.50	.05	.58	.66
-100 MESH	8.34	51.80	63.50	.14	2.92	1.84
CRUDE	16.11	100.00	65.90	.10	1.80	1.27

SVS HEMAT

Figure 4.3 Computerized bore hole log

MINE	CATEGORY	NUMBER	YEAR/LETTER	NS	EW	ELEVATION	AZIMUTH	DIP	SECTION	PAGE
MUT	GAL	414	0	424.599	201.380	1248.800	270.000	.000	400.000	1

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				424.599	202.880	1248.800	.00	3.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SI02
-2IN +1IN	-.10	50.60	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	19.90	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	70.50	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	26.70	-.10	-.10	-.10	-.10
-100 MESH	-.10	2.80	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	64.20	.09	-.10	1.50

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				424.599	205.880	1248.800	3.00	6.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SI02
-2IN +1IN	-.10	58.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	26.40	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	84.50	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	14.10	-.10	-.10	-.10	-.10
-100 MESH	-.10	1.40	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	63.50	.07	-.10	1.40

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				424.599	208.880	1248.800	6.00	9.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SI02
-2IN +1IN	-.10	43.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	38.40	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	81.50	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	16.50	-.10	-.10	-.10	-.10
-100 MESH	-.10	2.00	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	66.20	.07	-.10	.90

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				424.599	211.880	1248.800	9.00	12.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SI02
-2IN +1IN	-.10	42.70	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	34.80	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	77.50	-.10	-.10	-.10	-.10

Figure 4.4 Computerized adit log - example 1

MINE	CATEGORY	NUMBER	YEAR/LETTER	NS	EW	ELEVATION	AZIMUTH	DIP	SECTION	PAGE
MJT	GAL	901	0 S	900.614	278.934	1276.300	270.000	.000	900.000	1

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				900.614	280.434	1276.300	.00	3.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	-.10	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	-.10	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	.00	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	-.10	-.10	-.10	-.10	-.10
-100 MESH	-.10	-.10	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	-.10	-.10	-.10	-.10

MIS MISNG -1

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				900.614	283.434	1276.300	3.00	6.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	22.30	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	49.70	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	72.00	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	23.30	-.10	-.10	-.10	-.10
-100 MESH	-.10	4.70	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	63.50	.09	-.10	.50

M RIB 2300

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				900.614	286.434	1276.300	6.00	9.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	12.90	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	47.30	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	60.20	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	32.90	-.10	-.10	-.10	-.10
-100 MESH	-.10	6.90	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	65.10	.13	-.10	.40

M RIB 1311

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				900.614	289.434	1276.300	9.00	12.00	3.00	100.	3.00

SIZE FRACTION	WEIGHT	(%)	FE	P	AL203	SIO2
-2IN +1IN	-.10	5.30	-.10	-.10	-.10	-.10
-1IN +1/4IN	-.10	58.00	-.10	-.10	-.10	-.10
-2IN +1/4 IN	.00	63.30	-.10	-.10	-.10	-.10
-1/4IN +100MESH	-.10	31.20	-.10	-.10	-.10	-.10
-100 MESH	-.10	5.50	-.10	-.10	-.10	-.10
CRUDE	-.10	100.00	63.90	.13	-.10	.40

M RIB 2301

Figure 4.5 Computerized adit log - example 2

or waste; whenever a diamond drill hole hits waste, the data coding process should not be stopped at this point, but rather the coordinates of samples containing waste should be entered into the data file as well.

A proper data base management system, such as SIR or SPSS(see references) can be of assistance for performing data handling operations.

Graphical aids can be used both for debugging the data and for helping the geological interpretation. A colour graphics terminal could speed up the data checking process, so that costly and time-consuming plotters are eliminated at the initial stages. Later on, when the data have been suficiently verified, hardcopies can be produced through the plotter. The possibility of purging spurious data "on-line" is offered by computers, and should be considered as a potential auxiliary tool for the geologist.

5.Graphical representation and mapping of the geological data

5.1 Summary and conclusions

5.2 Input data

5.3 Graphical representation of bore hole logs

5.4 Comments on the bore hole log graphs

5.5 Mapping the geological data

5.6 Suggestions for further improvements

5.1 Summary and conclusions

This chapter starts with a brief comment on the input data, followed by a description of the graphs representing the bore hole logs. Subsequently, the maps showing the spatial position of the sample points are presented. Finally, some suggestions for further enhancement of the graphs have been made.

The conclusions of this chapter are listed below:

- the computer and its graphical aids are very useful tools for the geologists to visually check the geological data.
- different numerical codes can be used for the various types of material or, alternatively, if a colour graphics output device is available, different colours.
- geological interpretations and the corresponding graphical representation can be speeded up by using computer graphics.

5.2 Input data

The input data have been described in chapter 4. Figure 4.3 shows an example of the geological information which is represented graphically further in this chapter. Details on each variable are given in chapter 3, "Geological sampling, core logging and assaying".

Most of the variables shown in figure 4.3 are raw data and a few others result either from computations or from direct geological classification. Although the percentages in weight of the size fractions are not represented directly in the graphs, they are taken into consideration both for classifying the friability of the material and for calculating the crude ore analyses.

Figure 5.1 shows a crude ore data file, which is a sub-set of the more general data file exemplified in figure 4.3. That sub-division is only a way of simplifying the data handling process.

5.3 Description of the bore hole graphs

The graphical representation of the Mutuca Mine diamond drill holes is explained in detail in figure 5.2.

The length of the bore hole is represented in the vertical axis. Also in the y-axis one can see the length of each segment of the core which has been taken as a sample and assayed, and to which the resulting analyses have been associated. The longest bore hole

MINE	CATEGORY	NUMBER	YEAR/LETTER	NS	EW	ELEVATION	AZIMUTH	DIP	SECTION	PAGE
MJT	1111	5	84	300.000	406.190	1241.700	278.010	60.000	300.000	1

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				299.686	408.418	1237.803	.00	9.00	9.00	100.	9.00
SIZE FRACTION	WEIGHT	(%)	FE	P	Al2O3	SIO2					
CRUDE	16.86	100.00	65.30	.10	1.71	1.13				SVS	HEMAT 1610

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				299.164	412.131	1231.308	9.00	15.00	6.00	100.	6.00
CRUDE	15.60	100.00	66.80	.09	1.20	.84				SVS	HEMAT 1610

				NS	EW	ELEVATION	FROM	TO	LENGTH	REC(%)	ACTLTH
				298.850	414.360	1227.411	15.00	18.00	3.00	100.	3.00
CRUDE	10.67	100.00	56.60	.26	4.88	2.83				SVS	RAIF 2631

Figure 5.1 Crude ore data file

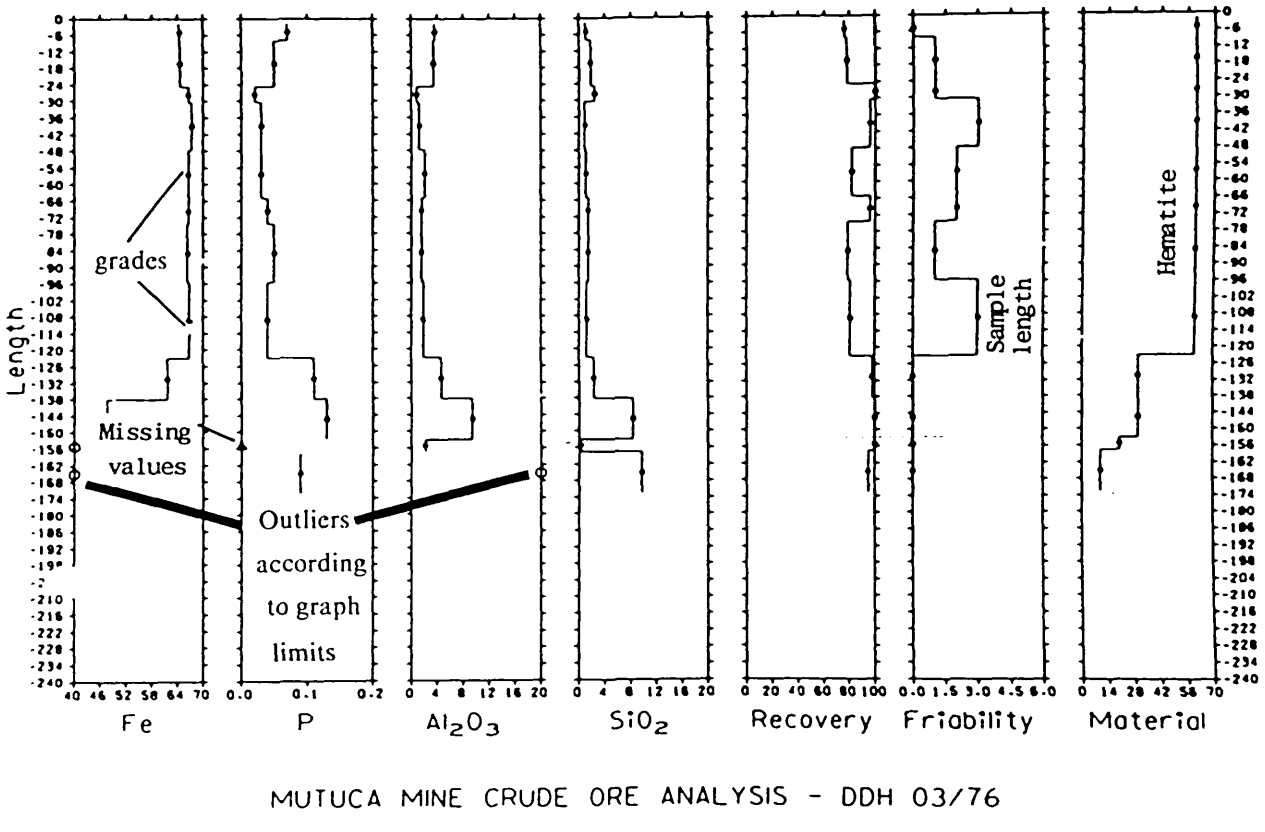


Figure 5.2 Explanation on the bore hole graphs

drilled in Mutuca dictated the maximum ordinate, and the number of intervals was chosen so as to produce multiples of 3m, which is the more frequent sample length in the geological exploration of the Mutuca ore body. The length is measured in metres.

There are eight different horizontal axes associated to the same vertical axis, representing, from left to right, Fe, P, Al_2O_3, SiO_2, recovery, friability and type of material. The grades and the recovery factor are given in percentage. The remaining variables have no measurement unit, as they are only numerical codes which have been adopted just for the sake of graphical representation.

The limits of the horizontal axes were fixed for Fe, P, Al_2O_3 and SiO_2 in order to compromise between a perceptible representation of the grades variation and the inclusion of most of the values in the graph. In other words, if the extreme cases of each variable had been chosen as the graph limits, all cases would have been plotted but, on the other hand, one would not have been able to identify grade differences in narrow zones which in fact contain most values and are of capital interest for the appraisal of the ore quality.

Table 5.1 shows the percentage of cases not represented in the graphs according to the limits selected for each variable.

Table 5.1 Outliers according to the graph limits (in %)

Mineral	Observed limits		Selected limits		Outliers
	Minimum	Maximum	Lower	Upper	
Fe	7.6	69.4	40.	70.	3.0
P	0.01	0.31	0.	0.20	1.5
Al_2O_3	0.10	34.3	0.	20.	1.1
SiO_2	0.20	51.3	0.0	20.	3.0

Note: the other three variables have no outliers.

The numerical codes adopted for the graphical representation of the friability are shown in table 5.2. The larger the value of the numerical code, the less friable is the material.

Table 5.2 Numerical codes adopted for friability

Code	Category
0	Missing value
1	Soft, very soft (SVS)
2	Mixed/soft (M/S)
3	Mixed (M)
4	Hard/Mixed (H/M)
5	Hard (H)

Table 5.3 shows the convention established for graphical representation of the different types of material. In general, the larger the value of the code, the better is the material.

Table 5.3 Numerical codes adopted for type of material

Code	Category
0	Missing values
10	Argillaceous iron formation (AIF)
20	Itabirite (IB)
30	Rich argillaceous iron formation (RAIF)
40	Rich itabirite (RIB)
50	Phosphorous hematite (PHem)
60	Hematite (Hem)

5.4 Comments on the graphs

In most cases one can detect a positive correlation between Fe and type of material, and negative correlations between Fe and P, Al_2O_3 and SiO_2, which are the contaminating elements (see chapter 6 for statistical results).

As a consequence of the inverted convention adopted for friability, the correlation between this variable and the recovery factor is positive.

These graphs can not only help the geologists in the interpretation of geological data, but help spot errors in the data file as well. Figure 5.3 is the graphical representation of the bore hole log shown in figure 4.3. Around the coordinate 200, in the y-axis, i.e., length, there is an anomaly which causes an inclined line to be plotted. This means there are some data missing, and the data file should be corrected.

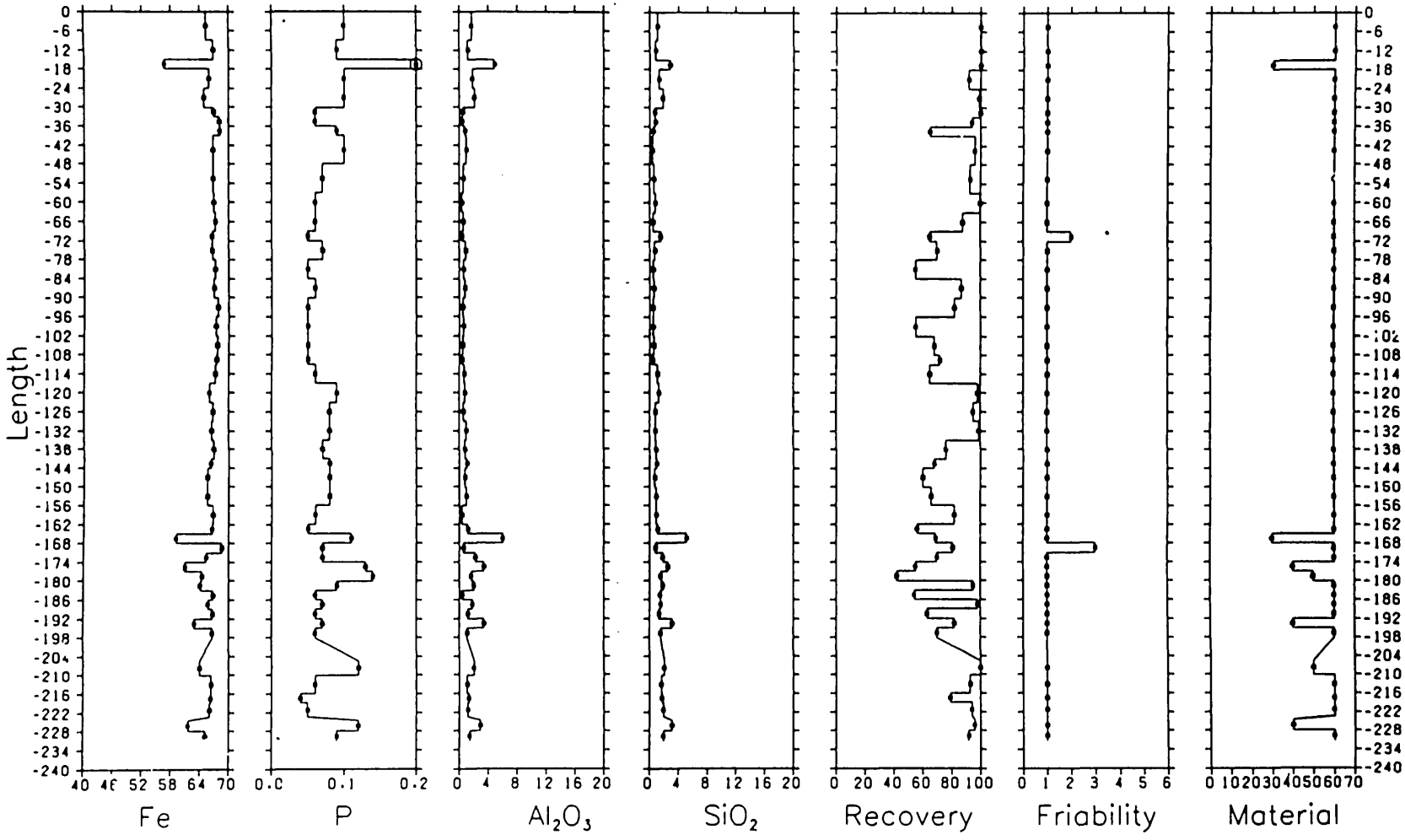


Figure 5.3 Example of bore hole graphs

5.5 Mapping of the geological data

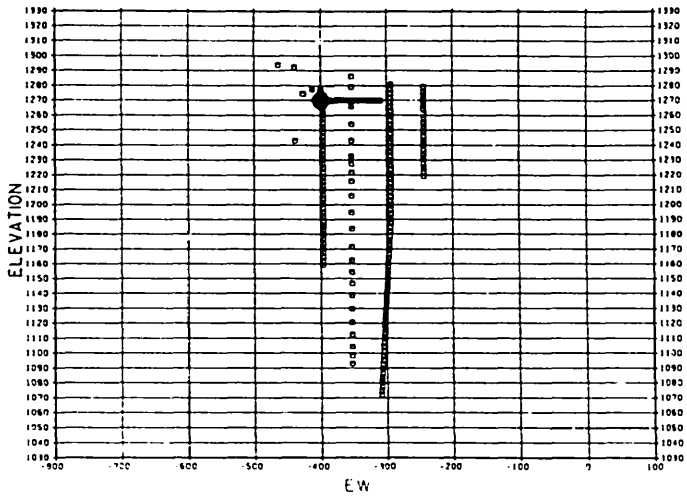
Figure 5.4 features the mapping of the sample points from the computerized geological data base. All data available are plotted, being them from diamond drill holes, trenches or adits. The graphs show cross and longitudinal sections, as well as bench maps of Mutuca's sample points.

The spatial position of all geological information can be checked by visual inspection, eliminating the risk of misplacing drill holes due to wrong input data, e.g., dip, bearing, and collar coordinates.

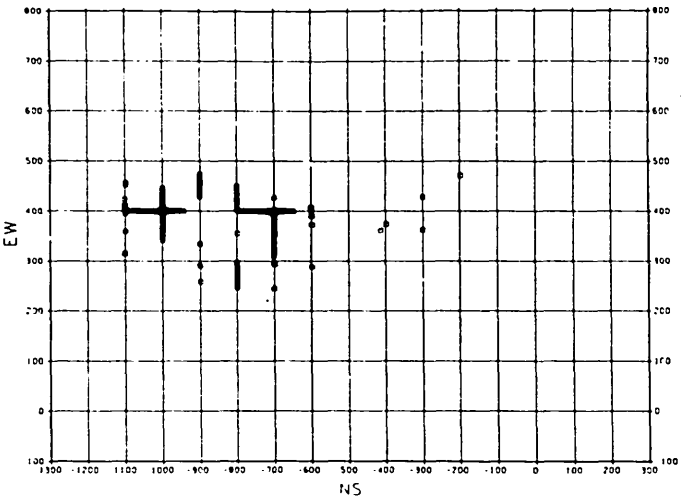
5.6 Suggestions for further improvements

The computer and its graphical aids could be used to produce a series of colour graphs showing Mutuca's different ore and waste types in plan maps, i.e., cross and longitudinal sections, as well as bench maps. Additionnally, both the representation of grades and types of material could be merged with the graphs showing the spatial position of sample points. This type of plot is usually made by displaying the grades in lines perpendicular to the bore hole longitudinal axis. For a vertical drill hole, these lines are horizontal, however, inclined drill holes are more difficult to represent. A computer program would have to be written to handle this problem.

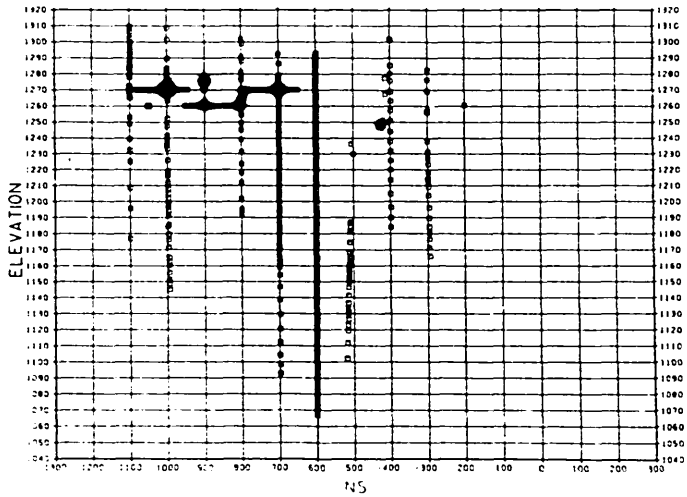
A digitiser would be very useful for linking the sample points so as to define the boundaries of distinct ore and waste zones. By exploring this possibility, the geological interpretation might be speeded up significantly.



[illegible] MINE CROSS-SECTION [illegible] FOC



[illegible] MINE BENCH MAP 1270



[illegible]

figure 5.4 mapping of the geological data

6. Classical statistical analysis

6.1 Summary and conclusions

6.2 Input data

6.3 Suggested approach for Mutuca Mine

6.4 Statistics on the crude ore

6.5 Testing the lognormality of the distributions for hematite's Fe, P, SiO_2, and Al_2O_3

6.6 Division of the raw data populations into two normal sub-populations(crude ore)

6.7 Statistics on the (−2";+1") size fraction

6.8 Statistics on the (−1";+¼") size fraction

6.9 Statistics on the (−2";+¼") size fraction

6.10 Statistics on the (−¼";+100 Mesh) size fraction

6.11 Statistics on the (−100 Mesh) size fraction

6.1 Summary and conclusions

This chapter presents a straightforward statistical analysis of the geological data (see Davis, 1973). Univariate descriptive statistics revealed the main characteristics of each variable and detected spurious values, enabling the data base to be thoroughly verified. The lognormality of the frequency distributions was analysed by way of goodness of fit tests. Additionally, the distributions were tentatively decomposed into two normal sub-populations. Multivariate statistics established the degree of correlation between variables.

The main conclusions of this chapter are summarized below:

- the chemical analyses of the minerals (Fe, P, SiO_2 and $Al2O_3$) follow patterns that are not significantly different from lognormal distributions, according to chi-squared goodness of fit tests which have been carried out.
- the size distributions in the four size fractions, expressed in percentage in weight, do not follow any particular pattern.
- multivariate analysis revealed that the Fe content has high negative correlations with $Al2O_3$ and SiO_2, but a poor linear relationship with the P grade.
- each cumulative frequency curve of the raw data chemical analyses has been tentatively split into two normal sub-populations, with no particulary interesting result other than the verification that 64%, for Fe, and 0.012%, for P, came out as the limits for the separation of the raw data populations into two component distributions. These limits are coincidentally the same as those adopted by MBR for the definition of hematite.
- in fact, the variables resulting from the segregation of hematite proved not to be significantly different from normally distributed values, allowing regular procedures to be used for their estimation through kriging (see chapter 8), which is a "robust"method of estimation (see David, 1977, page 114). For "robustness", see Wonnacott(1972, page 159).

6.2 Input data

The information contained in the geological data base, which has been described in chapter 4, was used as input data.

The statistical analysis had been made before the most recent geological data were available. However, as only a few of the latest drill holes had not been entered into the computer, the analysis which was made resulted in a reasonable description of the whole Mutuca geological data.

6.3 Suggested approach for Mutuca Mine

During the course of 1985, when the recent geological campaign was going on, a MINITAB (see list of references) program was created to run descriptive statistics on Mutuca's geological data. Each time a batch of new drill hole data had been added to the data base, the program was re-run, in a total of four times.

The statistical program can handle both individual drill holes and the complete data set. In that way, before the data for each drill hole is added to the computer file, they can be checked for spurious values.

The sequence of operations followed is briefly described below.

a) both raw data and the sub-population resulting from separating hematite received statistical treatment.

b) for size fractions, the following division was adopted:

crude ore (weighted average of all size fractions);

(-2";+1"),(-1";+¼") and (-2";+¼"), which is a weighted average of the previous two size fractions;

(-¼";+100 Mesh); and

(-100 Mesh).

c) for assays, the following variables were considered:

- % in weight; Fe; P; Al_2O_3; SiO_2 and (k-Fe), with k~70, for the three-parameter lognormal population (see Krige, 1981);

d) both the original variables and the logtransformed values were treated statistically;

e) the following statistics have been produced:

- number of cases (n); minimum; maximum; average (μ); weighted average (weighted by the sample lenght); standard deviation (σ); weighted stardard deviation; coefficient of variation (σ/μ); weighted coefficient of variation; median; 95% confidence interval for the mean (lower and upper limits); frequency distributions; goodness of fit tests; scattergrams and correlation matrices.

Table 6.1 summarizes the Mutuca Mine geological data which have been treated statistically as described above.

Table 6.1 - Mutuca Mine geological data

Size fraction	Proportion in weight (%)	Chemical analyses (%)			
		Fe	P	Al_2O_3	SiO_2
-2" ; +1"	16.9	67.3	0.04	1.3	1.3
-1" ; +1/4"	27.1	67.7	0.04	1.0	1.1
-2" ; +1/4"	41.3	67.6	0.04	1.0	1.1
-1/4 "; +100Mesh	36.0	67.3	0.04	1.0	1.5
-100Mesh	22.7	62.3	0.08	2.6	4.3
Crude	100.0	64.8	0.06	1.9	2.9

Notes : 1) The chemical analysis associated to the crude ore is a weighted average of the chemical results of the size fractions.

2) The data provided in the above table are mean values. Discrepancies are due to missing values :
total number of observations .....= 1378;
number of valid observations for crude ore = 1355 (Fe);
for(- 2 " ; + 1 ")size fraction .....= 794 (Fe);
for(- 1 " ; + 1/4 ")size fraction .....= 1098 (Fe);
for(- 2 " ; + 1/4 ")size fraction .....= 1098 (Fe);
for(- 1/4"; + 100 Mesh)size fraction .....= 1099 (Fe); and
for(- 100 Mesh).....= 1099 (Fe).

3) The(- 2 " ; + 1/4")size fraction is a weighted average of the (-2";+1") and (-1";+1/4") ones.

6.4 Statistics on the crude ore

From now on, a series of tables and graphs will be presented, showing the results of the statistical analysis. Both the raw and the logtransformed data statistics are presented first, followed by correlation matrices. This order of presentation will be followed throughout this chapter.

Tables 6.2, 6.3 and 6.4 feature the statistics on the crude ore.

Table 6.2 - Descriptive statistics on Mutuca Mine raw geological data
Size fraction: Crude Ore (grades in percentage)

6.2.1 - Without restriction other than missing values

Variable	Size	Fe	P	Al_2O_3	SiO_2	(k-Fe)(1)
Number of cases (n)	1358	1355	1354	1218	1355	1355
Minimum	100.	7.6	0.01	0.1	0.2	0.1
Maximum	100.	69.7	0.77	34.3	51.3	62.2
Average	100.	64.8	0.06	1.9	2.9	5.0
Weighted average	100.	65.1	0.06	1.9	2.5	4.3
Standard deviation	0.	6.8	0.05	3.2	5.7	6.8
Weighed standard deviation	0.	6.5	0.05	3.2	4.8	6.5
Coefficient of variation	0.	0.11	0.84	1.67	1.96	1.38
Weighed coeff. of variation	0.	0.10	0.89	1.66	1.95	1.49
Median	100.	67.0	0.04	1.1	1.1	2.8
95% Confidence :lower	100.	64.5	0.05	1.8	2.6	4.6
interval :upper	100.	65.2	0.06	2.1	3.2	5.3

6.2.2 - Selecting Hematite (2)

Variable	Size	Fe	P	Al_2O_3	SiO_2	(k-Fe)(1)
Number of cases (n)	1078	1078	1078	964	1078	1078
Minimum	100.	64.0	0.01	0.2	0.2	0.1
Maximum	100.	69.7	0.11	4.1	7.0	5.8
Average	100.	67.2	0.04	1.1	1.2	2.6
Weighted average	100.	67.2	0.04	1.1	1.2	2.3
Standard deviation	0.	1.2	0.02	0.7	1.0	1.2
Weighed standard deviation	0.	1.2	0.02	0.7	0.9	1.2
Coefficient of variation	0.	0.02	0.45	0.62	0.78	0.47
Weighed coeff. of variation	0.	0.02	0.46	0.61	0.71	0.52
Median	100.	67.5	0.04	0.9	0.9	2.3
95% Confidence :lower	100.	67.1	0.04	1.06	1.18	2.51
interval :upper	100.	67.3	0.05	1.15	1.29	2.66

Notes : 1) k = Max(Fe)~70.0
2) Data were selected for table 6.2.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.3 - Descriptive statistics on Mutuca Mine geological data - logtransformed
Size fraction: Crude Ore (grades in percentage)

6.3.1 - Without restriction other than missing values

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	1355	1354	1218	1355
Minimum	-2.30	-4.61	-1.61	-2.30
Maximum	4.13	-0.26	3.53	3.93
Average	1.15	-3.09	0.15	0.35
Standard deviation	0.85	0.61	0.89	1.00
Coefficient of variation	0.73	-0.20	6.06	2.87
Median	1.03	-3.22	0.10	0.10
95% Confidence :lower	1.1	-3.1	0.10	0.30
interval :upper	1.2	-3.0	0.20	0.40

6.3.2 - Selecting Hematite (2)

Variable	(k-Fe)(3)	P	Al_2O_3	SiO_2
Number of cases (n)	1078	1078	964	1078
Minimum	-2.3	-4.6	-1.6	-1.6
Maximum	1.76	-2.21	1.41	1.95
Average	0.83	-3.22	-0.08	0.007
Standard deviation	0.51	0.47	0.62	0.60
Coefficient of variation	0.62	-0.15	-7.32	80.89
Median	0.83	-3.22	-0.10	-0.10
95% Confidence :lower	0.80	-3.25	-0.12	-0.03
interval :upper	0.86	-3.19	-0.05	0.04

Notes : 1) This table contains statistics on the variables resulting from logtransformation of the raw data from table 6.2.

2) Data were selected for table 6.3.2 statistics as follows : $64.0 =< Fe =< 69.9$; $P < 0.12$.
Ore with the above characteristics is referred to as "Hematite" in MBR.

3) k = Max(Fe) ~70.00

Table 6.4 - Correlation matrices for raw and logtransformed data
Size fraction : Crude Ore (grades in percentage)

6.4.1 - Without restriction other than missing values

	Raw data			Logtransformed data (1)		
	Fe	P	SiO_2	k-Fe	P	SiO_2
P	-.386			.407		
SiO_2	-.781	.097		.812	.172	
Al_2O_3	-.779	.436	.410	.691	.535	.401

6.4.2 - Selecting Hematite (2)

	Raw data			Logtransformed data		
	Fe	P	SiO_2	k-Fe	P	SiO_2
P	-.297			.243		
SiO_2	-.604	-.017		.589	.006	
Al_2O_3	-.679	.210	.084	.696	.225	.178

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.4.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

6.5 Testing the lognormality of the distributions of the hematite's Fe, P, Al_2O_3 and SiO_2

Tables 6.5, 6.6, 6.7 and 6.8 show the chi-squared goodness of fit tests of Fe, P, Al_2O_3 and SiO_2, after the segregation of hematite. The Fe distribution is a three-parameter lognormal, therefore a constant (k~70) has been used to transform the original value, before the neperian logarithm (ln) is applied.

Table 6.5 - Chi-squared goodness of fit test - ln (k-Fe) Hematite

Class upper limits				Expected Frequency	Observed Frequency	(Ei-Oi)2 / Ei
Fe	k-Fe(1)	Logtransformed	Standardized	(Ei)	(Oi)	
68.79	0.71	-0.336	-1.456	29.9	29	0.027
68.49	1.01	0.014	-0.905	40.1	43	0.210
68.06	1.44	0.364	-0.354	77.7	77	0.006
67.46	2.04	0.714	0.197	89.1	92	0.094
66.66	2.90	1.064	0.748	79.3	60	4.697
65.39	4.11	1.414	1.299	53.2	63	1.805
64.00	5.50	1.700	1.780	39.7	45	0.708
		Total		409	409	7.548

Note : 1) k = Max(Fe) ~70.00

The chi-squared critical limit at 95% confidence level is 9.49, taking into account 4 degrees of freedom. Thus, since the calculated chi-squared statistic is less than the tabulated critical value, the null hypothesis (Ho) of no difference can not be rejected at a confidence level of 95%.
The number of degrees of freedom (df) was computed as follows :

$$df = n - p - 1$$

where n = number of interval classes in which the distribution has been divided; and

p = number of empirical or sample data population parameters necessary to calculate the expected frequencies.
At the present case, p = 2 as the normal distribution is characterized by mean and standard deviation.

Table 6.6 - Chi-squared goodness of fit test - ln(P) Hematite

Class upper limits			Expected Frequency	Observed Frequency	(Ei-Oi)2 / Ei
P	Logtransformed	Standardized	(Ei)	(Oi)	
0.01	-4.210	-2.060	8.1	8	0.000
0.02	-3.690	-0.910	66.1	66	0.000
0.03	-3.400	-0.250	89.9	101	1.370
0.04	-3.160	0.270	83.9	81	0.100
0.05	-2.940	0.750	68.3	74	0.480
0.07	-2.720	1.250	49.5	32	6.190
0.08	-2.550	1.620	21.7	26	0.850
0.09	-2.440	1.870	9.0	12	1.000
0.10	-2.300	2.110	12.6	9	1.030
	Total		409	409	11.020

The chi-squared critical limit at 95% confidence level is 12.6, taking into account 6 degrees of freedom (see table 6.5). Thus, since the calculated chi-squared statistics is less than the tabulated critical value,the null hypothesis (Ho) of no difference can not be rejected at a confidence level of 95%.

Table 6.7 - Chi-squared goodness of fit test - ln(SiO_2) Hematite

Class upper limits			Expected Frequency	Observed Frequency	(Ei-Oi)2 / Ei
SiO_2	Logtransformed	Standardized	(Ei)	(Oi)	
0.22	-1.529	-2.130	6.8	6	0.090
0.35	-1.042	-1.370	28.3	26	0.180
0.68	-0.393	-0.380	108.0	112	0.150
1.10	0.094	0.380	121.0	108	1.410
1.52	0.418	0.870	65.4	83	4.760
2.47	0.905	1.620	57.1	46	2.160
3.42	1.230	2.130	14.7	20	1.890
4.10	1.410	2.390	6.8	8	0.220
	Total		409	409	10.860

The chi-squared critical limit at 95% confidence level is 11.1, taking into account 5 degrees of freedom (see table 6.5). Thus, since the calculated chi-squared statistics is less than the tabulated critical value,the null hypothesis (Ho) of no difference can not be rejected at a confidence level of 95%.

Table 6.8 - Chi-squared goodness of fit test - $\ln(Al_2O_3)$ Hematite

Class upper limits			Expected Frequency	Observed Frequency	(Ei-Oi)2 / Ei
Al_2O_3	Logtransformed	Standardized	(Ei)	(Oi)	
0.51	-0.679	-1.370	34.9	34	0.020
0.66	-0.410	-0.870	43.6	41	0.160
0.75	-0.280	-0.620	30.8	33	0.160
0.86	-0.150	-0.370	36.1	42	0.960
1.27	0.240	0.390	121.0	124	0.070
1.64	0.500	0.880	65.0	64	0.020
1.86	0.620	1.120	23.7	18	1.370
2.12	0.750	1.370	18.9	16	0.440
6.70	1.900	3.580	34.9	37	0.130
	Total		409	409	3.330

The chi-squared critical limit at 95% confidence level is 12.6, taking into account 6 degrees of freedom (see table 6.5). Thus, since the calculated chi-squared statistics is less than the tabulated critical value,the null hypothesis (Ho) of no difference can not be rejected at a confidence level of 95%.

6.6 Division of the raw data population into two normal sub-populations (crude ore)

The variables have been split into two components, following a procedure suggested by Sinclair (1976). A program called GIRAF(Geological interactive frequency distribution analysis and decomposition based on probability scale cumulative curves), which had been developed by Earle(1977), was used. The results are listed in table 6.9.

The first set of three lines, in the table below, refer to the whole population (100%). The following two groups of three lines concern the two sub-populations, into which the parent distribution has been split. For instance, the Fe distribution (100%) can be the result of mixing two normal sub-populations, the first contributing with 71% of the cases and the second, with 29%.

Table 6.9 - Division of the raw data populations into two normal sub-populations (1) (grades in percentage)

Variable	Fe	P	Al_2O_3	SiO_2	(k-Fe)(2)
n	710	709	710	710	710
Population (%)	100.0	100.0	100.0	100.0	100.0
Average	63.99	0.06	2.3	3.0	5.5
Standard deviation	8.1	0.06	3.9	5.8	8.1
Sub-population 1 (%)	71.	65.	76.	76.	76.
Average	67.1	0.04	0.9	1.0	2.1
Standard deviation	1.1	0.01	0.4	0.3	1.0
Sub-population 2 (%)	29.	35.	24.	24.	24.
Average	56.7	0.10	5.5	0.92	13.3
Standard deviation	7.7	0.06	5.4	9.0	10.5

Notes: 1) The theoretical distribution of each variable was checked against the correspondent observed probability cumulative curve, by means of a Kolmogorov-Smirnov goodness of fit test. In all cases the null hypothesis Ho, of no difference, can not be rejected at a confidence level of 90 %.

2) k = Max(Fe) ~70.00

6.7 Statistics on the (−2";+1") size fraction

Tables 6.10, 6.11 and 6.12 describe the (−2";+1") size fraction

Table 6.10 - Descriptive statistics on Mutuca Mine geological raw data
Size fraction: -2 inch; +1 inch (grades in percentage)

6.10.1 - Without restriction other than missing values

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	1028	794	794	794	794	794
Minimum	0.3	32.4	0.01	0.1	0.1	0.1
Maximum	73.8	69.8	0.31	24.0	53.2	37.5
Average	16.9	67.3	0.04	1.3	1.3	2.6
Standard deviation	15.0	3.3	0.03	2.0	3.6	3.3
Coeff. of variation	0.89	0.05	0.68	1.49	2.76	1.27
Median	11.8	68.3	0.03	0.7	0.7	1.6
95% Confidence:lower	16.0	67.0	0.04	1.2	1.0	2.4
interval :upper	17.8	67.5	0.04	1.5	1.5	2.9

6.10.2 - Selecting Hematite (2)

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	726	726	726	726	726	726
Minimum	0.3	64.0	0.01	0.1	0.1	0.1
Maximum	73.8	69.8	0.11	6.2	8.1	5.9
Average	16.6	68.0	0.03	1.0	0.9	1.9
Standard deviation	14.6	1.1	0.02	0.9	0.8	1.1
Coeff. of variation	0.88	0.02	0.46	0.96	0.93	0.59
Median	11.6	68.4	0.03	0.6	0.7	1.5
95% Confidence:lower	15.5	67.9	0.03	0.9	0.8	1.8
interval :upper	17.7	68.1	0.04	1.0	1.0	2.0

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.10.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

Table 6.11 - Descriptive statistics on Mutuca Mine geological data - logtransformed
Size fraction: -2 inch; +1 inch (grades in percentage)

6.11.1 - Without restriction other than missing values

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	794	794	794	794
Minimum	-2.30	-4.61	-2.30	-2.30
Maximum	3.62	-1.17	3.18	3.97
Average	0.63	-3.43	-0.29	-0.26
Standard deviation	0.72	0.52	1.02	0.75
Coefficient of variation	1.14	-0.15	-3.51	-2.85
Median	0.47	-3.51	-.37	-.37
95% Confidence :lower	0.58	-3.47	-.36	-.32
interval :upper	0.68	-3.39	-.22	-.21

6.11.2 - Selecting Hematite (3)

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	726	726	726	726
Minimum	-2.30	-4.61	-2.30	-2.30
Maximum	1.77	-2.21	1.82	2.09
Average	0.49	-3.46	-0.41	-0.34
Standard deviation	0.54	0.46	0.87	0.61
Coefficient of variation	1.10	-0.13	-2.10	-1.80
Median	0.41	-3.51	-0.51	-0.36
95% Confidence :lower	0.45	-3.49	-0.48	-0.38
interval :upper	0.53	-3.42	-0.35	-0.29

Notes : 1) This table contains statistics on the variables resulting from logtransformation of the raw data from table 6.10.
2) k = Max(Fe) ~70.00
3) Data were selected for table 6.11.2 statistics as follows : 64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.12 - Correlation matrices for raw and logtransformed data
Size fraction: -2 inch; +1 inch (grades in percentage)

6.12.1 - Without restriction other than missing values

	Raw data				Logtransformed data (1)			
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2	
P	-.101				-.067			
SiO_2	-.735	-.096			.506	-.111		
Al_2O_3	-.638	.123	-.015		.649	.175	-.121	
Size	-.161	.116	-.051	-.178				

6.12.2 - Selecting Hematite (2)

	Raw data				Logtransformed data			
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2	
P	0.028				.017			
SiO_2	-.410	-.078			.408	-.042		
Al_2O_3	-.784	-.033	-.097		.689	.045	-.123	
Size	.195	.262	-.156	-.124				

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.12.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

6.8 Statistics on the (−1";+¼") size fraction

Tables 6.13, 6.14 and 6.15 feature the statistics on the (−1";+¼") size fraction.

Table 6.13 - Descriptive statistics on Mutuca Mine raw geological data
Size fraction: -1 inch; +1/4 inch (grades in percentage)

6.13.1 - Without restriction other than missing values

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe (1)
Number of cases(n)	1219	1098	1098	1098	1098	1098
Minimum	0.7	8.2	0.01	0.1	0.2	0.1
Maximum	76.2	69.7	0.52	12.8	35.5	61.6
Average	27.1	67.7	0.04	1.0	1.1	2.1
Standard deviation	13.2	3.3	0.03	1.1	2.5	3.3
Coeff. of variation	0.47	0.05	0.78	1.1	2.2	1.6
Median	27.7	68.3	0.03	0.7	0.7	1.5
95% Confidence:lower	26.3	67.5	0.04	0.9	1.0	1.9
interval :upper	27.8	67.9	0.04	1.0	1.3	2.3

6.13.2 - Selecting Hematite (2)

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe (1)
Number of cases(n)	1054	1054	1054	1054	1054	1054
Minimum	0.7	64.1	0.01	0.1	0.2	0.1
Maximum	76.2	69.7	0.11	10.5	6.7	5.7
Average	26.3	68.1	0.04	0.9	0.9	1.7
Standard deviation	13.4	0.9	0.02	0.8	0.7	0.9
Coeff. of variation	0.51	0.01	0.47	0.88	0.81	0.55
Median	26.4	68.4	0.03	0.6	0.7	1.4
95% Confidence:lower	25.5	68.0	0.03	0.8	0.8	1.6
interval :upper	27.1	68.2	0.04	0.9	0.9	1.8

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.13.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

Table 6.14 - Descriptive statistics on Mutuca Mine geological data - logtransformed
Size fraction: -1 inch; +1/4 inch (grades in percentage)

6.14.1 - Without restriction other than missing values

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	1098	1098	1098	1098
Minimum	-2.30	-4.61	-2.30	-1.61
Maximum	4.12	-0.65	2.55	3.57
Average	0.46	-3.43	-0.41	-2.76
Standard deviation	0.63	0.54	0.84	0.66
Coeff. of variation	1.36	-0.16	-2.06	-2.40
Median	0.41	-3.51	-0.36	-0.36
95% Confidence: lower	0.43	-3.47	-0.46	-0.31
interval : upper	0.50	-3.40	-0.36	-0.24

6.14.2 - Selecting Hematite (3)

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	1054	1054	1054	1054
Minimum	-2.30	-4.61	-2.30	-1.61
Maximum	1.74	-2.21	2.35	1.90
Average	0.39	-3.46	-0.45	-0.33
Standard deviation	0.52	0.49	0.79	0.54
Coeff. of variation	1.31	-0.14	-1.77	-1.63
Median	0.34	-3.51	-0.51	-0.36
95% Confidence: lower	0.36	-3.49	-0.50	-0.37
interval : upper	0.43	-3.43	-0.40	-0.30

Notes : 1) This table contains statistics on the variables resulting from logtransformation of the raw data from table 6.13.
2) k = Max(Fe) ~70.00
3) Data were selected for table 6.14.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.15 - Correlation matrices for raw and logtransformed data
Size fraction: -1 inch; +1/4 inch (grades in percentage)

6.15.1 - Without restriction other than missing values

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe) (1)	P	SiO_2
P	-.317				.245		
SiO_2	-.626	.036			.619	.087	
Al_2O_3	-.517	.358	.092		.610	.256	.037
Size	.070	-.135	.007	-.094			

6.15.2 - Selecting Hematite (2)

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe) (1)	P	SiO_2
P	-.175				.185		
SiO_2	-.509	.042			.484	.092	
Al_2O_3	-.682	.111	.012		.658	.154	.005
Size	.069	-.159	-.078	-.005			

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.15.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

6.9 Statistics on the (−2";+¼") size fraction

This granulometric bracket was generated by compositing both the (−2";+1") and the (−1";+¼") size fractions, the result being a weighted average. Tables 6.16, 6.17 and 6.18 show the statistical results obtained.

Table 6.16 - Descriptive statistics on Mutuca Mine raw geological data
Size fraction: -2 inch; +1/4 inch (grades in percentage)

6.16.1 - Without restriction other than missing values

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	1219	1098	1098	1098	1098	1098
Minimum	0.70	8.2	0.01	0.10	0.10	0.10
Maximum	96.9	69.7	0.52	13.0	39.1	61.6
Average	41.3	67.6	0.04	1.0	1.1	2.2
Standard deviation	22.1	3.4	0.03	1.2	2.6	3.4
Coeff. of variation	0.53	0.05	0.78	1.2	2.3	1.6
Median	40.3	68.3	0.03	0.7	0.7	1.5
95% Confidence:lower	40.1	67.4	0.04	1.0	1.0	2.0
interval :upper	42.6	67.8	0.04	1.1	1.3	2.4

6.16.2 - Selecting Hematite (2)

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	1049	1049	1049	1049	1049	1049
Minimum	0.70	64.0	0.01	0.10	0.10	0.10
Maximum	96.9	69.7	0.11	10.5	7.5	5.8
Average	39.5	68.1	0.04	0.9	0.9	1.7
Standard deviation	21.5	0.9	0.02	0.8	0.7	0.9
Coeff. of variation	0.54	0.01	0.47	0.87	0.81	0.55
Median	38.3	68.3	0.03	0.7	0.7	1.5
95% Confidence:lower	38.2	68.0	0.03	0.8	0.8	1.7
interval :upper	40.8	68.1	0.04	0.9	0.9	1.8

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.16.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.17 - Descriptive statistics on Mutuca Mine geological data - logtransformed
Size fraction: -2 inch; +1/4 inch (grades in percentage)

6.17.1 - Without restriction other than missing values

Variable	(k-Fe) (2)	P	Al_2O_3	SiO_2
Number of cases (n)	1098	1098	1098	1098
Minimum	-2.30	-4.61	-2.30	-2.30
Maximum	4.12	-0.65	2.56	3.67
Average	0.49	-3.42	-0.36	-0.26
Standard deviation	0.65	0.55	0.85	0.67
Coeff. of variation	1.32	-0.16	-2.36	-2.51
Median	0.41	-3.51	-0.36	-0.36
95% Confidence: lower	0.45	-3.45	-0.41	-0.30
interval : upper	0.53	-3.39	-0.31	-0.23

6.17.2 - Selecting Hematite (3)

Variable	(k-Fe) (2)	P	Al_2O_3	SiO_2
Number of cases (n)	1049	1049	1049	1049
Minimum	-2.30	-4.61	-2.30	-2.30
Maximum	1.78	-2.21	2.35	2.01
Average	0.41	-3.45	-0.41	-0.32
Standard deviation	0.52	0.49	0.78	0.55
Coeff. of variation	1.26	-0.14	-1.91	-1.69
Median	0.41	-3.51	-0.36	-0.36
95% Confidence: lower	0.38	-3.48	-0.46	-0.36
interval : upper	0.45	-3.42	-0.36	-0.29

Notes : 1) This table contains statistics on the variables resulting from logtransformation of the raw data from table 6.16.
2) k = Max(Fe) ~70.00
3) Data were selected for table 6.17.2 statistics as follows : 64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.18 - Correlation matrices for raw and logtransformed data
Size fraction: -2 inch; +1/4 inch (grades in percentage)

6.18.1 - Without restriction other than missing values

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe) (1)	P	SiO_2
P	-.302				.222		
SiO_2	-.625	.027			.595	.078	
Al_2O_3	-.531	.306	.074		.628	.242	.020
Size	.102	-.039	-.005	-.129			

6.18.2 - Selecting Hematite (2)

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2
P	-.127				.151		
SiO_2	-.489	.031			.462	.082	
Al_2O_3	-.699	.069	-.000		.672	.131	-.010
Size	.119	.042	-.104	-.041			

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.18.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

6.10 Statistics on the (−¼";+100 Mesh) size fraction

Tables 6.19, 6.20 and 6.21 display the statistical analysis of the (−¼";+100 Mesh) size fraction

Table 6.19 - Descriptive statistics on Mutuca Mine raw geological data
Size fraction: -1/4inch; +100 Mesh (grades in percentage)

6.19.1 - Without restriction other than missing values

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	1220	1099	1099	1099	1099	1099
Minimum	2.8	16.4	0.01	0.10	0.20	0.10
Maximum	94.7	69.9	0.55	16.8	34.0	53.6
Average	36.0	67.3	0.04	1.0	1.5	2.7
Standard deviation	13.4	3.4	0.03	1.2	2.7	3.4
Coeff. of variation	0.37	0.05	0.82	1.2	1.8	1.2
Median	35.5	68.0	0.03	0.7	0.9	2.0
95% Confidence:lower	35.3	67.1	0.04	0.9	1.3	2.5
interval :upper	36.8	67.5	0.04	1.0	1.6	2.9

6.19.2 - Selecting Hematite (2)

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe
Number of cases(n)	1040	1040	1040	1040	1040	1040
Minimum	2.9	64.2	0.01	0.10	0.20	0.10
Maximum	94.7	69.9	0.10	4.1	7.1	5.8
Average	37.4	67.8	0.03	0.9	1.1	2.2
Standard deviation	13.3	1.04	0.02	0.7	0.9	1.0
Coeff. of variation	0.36	0.02	0.48	0.76	0.85	0.47
Median	36.9	68.0	0.03	0.7	0.8	2.0
95% Confidence:lower	36.6	67.7	0.03	0.8	1.1	2.1
interval :upper	38.2	67.9	0.04	0.9	1.2	2.3

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.19.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

Table 6.20 - Descriptive statistics on Mutuca Mine geological data - logtransformed
Size fraction: -1/4 inch; +100 Mesh (grades in percentage)

6.20.1 - Without restriction other than missing values

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	1099	1099	1099	1099
Minimum	-2.30	-4.61	-2.30	-1.61
Maximum	3.98	-0.60	2.82	3.53
Average	0.77	-3.44	-0.38	-0.03
Standard deviation	0.58	0.54	0.82	0.74
Coeff. of variation	0.75	-0.16	-2.18	25.00
Median	0.69	-3.51	-0.36	-0.11
95% Confidence:lower	0.73	-3.47	-0.43	-0.07
interval :upper	0.80	-3.41	-0.33	-0.01

6.20.2 - Selecting Hematite (3)

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	1040	1040	1040	1040
Minimum	-2.30	-4.61	-2.30	-1.61
Maximum	1.76	-2.30	1.41	1.96
Average	0.69	-3.48	-0.42	-0.11
Standard deviation	0.46	0.47	0.76	0.61
Coeff. of variation	0.66	-0.14	-1.82	-5.75
Median	0.69	-3.51	-0.36	-0.22
95% Confidence: lower	0.66	-3.51	-0.46	-0.14
interval : upper	0.72	-3.45	-0.37	-0.07

Notes : 1) This table contains statistics on the variables resulting from logtransformation of the raw data from table 6.19.
2) k = Max(Fe) ~70.00
3) Data were selected for table 6.20.2 statistics as follows : 64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.21 - Correlation matrices for raw and logtransformed data
Size fraction: -1/4 inch; +100 Mesh (grades in percentage)

6.21.1 - Without restriction other than missing values

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2
P	-.368				.317		
SiO_2	-.727	.091			.730	.101	
Al_2O_3	-.654	.432	.228		.566	.328	.084
Size	.177	-.101	-.119	-.169			

6.21.2 - Selecting Hematite (2)

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2
P	-.249				.275		
SiO_2	-.619	.024			.634	.096	
Al_2O_3	-.653	.196	-.009		.618	.218	-.062
Size	.042	-.049	.017	-.054			

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.21.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

6.11 Statistics on the (−100 Mesh) size fraction

Tables 6.22, 6.23 and 6.24 feature the statistical results obtained for the (−100 Mesh) size fraction

Table 6.22 - Descriptive statistics on Mutuca Mine raw geological data
Size fraction: -100 Mesh (grades in percentage)

6.22.1 - Without restriction other than missing values

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	1220	1099	1098	1098	1099	1099
Minimum	0.20	12.5	0.01	0.1	0.1	0.1
Maximum	95.3	69.3	2.14	29.6	79.9	56.9
Average	22.7	62.3	0.08	2.6	4.3	7.1
Standard deviation	14.6	6.8	0.09	2.6	7.7	6.8
Coeff. of variation	0.64	0.11	1.07	1.01	1.80	0.96
Median	20.4	64.3	0.06	1.80	2.20	5.1
95% Confidence:lower	21.9	61.9	0.08	2.5	3.8	6.7
interval :upper	23.5	62.7	0.09	2.8	4.8	7.5

6.22.2 - Selecting Hematite (2)

Variable	Size	Fe	P	Al_2O_3	SiO_2	k-Fe(1)
Number of cases(n)	556	556	556	555	556	556
Minimum	0.60	64.0	0.01	0.1	0.1	0.1
Maximum	67.3	69.3	0.11	4.7	5.6	5.4
Average	23.0	66.4	0.05	1.2	1.5	3.0
Standard deviation	12.1	1.3	0.02	0.7	0.9	1.3
Coeff. of variation	0.53	0.02	0.42	0.6	0.64	0.44
Median	21.2	66.4	0.05	1.1	1.2	3.0
95% Confidence:lower	22.0	66.3	0.05	1.1	1.4	2.9
interval :upper	24.0	66.5	0.05	1.3	1.5	3.1

Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.22.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.

Table 6.23 - Descriptive statistics on Mutuca Mine geological data - logtransformed
Size fraction: -100 Mesh (grades in percentage)

6.23.1 - Without restriction other than missing values

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	1099	1098	1098	1099
Minimum	-2.30	-4.61	-2.30	-2.30
Maximum	4.04	0.76	3.39	4.38
Average	1.62	-2.74	0.56	0.83
Standard deviation	0.84	0.64	0.92	1.03
Coeff. of variation	0.52	-0.23	1.64	1.24
Median	1.63	-2.81	0.59	0.79
95% Confidence : lower	1.57	-2.78	0.51	0.77
: upper	1.67	-2.70	0.62	0.89

6.23.2 - Selecting Hematite (3)

Variable	(k-Fe)(2)	P	Al_2O_3	SiO_2
Number of cases (n)	556	556	555	552
Minimum	-2.30	-4.61	-2.30	-2.30
Maximum	1.67	-2.21	1.55	1.72
Average	0.98	-3.05	-0.04	0.19
Standard deviation	0.56	0.47	0.70	0.63
Coeff. of variation	0.57	-0.16	-20.01	3.23
Median	1.10	-3.00	0.10	0.18
95% Confidence: lower	0.93	-3.09	-0.09	0.14
: upper	1.02	-3.01	0.02	0.25

Notes : 1) This table contains statistics on the variables resulting from logtransformation of the raw data from table 6.22.
2) k = Max(Fe) ~70.00
3) Data were selected for table 6.23.2 statistics as follows : 64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR .

Table 6.24 - Correlation matrices for raw and logtransformed data
Size fraction: -100 Mesh (grades in percentage)

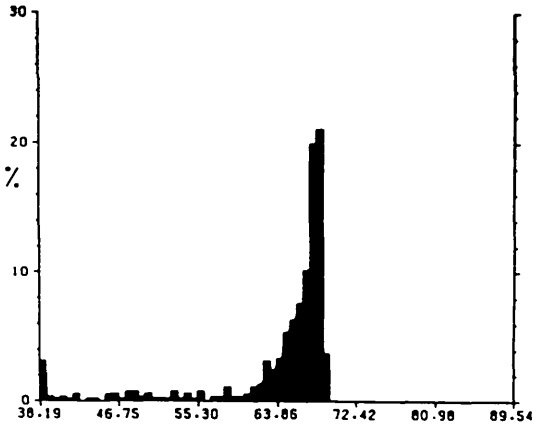
6.24.1 - Without restriction other than missing values

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2
P	-.225				.490		
SiO_2	-.850	.004			.811	.160	
Al_2O_3	-.591	.314	.153		.741	.529	.441
Size	-.130	.191	.043	.042			

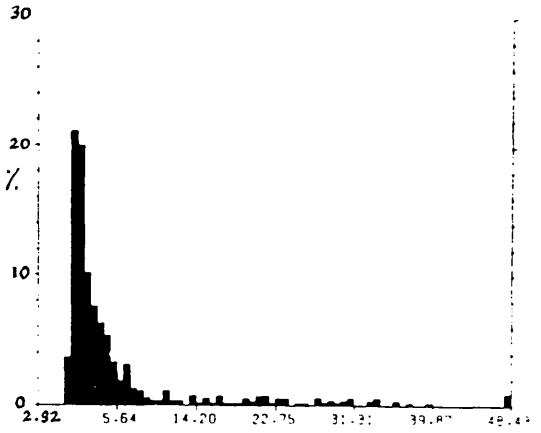
6.24.2 - Selecting Hematite (2)

	Raw data				Logtransformed data		
	Fe	P	SiO_2	Al_2O_3	(k-Fe)	P	SiO_2
P	-.442				.483		
SiO_2	-.575	-.029			.569	.024	
Al_2O_3	-.663	.263	.188		.673	.301	.238
Size	.028	-.083	-.163	-.227			

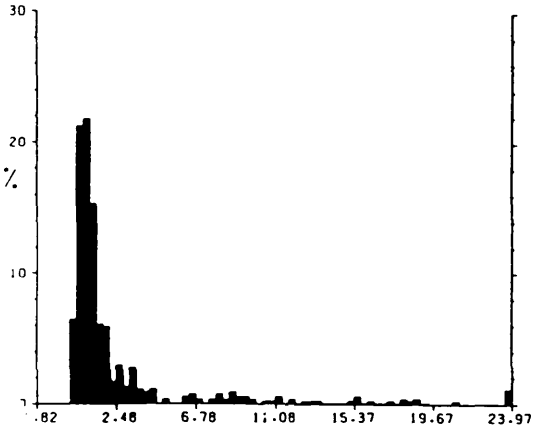
Notes : 1) k = Max(Fe) ~70.00
2) Data were selected for table 6.24.2 statistics as follows :
64.0 =< Fe =< 69.9 ; P < 0.12 .
Ore with the above characteristics is referred to as "Hematite" in MBR.



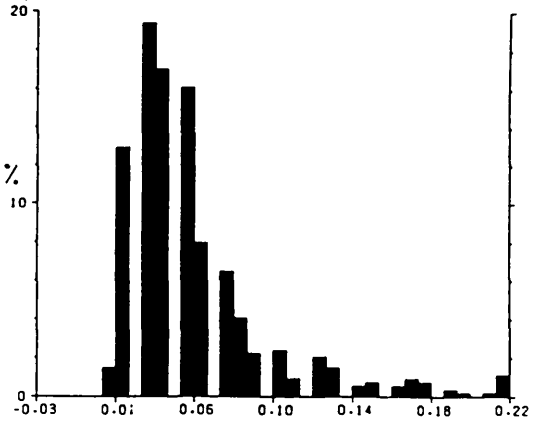
FE
NO. OF OBSERVATIONS = 540
MEAN = 63.86 STD DEV = 8.559



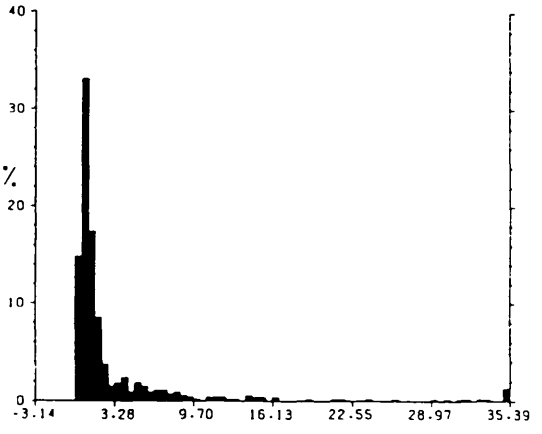
K – FE
NO. OF OBSERVATIONS = 540
MEAN = 5.64 STD DEV = 8.559



SI02
NO. OF OBSERVATIONS = 540
MEAN = 2.48 STD DEV = 4.297

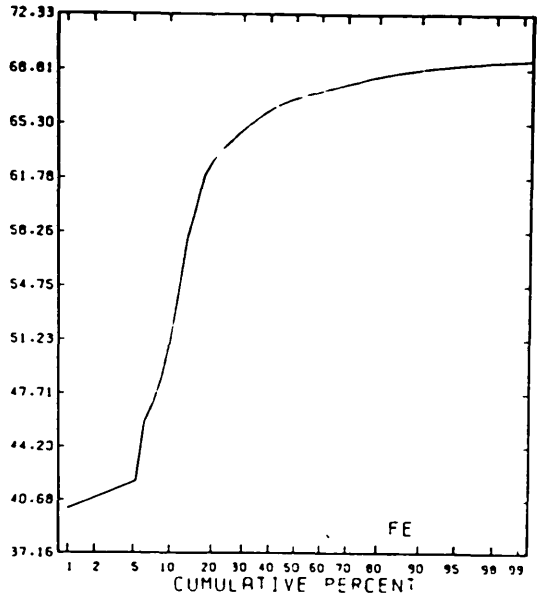


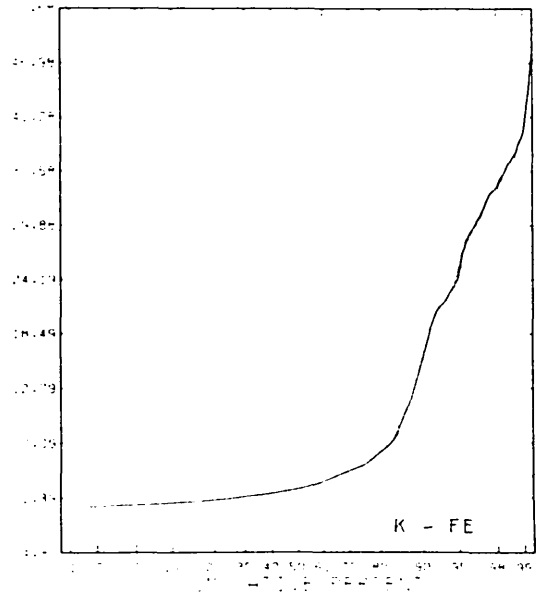
P
NO. OF OBSERVATIONS = 540
MEAN = 0.06 STD DEV = 0.041

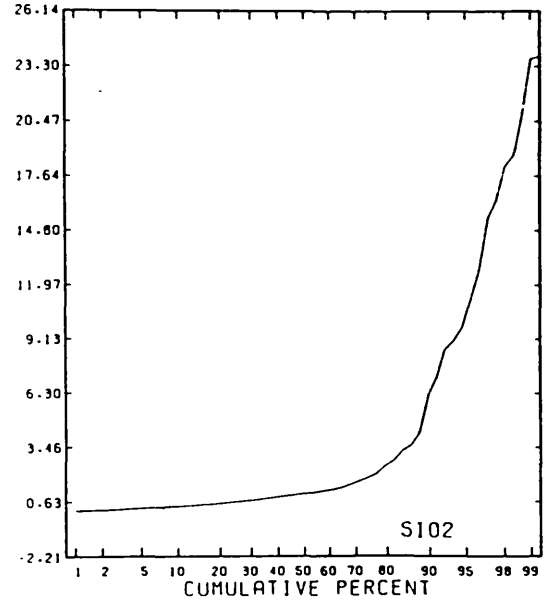


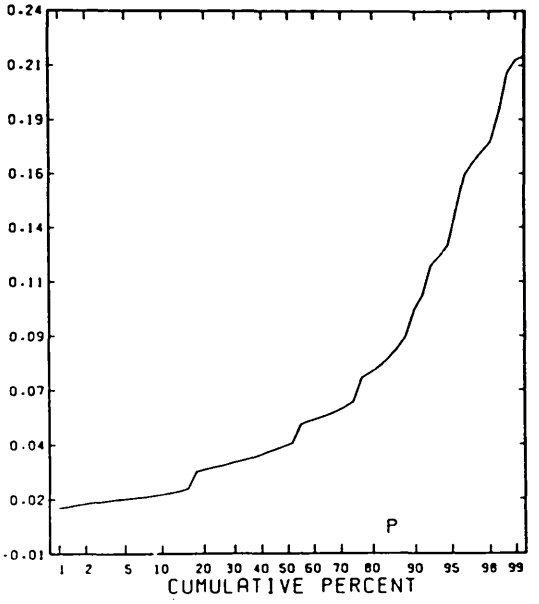
AL203
NO. OF OBSERVATIONS = 540
MEAN = 3.28 STD DEV = 6.422

Figure 6.1 Histograms of the raw data









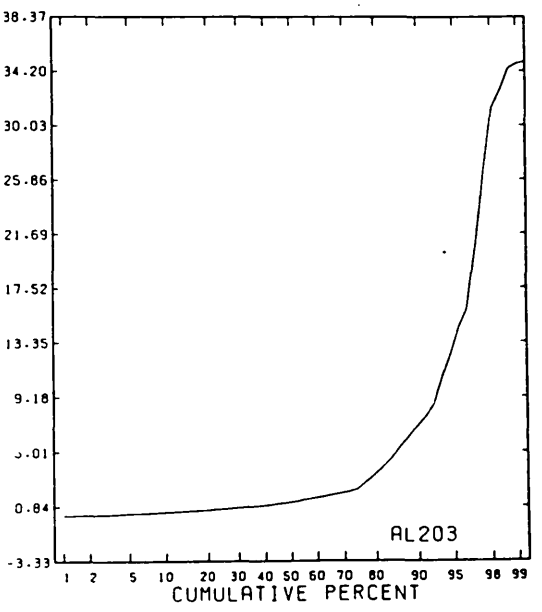
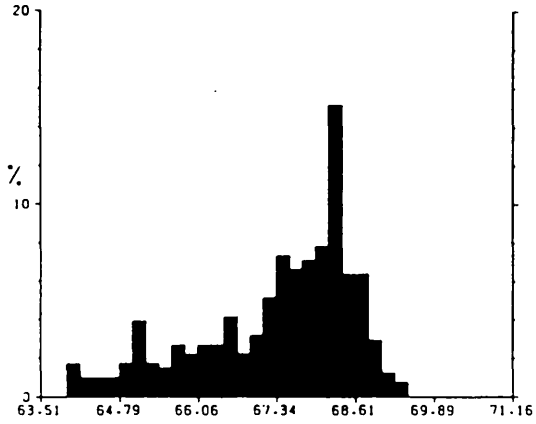
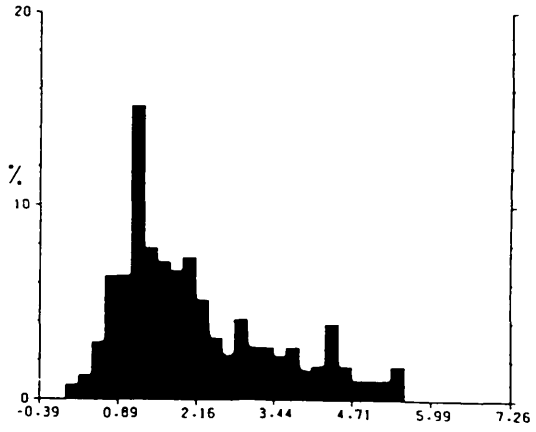


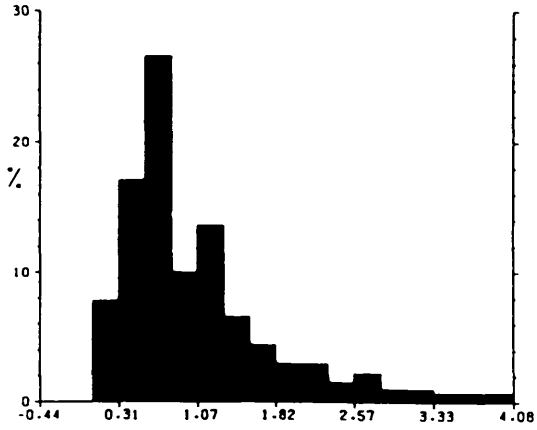
Figure 6.2 Normal cumulative probability graphs of the raw data



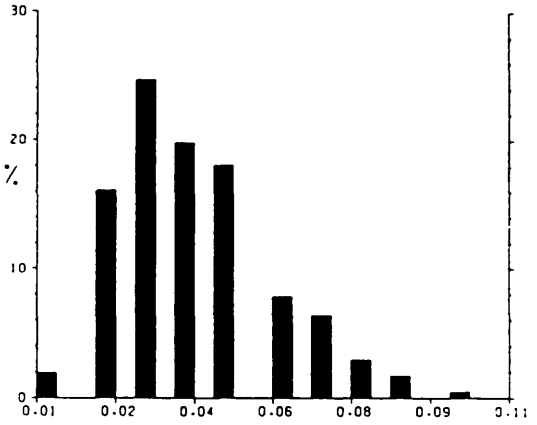
FE-HEM
NO. OF OBSERVATIONS = 409
MEAN = 67.34 STD DEV = 1.275



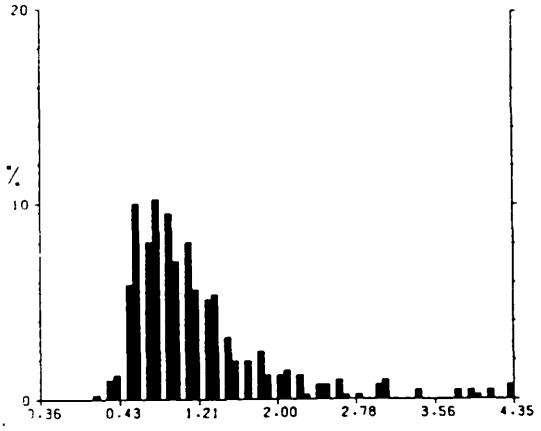
K-FE HEM
NO. OF OBSERVATIONS = 409
MEAN = 2.16 STD DEV = 1.275



SIO2-HEM
NO. OF OBSERVATIONS = 409
MEAN = 1.07 STD DEV = 0.754



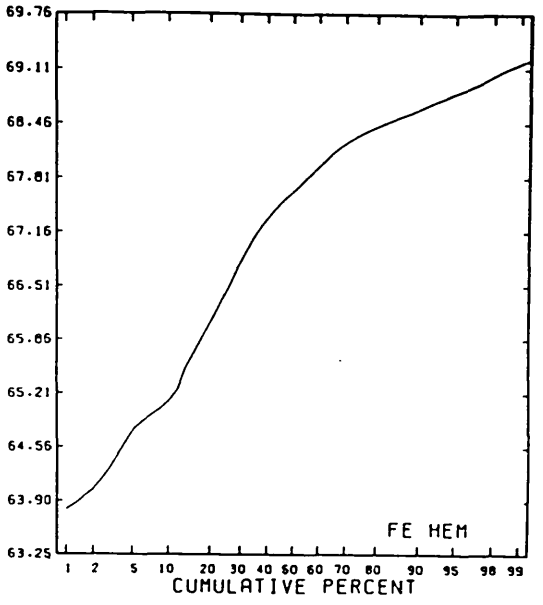
P-HEM
NO. OF OBSERVATIONS = 409
MEAN = 0.04 STD DEV = 0.018

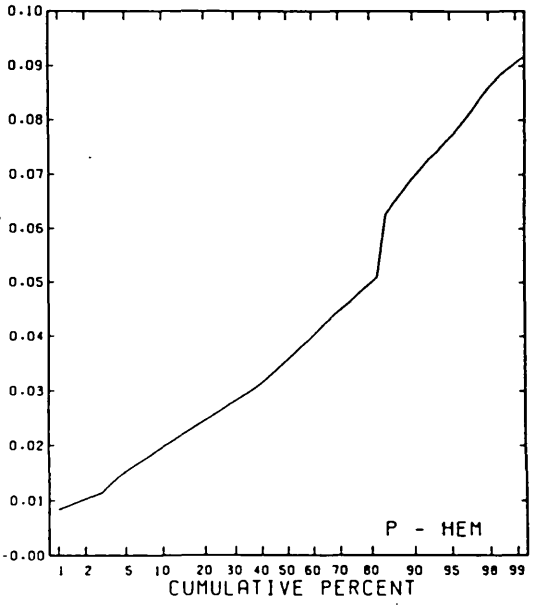


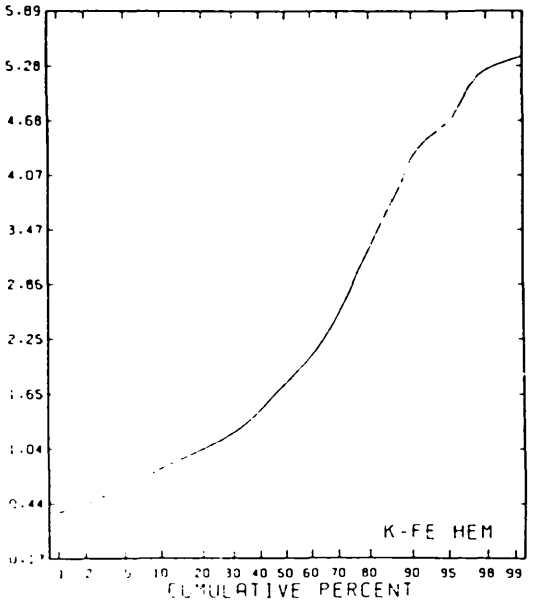
AL203-HEM
NO. OF OBSERVATIONS = 409
MEAN = 1.21 STD DEV = 0.784

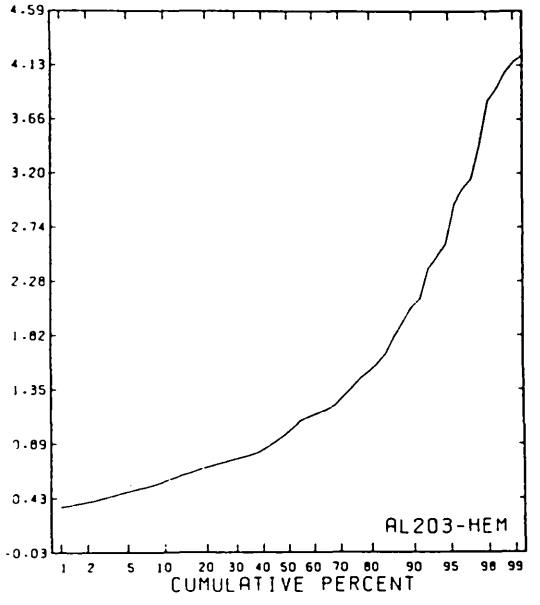
Figure 6.3 Histograms of the variables selected as follows:

64.0% =<Fe=<69.9% and P<0.12%









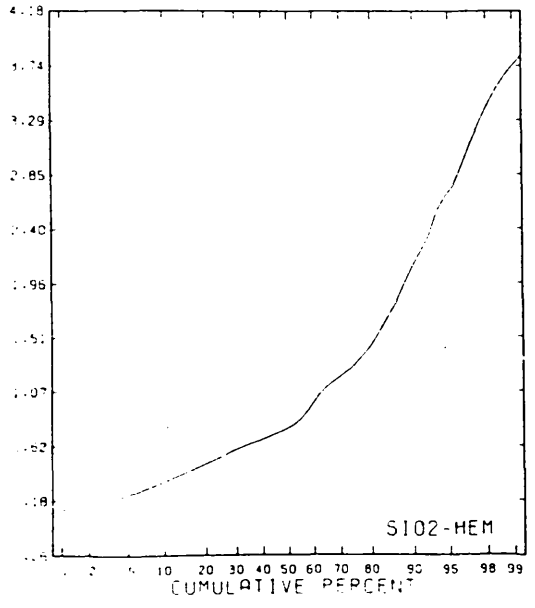
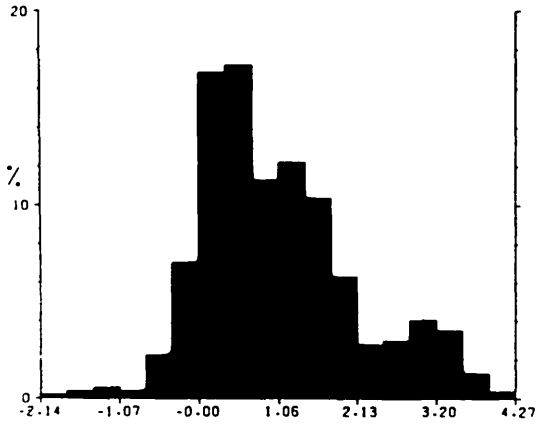
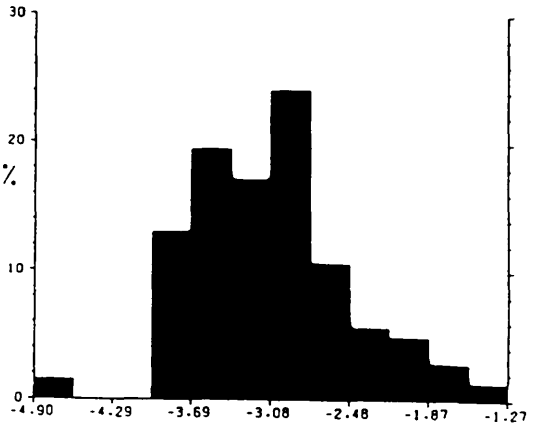


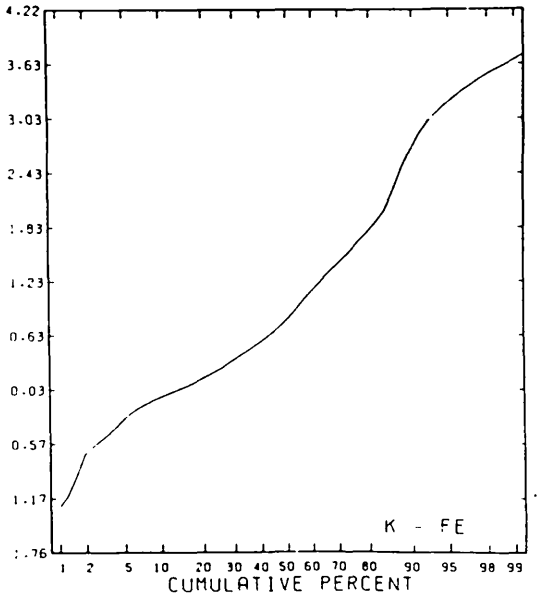
Figure 6.4 Normal cumulative probability graphs of the variables with 64.0% =<Fe=<69.9% and P<0.12%

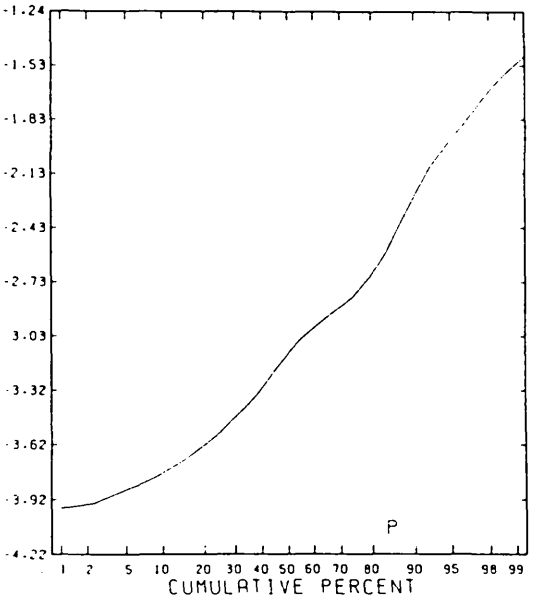


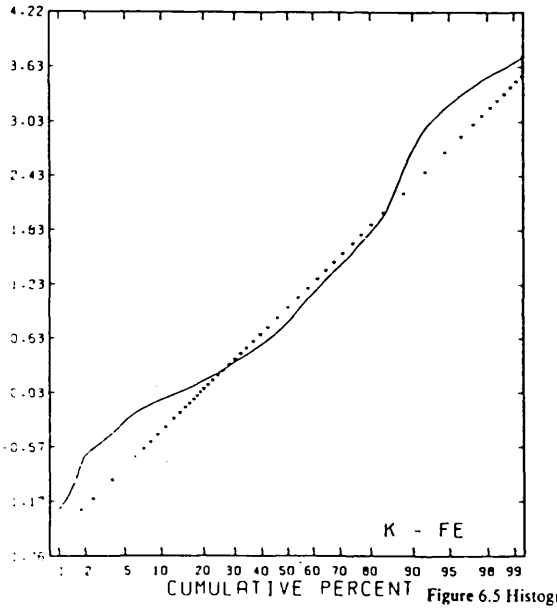
K - FE
NO. OF OBSERVATIONS = 540
MEAN = 1.06 STD DEV = 1.068
NO. OF LOG E TRANSFORMS = 1



P
NO. OF OBSERVATIONS = 540
MEAN = -3.08 STD DEV = 0.605
NO. OF LOG E TRANSFORMS = 1







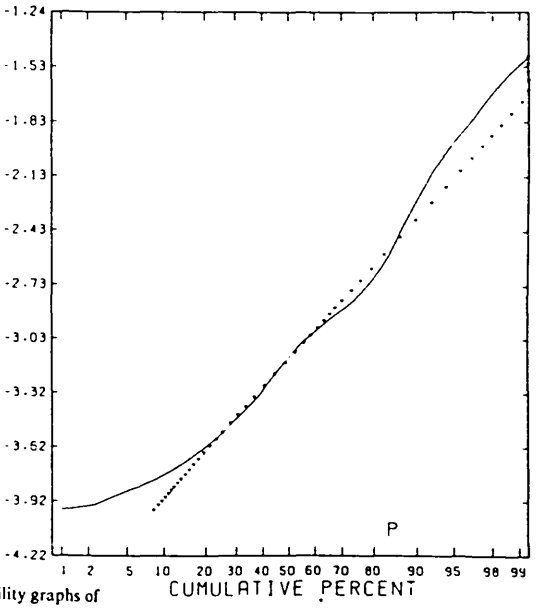
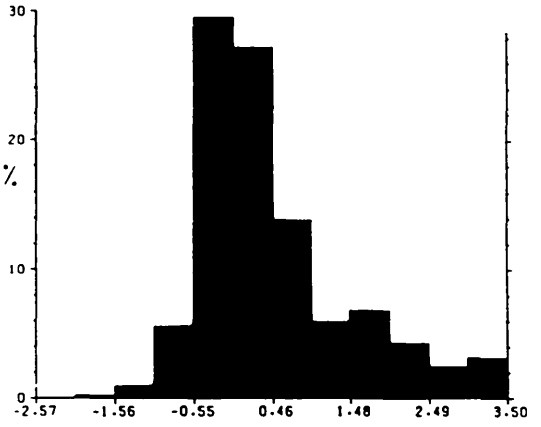
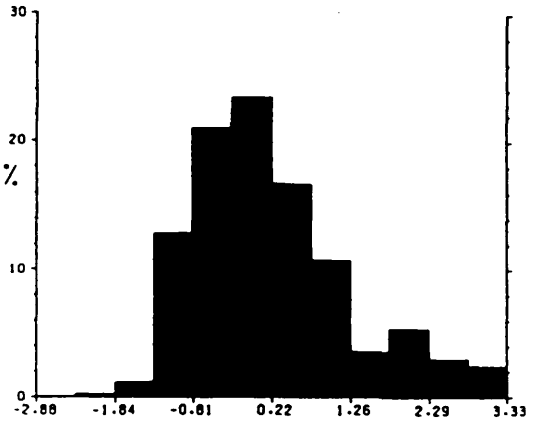


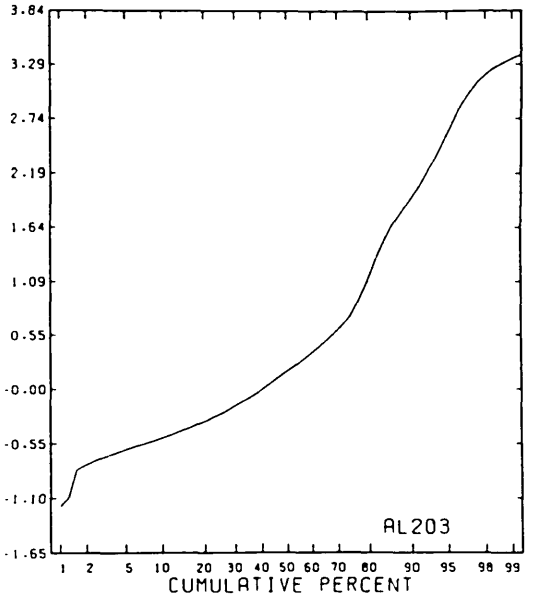
Figure 6.5 Histograms and probability graphs of the logtransformed variables - ln(k-Fe) and ln(P)

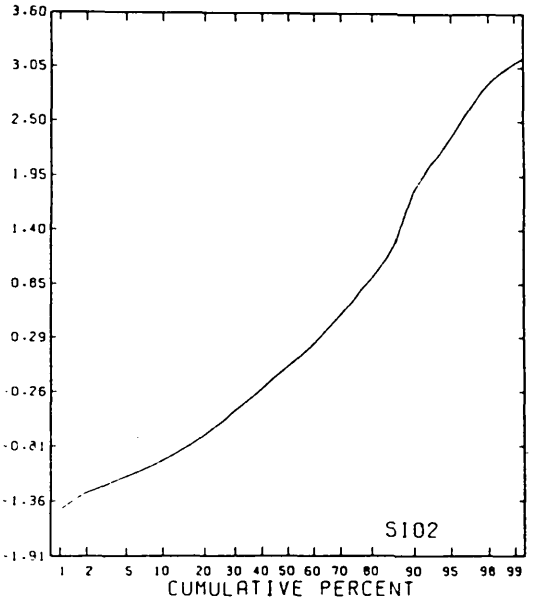


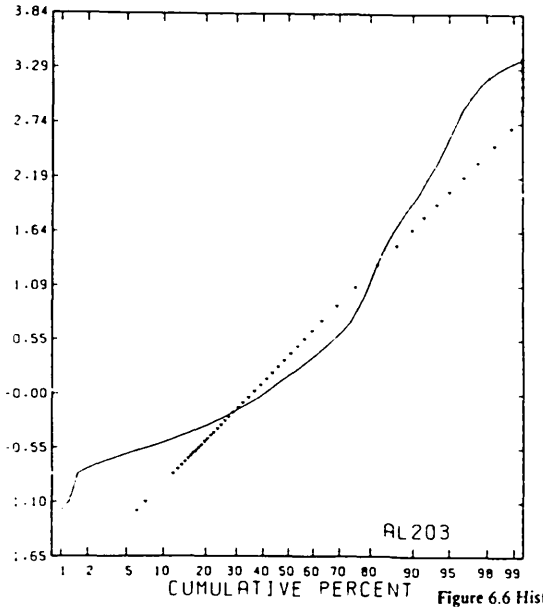
AL203
NO. OF OBSERVATIONS = 540
MEAN = 0.46 STD DEV = 1.012
NO. OF LOG E TRANSFORMS = 1



SIO2
NO. OF OBSERVATIONS = 540
MEAN = 0.22 STD DEV = 1.034
NO. OF LOG E TRANSFORMS = 1







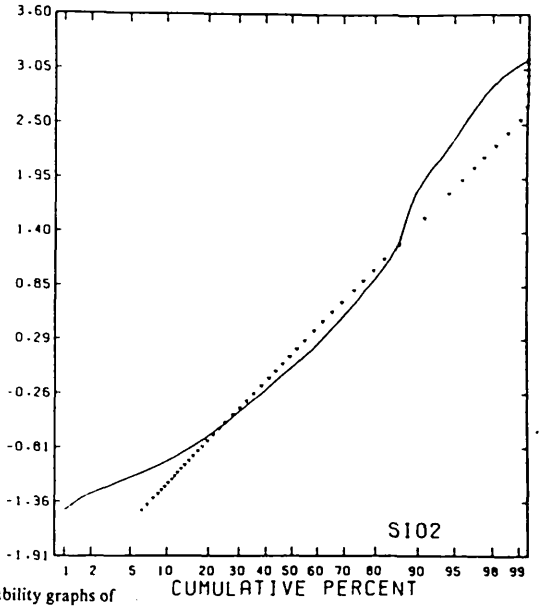
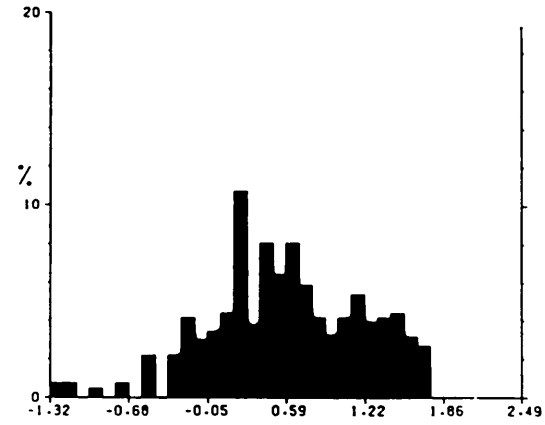
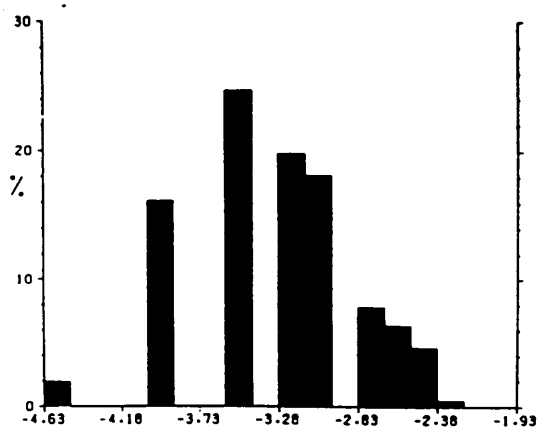


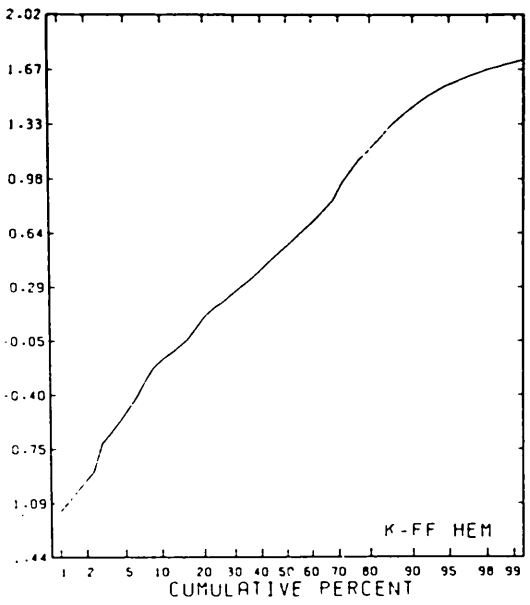
Figure 6.6 Histograms and probability graphs of the logtransformed variables - $\ln(SiO_2)$ and $\ln(Al_2O_3)$

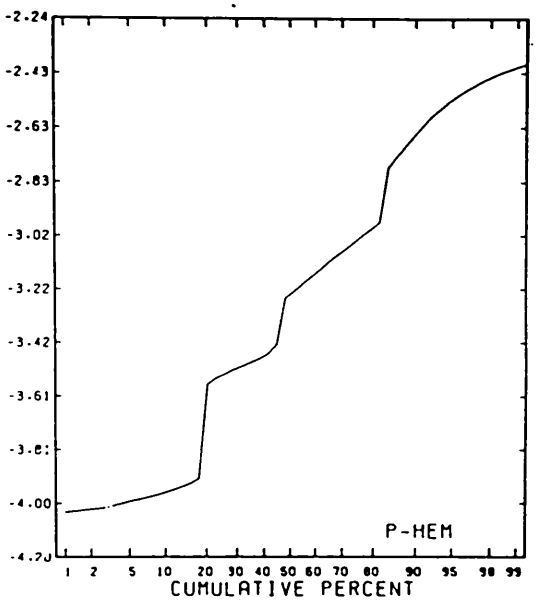


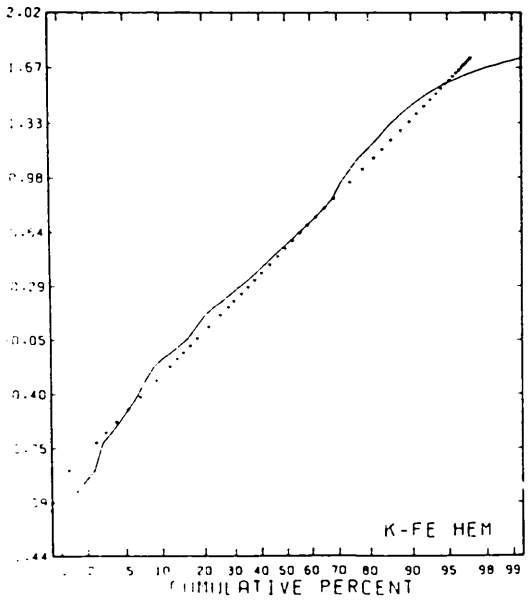
K-FE HEM
NO. OF OBSERVATIONS = 409
MEAN = 0.59 STD DEV = 0.635
NO. OF LOG E TRANSFORMS = 1



P-HEM
NO. OF OBSERVATIONS = 409
MEAN = -3.28 STD DEV = 0.451
NO. OF LOG E TRANSFORMS = 1







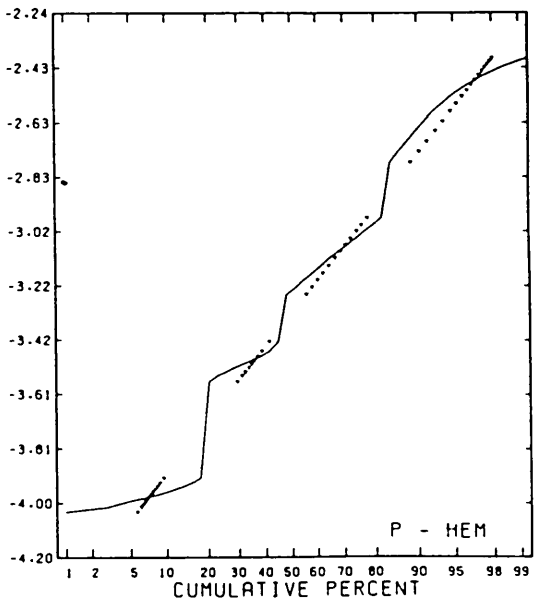


Figure 6.7 Histograms and probability graphs of the logtransformed variables with 64.0% =<Fe=<69.9 and P<.12% - ln(k-Fe) and ln(P)

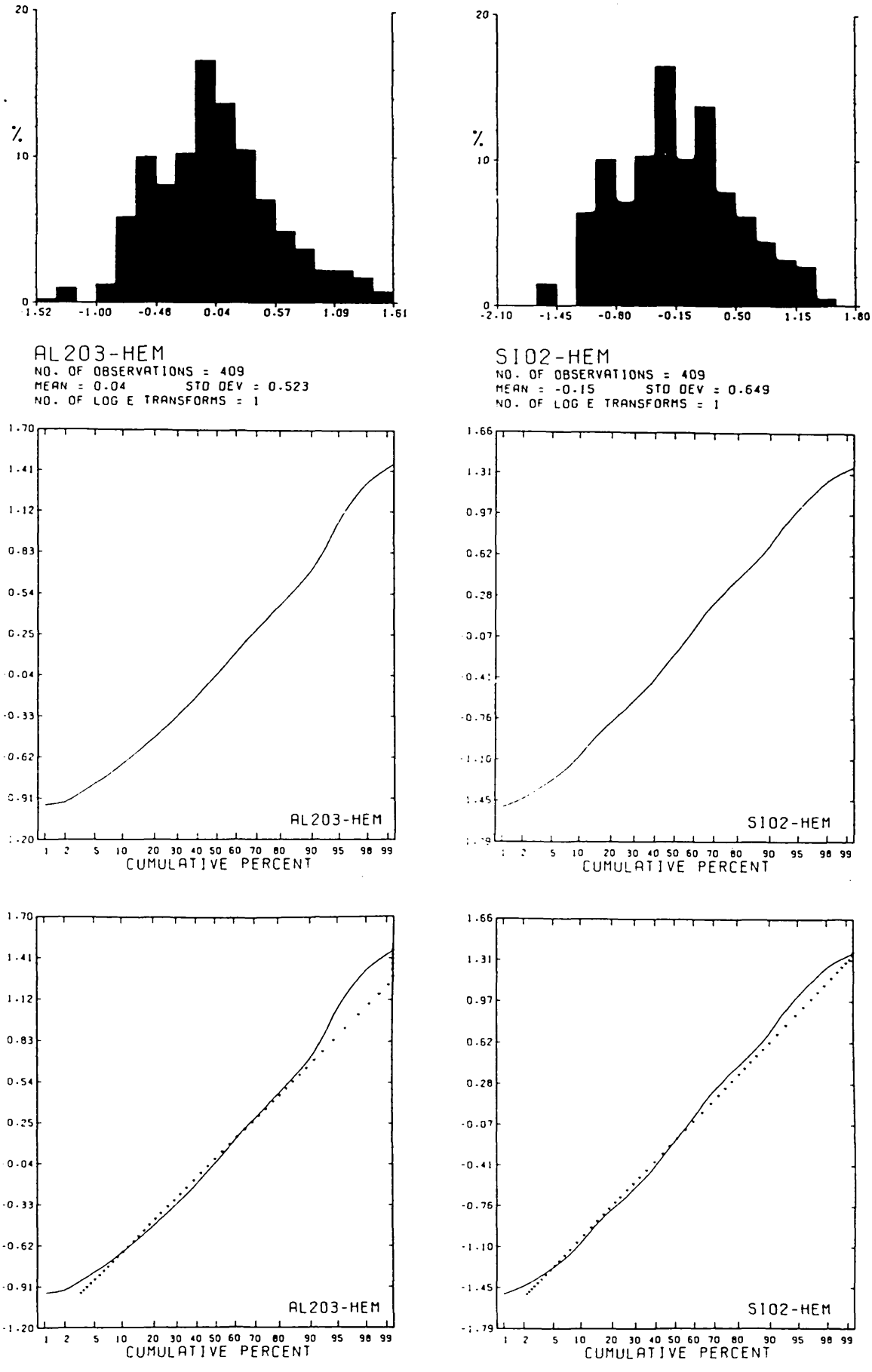
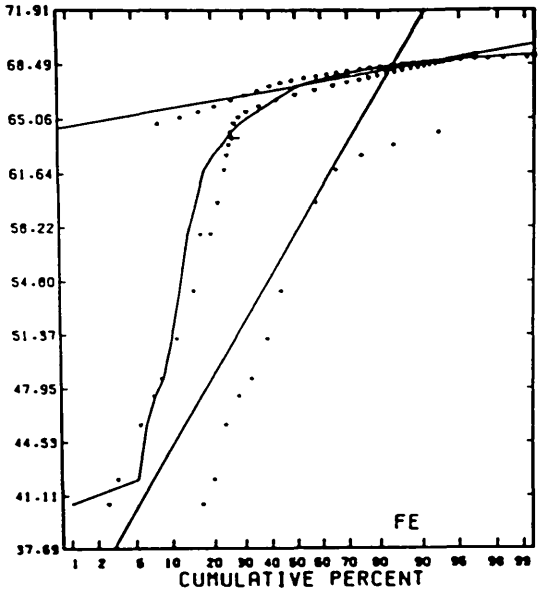
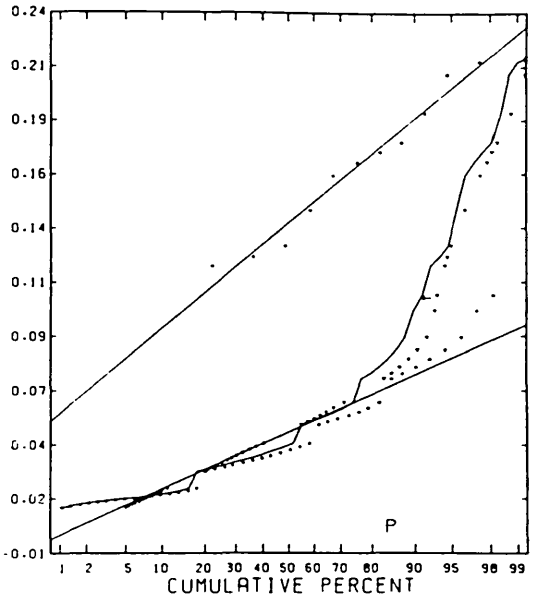
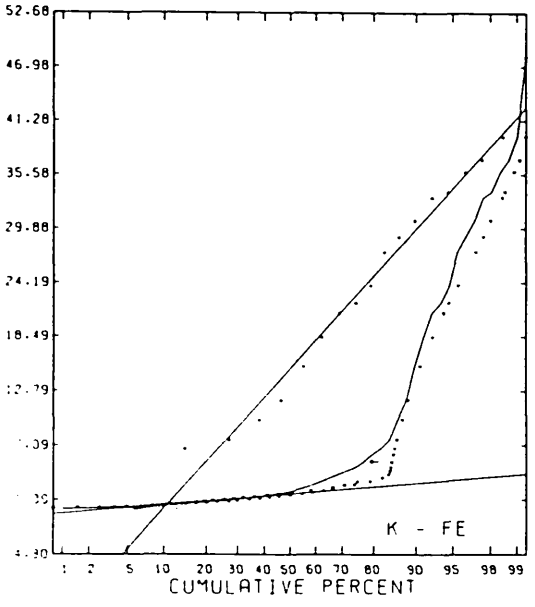
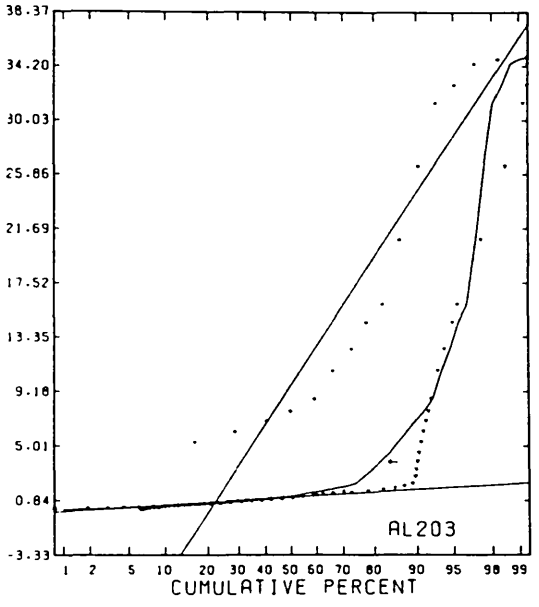


Figure 6.8 Histograms and probability graphs of the logtransformed variables with 64.0% =<Fe=<69.9 and P<.12% - $\ln(SiO_2)$ and $\ln(Al_2O_3)$









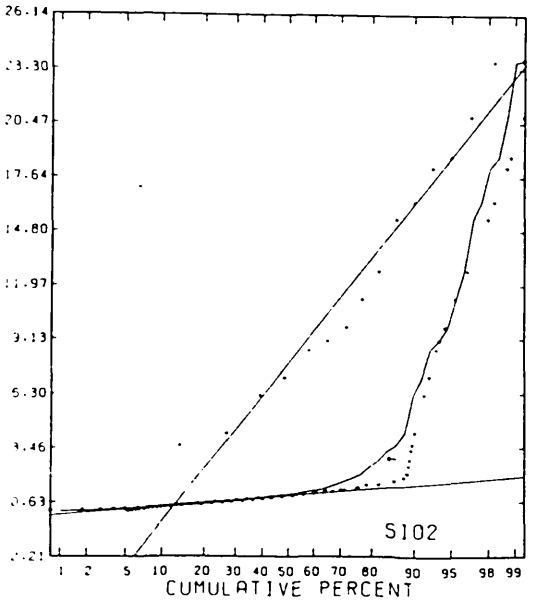


Figure 6.9 Fitting of the two normal sub-populations model to the probability curve of the raw data

7. Geostatistical analysis - variograms

7.1 Summary and conclusions

7.2 Introduction

7.2.1 Geostatistical analysis in general

7.2.2 Applications of geostatistics to iron ore reserves

7.2.3 Suggested approach for Mutuca Mine

7.3 Input data

7.3.1 Geological data base

7.3.2 Parameters for the variograms to be computed

7.4 Sector variograms of the crude ore's Fe content

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7.6.1 Isotropic variograms

7.6.2 Point kriging cross validation and statistical analysis

7.1 Summary and conclusions

A brief review of the literature related to geostatistical analysis is made both in general and as applied to iron deposits in particular, in the introduction to this chapter, together with the presentation of the suggested approach for Mutuca Mine's reserve. After that, a preliminary geostatistical analysis is made on the crude ore iron content, which is a weighted average of the Fe grade of all size fractions. Subsequently, a tentative geological interpretation by means of "categorical" or "indicator" kriging is reported and discussed, the results being compared with the conventional geological interpretation. From the three different size fractions assayed, only the (−2";+¼") one has been studied, and the following estimates have been produced for it: percentage in weight, iron content (Fe) and phosphorus content (P). Finally, the results are discussed, some recommendations for further improvements being made.

The main conclusions of this chapter are summarized below:

- despite its limitations, categorical kriging can be used as an auxiliary tool for geological interpretation.
- as far as the crude ore iron content is concerned, the Mutuca ore body has the same pattern of variability in all directions, that is to say, it behaves isotropically.
- this isotropic characteristic can be extended to the other variables that have been examined in this study. The statistical analysis which compared the sample point's actual values with their kriging estimates revealed differences that are not significant, at a 95% confidence level;
- now that the methodological framework has been established, it can be applied, exactly in the same way, for analysing other variables of Mutuca's geological data base, namely, Silica (SiO_2) and Alumina (Al_2O_3), and the other two size fractions could be estimated, i.e., (−¼";+100Mesh) and (−100Mesh), as well as the crude ore.

7.2 Introduction

In the first place there are two questions that need to be answered : "Why geostatistics?", and "Is it worth applying it to Mutuca Mine?".

Royle (1980, page 14) has answered the first question, and the discussion that follows approaches the subject in the context of the present applied research work.

According to Royle(1980, page 14), "a brief argument for the use of geostatistics would make the following points :

- Geostatistics is soundly based on good valuation practice, in that it provides a rational theoretical basis for intuitive mine valuation practices of the past.
- Recognition of the fact that total variation is partly random and partly spatial leads to estimates that are unbiased and have lower errors of estimation. Geostatistics also shows where to look for sources of errors.
- Geostatistics explains why some traditional types of estimator, such as the polygonal method, produce biased estimates. These biases may be removed by using other kinds of estimators, or kriging.
- Grade-control problems are amenable to geostatistical treatment because the number of working stopes needed to keep mill-feed grade fluctuations within predetermined limits can be easily determined.
- Valuation and mining operations can be kept under statistical control, and deviations, which must occur with some calculable frequency, can be forecast and dealt with. The unbiased nature of geostatistical estimations is of prime importance for such control. They do not create compensating errors that mask other deficiencies in ore estimation, mining or milling.
- Sampling and valuation programs can be designed economically, and geostatistics can help optimize sampling patterns".

Mendelshon (1980) divides the methods of ore reserve estimation into four groups: geometric, statistical, mathematical and geostatistical. After summarizing the characteristics of each system, he affirms: "Geostatistical kriging methods call for a lot more work and mathematics than the empirical methods, but give unbiased estimates, enable [kriged] variances to be calculated, and allow the derivation of unbiased grade-tonnage curves"(Mendelshon, 1980, page 20).

King (1982, page 14) lists the estimation procedures from the simplest to the more elaborated techniques:

1) an old style approach (the samples are lumped together for calculating average grade and width);

2) the polygonal method;

3) the triangular method;

4) a sectional method;

5) a contouring method;

6) a "sphere of influence" method; and

7) geostatistics.

He discusses the advantages and disadvantages of each approach with interesting remarks:

"(...) the simple, even if over-simple assumptions of the earlier procedures become, in the later, so wrapped up in mathematics and computer print-outs that the assumptions can be lost sight of. For example, in the sphere of influence method, one of the formulae that we have met is weighting of a peripheral sample by the inverse square of its distance from the central sample. Now the inverse square weighting is valid for light and for gravity but so far as we know it has no place in ore geology. At best, the inverse square weighting is a severe discounting of the influence of peripheral samples. At worst it suggests a scientific basis that it does not possess."

On the other hand, amongst other reasons for being specially interested in geostatistics, he mentions that it seems to offer a better estimate of block grades and, in addition, a measure of the accuracy of the estimates (King, 1982, page 18).

MBR has been using geostatistics regularly for iron ore reserve evaluation since 1982, thus its applicability in the case of Mutuca Mine obviously required investigation.

Other factors in favour of using geostatistics techniques were:

- a series of applications envisaged for the mineral inventory produced through kriging would potentially use the error of estimation (kriging error), for instance, computer-assisted open-pit design and economic evaluation;
- the speed and reliability of computerized methods, specially in view of the on-going nature of the drilling process.

No explanations on the theory of geostatistics are given in this text. The merits of kriging over other evaluation methods, as far as precision is concerned, have been discussed by Dunlop (1979), Rendu (1979, page 207) and Royle (1980).

7.2.1 Geostatistics in general

Early in the 1960's, Matheron (1963) published a paper introducing new concepts in the field of ore reserve estimation:

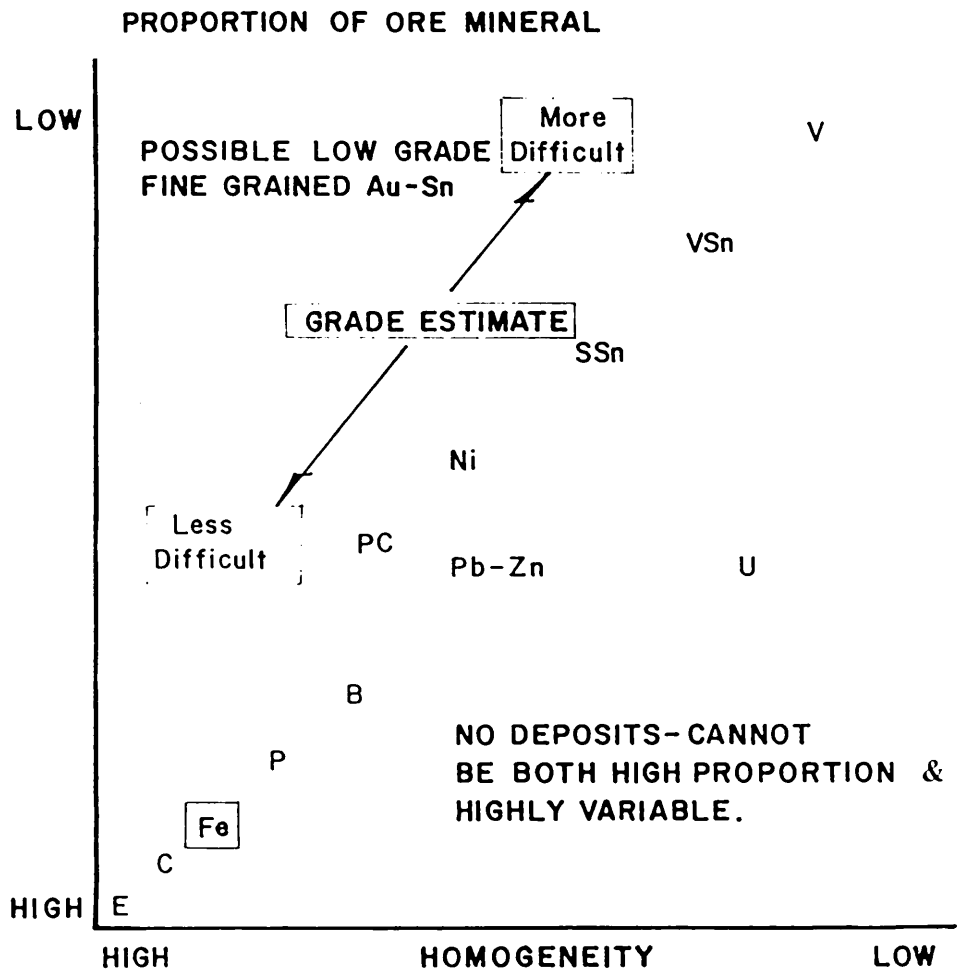
"Knowledge of ore grades and ore reserves, as well as error estimation of these values, is fundamental for mining geologists. Until now no appropriate scientific approach to those estimation problems has existed: geostatistics (...) constituted a new science leading to such an approach. The author criticizes classical methods still in use, and shows some of the main results given by geostatistics. Any ore deposit evaluation as well as proper decision of starting mining operations should be preceded by a geostatistical evaluation which may avoid economic failures (...)

(...) As a matter of fact, the South African school, which has recorded the most remarkable results with Krige, Sichel, used to say, and believed that they were applying classical statistics. But the methods they were developing differed more and more from classical statistics, and adjusted themselves spontaneously to their object."

Matheron's first article was followed by a series of other related papers, as attested by extensive bibliographies contained in the following publications: Krige (1981), Rendu (1981), Royle (1980), Bell (1979), Clark (1979), Pauncz (1978), Journel (1978), David (1977) and the AESI's special list on "Geostatistics..." (1983).

Geostatistics has been substantially developed since its creation, and there is a trend nowadays towards applying geostatistics in most evaluations of mineral reserves.

In parallel, the mining industry has been equipping itself with the tools and technical staff required for geostatistical analyses. See, for example, Magri (1979).



E - EVAPORITE
C - COAL
Fe - BEDDED IRON ORE
P - PHOSPHATE
B - BAUXITE
Pb-Zn - STRATIFORM LEAD-ZINC
Ni - STRATIFORM NICKEL
SSn - STRATIFORM TIN
PC - PORPHYRY COPPERS
VSn - TIN VEINS
V - GOLD SILVER VEINS
U - URANIUM

PROPORTION OF ORE MINERAL X HOMOGENEITY

Figure 7.1 Proportion of ore mineral vs homogeneity

Sources : King(1982); and Carras(1984).

7.2.2 Applications of geostatistics to iron ore reserves

When compared to the grades of other minerals, iron ore grades are relatively easy to estimate. "Direct shipping" iron ores have a particularly high homogeneity. The relationship represented in figure 7.1 illustrates this, indicating not only the degree of risk in estimating grade, but identifying a geological factor in this risk (see King, 1982; and Carras, 1984).

Dunlop (1979) describes the geostatistical modelling of an Australian iron ore body. Ore was classified as that material having an iron level of 55% [or more], and iron grades ranged from 40 to 60%, under the form of hematite, limonite and goethite. All holes were sampled at two meter intervals, and assayed for Fe, loss on ignition (LOI), SiO_2, Al_2O_3, P, TiO_2, S, CaO and MgO. Variograms were computed for Fe, LOI, SiO_2, Al_2O_3 and TiO_2. In the case of iron, it was observed clearly the absence of marked anisotropy. "The result was not unexpected, as many sedimentary iron deposits have been shown to be spherical in nature, with ranges between 300 and 500 metres (...)". All variables, except LOI, tended to be spherical in nature, with ranges of over 300 metres.

Figure 7.5 features a "model variogram", and additional details on the "variogram" are given by Barnes (1980, page 41). Both the "spherical model" and "anisotropy" are described in the same publication, respectively, in pages 45 and 43.

Suarez (1978) developed a methodology for multivariable estimation and simulation of large three-dimensional deposits, focusing on an iron ore mine in th the Pilbara region of Australia. Two geostatistical structures were identified for the iron content.

- Along the mineralisation axis:
  a) medium scale structure - range 260m, sill 189 $(\%)^2$Fe;
  b) large scale structure - range 1200m, sill 61 $(\%)^2$Fe.
- Across the mineralisation axis:
  a) medium scale structure - range 86.6m, sill 189 $(\%)^2$Fe;
  b) large scale structure - range 400m, sill 61 $(\%)^2$Fe.
- In the vertical direction:
  a) medium scale structure - range 52m, sill 189 $(\%)^2$Fe;
  b) large scale structure - range 240m, sill 61$(\%)^2$Fe.

Suarez(1978) has chosen the spherical model to represent all experimental variograms. The above parameters characterize an anisotropic behaviour of the iron content, which has an average of nearly 50%, ranging from 11% to 66%, with a negatively skewed distribution.

Marechal (1977) studied an iron ore body in Pilbara, Australia. Variograms were run for Fe, SiO_2, Al_2O_3, P and LOI. These experimental variograms "showed a strong anisotropy between the different horizontal directions and the vertical direction, and do not allow a good definition of the theoretical model over small distances."

The variability of the iron ore body was modelled as follows:

a) medium scale structure - spherical model with a 170m range for Fe, SiO_2 and Al_2O_3, 60m being the range for P and LOI; and

b) large scale structure - exponential model with a 340m parameter for Fe, SiO_2 and Al_2O_3, and 270m for P and LOI.

The iron ore bodies of the Pilbara region are formed by supergene enrichment of banded iron formation and rarely do such processes produce orebodies with uniform ore grade distributions.

O'Leary (1979) reported a geological/geostatistical approach to the problems of reserve estimation and grade control at a Canadian iron ore mine. The ore body, which is intensely folded, was divided into 5 units totalling approximately 650-750 ft thick and averaging 36% iron and 1.7% manganese. The ore undergoes two concentration stages, with the final concentrate requirements being 66.3% Fe and 2.25% SiO_2. The ore deposit was conceptually unfolded and the ore grades were analysed by way of statistics, polynomial

trend surfaces being used to estimate block grades.

David (1977, page 215) mentioned an example of a taconite iron mine with a Fe content around 33%. The model spherical variograms used horizontal ranges of 240 and 120 metres, and a vertical range of 36m.

Matheron (1963) wrote about an early application of geostatistics to the Lorraine iron ore mine, in France. No numerical data or results are given in that report, which has been mentioned here just for its historical value.

Bubenicek (1969) described the method of calculation of the iron ore reserves in the Lorraine deposit. Assaying covers Fe, CaO, SiO_2, Al_2O_3 and P. The spherical model was used to represent the experimental variograms. Both thickness and iron ore content were analysed in detail, the latter presenting discontinuity at the origin (nugget variance - C0); very marked anisotropy; a transition amplitude (C); and a well defined range (A), beyond which the semi-variance turns into a horizontal line. One variogram of the iron content features a range of 750 metres, a sill of 9.6 $(\%)^2$Fe and a nugget variance of 4.2, therefore presenting a nugget effect (C0/C) of around 0.9 (Bubeniceck, 1969, page 199).

Serra (1967, page 287) discussed the structural characteristics of the Toarcian deposit in the Lorraine area, showing the anisotropic behaviour of this ore body. The ranges of the variogram model for the directions NS and EW are, respectively, 550m and 200m. Both nugget variances are 4$(\%)^2$Fe, and the sills are the same for the two directions as well, 16.7$(\%)^2$Fe.

Barnes (1980, page 45) mentions that the spherical model represents adequately such diverse deposits as iron ore bodies, porphiry copper deposits, stratibound lead - zinc deposits, bauxite and lateritic nickel, as well as uranium and phosphate deposits.

Clark (1979, page 94) presented an example with a simulated deposit, which is a low grade sedimentary iron ore with an overall average of about 35% Fe, a standard deviation of 5% Fe, a range of influence of 100m and a sill of 25$(\%)^2$Fe. The model semi-variogram is spherical, with no nugget effect. This example was also discussed by Henley (1981) for introducing a non-parametric geostatistical approach to the problem of block estimates. Three different options were compared with the normal block kriging solution. The non-parametric methods are not as good as kriging at predicting block average values, however, they have a smaller bias (expressed as the algebraic sum of deviations).

The ore body of Águas Claras, an open-pit iron ore mine owned by MBR, in Brazil, was analysed by way of geostatistics. The model semi-variograms selected for Fe used the following parameters:

(−2";+¼"), nugget variance=0.05; sill=1.33(in %²Fe);

ranges(in m), EW=120; NS=40; Z=50.

(−¼";+100Mesh), nugget variance=0.10; sill=0.94(in %²Fe);

ranges(in m), EW=100; NS=55; Z=50.

(−100Mesh), nugget variance=0.15; sill=1.42(in %²Fe);

ranges(in m), EW=90; NS=70; Z=50.

This deposit presents geometric anisotropy. (Source: MBR; 1982; unpublished report). These results are particularly interesting because they were obtained for a high iron content hematite deposit located merely at a 15km distance from Mutuca Mine.

7.2.3 Suggested approach for Mutuca Mine

Mutuca's ore body is composed mainly of high grade hematite, a "direct shipping" ore(see Ohle, 1983, pages 50 and 57), therefore its iron content is not as variable as that of low grade iron ore deposits which are suitable for concentration processes. On the other hand, it is intensely folded (see geological description, chapter 3).

The geostatistical analysis used hematite data only, according to the classification adopted by MBR, and the geological data have not been composited on a benchwise basis, as the number of samples is not particularly large to be handled by a computer(O'Leary, 1985, personal communication). According to Carras(1984, page 10), if the the coefficient of variation of gold data is greater than 1.2, the data must be composited to reduce it. As the grades of Mutuca's ore present low coefficients of variation(see table 6.2), the geological data have not been composited.

Although data on % in weight, Fe, P, Al_2O_3 and SiO_2 for four different size fractions are available, as well as for the weighted average (crude ore), the geostatistical analysis was focused on three variables only: iron and phosphorus content and the percentage in weight, all these variables referring to the (−2"; +¼") size fraction. This granulometric bracket generates the chief product of Mutuca Mine, Lump Ore for Direct Reduction (LODR).

The Fe content has been selected to be studied in the first place because the ore price is calculated as a function of the iron content in most international sales contracts. Also, together with the P grade, it is used for the geological classification of hematite in MBR (see chapter 3).

A preliminary analysis of sector variograms of the Fe content revealed that the ore

body presents the same pattern of variability in all directions, that is to say, it behaves isotropically, therefore a spherical isotropic model has been chosen to represent the experimental variograms(see figures 7.2, 7.3, 7.4 and 7.5).

Subsequently, isotropic variograms were run for Fe, P and % in weight of the (−2";+¼") size fraction. Spherical models were fitted to the experimental variograms. The model variogram was calibrated by way of a procedure called "point kriging cross validation", which uses the data sorrounding each sample point for producing kriging estimates for the point. Afterwards, a statistical analysis compared the sample points' actual values with their kriging estimates, aiming at validating the model variogram.

7.3 Input data

7.3.1 Geological data base

The spatial position of each geological sample is given by the coordinates of its geometric centre, namely, EW, NS and elevation above sea level. Assays are available for Fe, P, Al_2O_3 and SiO_2, in the four size fractions: (−2";+¼"), (−1";+¼"), (−¼";+100 Mesh) and (−100 Mesh). The (−2";+¼") and (−1";+¼") size fractions were combined to form the (−2";+¼") composite, the resulting grades being weighted averages. Similarly, the crude ore grades are the weighted average of all size fractions'.

7.3.2 Parameters for the variograms to be computed

Sector variograms: if this option is to be used, both the number of sectors and the "angle of regularization" are needed. For instance, 4 sectors with an angle of regularization of +/− 22.5 degrees would cover 180 degrees. This, by symmetry, represents 360 degrees, that is to say, an entire plan, as samples are searched for in both orientations, in each direction, for the sake of pairing.

Lag distance, sampling intervals and number of lags: the average sampling distance in Mutuca is around 6 metres, with 3 metres being by far the most frequent sampling interval. The lag distance adopted to run the variograms was 12 metres, with 50 lags, covering a maximum distance of 600 metres. For the preliminary analysis, 30 lags of 12 metres had been used, with a maximum range of 360m.

7.4 Sector variograms of the crude ore's Fe content

A preliminary assessment of the ore body's variability was made using the iron content of the crude ore. Sector variograms were run each 45 degrees in three mutually perpendicular planes, with angles of regularization of +/− 22.5 degrees. For example, in the horizontal plane, moving clockwise from north (0 degrees), four sector variograms were computed, centered at lines lying at 0, 45, 90 and 135 degrees. A similar procedure was followed in vertical planes, across the ore body and along it, respectively, in the EW and NS directions.

All computations were made using the "MINPAK" library of subroutines, which has been developed by Forkes and O'Leary. Plots were produced through the "AGRAPH" package, which has been created by Raby (1984) and is available at the Imperial College Computer Centre(ICCC).

7.4.1 Numerical results and graphical outputs

Table 7.1 shows a typical output of a sector variogram. Figures 7.2, 7.3 and 7.4 revealed that the crude ore's iron content has the same pattern of variability in all directions. In other words, it is isotropic.

7.4.2 Tentative selection of a model variogram

A isotropic spherical model was preliminary selected for representing the experimental variograms. Its parameters were: range, 200 metres; nugget variance (C0), 0.80 $(\%)^2$Fe; and transition amplitude (C), 0.80 $(\%)^2$Fe. A nugget effect (C0/C) of 100% resulted.

7.4.3 Calibration of the model variogram through point kriging cross validation

Provided the model variogram fits well, there is no need for "back kriging"[point kriging cross validation], according to Armstrong (Centre de Géostatistique et de Morphologie Mathématique; 1985; Fontainebleau; France; personal communication). However, the "goodness of fit" of the model variogram which had been preliminary selected was appraised with the aid of a procedure called "point kriging cross validation"(see item 7.2.3).

The calibration exercise uses the ratio of estimated variance to kriging variance as a sort of goodness of fit index of the model chosen (see Royle, 1979, page 55). In that particular case, the nugget variance (C0) was changed by plus and minus 10%, the resulting aforesaid ratio being observed, as summarized in table 7.2.

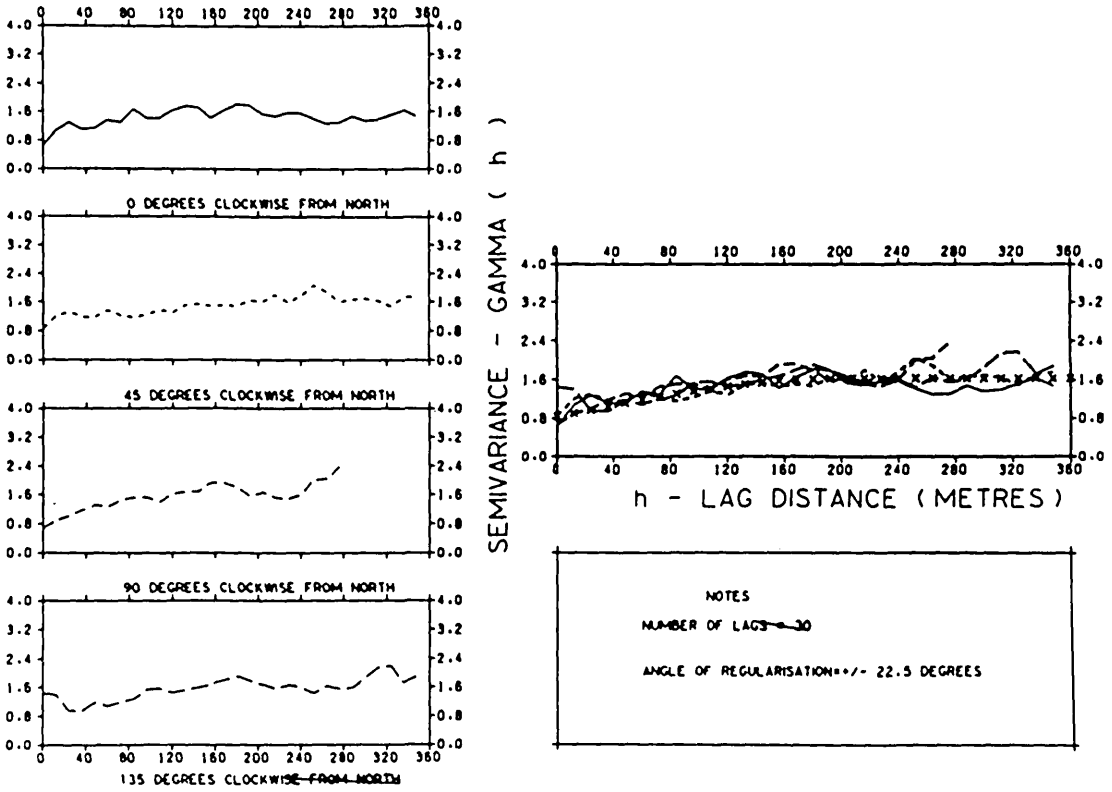
DIRECTIONAL SEMI-VARIOGRAM OF HEMFECRUDH

DIRECTION OF SEMI-VARIOGRAM: 45.000 DEGREES CLOCKWISE FROM NORTH

ANGLE OF REGULARISATION: +/- 22.500 DEGREES LAG DISTANCE = 12.00000

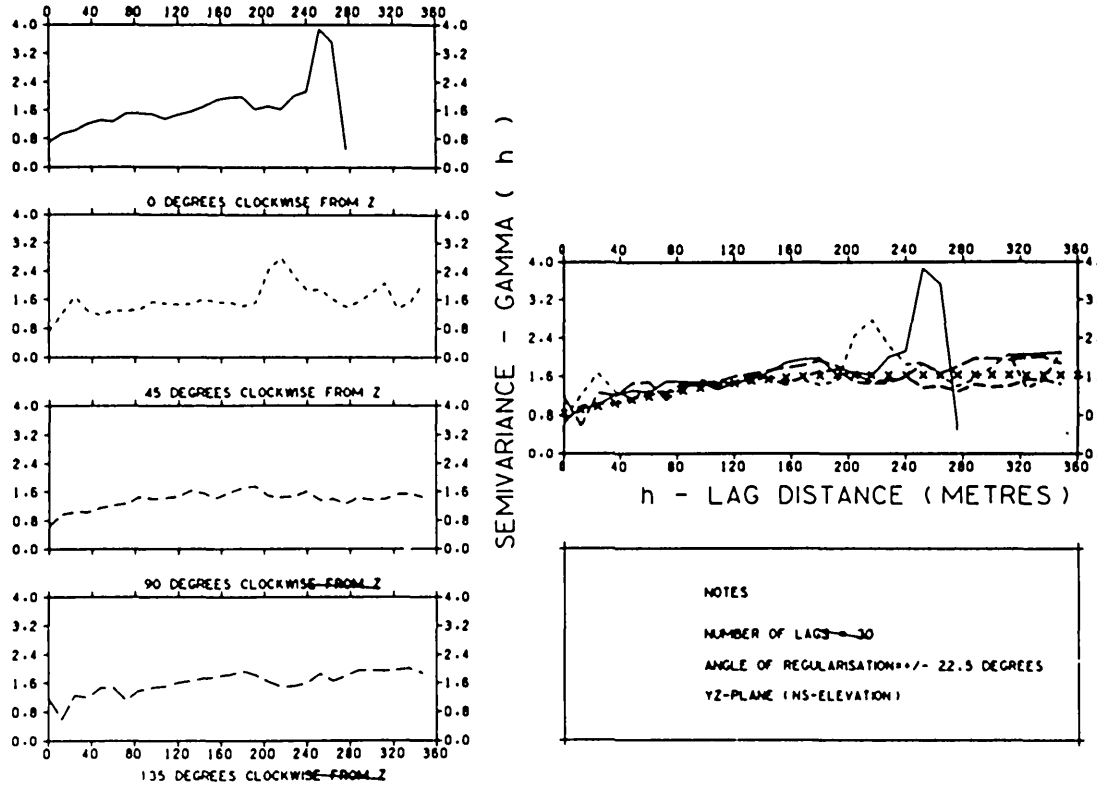
LAG NO.	LAG DISTANCE	DISTANCE	SEMIVARIANCE	NO. PAIRS	COVARIANCE	DRIFT	SILL VALUE
1	.000	4.005	.9423	15	.1465	.5800	.9888
2	12.000	13.238	1.1978	119	-.0220	.2849	1.1758
3	24.000	24.589	1.3075	277	.1751	.1245	1.4826
4	36.000	36.096	1.1717	399	.1118	-.0536	1.2835
5	48.000	48.601	1.1687	546	.0938	.1996	1.2625
6	60.000	60.196	1.3618	1220	-.0502	.3459	1.3116
7	72.000	71.814	1.2082	1119	.1041	.1437	1.3123
8	84.000	84.047	1.1556	1215	.0631	.1653	1.2188
9	96.000	95.931	1.2412	1269	.0849	.0162	1.3261
10	108.000	109.824	1.3641	2660	-.0424	.2525	1.3216
11	120.000	120.016	1.2956	4934	.0359	.1938	1.3315
12	132.000	131.979	1.5061	4679	-.0977	.1389	1.4083
13	144.000	143.908	1.5299	4140	-.0996	.0224	1.4304
14	156.000	155.927	1.4744	3810	-.0279	.0419	1.4464
15	168.000	167.854	1.5094	3606	-.0998	.0024	1.4096
16	180.000	179.790	1.4788	2910	-.0254	.0759	1.4535
17	192.000	191.950	1.5360	2517	.0051	.0074	1.6411
18	204.000	203.689	1.5022	2141	.0066	.1078	1.6089
19	216.000	217.056	1.8003	2644	-.1658	.4085	1.6345
20	228.000	227.987	1.5798	3754	.0823	.3150	1.6621
21	240.000	239.687	1.7704	3228	-.0303	.0206	1.7401
22	252.000	251.749	2.0551	2426	-.1036	-.0267	1.9515
23	264.000	263.523	1.8850	1659	.0260	-.0551	1.9110
24	276.000	275.636	1.5978	1214	.1550	-.0040	1.7528
25	288.000	287.562	1.5779	761	.1638	.4650	1.8416
26	300.000	300.446	1.5884	788	-.1216	.4901	1.5668
27	312.000	311.708	1.5443	831	-.3016	.7866	1.3427
28	324.000	324.771	1.4875	1441	-.2173	.4268	1.2702
29	336.000	335.722	1.7239	1602	-.1739	.3160	1.5500
30	348.000	347.618	1.7622	1096	.0352	-.0050	1.7974
31	360.000	359.562	1.7960	791	-.0146	-.2999	1.7814
32	372.000	371.507	1.7279	494	.0381	-.4231	1.7661
33	384.000	384.083	1.7474	415	-.0067	-.6400	1.7407
34	396.000	395.349	1.5412	234	.4363	-.7184	2.0774
35	408.000	408.972	1.9994	315	.0291	.0413	2.0285
36	420.000	419.303	2.2816	268	-.0316	.8101	2.2500
37	432.000	433.415	2.3083	309	-.0156	.0045	2.2927
38	444.000	444.517	3.0426	329	-.5394	-1.5726	2.5031
39	456.000	454.976	2.9068	227	-.5184	-1.3502	2.3883
40	468.000	467.363	2.5652	90	-.1659	-.8978	2.4993
41	480.000	479.045	1.5835	42	.3687	.0167	1.9521
42	492.000	491.240	3.9731	13	-1.5165	.0000	2.4566
43	504.000	502.038	2.7946	13	-.7183	-2.1231	2.0763
44	516.000	518.420	2.1773	73	.2718	-.0055	2.4491
45	528.000	527.207	2.1863	72	.7741	.7236	2.9604
46	540.000	537.494	2.5577	15	1.3390	1.0067	3.9967
47	552.000	552.626	3.5696	27	-1.4857	-2.1704	2.1839
48	564.000	562.813	2.9458	18	-1.3293	-1.5833	1.6165
49	576.000	573.770	2.5592	6	-1.0704	-2.1500	1.4888
50	588.000	NO PAIRS AT THIS LAG					

Table 7.1 "MINPAK" typical output of a sectorial variogram



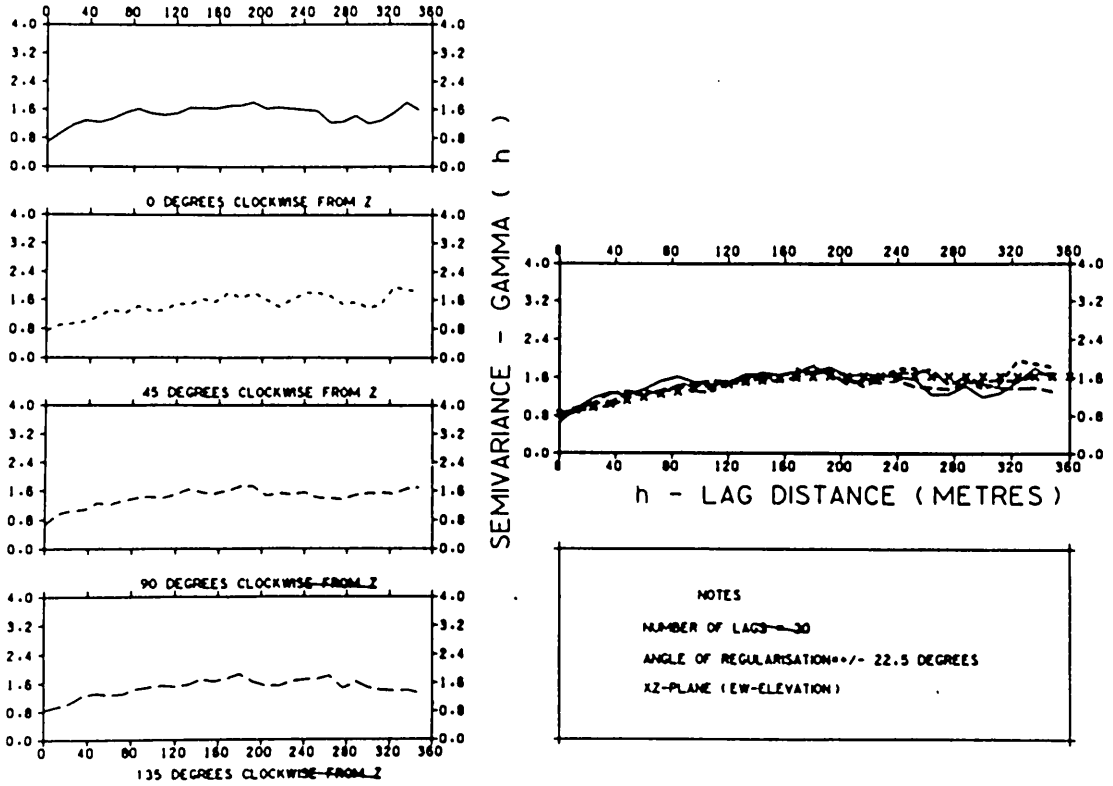
SECTOR SEMIVARIOGRAMS OF Fe CRUDE - SELECTING HEMATITE

Figure 7.2 Experimental sector variograms of the crude ore's Fe content in the horizontal plane



SECTOR SEMIVARIOGRAMS OF Fe CRUDE - SELECTING HEMATITE

Figure 7.3 Experimental sector variograms of the crude ore's Fe content in the vertical plane along the ore body



SECTOR SEMIVARIOGRAMS OF Fe CRUDE - SELECTING HEMATITE

Figure 7.4 Experimental sector variograms of the crude ore's Fe content in the vertical plane across the ore body

Table 7.2 Calibration of the model semi-variogram through point kriging cross validation

Nugget variance (CO)(%)^{2}Fe	0.72	0.80	0.88	0.83
Variation of CO (%)	−10	0	+10	+4
C =Sill−CO (C)(%)^{2}Fe	0.8	0.8	0.8	0.8
Sill (C+CO)(%)^{2}Fe	1.52	1.60	1.68	1.63
Mean Kriging Variance (MKV)(%)^{2}Fe	0.8529	0.9399	1.0267	0.9725
Mean Estimated Variance(MEV)(%)^{2}Fe	0.9561	0.9610	0.9654	0.9627
Ratio MEV/MKV (%)	112	102	94	99

By manipulating the values of the nugget variance, transition amplitude and range, one can calibrate the model variogram aiming at a unitary ratio of variances (estimated to kriging). The model selected in the present case is shown in figure 7.5, and figure 7.6 features a summary of point kriging results, as produced by way of the "MINPAK" library of subroutines (op.cit.).

7.4.4 Validation of the model variogram by way of statistical analysis

Once calibrated, the model can be validated, which means the kriged estimates are compared with the actual grades in more detail through statistical analysis as shown in figure 7.7, which displays an output produced by "MINITAB" (1985), a package available at the ICCC. Figure 7.8 is a graphical comparison of both actual and estimated grades, which features the so-called "boxplots" (see MINITAB, 1985). Table 7.3 compares the descriptive statistics of actual grades and their point kriging estimates.

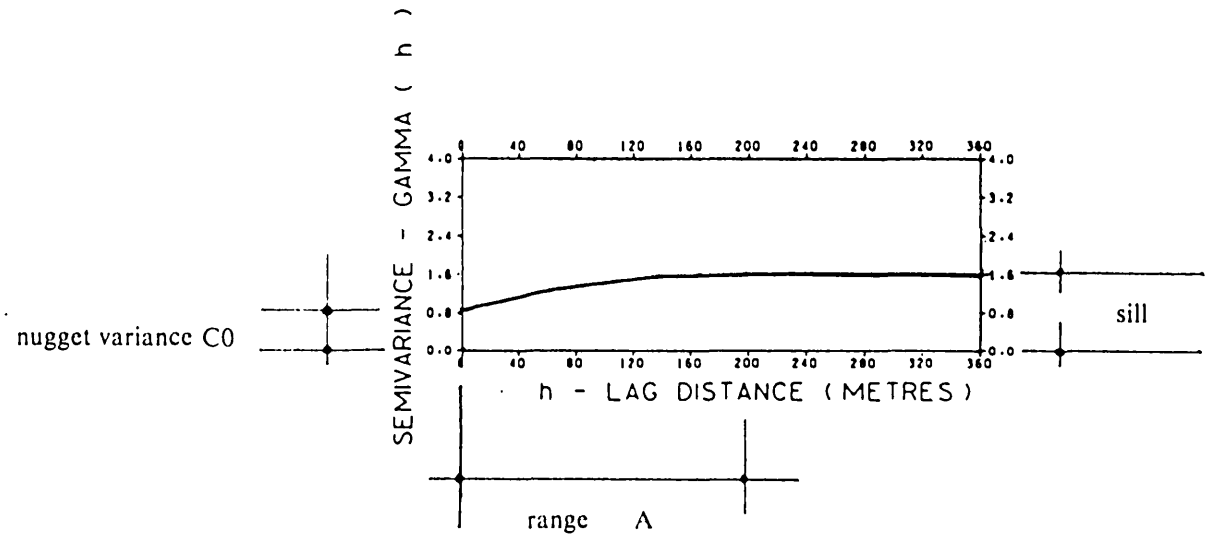


Figure 7.5 Isotropic spherical model variogram of the crude ore's Fe content

STATISTICS:

NUMBER OF POINTS KRIGED = ***

NUMBER OF POINTS WHICH WERE NOT KRIGED DUE TO A LACK OF POINTS = 0

NUMBER OF POINTS WHICH WERE NOT KRIGED DUE TO TOO MANY POINTS = 0

MEAN OF POINTS USED FOR KRIGING = 67.2172 MEAN OF KRIGED ESTIMATES = 67.2447

VARIANCE OF THE POINTS USED FOR KRIGING = 1.4985 MEAN KRIGING VARIANCE = .9725

% ERROR DUE TO PARAMETERS = -.04 MEAN ERROR OF ESTIMATION = 1.13 %

RATIO OF KRIGING VARIANCE : ESTIMATES VARIANCE = .99

MEAN (Z-Z*) = -.0375

MEAN ABSOLUTE (Z-Z*) = .7560

MEAN ESTIMATED VARIANCE = .9627

SPHERICAL SCHEME MODEL FOR HEMELCRUDH

NUGGET VARIANCE = .83000 C = .80300 RANGE = 200.00000 C1 = .03000 RANGE 2 = 100.00000

LAG NO	LAG DISTANCE	GAMMA (H)
0	.0000	.83000
1	11.0000	.90191
2	24.0000	.97331
3	36.0000	1.04367
4	48.0000	1.11247
5	60.0000	1.17920
6	72.0000	1.24334
7	84.0000	1.30436
8	96.0000	1.36176
9	108.0000	1.41501
10	120.0000	1.46360
11	132.0000	1.50700
12	144.0000	1.54470
13	156.0000	1.57618
14	168.0000	1.60092
15	180.0000	1.61840
16	192.0000	1.62811
17	204.0000	1.63000
18	216.0000	1.63000
19	228.0000	1.63000
20	240.0000	1.63000
21	252.0000	1.63000
22	264.0000	1.63000
23	276.0000	1.63000
24	288.0000	1.63000
25	300.0000	1.63000
26	312.0000	1.63000
27	324.0000	1.63000
28	336.0000	1.63000
29	348.0000	1.63000
30	360.0000	1.63000

Figure 7.6 "MINPAK" summary output of point kriging cross validation results

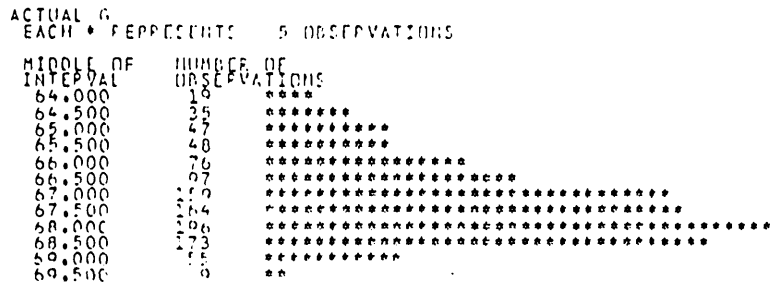
Table 7.3 Descriptive statistics of the sample points' actual values and kriging results - Crude ore's Fe

Sample points' Fe grades (%)		Actual	Estimated
Minimum		64.0	65.3
Maximum		69.7	69.0
Average		67.22	67.24
Standard deviation		1.22	0.73
Coefficient of variation (not in %)		0.02	0.01
Median		67.5	67.3
95% confidence interval for the mean	lower	67.1	67.3
	upper	67.2	67.3
Correlation coefficient		0.6	
Number of sample points		1078	

The "regression effect" (see Krige, 1981, page 21) could be observed in figures 7.7 and 7.8, as well as in table 7.3 : the variance of the point kriged values is lower than that of the actual grades, the same applying for the standard deviation.

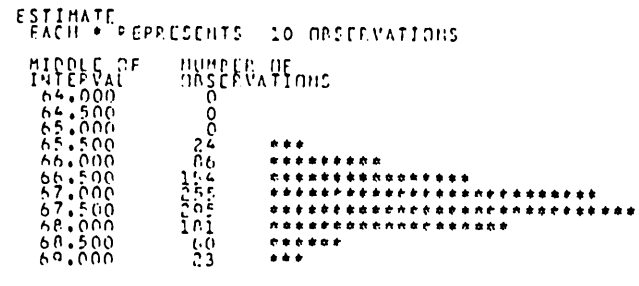
Figure 7.7 "MINITAB" output - statistical analysis of the results of point kriging cross validation

A - Frequency distributions



```
ACTUAL G
 EACH * REPRESENTS   5 OBSERVATIONS

MIDDLE OF    NUMBER OF
INTERVAL     OBSERVATIONS
  64.000          19     ****
  64.500          35     *******
  65.000          47     **********
  65.500          48     **********
  66.000          76     ****************
  66.500          97     ********************
  67.000         150     ******************************
  67.500         194     ****************************************
  68.000         196     ****************************************
  68.500         173     ***********************************
  69.000          55     ***********
  69.500           9     **
```



```
ESTIMATE
 EACH * REPRESENTS  10 OBSERVATIONS

MIDDLE OF    NUMBER OF
INTERVAL     OBSERVATIONS
  64.000           0
  64.500           0
  65.000           0
  65.500          24     ***
  66.000          86     *********
  66.500         154     ****************
  67.000         355     ************************************
  67.500         395     ****************************************
  68.000         181     ******************
  68.500          60     ******
  69.000          23     ***
```

Figure 7.7 "MINITAB" output - statistical analysis of the results of point kriging cross validation

B - Regression analysis

```
THE REGRESSION EQUATION IS
Y =    43.2 +  .357 X1

                                                ST. DEV.      T-RATIO =
        COLUMN       COEFFICIENT              OF COEFF.      COEF/S.D.
                                                                 44.35
X1      ACTUAL C        0.35721                 0.01460          24.47

THE ST. DEV. OF Y ABOUT REGRESSION LINE IS
S = 0.5865
WITH (1078- 2) =1076 DEGREES OF FREEDOM

R-SQUARED = 35.7 PERCENT
R-SQUARED = 35.7 PERCENT, ADJUSTED FOR D.F.

ANALYSIS OF VARIANCE

 DUE TO      DF            SS          MS=SS/DF
REGRESSION    1        [illegible]9304    [illegible].9304
RESIDUAL   1076        370.1425             0.3440
TOTAL      1077        [illegible]0.0729
```

Figure 7.7 "MINITAB" output - statistical analysis of the results of point kriging cross validation

C -Analysis of variance

```
ANALYSIS OF VARIANCE

DUE TO          DF          SS        MS=SS/DF      F-RATIO
FACTOR           1          .41             .41          .40
ERROR         2154      2189.98            1.02
TOTAL         2155      2190.39

LEVEL            N          MEAN        ST. DEV.
C3            1078         67.22             1.32
C4            1078         67.24              .73

POOLED ST. DEV. =          1.01

INDIVIDUAL 95 PERCENT C. I. FOR LEVEL MEANS
(BASED ON POOLED STANDARD DEVIATION)
                +---------+---------+---------+---------+---------+---------+
C3              I*********************I*********************I
C4                        I*********************I*********************I
                +---------+---------+---------+---------+---------+---------+
              67.140    67.170    67.200    67.230    67.250    67.290    67.320

   C5          N =1078      MEAN =      .027543      ST.DEV. =       .981

   TEST OF MU =      0.0000 VS. MU N.E.      0.0000
   T =  1.022
   THE TEST IS SIGNIFICANT AT    .3069
   CANNOT REJECT AT ALPHA = 0.05

   ACTUAL G  N =1078      MEAN =       67.217      ST.DEV. =       1.22
   ESTIMATE  N =1078      MEAN =       67.245      ST.DEV. =       .731

   DEGREES OF FREEDOM =2154

   A 95.00  PERCENT C.I. FOR MU1-MU2 IS (      -.1127,       .0576)

   TEST OF MU1 = MU2 VS. MU1 N.E. MU2
   T =  -.634
   THE TEST IS SIGNIFICANT AT    .5260
   CANNOT REJECT AT ALPHA = 0.05

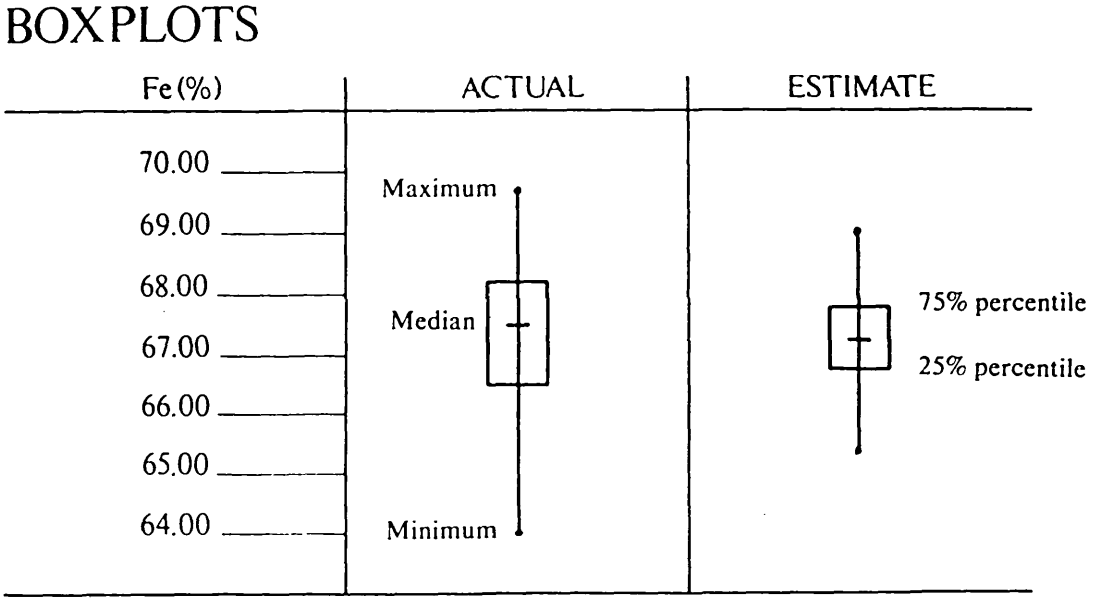
   ACTUAL G  N =1078      MEAN =       67.217      ST.DEV. =       1.22
   ESTIMATE  N =1078      MEAN =       67.245      ST.DEV. =       .731

   APPROX. DEGREES OF FREEDOM =1751

   A 95.00  PERCENT C.I. FOR MU1-MU2 IS (      -.1127,       .0577)

   TEST OF MU1 = MU2 VS. MU1 N.E. MU2
   T =  -.634
   THE TEST IS SIGNIFICANT AT    .5260
   CANNOT REJECT AT ALPHA = 0.05
```

Student's t-test



"Regression effect" : kriged estimates are less spread around the mean than the actual values.

Figure 7.8 "Boxplots": comparison between the sample points' actual values and kriging estimates

The statistical analysis shown in figure 7.7 can be summarized as follows:

a) Analysis of variance (ANOVA - see Wonnacott, 1972, page 221): the null hypothesis, H_o, of no differences in sample variances can not be rejected at a 5% level of significance, which means the difference in sample variance is very small relative to chance fluctuations. Therefore, both actual and estimated grades can be considered as having variances of the same order of magnitude, and the t-test on the difference between two means can be run.

b) Paired t-test: the samples are considered to be not independent one from each other. The test is significant at 0.3569. Hence, the null hypothesis $H_o : \mu=0$, of no difference between samples, can not be rejected at a confidence level of 95%.

c) Difference in two means - independent samples: now, if the samples are to be considered independent from each other, the null hypothesis $H_o:\mu_1-\mu_2=0$ of no difference between the means can not be rejected at a confidence level of 95%. Therefore, the samples being considered either independent or not, the null hypothesis is always rejected.

d) Regression analysis: a straight line was fitted to the data, with the estimated grades (X) as a function of the actual grades (X1). The adjusted angular coefficient, β, is the marginal effect that X1 has in Y, for the sake of prediction. The null hypothesis, $H_o:\beta=0$ has been rejected at the 95% confidence level. In other words, the marginal effect of the actual grades over the estimated values is not null, as far as forecasting is concerned.

e) Correlation analysis: the 95% confidence interval for the correlation of the population (ρ) is [0.54;0.64], therefore one rejects the hypothesis $H_o:\rho=0$ at a 5% level of significance (see Wonnacott, 1972, page 335).

The final conclusion of the statistical analysis is that the model variogram has been validated, therefore it can be used for block kriging. The ore reserve is completely divided into 3-D blocks and those containing ore can be evaluated in terms of crude ore Fe content.

7.5 Preliminary geological interpretation through categorical kriging

7.5.1 Brief description of the procedure adopted

At the end of 1985, although the drilling campaign had been finished, the geological re-interpretation of Mutuca Mine's ore body was not yet available. However, most data of the recent drill holes had already been entered into the computerized data base. From the 70 million tons which used to be considered Mutuca's reserve, an additional 30 million tons of ore were expected, totalling around 100 million tons. Nevertheless, the ore body boundaries had not yet been located.

On the other hand, before starting the block kriging process(see chapter 8), the so called "rock matrix" is required, in order to indicate whether a block contains either ore, therefore needing to be evaluated, or waste, in which case it does not have to be kriged at all.

Aiming at producing at least a temporary solution for the rock matrix problem, to be eventually checked against the results obtained by way of the conventional geological interpretation, a trial categorical kriging exercise was made. First, the sample points were divided into two categories: ore (1) and waste (0). "Ore", in this context, is hematite, as it is classified by MBR (see chapter 3), and the block dimensions in each direction are: NS-25m; EW-12.5m; and height - 10m.

After that, the 0's and 1's resulting from the above classification were treated as regular variables, as far as variograms are concerned. Then, the blocks were identified by their geometric centres and point kriging estimates were made for each block, fractional values ranging from 0.0 to 1.0 resulting from that procedure. Finally, a 0.5 cutoff limit was used to divide blocks into ore and waste.

7.5.2 Generation of a rock matrix through categorical kriging

Figure 7.9 shows the isotropic variogram resulting from the aforementioned approach. The model adopted to represent the experimental variogram is as follows: nugget variance (C0), 0.09; transition amplitude (C), 0.08; and range (A), 150m. A similar application has been described by Kwa (1984, pages 360, 361 and 363) for treating gold deposits in Nevada. The sill of the "indicator" semi-variograms is around 0.2, approximately the same value as that obtained for Mutuca.

Figure 7.10 displays the conventional geological interpretation of the mine bench 1280 and figure 7.11 shows the corresponding rock matrix as generated through categorical kriging. The overall shapes are not markedly different one from each other. In order to

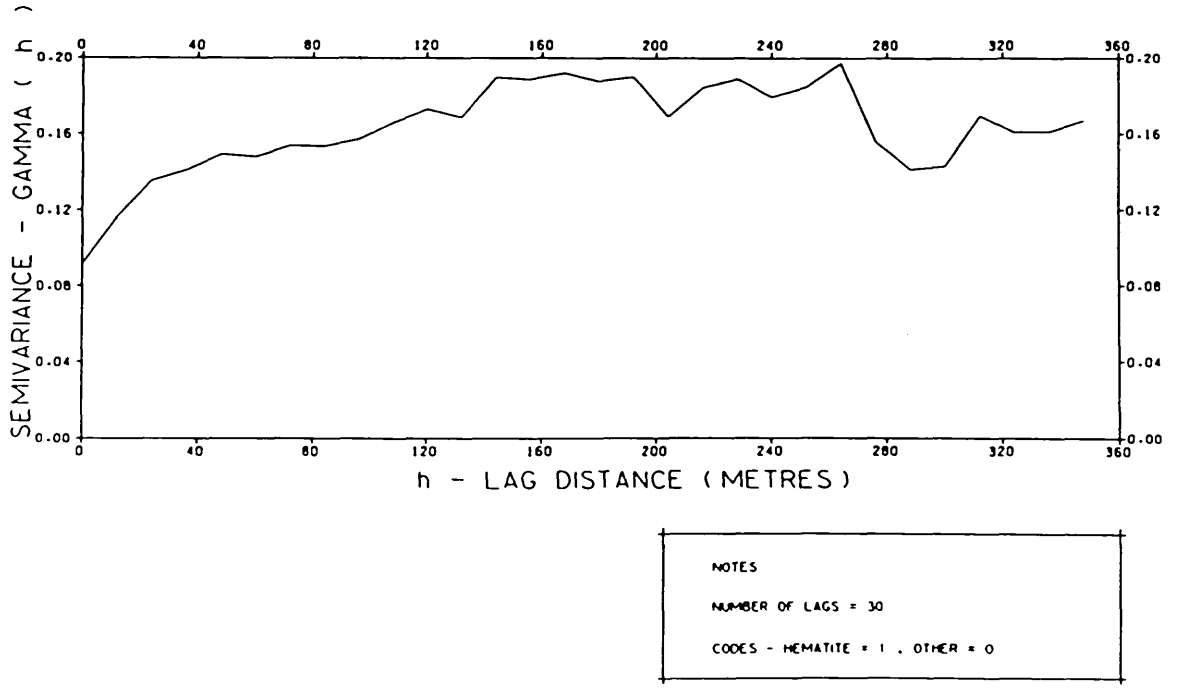
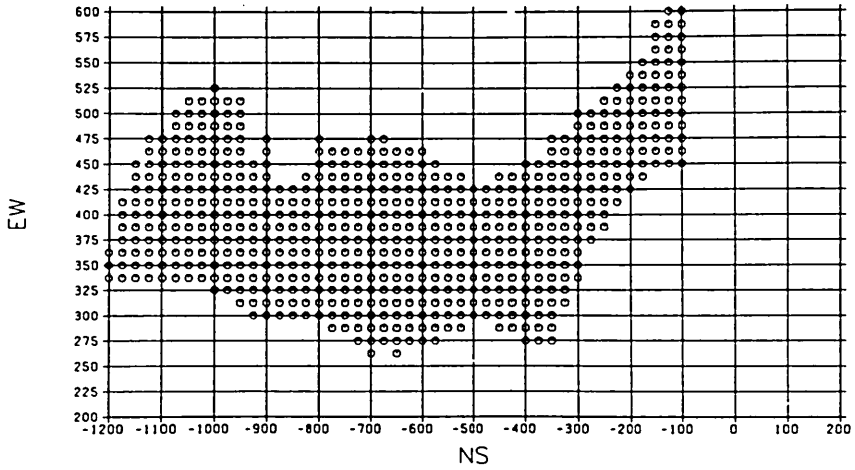


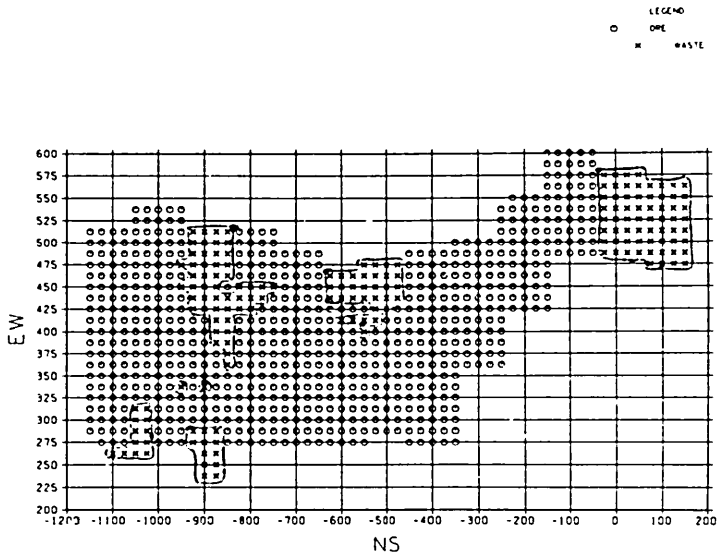
Figure 7.9 Isotropic variogram of type of material as divided into two categories only



BENCH 1280 – ROCK MATRIX

CONVENTIONAL GEOLOGICAL INTERPRETATION

Figure 7.10 "Rock matrix" resulting from conventional geological interpretation



BENCH 1280 - ROCK MATRIX

CATEGORICAL KRIGED ESTIMATES

Figure 7.11 "Rock matrix" resulting from categorical kriging

make sure that the whole extension of the ore body was covered, in the data file used as input for the block kriging program all blocks of figure 7.11 have been considered as if they were ore.

Categorical kriging estimates were produced for the bench elevations 1280m, 1250m, 1200m, 1150m, 1100m, 1050m and 1000m. For the purpose of block kriging, the categorical point kriging estimates of these benches were used as described before to represent a series of benches, as follows: bench 1280 represented benches 1280, 1270 and 1260; bench 1250, from 1250 down to bench 1210; and so on. From 1100m down to the lowest mine level, all benches were represented by bench 1100, in order to have a reasonable number of blocks to be kriged. A maximum searching radius of 50m, in three mutually perpendicular directions, was used for the categorical kriging exercise, and the permitted number of sample points considered for point kriging ranged from 2 to 6.

7.5.3 Discussion of the results and recommendations

The application of categorical kriging for generating the rock matrix which has been described has serious limitations. It lacks the human component that geologists can incorporate to their interpretations. On the other hand, a series of percussion drills, from early geological explorations of Mutuca Mine, has not been included in the geological data base (see chapter 3). Also, the information that can be obtained by geologists simply by visual inspection, given that the ore is exposed at Mutuca Mine, has not been included in the data file. Therefore, the automatic procedure does not have access to as many pieces of information as the geologists have.

Another constraint to the categorical kriging approach presented herein is that the "MINPAK" version that was used adopts a rectangular searching procedure, hence the resulting outline of the kriged points is rectangular as well, whereas the conventional geological interpretation produces a much smoother shape. New versions of "MINPAK" incorporate other searching options [e.g., paraboloid or ellipsoid] (Forkes; 1986; RTZ Technical Services; Bristol - UK; personal communication).

Nevertheless, the categorical kriging procedure might be an auxiliary tool for geologists. As in the case of computerized open-pit shell generation, the "manumatic" approach(see chapter 9) may produce a technique of great potential. In this respect, the following recommendations are made :

- include in the geological data base all information available, that is to say, up-to-date mine surface data; data from the early percussion drill holes, despite their inadequacy for producing block kriging estimates of grades in the various size fractions (see chapter 3); any known or implied boundaries of the ore body, in the form of coordinates (EW, NS, elevation) of points with the corresponding code, i.e., 1 for ore and 0 for waste; whenever a diamond drill hole hits waste, the data coding process should not be stopped at this point - the coordinates of samples containing waste should be entered into the data file as well.
- an interactive graphical workstation, with a digitiser and possibly a colour graphics terminal, would help the geologists to produce an easy and quick preliminary geological interpretation, which could be re-assessed whenever significant changes in the geological data base were observed. Categorical kriging could be used in this process as an auxiliary tool, the geologists interacting with the result of the automatic interpretation by means of the digitiser.
- use the permitted range for the number of samples considered for kriging, the maximum searching radii and the cut-off between ore and waste as control variables, aiming at making a sensitivity analysis of the resulting ore body's shape to variations of these numerical values. Through this approach, geologists could visualize the differences between their subjective interpretation and the product of an automatic procedure.
- analyse contact zones between ore and waste by way of categorical kriging, using different input parameters, and determine how the ore body boundaries are affected, specifically in these zones.

The principal outcome of this suggested method is the possibility for geologists to see how far they are extrapolating geological information by way of their subjective interpretation. Significant differences in the two results could highlight excessively subjective judgements, or alternatively could call on geologists to justify more strongly their analysis or review their assumptions.

7.6 Analysis of the (−2";+¼") size fraction: % in weight, Fe and P

The main product of Mutuca Mine, Lump Ore for Direct Reduction (LODR) is generated from the (−2";+¼") size fraction, which for this reason has been selected to be treated in this research work. All values associated to it are the result of a weighted average of the (−2";+1") and (−1";+¼") granulometric brackets.

The percentage in weight, the iron and phosphorus contents of the (−2";+¼") size fraction were analysed only. Fe has been selected to be studied in the first place because the ore price is calculated as a function of the iron content. Also, together with the P grade, it is used for the geological classification of hematite (see chapter 3).

The study could be extended to deal with other variables, i.e., Al_2O_3 and SiO_2, in all size fractions plus the crude ore. Now that a methodological framework has been established, this could be done without any major difficulties.

7.6.1 Isotropic variograms

Figure 7.12 features the experimental isotropic variogram of the percentage in weight of the (−2";+¼") size fraction. Figures 7.13 and 7.14 display, respectively, the corresponding variograms for Fe and P. Preliminary model variograms have been selected to represent these experimental variograms.

7.6.2 Point kriging cross validation and statistical analysis

The tentative model variograms were calibrated by means of point kriging and the results were analysed statistically, the models having been validated.

Table 7.4 shows a summary of the point kriging input parameters and the consequent results for the percentages of the (−2";+¼") size fraction, iron grade and phosphorus content. Those parameters have been adopted after a sensitivity analysis similar to that one described in item 7.4.3.

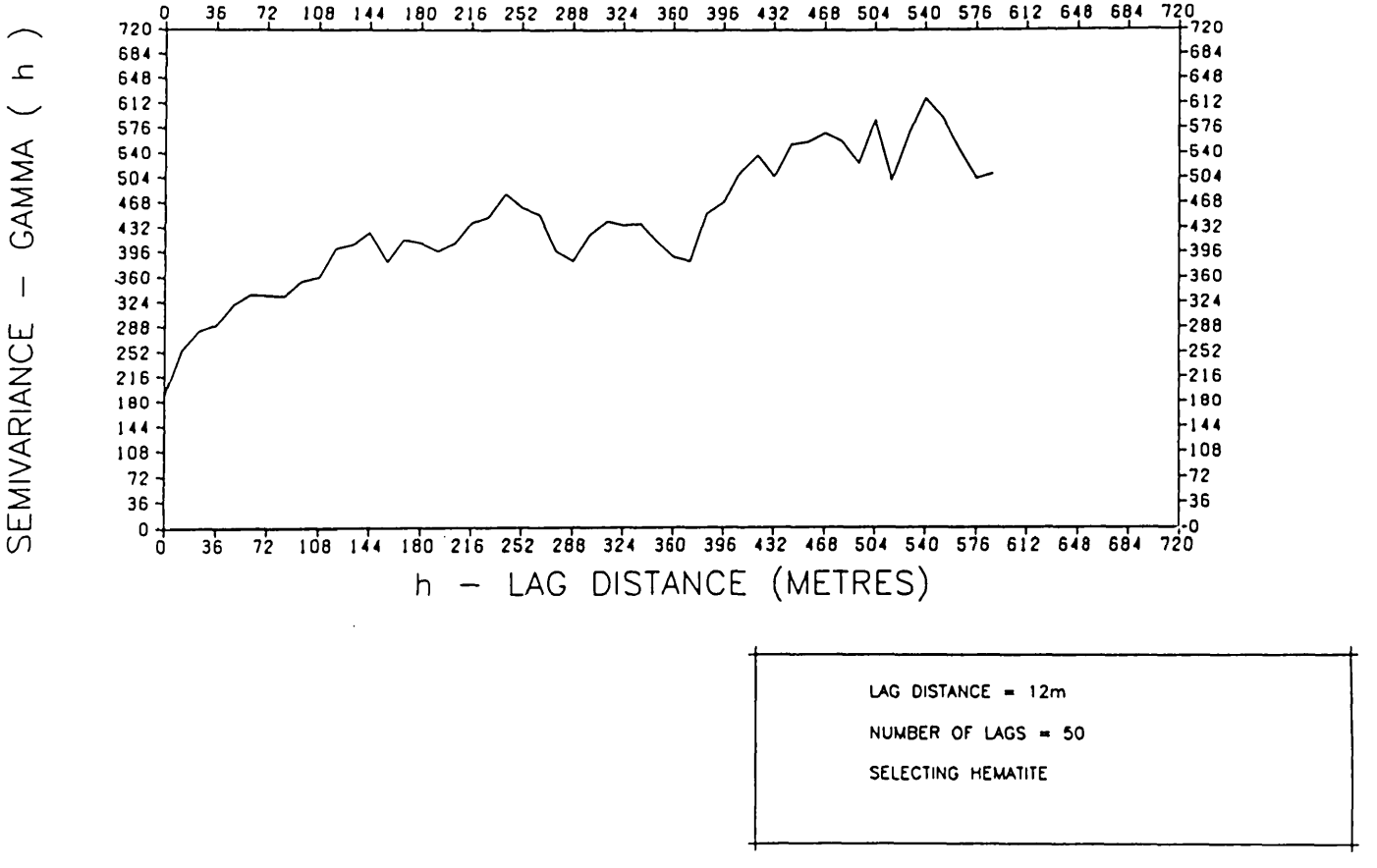


Figure 7.12 Experimental isotropic variogram of the percentage in weight of the (−2";+¼") size fraction

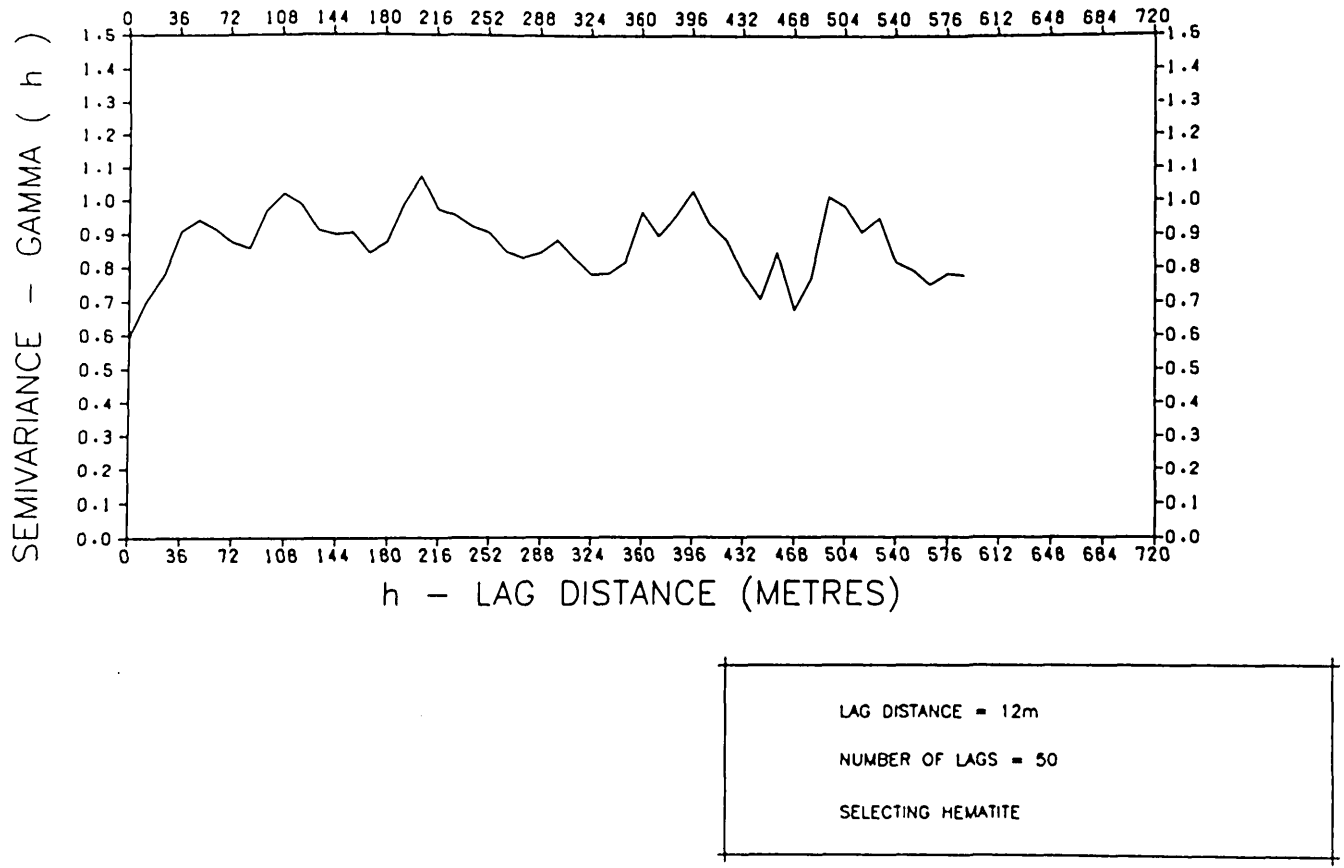


Figure 7.13 Experimental isotropic variogram of the Fe content of the (−2";+¼") size fraction

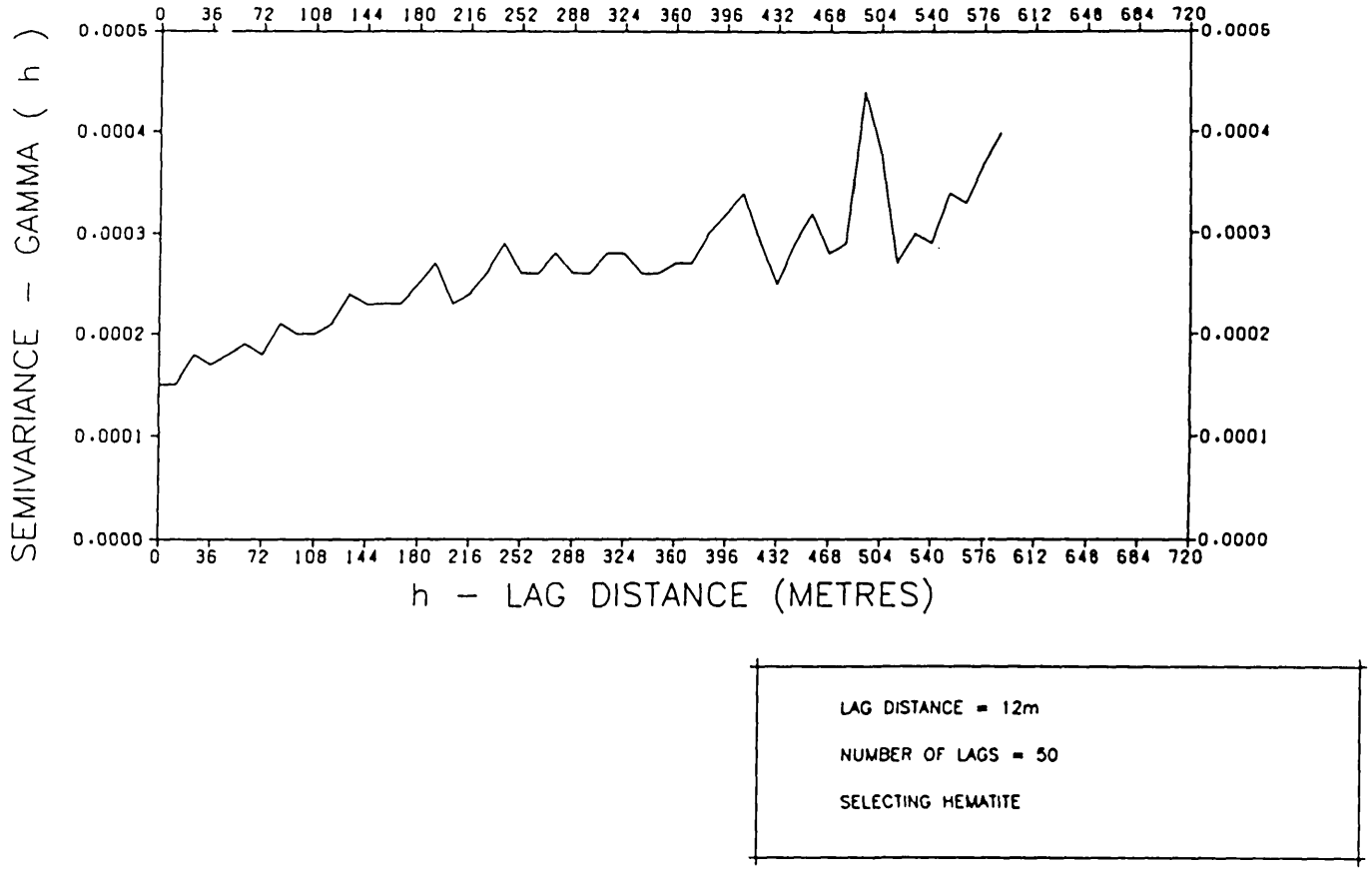


Figure 7.14 Experimental isotropic variogram of the P content of the (−2";+¼") size fraction

Table 7.4 Summary of the point kriging input parameters and corresponding results: (−2";+¼") size fraction

Variable		%(−2";+¼")	Fe	P
Input parameters				
Nugget variance	$(C0)(\%)^2$	250	0.6	0.00015
C =Sill−C0	$(C)(\%)^2$	200	0.4	0.00014
Range	(A)(m)	200	100	200
Searching radius	(m)	200	100	200
Number of sample points (hematite)		1175	1049	1049
Results				
Correlation of actual vs estimated		0.63	0.53	0.60
Mean kriging variance (MKV)	$(\%)^2$	288	0.72	0.0002
Mean Estimated Variance (MEV)	$(\%)^2$	281	0.74	0.0002
Ratio of MKV to MEV	(%)	98	103	103

Notes:

a) Permitted range for the number of sample points considered for kriging: from 4 to 15;

b) Total number of sample points: 1378.

The statistical analysis that compared the actual values of the sample points with their estimates is summarized in table 7.5.

Table 7.5 Statistical analysis of the sample points' actual values and kriging estimates: (−2";+¼") size fraction

Variable	%(−2";+¼")		Fe		P	
Sample point values	Actual	Estimated	Actual	Estimated	Actual	Estimated
Minimum	0.70	7.02	65.10	66.09	0.010	0.012
Maximum	96.90	80.36	69.70	69.12	0.11	0.07
Average	37.89	37.71	68.11	68.09	0.036	0.036
Standard deviation	21.67	13.36	0.88	0.48	0.017	0.011
Coefficient of variation	0.57	0.35	0.013	0.007	0.47	0.30
Median	35.90	38.37	68.35	68.11	0.030	0.033
95% conf. interval} lower	36.65	36.94	68.05	68.06	0.034	0.035
for the mean } upper	39.13	38.48	68.16	68.12	0.037	0.036
Correlation coefficient	0.63		0.44		0.60	
Number of sample points selected as hematite	1175		1038		1049	

Figure 7.15 shows the frequency distributions of both actual values and kriging estimates of the sample points' percentage in weight, Fe and P, for the (−2";+¼") size fraction. The "regression effect" (see item 7.4.3) could be observed again in this figure.

Figure 7.15 Frequency distributions of the sample points' actual values and kriging estimates

A – Fe Iron content

ACTUAL G
EACH * REPRESENTS 10 OBSERVATIONS

MIDDLE OF INTERVAL	NUMBER OF OBSERVATIONS	
64.000	2	*
64.500	6	*
65.000	10	*
65.500	16	**
66.000	31	****
66.500	38	****
67.000	69	*******
67.500	120	**********
68.000	179	******************
68.500	329	**********************************
69.000	233	************************
69.500	16	**

ESTIMATE
EACH * REPRESENTS 10 OBSERVATIONS

MIDDLE OF INTERVAL	NUMBER OF OBSERVATIONS	
64.000	0	
64.500	0	
65.000	0	
65.500	0	
66.000	3	*
66.500	12	**
67.000	30	***
67.500	194	********************
68.000	373	**
68.500	366	***
69.000	69	*******

Figure 7.15 Frequency distributions of the sample points' actual values and kriging estimates

B — P Phosphorus content

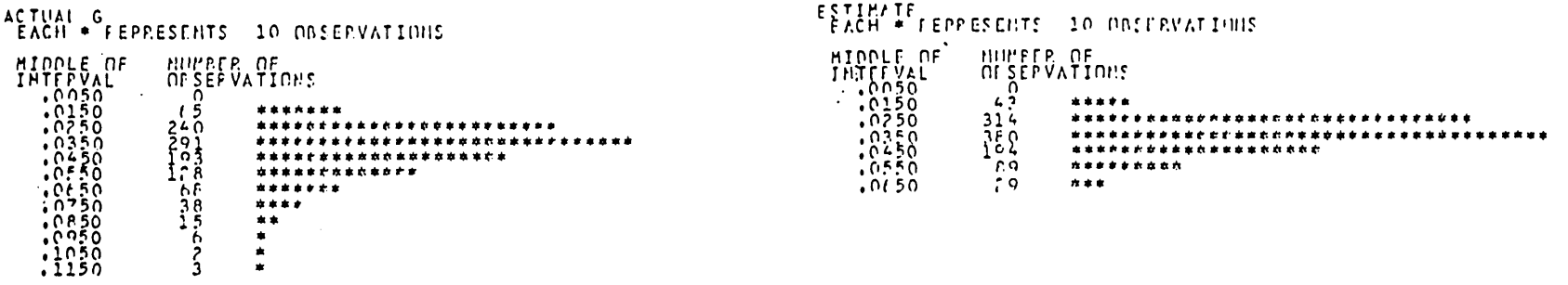
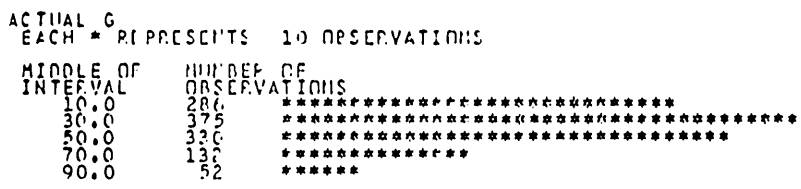
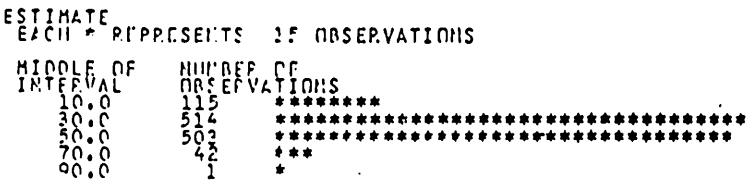


Figure 7.15 Frequency distributions of the sample points' actual values and kriging estimates

C — Percentage in weight





8. Block kriging - Mineral inventory

8.1 Summary and conclusions

8.2 Introduction to this chapter

8.2.1 Block dimensions in general

8.2.2 Block dimensions for iron ore reserves

8.2.3 Suggested solution for Mutuca Mine

8.3 Input data required for block kriging computation

8.3.1 Information on the blocks to be kriged

8.3.2 Geological data base

8.3.3 Kriging parameters

8.3.4 Probabilistic cut-off target

8.4 Brief description of the procedure adopted

8.4.1 Block kriging

8.4.2 Kriging error and cut-off target

8.5 Numerical results and graphical outputs

8.6 Mineral inventory

8.7 Suggestions and recommendations

8.1 Summary and conclusions

First, block kriging, in general, and of iron ore reserves, in particular, are discussed. After that, the suggested solution for Mutuca Mine is presented, as an introduction to this chapter. Subsequently, the input data required to run the kriging programs are listed. Following that, the numerical results and their corresponding graphical outputs are presented, the "mineral inventory" generated by assembling all results in one single data file being exhibited. Finally, some suggestions and recommendations have been made.

The main conclusions of this chapter are summarized below:

- both the contents of iron and phosphorus of the (−2";+¼") size fraction will keep the present range of values in the future, that is to say, in lower mine benches(see figures 8.2 and 8.3).
- the estimated percentage in weight of the (−2";+¼") size fraction decreases with depth(see figure 8.4), therefore the proportion of Lump Ore for Direct Reduction(LODR), Mutuca's chief product, is likely to be affected.
- in view of the previous conclusion, it is recommended a study on the relationship between the kriged estimates and the values actually observed, in order to assess the reliability of the evaluation process as a whole, starting from geological sampling and assaying.
- the concept of a "probabilistic cut-off target", as opposed to the deterministic cut-off grade, can be explored in more detail. By using both the kriged estimate and its associated kriging error, this proposed approach considers not only the quality of ore, but also the quality of information(see topic 8.4.2).
- a computerized mineral inventory is a very useful source of information, as far as mine planning and economic studies are concerned(see item 8.6).

8.2 Introduction to this chapter

8.2.1 Block dimensions in general

Johnson(1969, page 543) discussed the block dimensions, in general. Factors influencing their choice are: equipment capabilities, geological structure, allowable wall slopes, precision of sample data, manner of mining, desired use of block, and capability of manipulating a huge number of blocks with or without a computer. The smaller the block size, the more flexibility in planning is available and also more refinement possible, but the number of blocks increases. The height of a block should usually be taken as the bench height, since this is the way it is mined. Bench heights in present mining practices are in the range of 40-50 feet [roughly 12 to 15 metres], according to Johnson(1969), but perhaps this would be too wide a generalization. The same author gave some examples of block sizes that have been used in previous studies : 30×30×12m, 12×12×12m and 30×15×15m.

As far as geostatistics is concerned, David(1977, page 283) made the following remarks on choosing the right kind of block size:

> "Typically in the profession, people like to know as much as possible about their deposit and consequently they ask for detailed estimation on the basis of the smallest possible blocks. This tendency, besides being possibly unnecessarily expensive, will also bring disapointing results. One will find that small neighbouring blocks are given very similar grades. One should remember that as the size of a block diminishes, the error of estimation of that block increases. Also, dividing the linear dimension of a block by 2, multiplies the number of blocks to be estimated, and probably the number of systems of equations to be solved by 8! As a rule of thumb, the minimum size of a block should not be less than ¼ of the average drill hole interval, say 50' blocks for a 200' drilling grid and 200' for an 800' drilling grid."

Krige(1981, page 39) adds to the above comment the following: "when the data are highly variable, such as in gold mines, the minimum size will be substantially larger."

8.2.2 Block dimensions for iron ore reserve estimation

It is a generally accepted practice to divide the reserve into regular blocks, for kriging purposes. In this respect, a series of publications on estimation of iron ore reserves is briefly mentioned below.

Dunlop(1979) used 25 metre square blocks for kriging an iron ore body. His justification for choosing this block size was that it fitted integrally into the 50 metre drill grid, was suitably small for mine planning whilst the kriging variances were not too large.

Marechal(1977) studied an iron ore body in which the information grid in two zones of

the deposit were: west zone, complete, regular, 243.8m× 61m; and East zone, incomplete, irregular, 121.9m×30.5m. Therefore, he decided to make the local estimation by kriging different panel sizes, with dimensions given by the information grid in each zone. The data were composited over benches 10 and 15m high. The global reserves for various Fe cut-off grades were computed assuming that the selection, when mining, would be made on the basis of 10m×10m×10m or 15m×15m×15m blocks.

David(1977, page 177) mentioned two block sizes, namely, large, 60×30×15m ; and small, 15×7.5×15m, for kriging the reserve of a Taconite iron mine (33% Fe on average). The drilling grid was 60×120m, and the bench height, 15m. A 100,000t block could be 15×30×60m. Similarly, a one-day production for a shovel could be 3,000t, which is, for instance, a 7.5×7.5×15m block (3125t) half a bench height to allow for blast dispersion (David, 1977, page 216).

Kosovac(1978) reported the use of 15×15×12m blocks, resulting in 1 million blocks, for the computerized analysis of an open-pit iron ore mine in Canada. The basic geologic information is obtained at 122-m intervals.

Wilke(1977) presented an optimization schedule procedure for an open-pit iron mining operation. "To model the deposit the well known block approach was used. (...) The ore body and the surrounding waste/overburden is divided into regularly shaped blocks, the height of which equalling the bench height."

Suarez(1978) studied an iron ore deposit which proved to be anisotropic. Elementary panels of 12×12×6m were used, 6 metres being the block height. When their dimensions were multiplied by the pertinent anisotropic coefficients, they turned to be, respectively, 12×36×30m (page 40). Also, in page 69, he referred to 10×10×10m blocks for the sake of simulation of Fe grades.

Caporalli(1982) presented a computerized mining planning system with three modules: tonnage, quality and economic. It is used by CVRD(Companhia Vale do Rio Doce), Brazil's largest producer and exporter of iron ore. One example of block dimensions adopted in this system is: 33×33×10m, the bench height being 10m. The number of blocks, depending on the mine, varies from 10,000 to 40,000.

The author of the present research work participated in geostatistical studies applied to a high-grade open-pit iron ore mine, Águas Claras. The hematite reserve of that mine is located in the same region of Mutuca Mine, therefore it is an important reference. The yearly output of Águas Claras is nearly 2.5 times the annual production planned for Mutuca's expansion, the respective reserves having roughly the same relative size. The first

block kriging exercise made for Águas Claras considered 50×25×13m blocks, the bench height being 13m. After some years of use of the computerized mineral inventory, the mining planning engineers asked for the original blocks to be divided into 4 smaller blocks, keeping the same bench height, which means the linear dimensions were divided by two. Such a reduction in size was required chiefly because the majority of the large size blocks had been mined only fractionally, in the course of each planning period, and the engineers would like the blocks representing a unitary volume more in accordance with the tonnages physically mined, for the sake of better planning.

8.2.3 Suggested solution for Mutuca Mine

The block dimensions adopted were, in each direction :

NS - 25m;

EW - 12.5m;

vertical - 10m, coinciding with the bench height.

This block size corresponds roughly to one day's production, at the present plant feed rate of Mutuca. The ore body dimensions in the NS and EW directions have approximately a 2:1 ratio, therefore the above mentioned block size will implicate roughly the same number of intervals in each direction, as far as the rectangular block grid is concerned.

Mutuca's reserve can be represented by nearly 10,000 ore blocks. On the other hand, the deepest optional pit considered in this thesis, going down to elevation 960m, contains around 20,000 waste blocks (see chapter 9).

8.3 Input data required for block kriging computation

8.3.1 Information on the blocks to be kriged

-Block dimensions: NS-25m; EW-12.5m; and block height-10m(bench height).

-Coordinates of ore blocks to be kriged: EW,NS,elevation, locating the geometric centre of the ore blocks("rock matrix").

8.3.2 Geological data base

Coordinates of the geometric centres of sample points, in the same order as above, plus the assay values and percentage in weight for each size fraction and for the crude ore as well.

In this particular case, the geological samples were classified into hematite and waste, the former category only being used as input data for block kriging. This classification has been explained in chapter 3.

8.3.3 Kriging parameters

-Permitted range for the number of samples considered for kriging: from 4 to 15(O'Leary; 1985; personal communication).

-Model variograms: for each one of three mutually perpendicular directions, the nugget variance (C0), the transition amplitude (C) and the range(A) are given(see chapter 7). If a nested model is used, the information on the transition amplitudes (CC) and ranges (AA) of the second structures are needed, again in 3-D.

-Searching radii: different radius in each direction can be used for 3-D searching of sample points.

8.3.4 Probabilistic cut-off target

The use of this information will be explained in item 8.4.2 . For the time being, it will only be mentioned that this variable is needed for the block kriging program to run.

8.4 Brief description of the procedure adopted

8.4.1 Block kriging

A "rock matrix" with the coordinates of ore blocks indicates the location of the unitary volumes to be kriged. "Ore", in this context, is "hematite", as classified by MBR (see chapter 3).

Hematite data only have been admitted to the block kriging process, the model

variograms being based on the same sub-set of samples of the geological data base.

"MINPAK", the geostatistical library of subroutines which has been used, is an "open package", in the sense that users can create their own FORTRAN main programs.

Figure 8.1 is an example of a block kriging output of "MINPAK", featuring for each block:

- block coordinates (X,Y,Z): EW,NS, elevation(in this order);
- kriged value (Z*): grade or % in weight, in this case;
- kriging variance (KRIG VAR): in $(\%)^2$;
- number of samples used for kriging (NO. PTS.): from 4 to 15;
- cumulative normal probability of observing a real value beyond a given cut-off target (PB VAL%): see underlying explanation; and
- kriging error (KR ERROR): grade or % in weight.

8.4.2 Kriging error and cut-off target

The concept of a probabilistic cut-off target is an extra feature that has been incorporated to the standard "MINPAK" block kriging program, and requires a special explanation which is given below. An introductory note both on statistical and geostatistical concepts is also included.

- **statistical concepts**: standardization

Let

Z = standard normal variable

μ = population mean

X = random normal variable

σ = population standard deviation

s = sample standard deviation. In order to use the tables on the standard normal cumulative probability, one has to transform any normal variable X into the standard form by defining: $Z = (X - \mu) / \sigma$.

Z is just a linear transformation of X. Whereas the mean and standard deviation of a general normal variable X can take on any values, the standard normal variable Z is unique -with mean 0 and standard deviation 1 (Wonnacott, 1972, page 77).

1 RESULTS

X-COORD	Y-COORD	Z-COORD	Z*	KRIG VAR	NO. PTS.	PB.VAL.%	KR ERROR
-600.0000	-50.0000	1240.0000	67.9718	1.11895	15	.135	1.058
-600.0000	-75.0000	1240.0000	67.9800	.89980	15	.135	.949
-600.0000	-100.0000	1240.0000	67.9886	.75265	15	.135	.868
-600.0000	-125.0000	1240.0000	67.9800	.89980	15	.135	.949
-600.0000	-150.0000	1240.0000	67.9718	1.11895	15	.135	1.058
-587.5000	-50.0000	1240.0000	67.9611	1.11106	15	.135	1.054
-587.5000	-75.0000	1240.0000	67.9564	.88371	15	.135	.940
-587.5000	-100.0000	1240.0000	67.9368	.72254	15	.135	.850
-587.5000	-125.0000	1240.0000	67.9564	.88371	15	.135	.940
-587.5000	-150.0000	1240.0000	67.9611	1.11106	15	.135	1.054
-575.0000	50.0000	1240.0000	68.3267	1.09783	15	.135	1.048
-575.0000	25.0000	1240.0000	68.7367	1.10264	15	.135	1.050
-575.0000	.0000	1240.0000	68.7276	1.06135	15	.135	1.030
-575.0000	-25.0000	1240.0000	68.6099	1.06113	15	.135	1.030
-575.0000	-50.0000	1240.0000	67.9737	1.09310	15	.135	1.046
-575.0000	-75.0000	1240.0000	67.9600	.88566	15	.135	.941
-575.0000	-100.0000	1240.0000	67.9431	.75906	15	.135	.871
-575.0000	-125.0000	1240.0000	67.9591	.88774	15	.135	.942
-575.0000	-150.0000	1240.0000	67.9729	1.09539	15	.135	1.047
-562.5000	150.0000	1240.0000	67.7855	1.32105	4	.135	1.149
-562.5000	125.0000	1240.0000	67.7792	1.19164	4	.135	1.092
-562.5000	100.0000	1240.0000	68.1232	1.02302	15	.135	1.011
-562.5000	75.0000	1240.0000	68.1813	1.05158	15	.135	1.025
-562.5000	50.0000	1240.0000	68.3132	1.05878	15	.135	1.029
-562.5000	25.0000	1240.0000	68.6759	1.03683	15	.135	1.018
-562.5000	.0000	1240.0000	68.6696	.98196	15	.135	.991
-562.5000	-25.0000	1240.0000	68.5246	1.01575	15	.135	1.008
-562.5000	-50.0000	1240.0000	67.7838	1.08536	15	.135	1.042
-562.5000	-75.0000	1240.0000	67.7776	.88441	15	.135	.940
-562.5000	-100.0000	1240.0000	67.7680	.76446	15	.135	.874
-562.5000	-125.0000	1240.0000	67.7607	.88305	15	.135	.940
-562.5000	-150.0000	1240.0000	67.7626	1.08098	15	.135	1.040
-550.0000	150.0000	1240.0000	67.7855	1.27933	4	.135	1.131
-550.0000	125.0000	1240.0000	67.7790	1.11584	4	.135	1.056
-550.0000	100.0000	1240.0000	68.0686	.95807	15	.135	.979
-550.0000	75.0000	1240.0000	68.1430	1.00498	15	.135	1.002
-550.0000	50.0000	1240.0000	68.2971	1.02802	15	.135	1.014
-550.0000	25.0000	1240.0000	68.5061	.98633	15	.135	.993
-550.0000	.0000	1240.0000	68.5271	.91290	15	.135	.955
-550.0000	-25.0000	1240.0000	68.4891	1.00236	15	.135	1.001
-550.0000	-50.0000	1240.0000	68.0551	.98691	15	.135	.993
-550.0000	-75.0000	1240.0000	67.7373	.88340	15	.135	.940
-550.0000	-100.0000	1240.0000	67.7234	.74227	15	.135	.862
-550.0000	-125.0000	1240.0000	67.7256	.88299	15	.135	.940
-550.0000	-150.0000	1240.0000	67.7223	1.08273	15	.135	1.041
-550.0000	-175.0000	1240.0000	68.2035	1.12284	15	.135	1.060
-550.0000	-200.0000	1240.0000	68.1844	1.08667	15	.135	1.042
-550.0000	-225.0000	1240.0000	68.1838	1.13658	15	.135	1.066
-537.5000	150.0000	1240.0000	67.7871	1.25231	4	.135	1.119
-537.5000	125.0000	1240.0000	67.7835	1.06030	4	.135	1.030
-537.5000	100.0000	1240.0000	67.9958	.89274	15	.135	.945
-537.5000	75.0000	1240.0000	68.0786	.96466	15	.135	.982
-537.5000	50.0000	1240.0000	68.2405	1.00208	15	.135	1.001
-537.5000	25.0000	1240.0000	68.4469	.95671	15	.135	.978

Figure 8.1 Typical block kriging output of "MINPAK"

- geostatistical concepts:

Let

Z_v = true grade

Z_v^* = kriged estimate

cot = cut-off target(arbitrarily fixed)

σ_k = kriging error. Now assuming σ_k has a normal distribution, the standardized kriged estimate is: $Z = (Z_v^* - cot) / \sigma_k$

After Z is calculated, the cumulative probability of the actual value (Z_v) lying beyond the cut-off target can be obtained from any table on the standard normal curve. Table 8.1 shows examples on how this probability can vary as a function of the above referred parameters.

The next step would be the use of a "probabilistic cut-off target", so that the number of ore blocks would vary as a function of the arbitrarily fixed cut-off target, the ore body limits expanding or contracting accordingly.

Before going further in the description of the probabilistic cut-off target, it should be pointed out that the kriging error has an interesting characteristic : it increases as the density of geological information in space decreases, that is to say, as the data turns to be more and more scarce. This can be observed in figure 8.7, where the kriging variance -and consequently the kriging error- increases with the distance from the centre of the ore body, where the geological information is more concentrated.

Similar approaches have been used by other authors:

Godwin and Sinclair(1977, page 147) studied a porphyry copper-molibdenium deposit through multiple regression. Both estimated grade and the estimated grade minus one standard deviation were calculated. In addition to that, the concept of isoprobability was used for classifying the blocks' Cu content as follows : 84% chance cell grade >= 0.1% ; 84% chance cell grade >= 0.2%.

The 84% limit is the cumulative probability for values above ($Z-\sigma$) in the standard normal curve.

David(1977, page 217) uses a 95% confidence interval for the mean of a block assuming it has a normal distribution and taking minus and plus twice the square root of the estimation variance around the mean :

$I.C._{95\%} = \mu \; +/- \; \{2 \times (\text{estimation error})\}$

Note: in fact, 2 is an approximation for the value 1.96.

Table 8.1 Probabilistic cut-off target : examples for Fe content

Kriged Estimate	Kriging Errror	Probability values $P[Z_v <= cot]$ Fe cut-off targets (cot) in %		
		66.0	66.5	67.0
66.0	1.0	50	69	84
66.5	1.0	31	50	69
67.0	1.0	16	31	50
67.5	1.0	7	16	31
68.0	1.0	3	7	16
66.0	1.2	50	66	80
66.5	1.2	34	50	66
67.0	1.2	20	34	50
67.5	1.2	11	20	34
68.0	1.2	5	11	20

The approach suggested in this research work is for a probabilistic cut-off limit to be fixed, instead of the more usual deterministic cut-off grade. For instance, blocks with 15% or more probability of its real grade lying beyond the cut-off target are considered to be waste. This method would penalize not only low-grade blocks, but also those with high kriging errors, that is to say, both **quality of ore** and **quality of information** would be taken into account.

One possible application of the probabilistic cut-off target approach is to visualize the ore body "fuzzy" boundaries by means of isoprobability curves, therefore identifying transition zones between "ore" and "waste", that is to say, between grades which present a

gradient of variation and have limit probabilities separating them. The need for additional drilling in zones which are not clearly identified as ore or waste could be justified in that manner.

The above concept could also be used for both computer-assisted open-pit design and economic analysis of the optional pits generated. For pit shell generation, it will have an impact over the ore body limits, whereas for economic analysis, it could influence the ore price, which is a function of the iron content. A range of values, with their associated probabilities, can be used instead of the deterministic single value approach.

8.5 Numerical results and graphical outputs

Benchwise : Table 8.2 is a summary of the block kriging results, bench by bench. Figures 8.2, 8.3 and 8.4 are the corresponding graphical representation, respectively, for Fe content, P grade an % in weight , all variables referring to the (−2";+¼") size fraction.

Blockwise : The kriged values for the Fe content of blocks lying in bench 1240, which is the elevation being mined at present in Mutuca Mine, are shown in detail in figure 8.5 , as an example of blockwise representation through the line printer. The corresponding contour curves on the iron content and its associated kriging variances are shown in figures 8.6 and 8.7.

As for the P content, the numerical results are featured in figure 8.8, the contour curves of the grades are shown in figure 8.9, and of the kriging variances, in figure 8.10. The results refer to bench 1240, in general.

Finally, for the % in weight of the (−2";+¼") size fraction the graphs are, respectively, 8.11, 8.12 and 8.13.

8.6 Mineral inventory

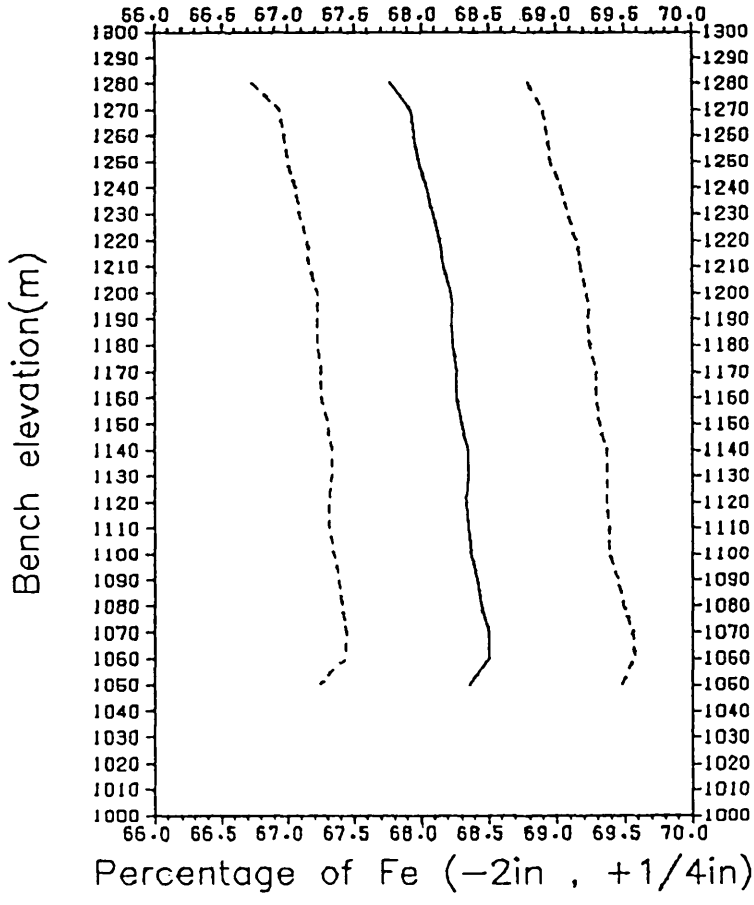
The variables which have been mentioned so far have been treated individually, for variograms, block kriging and numerical and graphical outputs. From now on, the concept of "mineral inventory" is introduced.

According to Barnes(1980, page 51), "a mineral inventory is an estimated inventory of mineral in place and is usually achieved by dividing the deposit into regularly spaced gridded blocks to which are assigned estimation values using various extension techniques. A mineral inventory makes no presumptions about the mineability of the blocks or their

Table 8.2 Summary of the block kriging results : % in weight, Fe and P for the (-2";+¼") size fraction (in %)

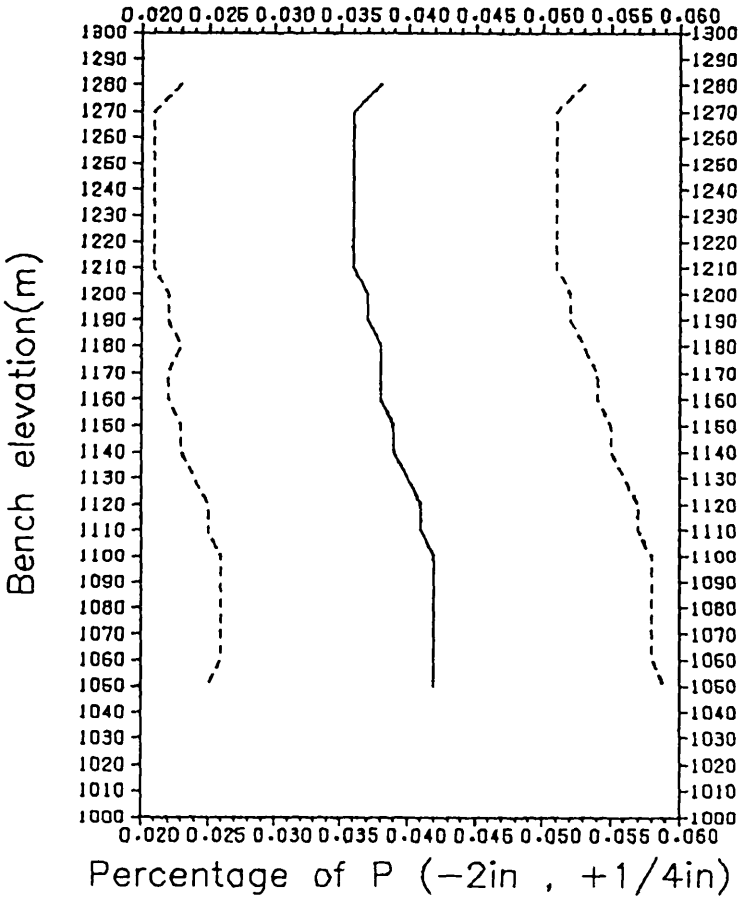
Bench	Iron content			Phosphorus content			Percentage in weight		
(m)	$Z^*-\sigma_k$	Z^*	$Z+\sigma_k$	$Z^*-\sigma_k$	Z^*	$Z^*+\sigma_k$	$Z^*-\sigma_k$	Z^*	$Z^*+\sigma_k$
1280.	66.738	67.768	68.798	.023	.038	.053	27.660	46.460	65.260
1270.	66.940	67.920	68.900	.021	.036	.051	27.578	46.078	64.578
1260.	66.970	67.950	68.930	.021	.036	.051	26.820	45.320	63.820
1250.	66.999	67.979	68.959	.021	.036	.051	25.740	44.240	62.740
1240.	67.053	68.043	69.033	.021	.036	.051	24.450	43.150	61.850
1230.	67.088	68.088	69.088	.021	.036	.051	22.593	41.493	60.393
1220.	67.140	68.140	69.140	.021	.036	.051	21.121	40.121	59.121
1210.	67.160	68.170	69.180	.021	.036	.051	19.200	38.400	57.600
1200.	67.220	68.220	69.220	.022	.037	.052	16.650	35.550	54.450
1190.	67.220	68.230	69.240	.022	.037	.052	14.500	33.600	52.700
1180.	67.230	68.240	69.250	.023	.038	.053	13.100	32.300	51.500
1170.	67.250	68.270	69.290	.022	.038	.054	12.100	31.500	50.900
1160.	67.250	68.270	69.290	.022	.038	.054	10.800	30.400	50.000
1150.	67.300	68.310	69.320	.023	.039	.055	9.750	29.050	48.350
1140.	67.330	68.350	69.370	.023	.039	.055	8.500	28.000	47.500
1130.	67.330	68.350	69.370	.024	.040	.056	7.700	27.500	47.300
1120.	67.310	68.340	69.370	.025	.041	.057	7.500	27.600	47.700
1110.	67.310	68.350	69.390	.025	.041	.057	7.200	27.600	48.000
1100.	67.350	68.370	69.390	.026	.042	.058	8.250	27.750	47.250
1090.	67.390	68.420	69.450	.026	.042	.058	8.100	27.900	47.700
1080.	67.410	68.450	69.490	.026	.042	.058	7.400	27.600	47.800
1070.	67.440	68.500	69.560	.026	.042	.058	7.100	27.700	48.300
1060.	67.430	68.500	69.570	.026	.042	.058	6.400	27.500	48.600
1050.	67.240	68.360	69.480	.025	.042	.059	5.800	27.400	49.000

Note : Both Z^*_k and σ_k are bench averages, respectively, for the kriged estimate and kriging error.



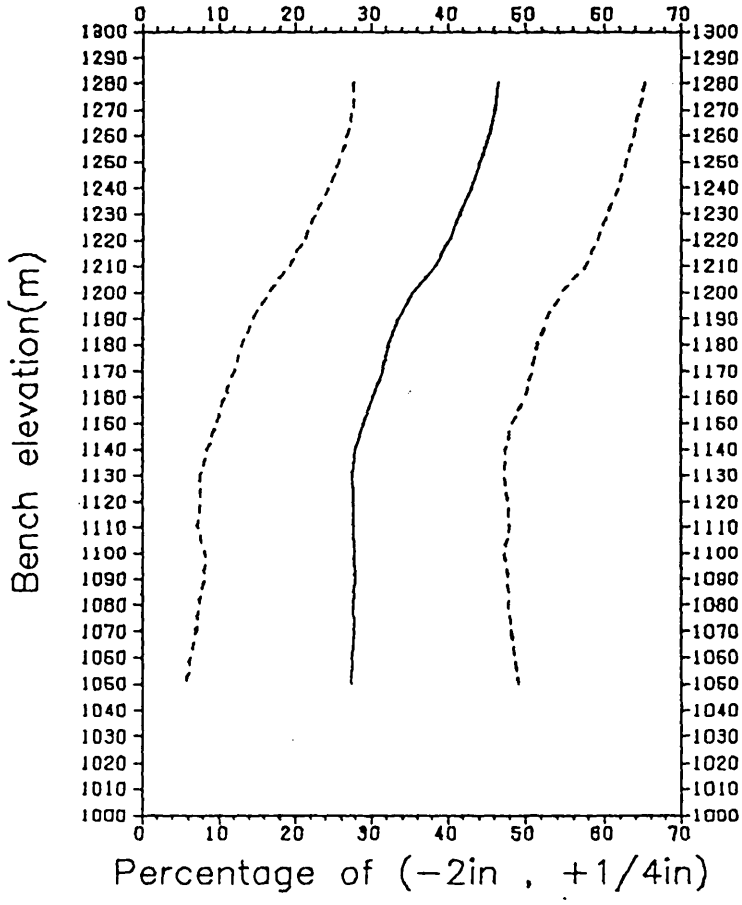
MUTUCA MINE – BLOCK KRIGING RESULTS

Figure 8.2 Graphical representation of block kriging results - Fe content of the (−2";+¼") size fraction



MUTUCA MINE – BLOCK KRIGING RESULTS

Figure 8.3 Graphical representation of block kriging results - P content of the (−2";+¼") size fraction



MUTUCA MINE − BLOCK KRIGING RESULTS

Figure 8.4 Graphical representation of block kriging results - % in weight of the (−2";+¼") size fraction



Notes : each block is represented by two digits, i.e. two columns;

Fe percentage "67" represents "67%"

size fraction (−2";+¼")

Figure 8.5 Fe content of the (−2";+¼") size fraction
line printer output - block grades

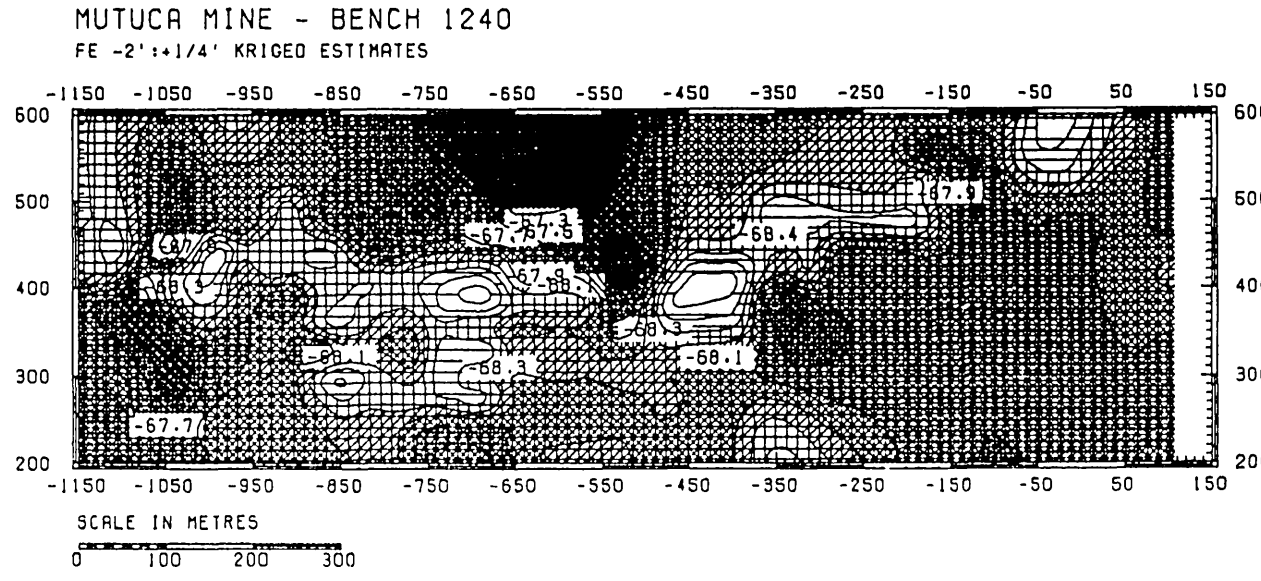


Figure 8.6 Fe content of the (− 2"; + ¼") size fraction

contour curves - kriged estimates

Note : zones with lighter shades correspond to higher values.

Note : values outside the rock matrix are meaningless (see figure 8.8)

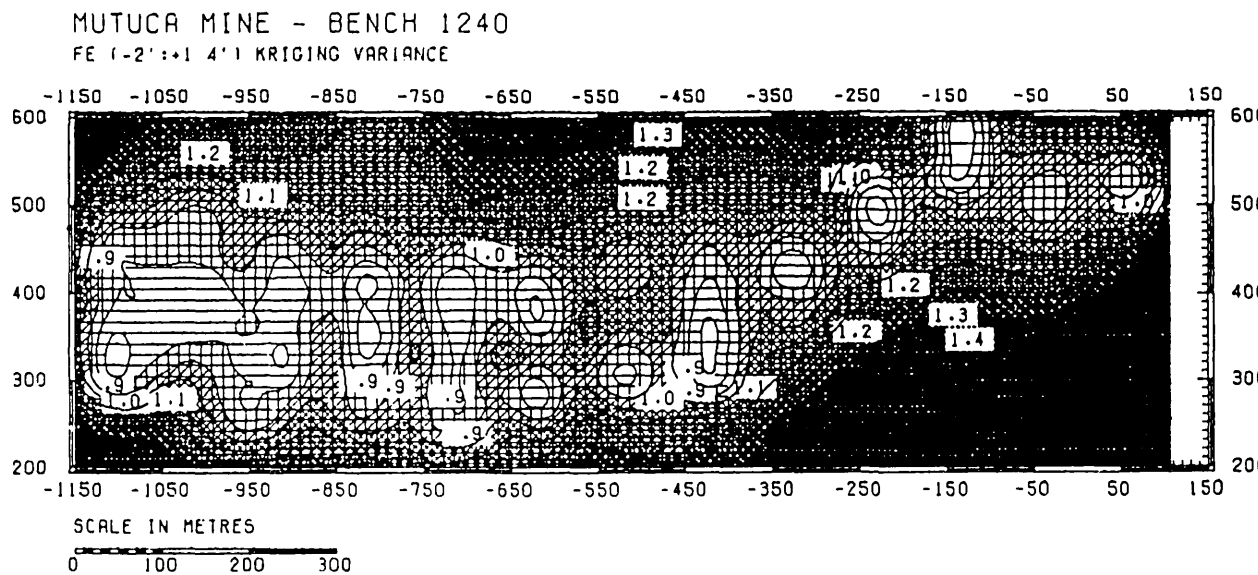


Figure 8.7 Fe content of the (− 2"; + ¼") size fraction contour curves - kriging variances

Note : zones with lighter shades correspond to lower values.

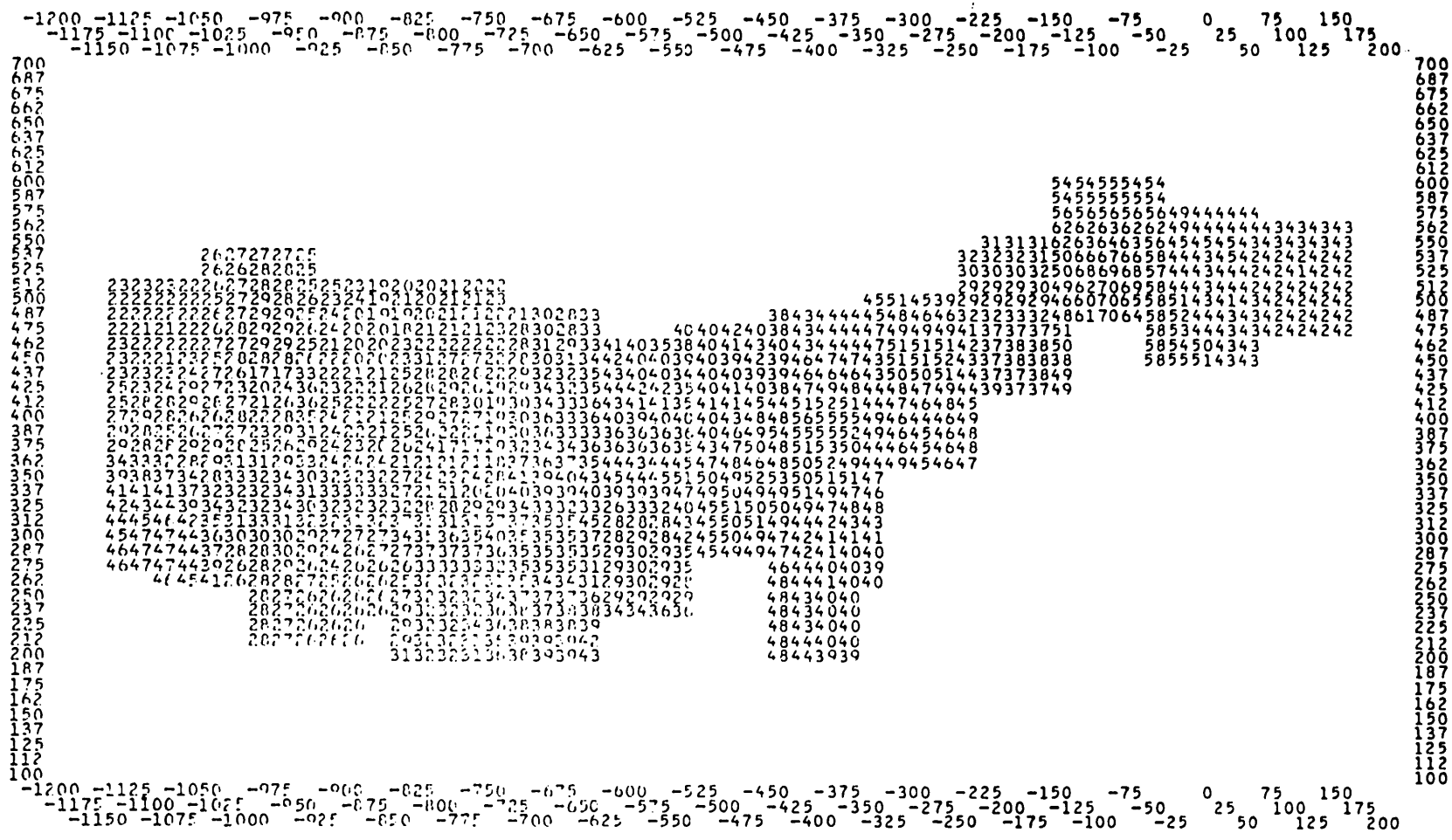


Figure 8.8 P content of the (−2";+¼") size fraction

line printer output - block grades

Notes : each block is represented by two digits, i.e. two columns;
values are multiplied by 10^{-3}
"41" represents "0.041%" Phosphorus - P

size fraction (−2";+¼")

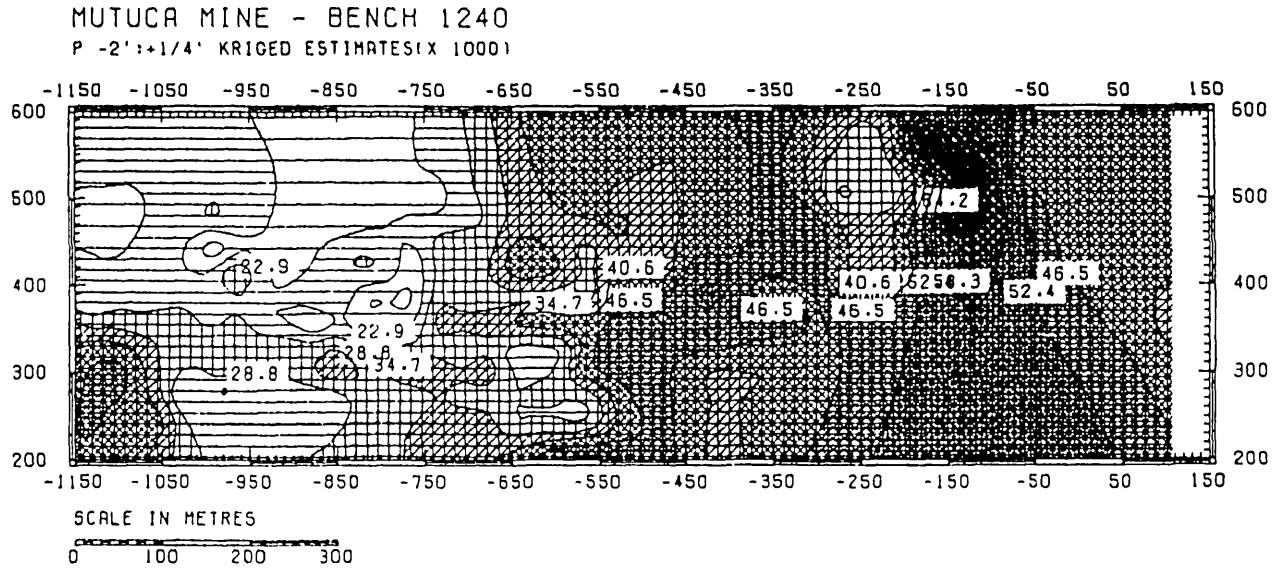


Figure 8.9 P content of the (− 2"; + ¼") size fraction

contour curves - kriged estimates

Note : zones with lighter shades correspond to lower values.

Note : values outside the rock matrix are meaningless (see figure 8.8)

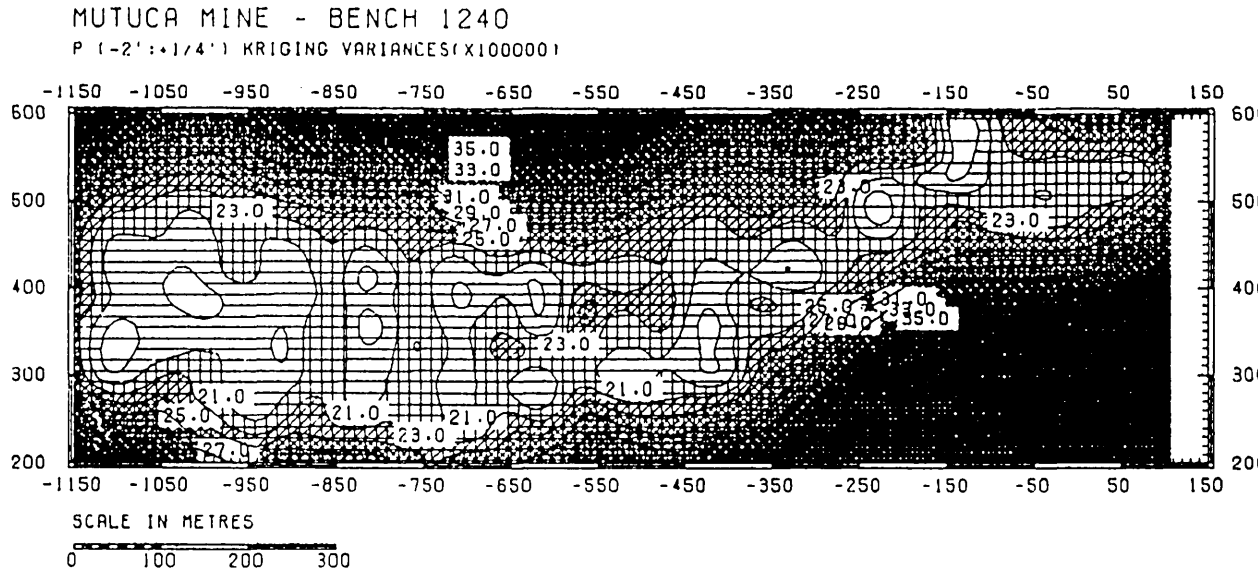


Figure 8.10 P content of the (− 2"; + ¼") size fraction

contour curves - kriging variances

Note : zones with lighter shades correspond to lower values.

Note : values outside the rock matrix are meaningless (see figure 8.8)

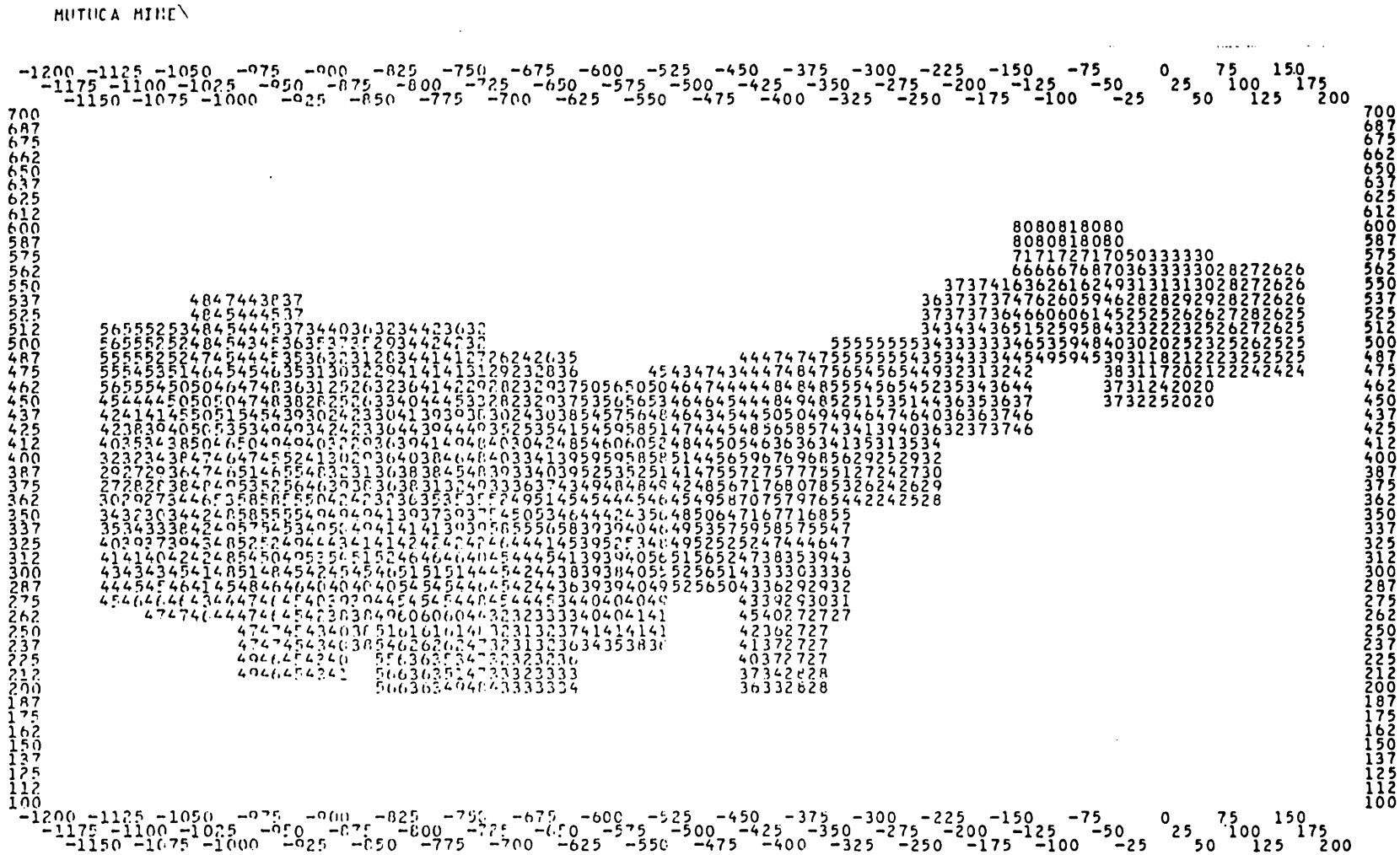


Figure 8.11 % in weight of the (−2";+¼") size fraction line printer output - block grades

Notes : each block is represented by two digits, i.e. two columns; "48" represents "48%" Size fraction (−2";+¼")

size fraction (−2";+¼)

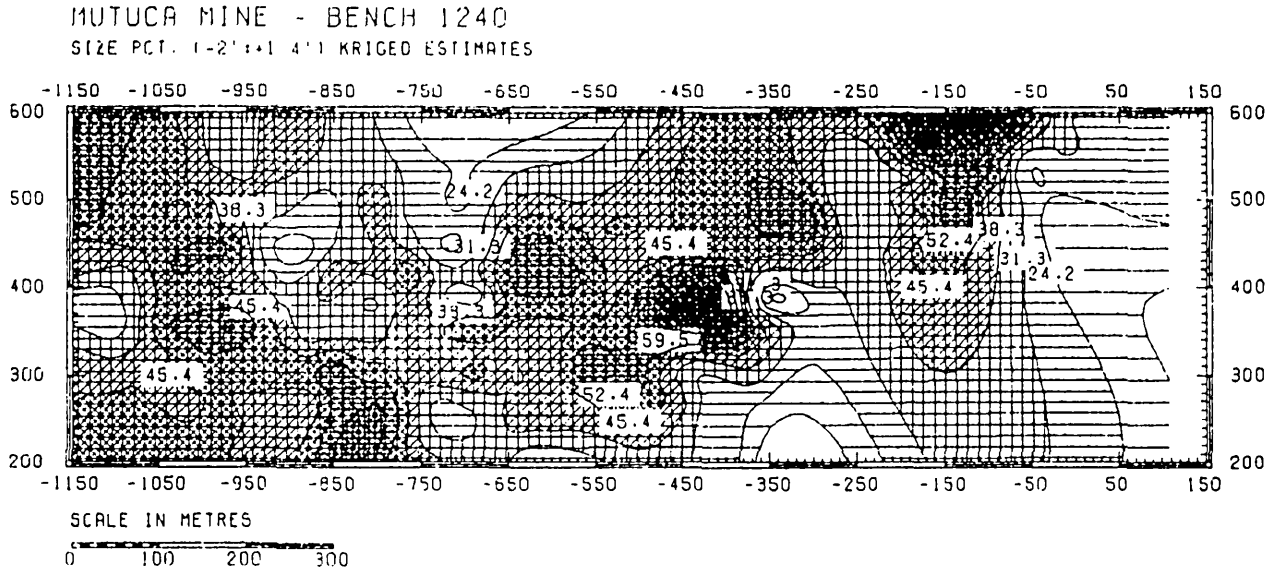


Figure 8.12 % in weight of the (− 2"; + ¼") size fraction

contour curves - kriged estimates

Note : zones with lighter shades correspond to lower values.

Note : values outside the rock matrix are meaningless (see figure 8.8)

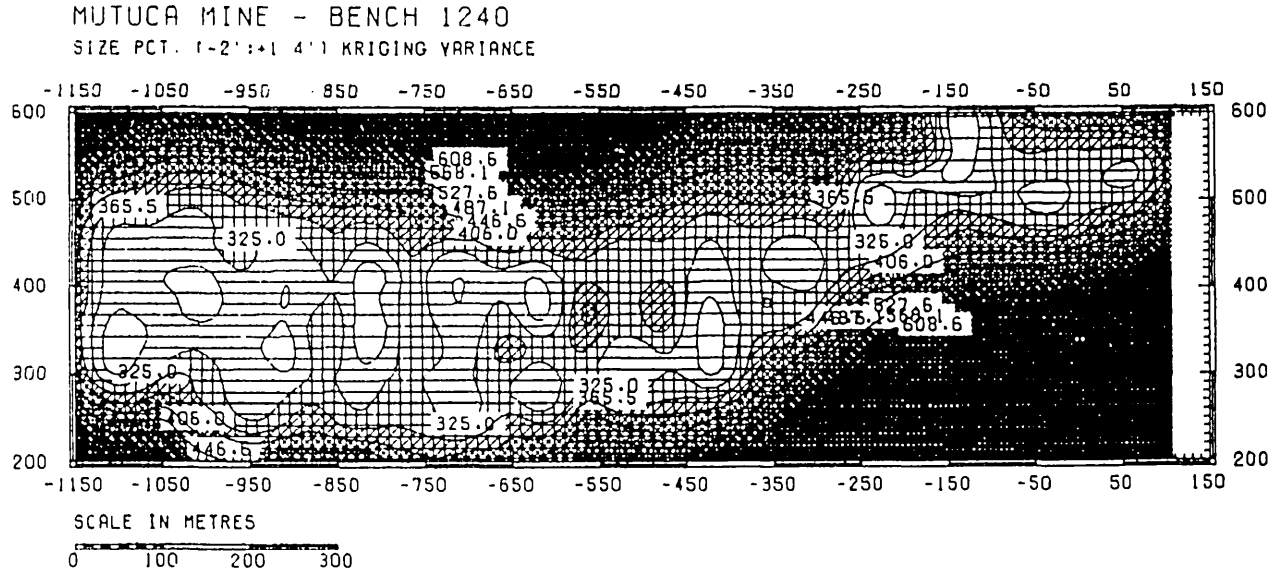


Figure 8.13 % in weight of the (− 2"; + ¼") size fraction

contour curves - kriging variances

Note : zones with lighter shades correspond to lower values.

Note : values outside the rock matrix are meaningless (see figure 8.8)

cut-off grade. Such parameters are implied with the use of the term ore reserve".

In the particular case of this research work, where the (−2";+¼") size fraction has been studied only, the mineral inventory is not complete, although its concept has been used, as shown in figure 8.14, which is a hardcopy of a microfiche. This is a very useful output modality for mineral inventories. Not only thick computer print-outs are eliminated, but it is much more easy and fast to examine the results, provided a microfilm reader machine is available. Other obvious option is the access to the mineral inventory being made by way of a data base management system, possibly using a VDU terminal and a line printer.

Figure 8.14 Mineral inventory - typical output

MUTUCA MINE - MINERAL INVENTORY PAGE 4

E	NS	ELEV	Z*	KE	PV	NP
-- 437.5	950.	1060.				
(%) -2";+1/4" (PV>50%)			44.9141	23.0480	41.2640	15.
FE (PV < 66%)			68.7618	1.2170	1.1619	5.
P (PV > 0.06%)			.0304	.0180	5.0056	15.
-- 425.0	950.	1060.				
(%) -2";+1/4" (PV>50%)			42.6565	22.8630	37.4050	15.
FE (PV < 66%)			68.5442	1.1040	1.0564	14.
P (PV > 0.06%)			.0327	.0170	5.4153	15.
-- 412.5	950.	1060.				
(%) -2";+1/4" (PV>50%)			39.0807	22.7570	31.5670	15.
FE (PV < 66%)			68.5446	1.1040	1.0553	14.
P (PV > 0.06%)			.0349	.0170	6.9894	15.
-- 400.0	950.	1060.				
(%) -2";+1/4" (PV>50%)			38.9537	22.6780	31.3120	15.
FE (PV < 66%)			68.5449	1.1040	1.0545	14.
P (PV > 0.06%)			.0366	.0170	8.4329	15.
-- 387.5	950.	1060.				
(%) -2";+1/4" (PV>50%)			32.3922	22.8010	21.9950	15.
FE (PV < 66%)			68.5436	1.1040	1.0580	14.
P (PV > 0.06%)			.0391	.0170	10.9412	15.
-- 375.0	925.	1060.				
(%) -2";+1/4" (PV>50%)			36.4609	22.6420	27.4970	15.
FE (PV < 66%)			68.5094	1.1010	1.1324	14.
P (PV > 0.06%)			.0406	.0170	12.6865	15.
-- 362.5	925.	1060.				
(%) -2";+1/4" (PV>50%)			23.9278	23.8440	13.7150	15.
FE (PV < 66%)			68.4878	1.0990	1.1789	14.
P (PV > 0.06%)			.0495	.0180	27.9867	15.
-- 350.0	925.	1060.				
(%) -2";+1/4" (PV>50%)			23.6838	23.6080	13.2510	15.
FE (PV < 66%)			68.4710	1.0970	1.2125	14.
P (PV > 0.06%)			.0496	.0180	28.1733	15.
-- 337.5	925.	1060.				
(%) -2";+1/4" (PV>50%)			23.5691	23.4990	13.0350	15.
FE (PV < 66%)			68.4629	1.0960	1.2285	14.
P (PV > 0.06%)			.0496	.0180	28.1733	15.

MUTUCA MINE - MINERAL INVENTORY PAGE 4

8.7 Suggestions and recommendations

A basic framework for geostatistical estimation of reserves has been established, with extensive use of the computer and its graphical aids. Only a fraction of the variables was analysed in this thesis, however, all other variables could be rapidly assessed by means of exactly the same approach. A complete mineral inventory could then be created to be used by both geologists and mining engineers for many useful applications.

It is recommended that a study on the relationship between the kriged estimates and the values actually observed should be made. The data on the product quality being available for Mutuca, which has been mined for more than 25 years, a comparison between the estimates and the results would be possible to be made.

The potential applications of the "probabilistic cut-off target" could be explored in more detail, in the future.

9. Computer-assisted open-pit design (OPD)

9.1 Summary and conclusions

9.2 Input data

9.2.1 Geological data

9.2.2 Rock engineering data

9.2.3 Topographical data

9.2.4 Geostatistical estimates

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9.2.7 Other physical boundaries

9.3 The "Litzinger" algorithm for OPD

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9.3.2 Adaptations and implementations introduced

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9.1 Summary and conclusions

A computerized open-pit shell generation system is introduced in this chapter. The input data for running the OPD program is described in detail, a brief comment is made on the algorithm that has been used used, and both numerical and graphical outputs are presented and discussed. In addition to the generated pits, other existent physical boundaries have been plotted as well, such as the Mutuca Mine dam, its thickener, the location where the road comes out of the pit, and the site prepared for the first stage of the semi-mobile primary crusher. The computer-generated outputs have been compared with hand-made pit designs and numerical results. A preliminary economic analysis has been made using the concept of the economic break-even stripping ratio(BESR), as defined by Miller(1982), aiming at establishing the lowest acceptable level for the ultimate pit.

The importance of the computer and its graphical aids was highlighted, the final conclusion being that the OPD system which has been presented is a useful tool for the mining engineer, with a great potential for future applications, provided the suggested refinements to the system are implemented.

9.2 Input data

The input data required for running the computer program that generates pit shells are listed below.

- geological data
- rock engineering data
- topographical data
- geostatistical estimates
- production rates
- pit design parameters
- other existent physical boundaries.

The following items give details on the input data needed to run the computer-assisted open-pit design system.

9.2.1 Geological data

The geological interpretation of Mutuca Mine was revised by MBR's Geology Department late in the year of 1985. Bench maps were received in London early in 1986(Source: Tessari; 1985; MBR, Geology Department; personal communication).

The graphical information contained in these conventional geological bench maps was entered into a computer file with the aid of a digitiser at the Royal School of Mines. The details on different types of ore and waste were not taken into account, the outline of the ore body being digitised only. A computer program converted the irregular shape of the mineralized zone into a grid of unit blocks (25m×12.5m×10m) and all benches, from 950m up to 1300m, were processed. The resulting data file contains the coordinates of hematite blocks, as follows :

East-West;
North-South; and
Elevation.

9.2.2 Rock engineering data

Slopes - the OPD system has three optional data entry modes, as follows:

a) blockwise - each block, in a plan map, receives a slope to be used in the generation of the pit shell (see figure 9.1). The resulting data file contains the block coordinates (EW, NS and elevation) plus the corresponding slope.

b) sectorwise - the mine is divided into sectors, that is to say, angles in the horizontal plane that add up to 360 degrees, a different slope being assigned to each sector. Figure 9.2 shows the sectors in which the Mutuca Mine has been divided (Source: Souza; 1985; MBR Mining Engineer; personal communication).

c) as a function of type of material - each block receives a slope according to its type of material, which in turn is given by the geological interpretation.

Inter-ramp slope configurations : Table 9.1 shows the parameters adopted for Mutuca Mine's pit, using the nomenclature applied by Hendricks(1979). These parameters have been chosed based on a report on the slopes by a Brazilian consultant company, Victor de Mello(1982), and on the actual slopes that have been tested at the mine site by MBR technical staff. The figures which have been adopted were the object of personal

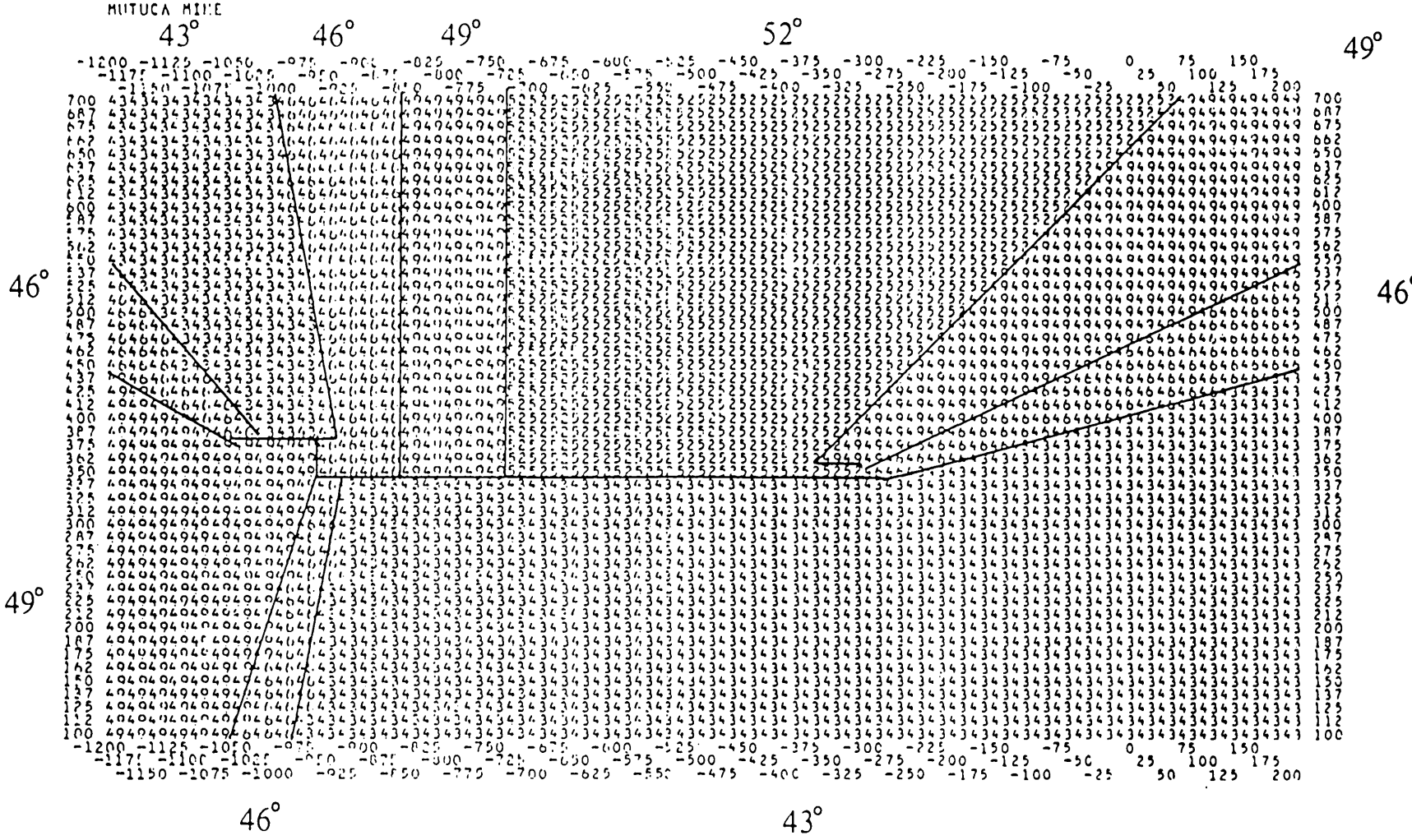


Figure 9.1 Blockwise definition of slopes

communications with Mr. Adriano Souza, senior Mining Engineer working for MBR, early in the year of 1986.

Table 9.1 Inter-ramp slope data

Pit wall	Overall slope (degrees)	Face angle (degrees)	Berm width (m)
West	52	65	6.5
East	43	60	10.5

In-pit Road : the pit slopes are flattened to account for the haulage road, according to road width and number of laps, as well as maximum terrain elevation in each sector. For example, given a road width of 16 metres, a maximum terrain elevation of 1260 metres , an original slope of 43 degrees and 2 road laps in the sector, the resulting slope is approximately 40 degrees for pit 980 (see table 9.2).

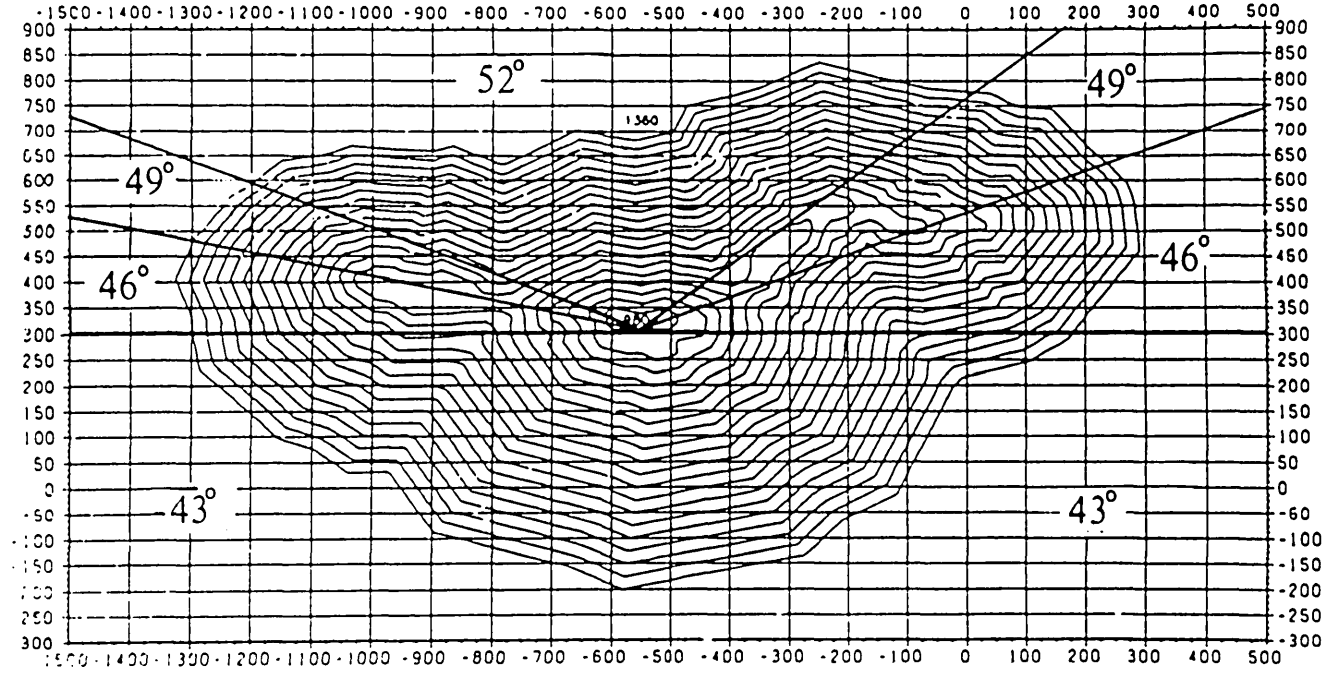


Figure 9.2 Sectorwise definition of slopes

Table 9.2 Sectorwise definition of slopes and resulting flattened slopes for pit 980

sector from (degrees)	to	original slope (degrees)	maximum terrain elevation (metres)	no. of road laps	resulting flattened slope (degrees)
0	70	52	1320	2	48.8
70	78	49	1320	2	46.1
78	90	46	1320	2	43.3
90	270	43	1260	2	40.1
270	288	46	1260	2	42.8
288	305	49	1260	2	45.5
305	360	52	1320	2	48.8

The maximum road grade was fixed at 8%. The values of this parameter and of the the road width, which were established by MBR personnel, are in accordance with the recommendations by Couzens(1979, page 225).

Specific gravity : two densities were used only, 3.5 for ore and 1.8 for waste, in order to have a simple and quick assessment of the pit tonnages. The information was supplied by MBR personnel, and might be the object of future revision, particularly the waste **in situ** density, which is likely to increase with depth. In the future, the specific gravities of different materials should be taken into account(see table 9.3).

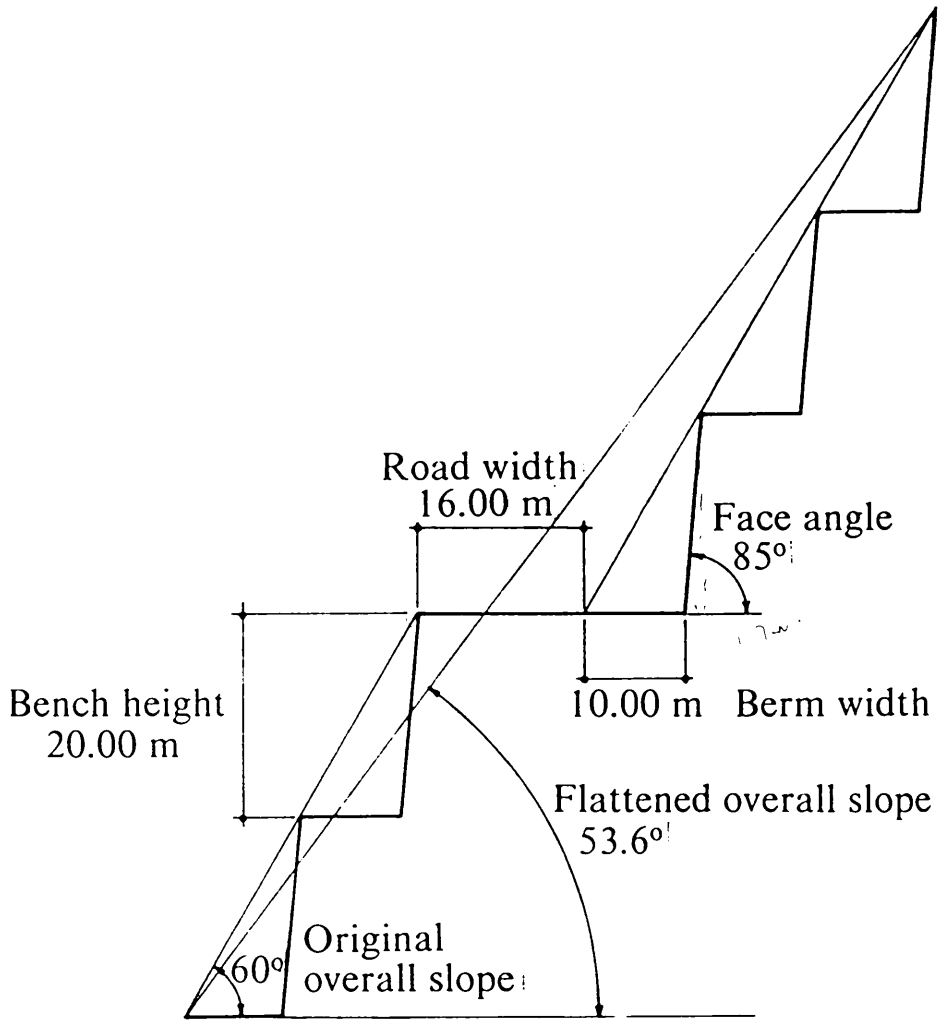


Figure 9.3 Inter-ramp slope and in-pit road

Table 9.3 Densities of Mutuca Mine materials

Material	Density
Hard ore	4.5
Hard ore/mixed ore	4.0
Mixed ore	3.5
Soft ore	3.2
Debris ore	3.5
Itabirite	3.0
Argillaceous iron formation(AIF)	2.0
Carbonate iron formation(CIF)	2.0
Basic intrusive rock	2.0
Schist, Phyllite	1.75

9.2.3 Topographical data

The topography observed in November, 1985 was recorded(figure 9.4). The mine was divided into regular blocks measuring 25m in the NS direction, 12.5m in the EW direction, and 10.0m in the vertical plane. The data file contains, for all blocks, eastings, northings and the terrain elevation.

UMLKAUZ ZU.11.U1. 86/05/16. AEXA 1 10J2D

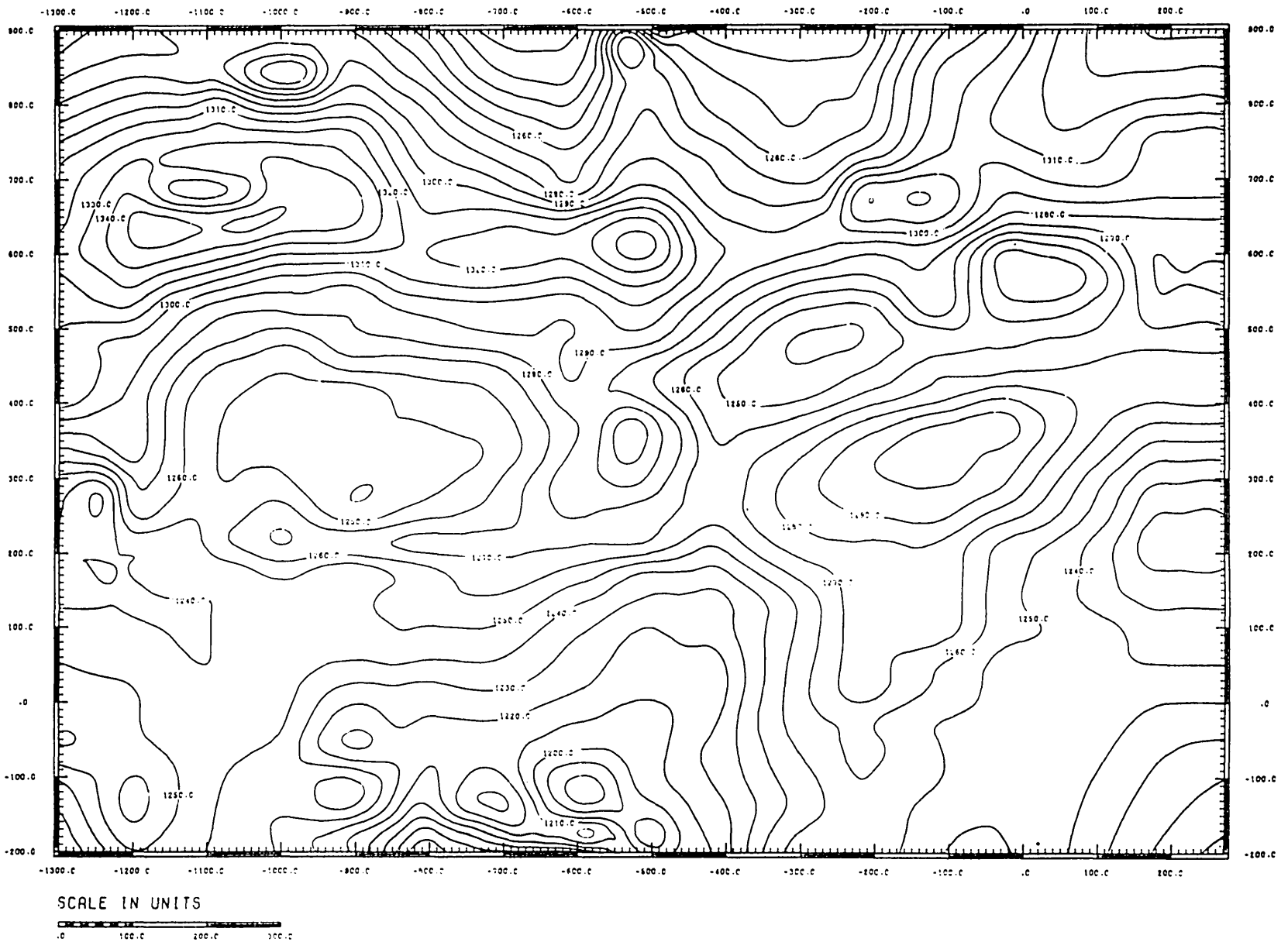


Figure 9.4 Topographic map as of November, 1985

9.2.4 Geostatistical estimates

The kriged estimates, obtained as described in chapter 7, constitute another data file, which is organised as follows:

EW, NS, Elevation (m)

Size fraction (%)

Fe, P, Al_2O_3, and SiO_2 grades (%)

For each block, four different size fractions are assayed, the crude ore analysis resulting from weighted average of the size fractions (see chapter 3).

9.2.5 Production rates

Extraction of ore: as the plant recovery index is 94%, in order to meet the planned 5.95 Mtpy of product, the ore dressing installation should be fed at a constant rate of 6.5 Mtpy.

Stripping: varies with the waste tonnages of each optional pit design and may change for different working pit sequences of a particular ultimate pit.

9.2.6 Pit design parameters

Base polygon : It is defined through its points. The base polygon should have some minimum dimensions, which are established as a function of the requirements for equipment access to the lowest pit bench.

The shape of the base polygon is dictated either by the ore body outline or by physical boundaries lying in upper benches that ought to be respected.

Permitted range for the internal angles : the computer-generated polygon, in each bench, is smoothed out, in order not to have angles too sharp or too wide, therefore, a range needs to be established by the user, the recommended values being : 120 degrees, minimum; and 240 degrees, maximum.

Bench height : although the actual value for mining is 10m, 20m was used as the bench height. The number of points generated is virtually halved, by doing this. The mining engineers follow a similar procedure at MBR(Source: Souza; 1986; MBR Mining Engineer; personal communication).

Number of benches to be generated : it is necessary for establishing at what elevation the program should stop.

Number of permitted horizontal expansions : controls the horizontal search for ore blocks, in each bench.

Cord length : Controls the maximum permitted distance between two adjacent points. If the distance is greater than the maximum, the necessary number of points is interpolated(see figure 9.18).

9.2.7 Other physical boundaries

Many physical boundaries are present at Mutuca Mine, as the dam, the thickener, the point where the haulage road comes out of the pit, and the site prepared for receiving the first stage of the semi-mobile in-pit crusher(see figure 9.14).

9.3 The "Litzinger" algorithm

Both the idea of having a program for generating open-pit shells and the necessary algorithm for that were entirely conceived by Dr. Bernardo Litzinger, Planning Director of MBR, in 1976. Mrs. Maria Aparecida V. Castro, a consultant Systems Analyst, converted the idea into a program that was originally focused on the Águas Claras Mine (ACM). That mine has a situation, in terms of orientation of coordinated system, completely different from Mutuca's(compare figures 9.5 and 9.6). At that time, the program was just a pit shell generator, not only needing itself some improvements, but also requiring the development of auxiliary programs to transform it in an embryonic OPD system. Castro's original program has been adapted for general use as well as specifically for Mutuca Mine.

9.3.1 Brief description of the "Litzinger" algorithm

The logic of the algorithm might be summarized as follows(see figure 9.31) :

a) It starts at the lowest pit bench, scanning the points of a given base polygon. Three points are picked at a time, the bisector of the angle formed by these three points being computed, so as to establish the direction for projecting the central point. All points of the polygon are projected outwards and upwards into the following bench plane, according to the slopes given.

b) The algorithm tries to expand horizontally the polygon that has initially being established in the upper bench, through successive iterations. In each loop, the points are displaced radially approximately by the component of the block dimensions in the direction of expansion. The program keeps the polygon expanding until it hits waste in all directions, therefore most ore blocks lying in the bench are likely to be included in the pit boundary.

c) Finally, the former upper bench is used as the lower bench for the following iteration, and the new upper bench is the one lying above the bench which has just being processed. The whole operation is repeated until the maximum elevation is hit, the resulting output data file containing, bench by bench, the points which define the pit surface.

Up to that point, no attention had been given to whether the upper benches of the pit shell would have hit the air, that is to say, would have gone beyond the topographical boundaries, or not.

It is worth mentioning that if an ore body has a peculiar conformation, in a certain horizontal section, this situation still can be handled by the Litzinger algorithm, with the aid of an artifice. The mining engineer can use an auxiliary geological data file, with artificial blocks of waste or ore, depending on what is required, in order to control the horizontal expansion of the pit limits. This artificial data file would only be used at the open-pit design stage. Subsequently, for the ore reserve summation, the actual geological data file can be used as input data.

For instance, if a group of ore blocks of a mine bench is encompassed by waste blocks, the Litzinger algorithm will not pick it up. However, the mining engineer can turn some waste blocks into ore, in the auxiliary data file, therefore creating a leading way towards the ore blocks that are to be included in the pit limits.

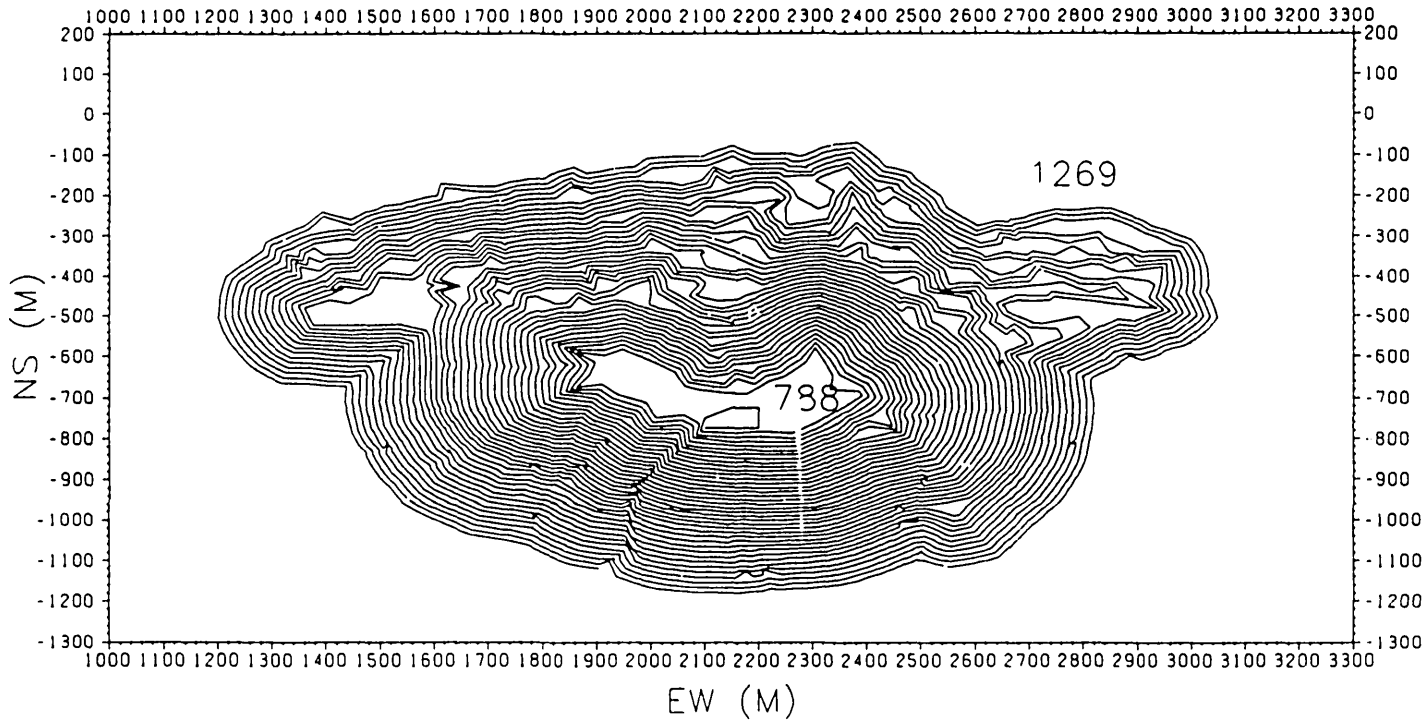
Another example is that of an ore body with an unusually elongated shape in a certain horizontal section. The Litzinger algorithm will include most ore blocks in the bench considered. However, as that elongated pattern is not observed in upper benches, excessive waste will end up included in the pit boundaries. Again using an auxiliary geological data file, the mining engineer can convert the ore blocks lying in the hypotetical elongated portion into waste, so that they will not be picked up by horizontal expansions of the pit limits.

9.3.2 Adaptations and implementations introduced

The program was originally conceived to treat the Águas Claras Mine data, which has different characteristics from Mutuca, as far as the system of coordinates is concerned(see figure 9.5). Therefore, it first needed to be adapted to handle the Mutuca Mine data (see figure 9.6).

Following that, a correction was made in the routine that projects upwards the polygon lying in the lower bench. At the beginning, the program used to take the slopes as they were, independently of the internal angles of the polygon. However, it is easy to verify that, if a horizontal angle is too sharp, e.g., 10 degrees, the slope is only respected in the vertical projection of the tip of the polygon, but not in the projection of the two adjacent chords, resulting in a slope much steeper than what it is meant to be.

The problem of "projection of points", as opposed to "projection of chords", was mentioned by O'Leary(1986; Royal School of Mines; personal communication). Figure 9.31 shows the solution adopted by the author of this research work. Savage(1986; RTZ Technical Services; personal communication) mentioned, about the



AGUAS CLARAS MINE PIT 788

37 LEVELS - BENCH HEIGTH = 13M

Figure 9.5 Output of the original version of the OPD program - Águas Claras Mine, pit 788

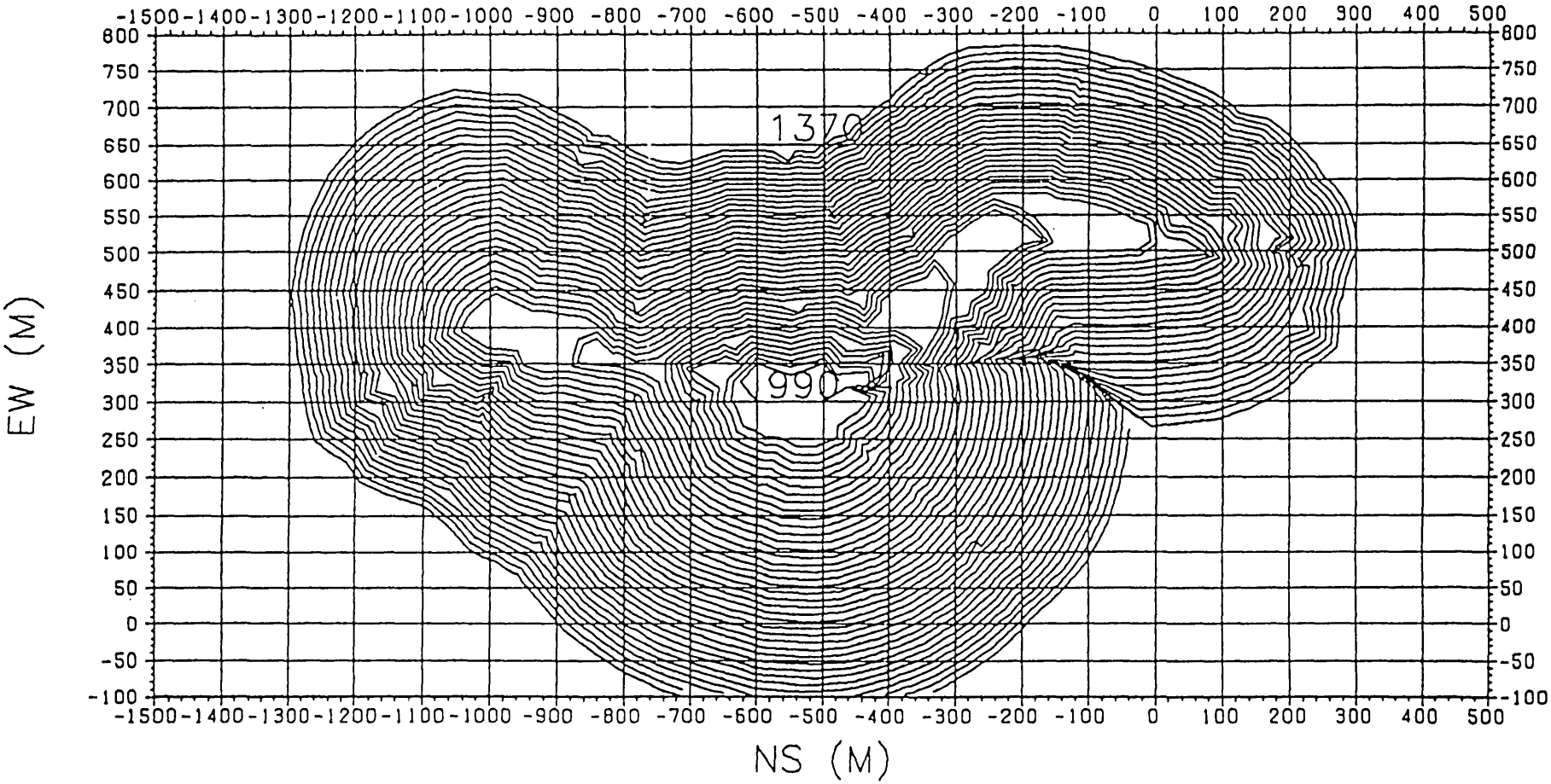


Figure 9.6 Output of an early version of the OPD program - Mutuca Mine, pit 990

MUTUCA MINE PIT 990

BENCH HEIGHT = 10M

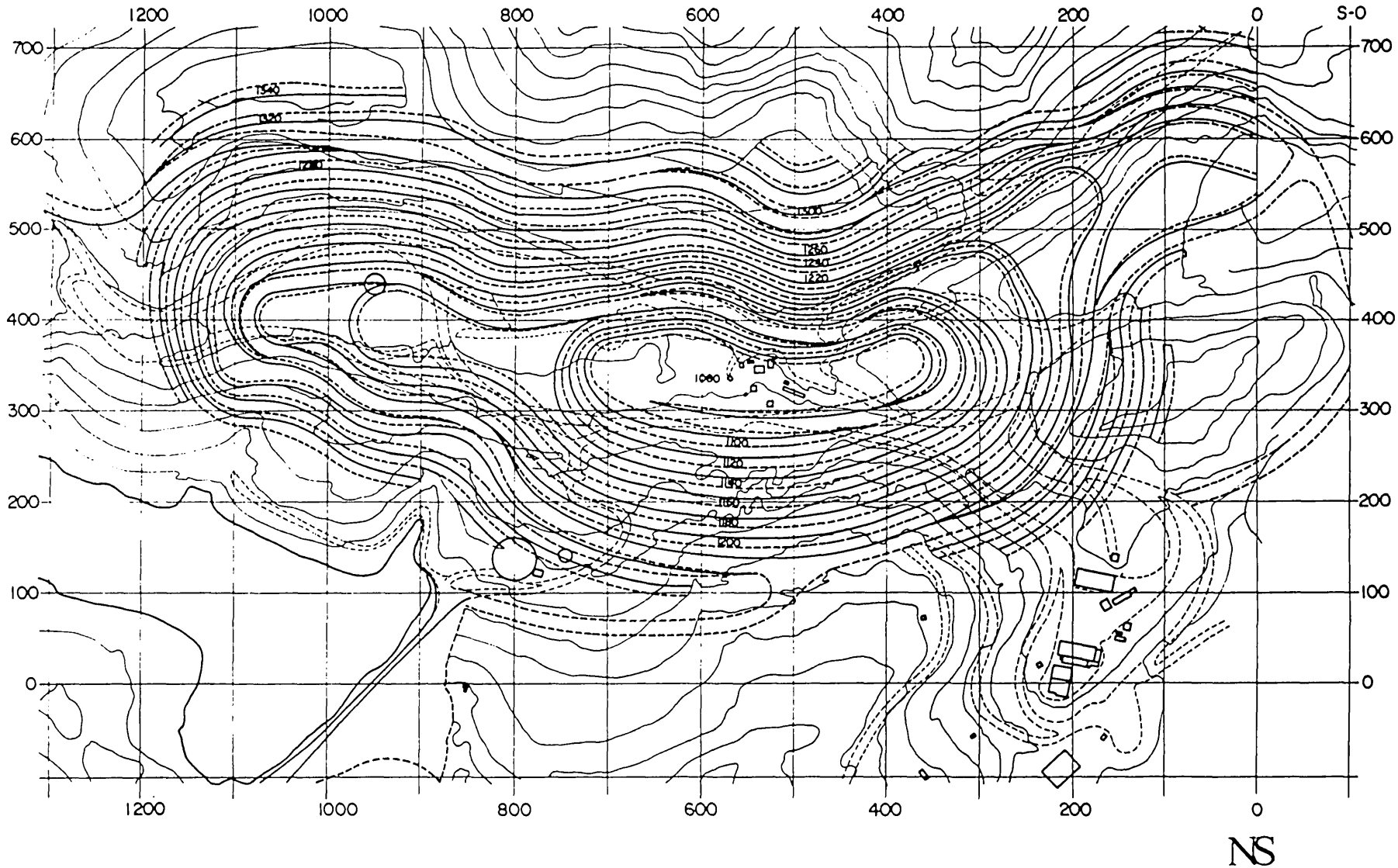


Figure 9.7 Hand-generated pit - early version of pit 1080

Figure 9.8 Computer-generated pit - present version of pit 1080

ORIGINAL 11.11.13. 00/00/77. AAIIT 1 13536

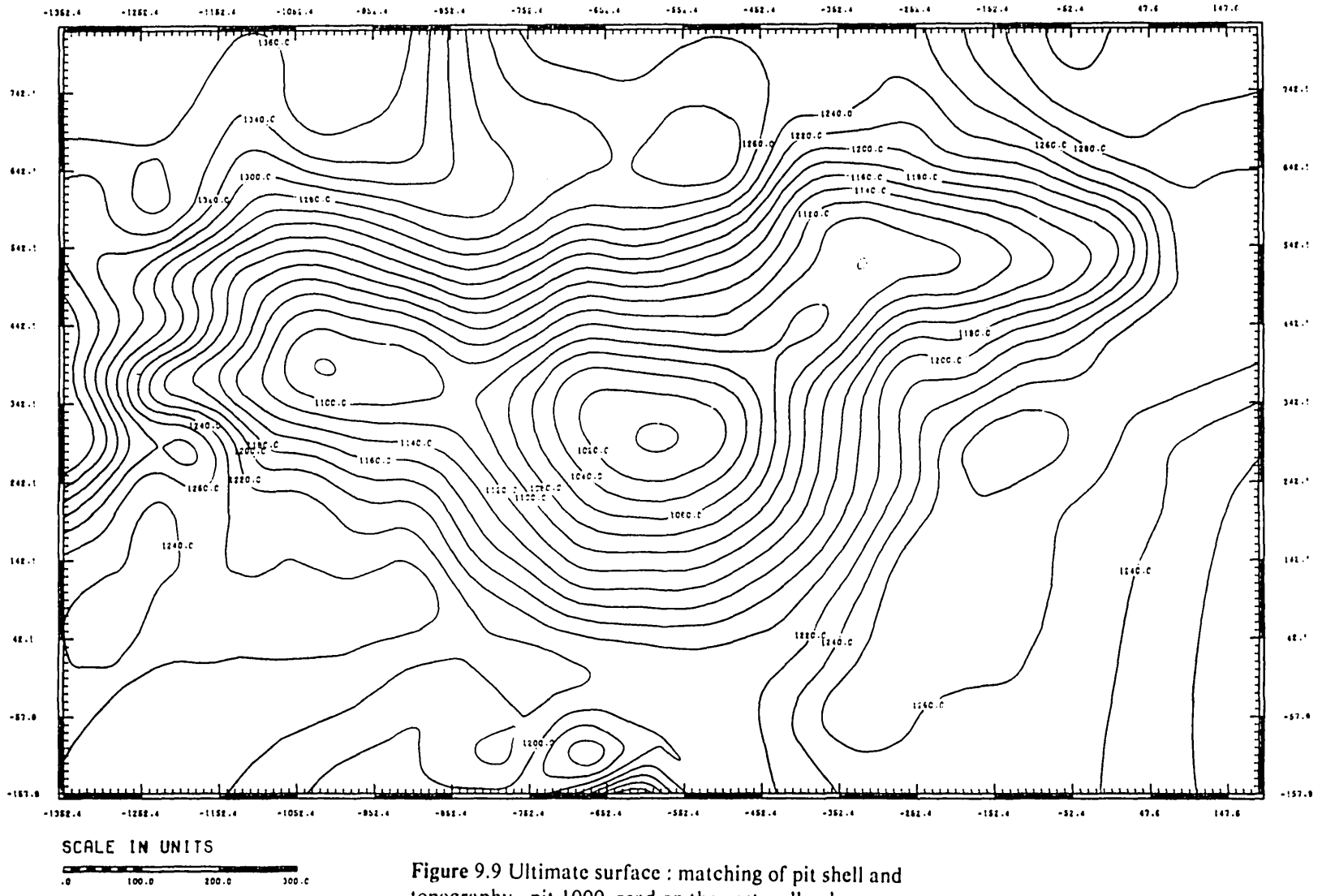


Figure 9.9 Ultimate surface : matching of pit shell and topography - pit 1000, road on the east wall only

same problem, that provided the maximum cord length is respected by way of interpolating points(see item 9.2.6), most angles will have 180 degrees, because these interpolated points lie in straight lines.

The third major change was the introduction of a procedure to smooth out the expanded polygons in each bench, by rearranging them so that their internal angles have no impractical values, for being either too sharp or too wide. This solution was suggested by Savage(1986; RTZ Technical Services; personal communication), who also mentioned the possibility of using the break-even stripping ratio as an economic criterion for pit selection.

Figure 9.7 shows a hand-generated pit which was designed by MBR personnel in Brazil. Its shape can be compared, in general terms, with the computer-generated design displayed in figure 9.8, which was obtained with the slopes that have been adopted recently. As these two figures were obtained with different slopes, they are not entirely comparable. The pit in figure 9.7 was designed with the slopes which were considered in January, 1986, whereas that in figure 9.8 was generated with those slopes adopted in March, 1986(Source: Souza; 1986; MBR Mining Engineer; personal communication).

A series of other improvements were introduced progressively, both in the OPD program itself and by way of other auxiliary programs, as follows :

- program to handle the digitised data of the ore body outline.
- program to compute the ultimate surface resulting from the matching of the pit shell with the terrain surface(see figure 9.9).
- program to count the blocks lying between the pit shell and the terrain surface.
- program to convert the number of blocks into tonnages of ore and waste.
- TSC instructions to plot the resulting ultimate surface. TSC is a graphical package created by Sibson and Thomson(1982), available at the Imperial College Computer Centre, accessed through a library call to "TSC77".

- AGRAPH instructions to plot the cross and longitudinal sections as well as the bench maps of the pits generated. AGRAPH is a graphical package created by Raby(1984). It is available at .the ICCC and has an on-line documentation manual for reference.

9.3.3 Suggestions for further improvements

A number of consultant companies offer software packages for OPD (see chapter 1), each one having special attractions. From those facilities being offered, a series of applications might have the key to solve some of the specific problems of iron ore open-pit mining, yet not necessarily meeting the exact requirements of a special mine. On the other hand, as the computerized system which has been introduced has no "black box" characteristics, being expandable in a modular fashion, the author has included some suggestions for further improvements in the embryonic OPD system,as follows:

- treatment of all data files with the aid of a proper "Data Base Management System", as mentioned by Henley(1983,pages 325 and 326). "SIR" and "SPSSX" are two packages available at the ICCC which have this capability(see list of references, Imperial College Computer Centre Bulletins).
- improvement in the accuracy of the area calculation. This can be done either by taking into account fractions of blocks, which would be an approximated solution with a higher level of detail than the present one, or by computing the exact area of the bench polygons, using the point coordinates.
- treatment of all types of material, including different categories of ore and waste(see table 9.3).
- addition of shape-controlling devices, such as number of permitted horizontal expansions, sector by sector and bench by bench, so that the mining engineer can control the pit designing process by varying the input data.
- Improvements in the interactivity of the system, that is to say, in its "manumatic" characteristics, so that manual improvements to the preliminary pit shell generated can be tentatively introduced.
- incorporation of facilities capable of taking into account the in-pit haulage road.

9.4 Optional pit shells considered for analysis

Five different minimum levels were considered for generation of optional pits : 960m, 980m, 1000m, 1020m, 1080m. For each bottom level, two alternative layouts for the in-pit road were used : a) switchbacks on the east wall only; and b) road spiralling around the pit. The details are described in the following items.

9.4.1 Minimum pit levels and base polygon

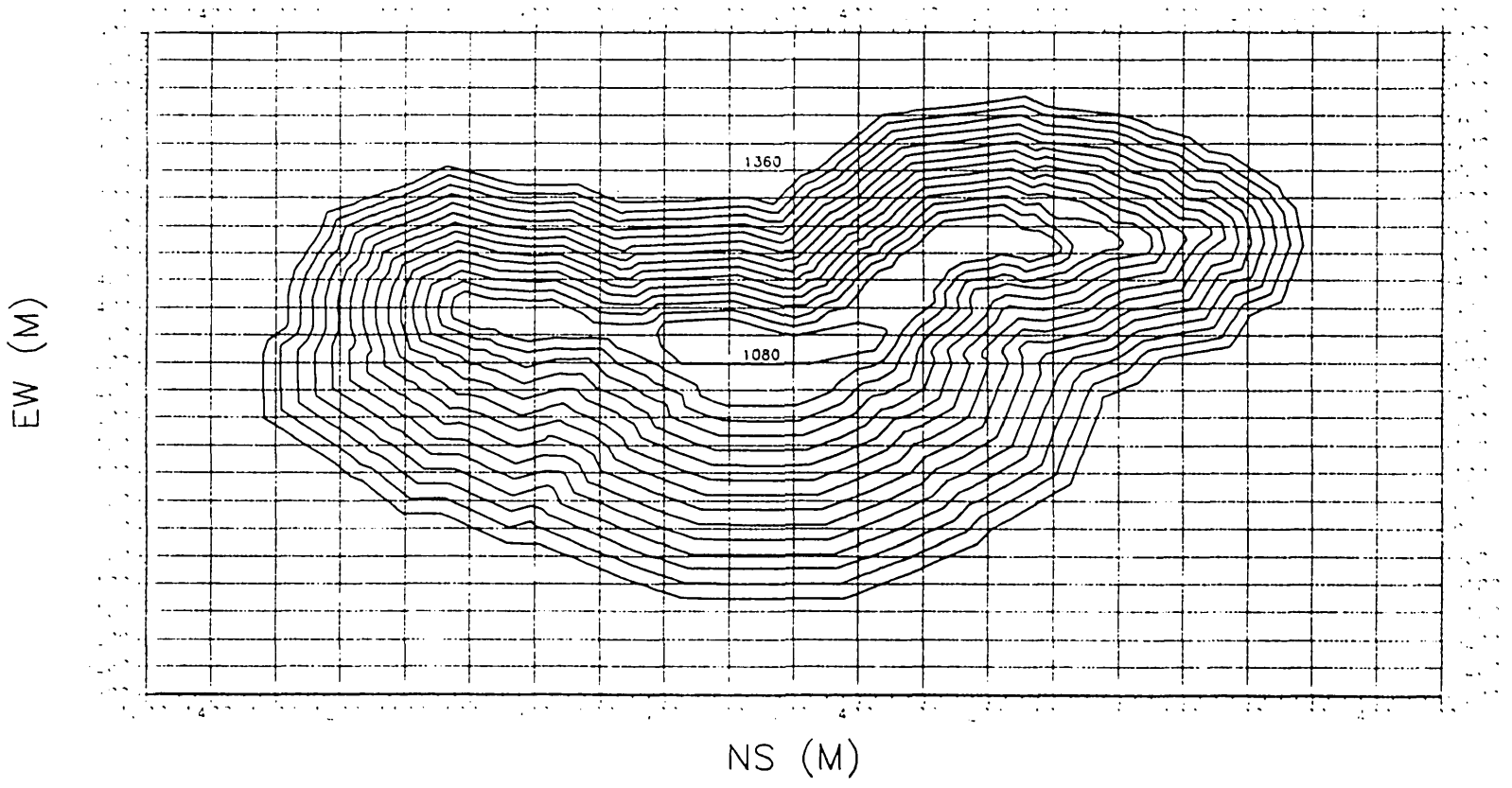
The shape of the base polygon, for each pit, was dictated by minimum operational width, ore body outline and the presence of physical boundaries in upper benches which ought to be respected. Both the position and shape of the initial polygon are reflected throughout the benches, up to the top level, therefore they should be carefully chosen.

For the pit 960, for instance, the deepest one, minimum operational width was the determinant factor, whereas for pit 1080, the shallowest one, the ore body shape was predominant, as far as the layout of the base polygon is concerned(see figures 9.10 and 9.11).

9.4.2 Layout of the in-pit road

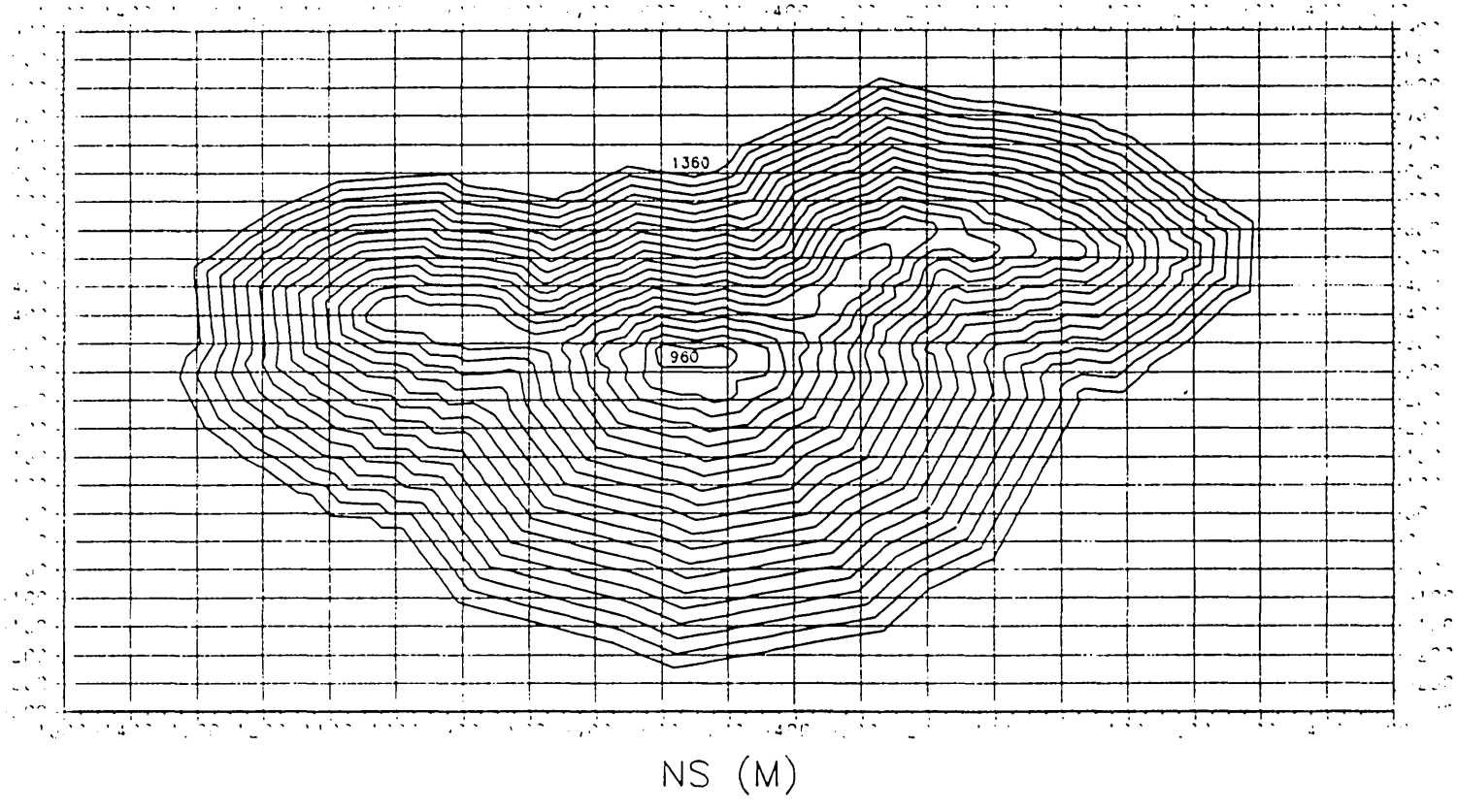
The layout selected for the in-pit road was the one with switchbacks on the east wall only, as opposed to spiralling the road around the pit. As a consequence of that, the waste tonnages were reduced roughly by 10 million tons. The east side, besides being the flattest one, has a maximum elevation much lower than the top level of the west side, therefore represents a smaller risk of having the road blocked by a slump. As a consequence of the road layout, the contour curves on the west wall of the pit displayed in figure 9.12 show a steeper angle than the correspondent lines in figure 9.13, the opposite applying to the east wall, in the same figures.

This decision on the road layout has many practical implications, such as the interference with the present eastern location of the dam, as well as the need to connect the in-pit road with the main entrance of the mine property, to the west, which links it to the local roadway network. These consequences were not analysed in detail in this study, for this would be beyond its scope. However, even if the feasibility of the "switchback" option remains to be proved, in this particular case, it has been favoured as a more generally accepted practice, as mentioned by Couzens(1979).



MUTUCA MINE PIT 1080/1

Figure 9.10 Pit 1080 - road on the east wall only



MUTUCA MINE PIT 960/1

BASIC SLOPE ANGLES (IN DEGREES) 43E 52W

3 ROAD LAPS ON THE EAST WALL ONLY

Figure 9.11 Pit 960 - road on the east wall only

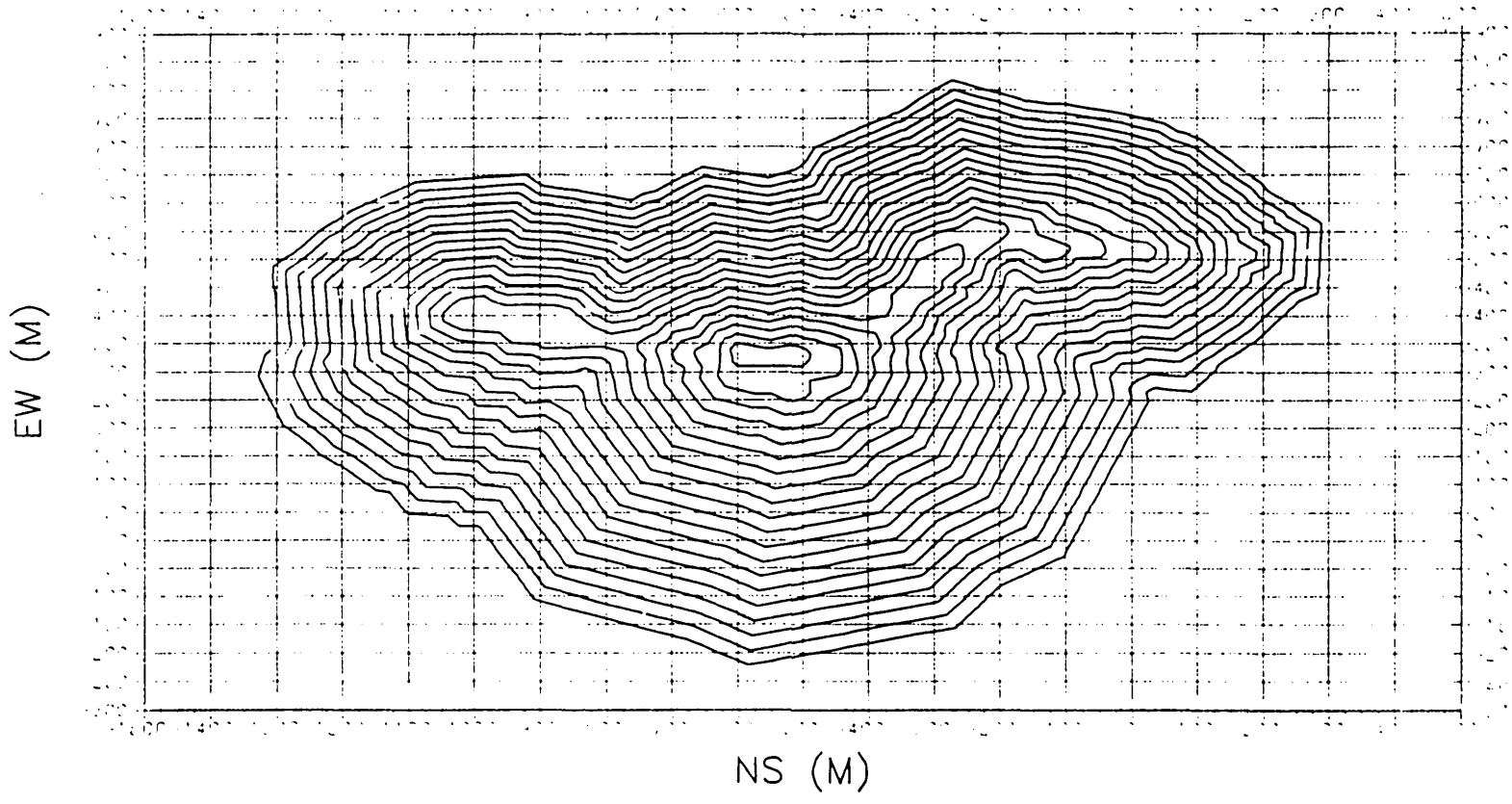
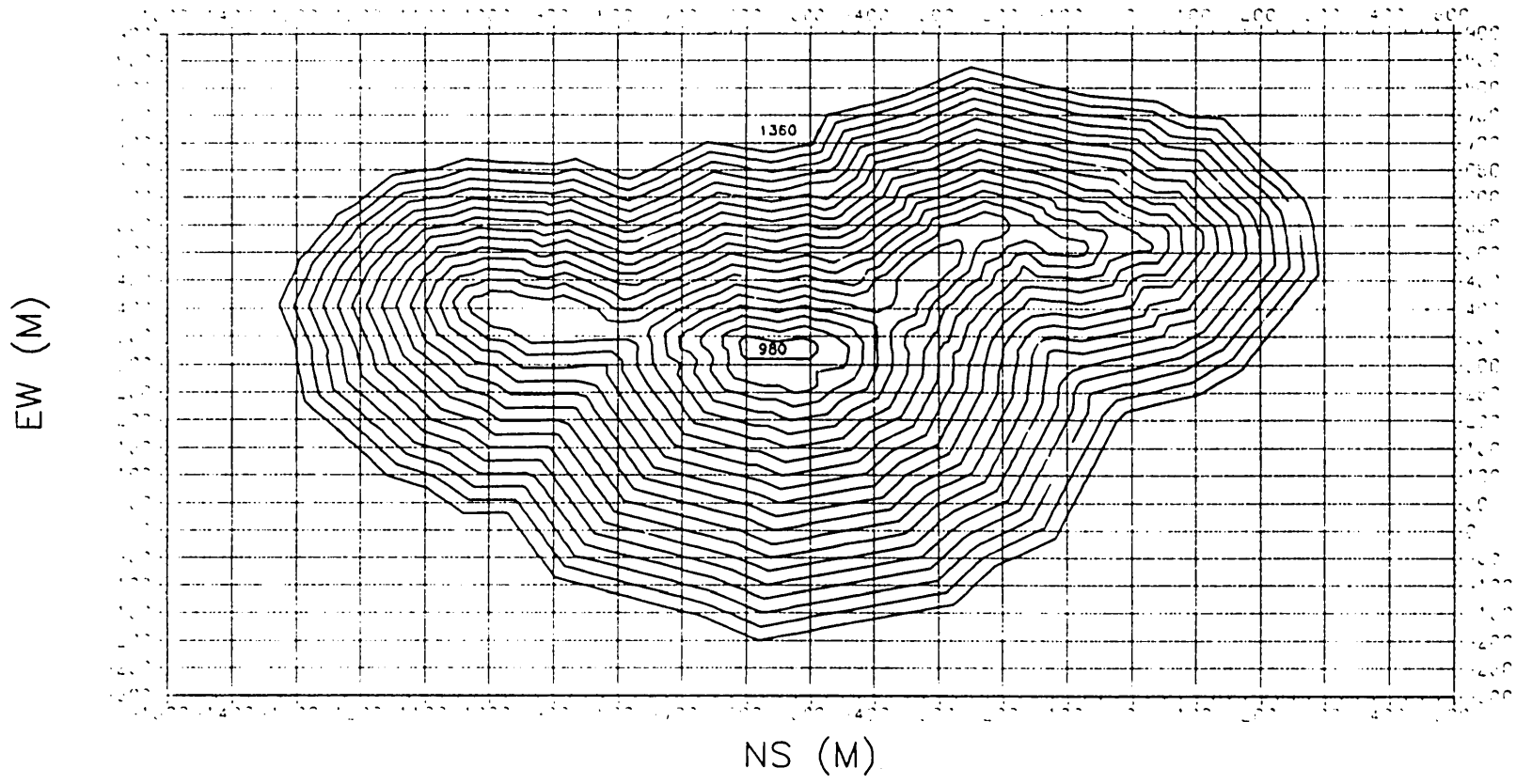


Figure 9.12 Pit 980 - road on the east wall only

UMCKA02 13.13.54. 66/06/20. ANON 1 13463



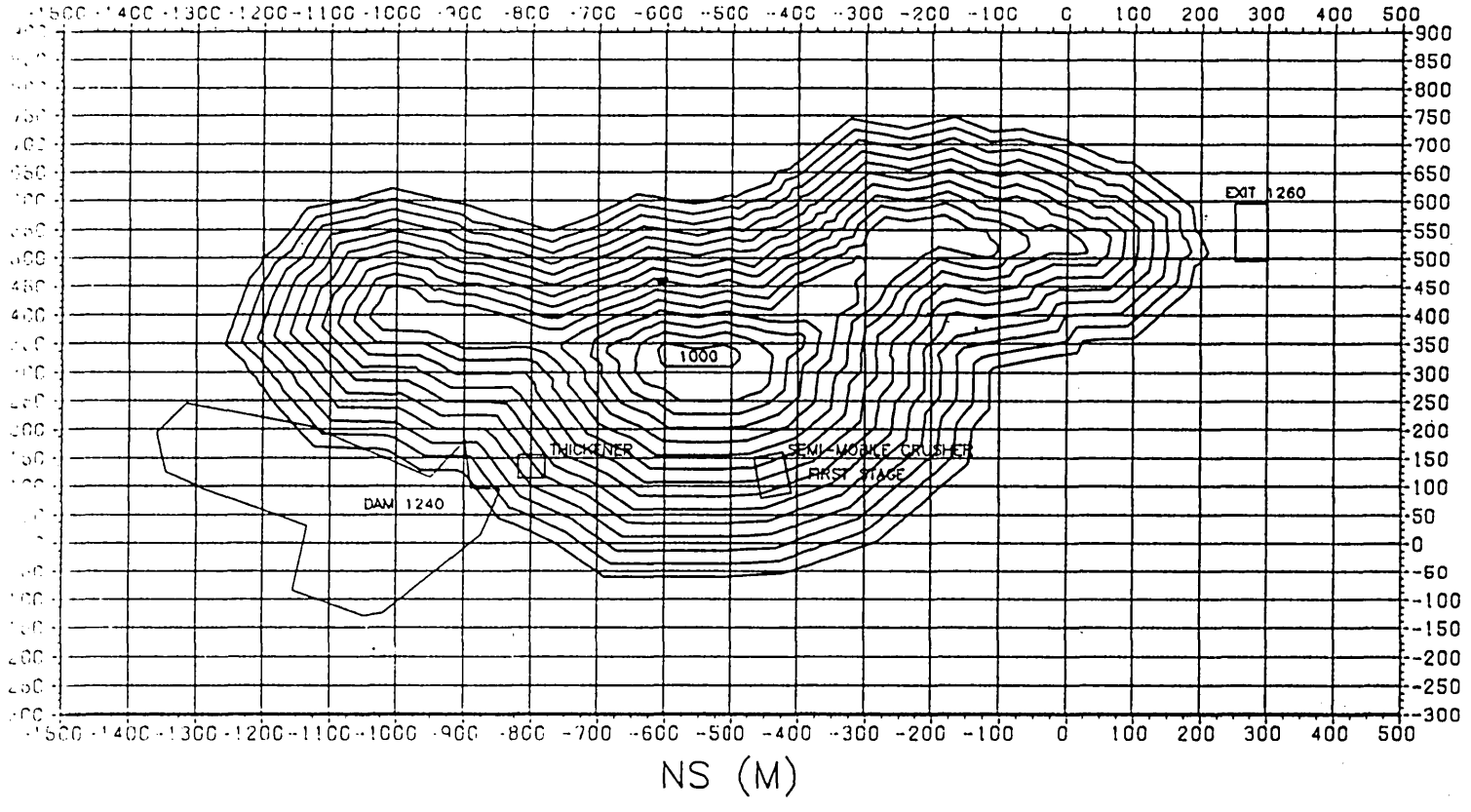
MUTUCA MINE PIT 980

BASIC SLOPE ANGLES (IN DEGREES) 43 E 52 W

2 ROAD LAPS AROUND THE PIT

Figure 9.13 Pit 980 - road spiralling around the pit

JMCKA02 17.37.48. 86/06/22. AASW 2 13530



MUTUCA MINE PIT 1000

Figure 9.14 Physical boundaries and pit 1000 - road spiralling around the pit

9.5 Numerical results - Quantitative analysis

Each one of the optional pits was evaluated separately, in terms of ore and waste volumes and tonnages. A series of tables was created, containing both top-to-bottom and bottom-up inventories of ore, waste and ore to waste ratio, bench by bench and cumulatively. The outputs were compared with hand-produced results which were calculated by MBR personnel, in parallel with the present work. An incremental analysis was made, using the concept of the economic break- even stripping ratio(BESR), as defined by Miller(1982), in order to have a preliminary assessment on what should be the ultimate pit lowest level.

9.5.1 Ore and waste tonnages, waste to ore ratio

Tables 9.4 and 9.5 summarize the outcome of the analysis on the layout of the in-pit road, as far as total tonnages are concerned. The option with switchbacks on the east wall only was preferred, as the stripping requirements are reduced roughly by ten million tons. Tables 9.6, 9.7, 9.8 and 9.9 show the detailed results of a given pit, both in terms of volumes and tonnages. Figures 9.15 and 9.16 show ore and waste tonnages for pits 1080 and 960.

Table 9.4 Analysis of total tonnages-switchbacks on the east wall only

Minimum level (m)	total number of blocks (in thousands)	total tonnage (Mt) Hematite	Waste	Ore to Waste Ratio O/W (t/t)	W/O (t/t)
1080	19.7	85.00	67.2	1.27	0.79
1020	23.3	93.30	83.0	1.12	0.89
1000	25.8	95.90	95.8	1.00	1.00
980	26.4	96.16	99.3	0.97	1.03
960	26.9	96.22	102.0	0.94	1.06

Table 9.6 Pit 1020 - Volumes, top to bottom

MUTUCA MINE , PIT 1020. - BLOCKS .

		ALL BLOCKS				HEMATITE				WASTE				ORE/WASTE	
		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH CUMUL.	
I	LEVEL	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	(BLOCKS)	
35	1360.	6.	.00	6.	.00	.00	.00	.00	.00	6.00	1.00	6.00	1.00	.00	.00
34	1350.	16.	.00	22.	.00	.00	.00	.00	.00	16.00	1.00	22.00	1.00	.00	.00
33	1340.	26.	.00	48.	.00	.00	.00	.00	.00	26.00	1.00	48.00	1.00	.00	.00
32	1330.	58.	.00	106.	.00	.00	.00	.00	.00	57.50	1.00	105.50	1.00	.00	.00
31	1320.	89.	.00	195.	.01	.00	.00	.00	.00	89.00	1.00	194.50	1.00	.00	.00
30	1310.	177.	.01	371.	.02	.00	.00	.00	.00	176.50	1.00	371.00	1.00	.00	.00
29	1300.	264.	.01	635.	.03	.00	.00	.00	.00	264.00	1.00	635.00	1.00	.00	.00
28	1290.	459.	.02	1094.	.05	29.00	.06	29.00	.03	430.00	.94	1065.00	.97	.07	.03
27	1280.	654.	.03	1748.	.08	58.00	.09	87.00	.05	596.00	.91	1661.00	.95	.10	.05
26	1270.	898.	.04	2646.	.11	135.50	.15	222.50	.08	762.00	.85	2423.00	.92	.18	.09
25	1260.	1141.	.05	3787.	.16	213.00	.19	435.50	.12	928.00	.81	3351.00	.88	.23	.13
24	1250.	1405.	.06	5192.	.22	381.50	.27	817.00	.16	1020.00	.73	4371.00	.84	.37	.19
23	1240.	1669.	.07	6860.	.29	550.00	.33	1367.00	.20	1112.00	.67	5483.00	.80	.49	.25
22	1230.	1636.	.07	8496.	.36	560.50	.34	1927.50	.23	1069.50	.66	6552.50	.77	.52	.29
21	1220.	1608.	.07	10078.	.43	571.00	.36	2498.50	.25	1027.00	.64	7579.50	.75	.56	.33
20	1210.	1503.	.06	11581.	.50	520.00	.35	3018.50	.26	982.50	.65	8562.00	.74	.53	.35
19	1200.	1407.	.06	12988.	.56	469.00	.33	3487.50	.27	938.00	.67	9500.00	.73	.50	.37
18	1190.	1308.	.06	14296.	.61	470.00	.37	3965.50	.28	830.00	.63	10330.00	.72	.56	.38
17	1180.	1209.	.05	15505.	.67	487.00	.40	4452.50	.29	722.00	.60	11052.00	.71	.67	.40
16	1170.	1110.	.05	16615.	.71	482.00	.43	4934.50	.30	628.00	.57	11680.00	.70	.77	.42
15	1160.	1011.	.04	17626.	.76	477.00	.47	5411.50	.31	534.00	.53	12214.00	.69	.89	.44
14	1150.	940.	.04	18566.	.80	420.50	.46	5832.00	.31	499.50	.54	12713.50	.69	.84	.46
13	1140.	869.	.04	19435.	.83	364.00	.44	6196.00	.32	465.00	.56	13178.50	.68	.78	.47
12	1130.	820.	.03	20255.	.86	343.00	.47	6539.00	.33	394.50	.53	13573.00	.67	.87	.48
11	1120.	646.	.03	20901.	.89	322.00	.50	6861.00	.33	324.00	.50	13897.00	.67	.99	.49
10	1110.	606.	.02	21507.	.92	325.00	.57	7186.00	.34	241.00	.43	14138.00	.65	1.35	.51
9	1100.	486.	.02	21993.	.94	320.00	.67	7514.00	.34	158.00	.33	14296.00	.66	2.08	.53
8	1090.	395.	.02	22388.	.95	258.00	.65	7772.00	.35	136.50	.35	14432.50	.65	1.89	.54
7	1080.	303.	.01	22691.	.97	188.00	.62	7960.00	.35	115.00	.38	14547.50	.65	1.63	.55
6	1070.	253.	.01	22944.	.98	161.50	.64	8121.50	.36	91.00	.36	14638.50	.64	1.77	.55
5	1060.	202.	.01	23146.	.99	135.00	.67	8256.50	.36	67.00	.33	14705.50	.64	2.01	.56
4	1050.	147.	.01	23293.	.99	109.50	.74	8366.00	.36	37.50	.26	14743.00	.64	2.92	.57
3	1040.	92.	.00	23385.	1.00	84.00	.91	8450.00	.36	8.00	.09	14751.00	.64	10.50	.57
2	1030.	58.	.00	23443.	1.00	53.50	.93	8503.50	.37	4.00	.07	14755.00	.63	13.38	.58
1	1020.	23.	.00	23466.	1.00	23.00	1.00	8526.50	.37	.00	.00	14755.00	.63	R	.58

Table 9.7 Pit 1020 - Volumes, bottom-up

MUTUCA MINE , PIT 1020. - BLOCKS .

	ALL BLOCKS					HEMATITE				WASTE				ORE/WASTE	
		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH CUMUL.	
I	LEVEL	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	(BLOCKS)	
	1000.	23.	.00	23.	.00	23.00	1.00	23.00	1.00	.00	.00	.00	.00	R	R
	1010.	53.	.00	76.	.00	53.50	.93	76.50	.95	4.00	.07	4.00	.05	13.38	19.13
	1020.	84.	.00	160.	.01	84.00	.91	160.50	.93	8.00	.09	12.00	.07	10.50	13.38
	1030.	147.	.01	307.	.01	109.50	.74	270.00	.85	37.50	.26	49.50	.15	2.92	5.45
	1040.	135.	.01	442.	.02	135.00	.67	405.00	.78	67.00	.33	116.50	.22	2.01	3.48
	1050.	124.	.01	566.	.03	161.50	.64	566.50	.73	91.00	.36	207.50	.27	1.77	2.73
	1060.	188.	.01	754.	.05	188.00	.62	754.50	.70	115.00	.38	322.50	.30	1.63	2.34
	1070.	256.	.01	1012.	.06	256.00	.65	1012.50	.69	136.50	.35	459.00	.31	1.89	2.21
	1080.	328.	.02	1340.	.08	328.00	.67	1340.50	.68	158.00	.33	617.00	.32	2.08	2.17
	1090.	325.	.02	1665.	.11	325.00	.57	1665.50	.66	241.00	.43	858.00	.34	1.35	1.94
	1100.	322.	.02	1987.	.14	322.00	.50	1987.50	.63	324.00	.50	1182.00	.37	.99	1.68
	1110.	343.	.02	2330.	.17	343.00	.47	2330.50	.60	394.50	.53	1576.50	.40	.87	1.48
	1120.	364.	.02	2694.	.20	364.00	.44	2694.50	.57	465.00	.56	2041.50	.43	.78	1.32
	1130.	420.	.03	3115.	.24	420.50	.46	3115.00	.55	499.50	.54	2541.00	.45	.84	1.23
	1140.	477.	.03	3592.	.28	477.00	.47	3592.00	.54	534.00	.53	3075.00	.46	.89	1.17
	1150.	482.	.03	4074.	.32	482.00	.43	4074.00	.52	628.00	.57	3703.00	.48	.77	1.10
	1160.	487.	.03	4561.	.36	487.00	.40	4561.00	.51	722.00	.60	4425.00	.49	.67	1.03
	1170.	478.	.03	5039.	.39	478.00	.37	5039.00	.49	930.00	.63	5255.00	.51	.58	.96
	1180.	469.	.03	5508.	.43	469.00	.33	5508.00	.47	938.00	.67	6193.00	.53	.50	.89
	1190.	520.	.04	6028.	.47	520.00	.35	6028.00	.46	982.50	.65	7175.50	.54	.53	.84
	1200.	571.	.04	6599.	.51	571.00	.36	6599.00	.45	1027.00	.64	8202.50	.55	.56	.80
	1210.	560.	.04	7159.	.55	560.50	.34	7159.50	.44	1069.50	.66	9272.00	.56	.52	.77
	1220.	550.	.04	7709.	.59	550.00	.33	7709.50	.43	1112.00	.67	10384.00	.57	.49	.74
	1230.	381.	.03	8091.	.62	381.50	.27	8091.00	.42	1020.00	.73	11404.00	.58	.37	.71
	1240.	213.	.02	8304.	.64	213.00	.19	8304.00	.40	928.00	.81	12332.00	.60	.23	.67
	1250.	135.	.01	8439.	.65	135.50	.15	8439.50	.39	762.00	.85	13094.00	.61	.18	.64
	1260.	58.	.00	8497.	.65	58.00	.09	8497.50	.38	596.00	.91	13690.00	.62	.10	.62
	1270.	29.	.00	8526.	.65	29.00	.06	8526.50	.38	430.00	.94	14120.00	.62	.07	.60
	1280.	.00	.00	8526.	.65	.00	.00	8526.50	.37	264.00	1.00	14384.00	.63	.00	.59
	1290.	.00	.00	8526.	.65	.00	.00	8526.50	.37	176.50	1.00	14560.50	.63	.00	.59
	1300.	.00	.00	8526.	.65	.00	.00	8526.50	.37	89.00	1.00	14649.50	.63	.00	.58
	1310.	.00	.00	8526.	.65	.00	.00	8526.50	.37	57.50	1.00	14707.00	.63	.00	.58
	1320.	.00	.00	8526.	.65	.00	.00	8526.50	.37	26.00	1.00	14733.00	.63	.00	.58
	1330.	.00	.00	8526.	.65	.00	.00	8526.50	.37	16.00	1.00	14749.00	.63	.00	.58
	1340.	.00	.00	8526.	.65	.00	.00	8526.50	.37	6.00	1.00	14755.00	.63	.00	.58

Table 9.8 Pit 1020 - Tonnages, top to bottom

MUTUCA MINE , PIT 1020. - TONNAGES :

(in thousand tons)

	ALL BLOCKS					HEMATITE				WASTE				ORE/WASTE	
		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH CUMUL.	
I	LEVEL	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	(T / T)	
35	1160.	6.	.00	6.	.00	.00	.00	.00	.00	33.75	1.00	33.75	1.00	.00	.00
34	1150.	16.	.00	22.	.00	.00	.00	.00	.00	90.00	1.00	123.75	1.00	.00	.00
33	1140.	48.	.00	70.	.00	.00	.00	.00	.00	146.25	1.00	270.00	1.00	.00	.00
32	1130.	106.	.00	176.	.00	.00	.00	.00	.00	323.44	1.00	593.44	1.00	.00	.00
31	1120.	189.	.01	365.	.01	.00	.00	.00	.00	500.63	1.00	1094.06	1.00	.00	.00
30	1110.	276.	.01	641.	.02	.00	.00	.00	.00	992.81	1.00	2086.88	1.00	.00	.00
29	1100.	294.	.01	935.	.03	.00	.00	.00	.00	1485.00	1.00	3571.88	1.00	.00	.00
28	1090.	459.	.02	1394.	.05	317.19	.12	317.19	.05	2418.75	.88	5990.63	.95	.13	.05
27	1080.	754.	.03	2148.	.08	634.38	.16	951.56	.09	3352.50	.84	9343.12	.91	.19	.10
26	1070.	1250.	.05	3398.	.11	1402.03	.26	2433.59	.15	4286.25	.74	13629.38	.85	.35	.18
25	1060.	1941.	.06	5339.	.16	2329.69	.31	4763.28	.20	5220.00	.69	18849.38	.80	.45	.25
24	1050.	2833.	.09	8172.	.22	4172.66	.42	8935.94	.27	5737.50	.58	24586.88	.73	.73	.36
23	1040.	3822.	.12	11994.	.29	6015.63	.49	14951.56	.33	6255.00	.51	30841.88	.67	.96	.48
22	1030.	4815.	.15	16809.	.36	6130.47	.50	21082.03	.36	6015.94	.50	36857.81	.64	1.02	.57
21	1020.	5811.	.18	22620.	.44	6245.31	.52	27327.34	.39	5776.88	.48	42634.69	.61	1.08	.64
20	1010.	6807.	.21	29427.	.50	6607.50	.51	33014.84	.41	5526.56	.49	48161.25	.59	1.03	.69
19	1000.	7800.	.24	37227.	.57	6129.69	.49	38144.53	.42	5276.25	.51	53437.50	.58	.97	.71
18	990.	8796.	.27	46023.	.61	6220.13	.53	43372.66	.43	4668.75	.47	58106.25	.57	1.12	.75
17	980.	9805.	.30	55828.	.67	5326.56	.57	48699.22	.44	4061.25	.43	62167.50	.56	1.31	.78
16	970.	10810.	.33	66638.	.72	5271.88	.60	53971.09	.45	3532.50	.40	65700.00	.55	1.49	.82
15	960.	11826.	.36	78464.	.76	5217.19	.63	59188.28	.46	3003.75	.37	68703.75	.54	1.74	.86
14	950.	12845.	.40	91309.	.80	4599.22	.62	63787.50	.47	2809.69	.38	71513.44	.53	1.64	.89
13	940.	13875.	.43	105184.	.83	3981.25	.60	67768.75	.48	2615.63	.40	74129.06	.52	1.52	.91
12	930.	14930.	.46	120114.	.86	3751.56	.63	71520.31	.48	2219.06	.37	76348.13	.52	1.69	.94
11	920.	15988.	.50	136102.	.90	3521.88	.66	75042.19	.49	1822.50	.34	78170.62	.51	1.93	.96
10	910.	17049.	.53	153151.	.94	3541.69	.72	78596.88	.50	1355.63	.28	79526.25	.50	2.62	.99
9	900.	18110.	.56	171261.	.98	3507.50	.80	82184.38	.51	888.75	.20	80415.00	.49	4.04	1.02
8	890.	19165.	.59	190426.	1.02	2821.88	.79	85006.25	.51	767.81	.21	81182.81	.49	3.68	1.05
7	880.	20218.	.63	210644.	1.07	2056.25	.76	87062.50	.52	646.87	.24	81829.69	.48	3.18	1.05
6	870.	21266.	.66	231910.	1.12	1766.41	.78	88828.91	.52	511.88	.22	82341.56	.48	3.45	1.08
5	860.	22309.	.69	254219.	1.17	1476.56	.80	90305.47	.52	376.89	.20	82718.44	.48	3.92	1.09
4	850.	23347.	.72	277566.	1.22	1197.66	.85	91503.13	.52	210.94	.15	82929.37	.49	5.68	1.10
3	840.	24379.	.75	301945.	1.27	910.75	.95	92421.88	.53	45.00	.05	82974.37	.47	20.42	1.11
2	830.	25405.	.78	327350.	1.32	585.16	.96	93007.03	.53	22.50	.04	82996.88	.47	26.01	1.12
1	820.	26423.	.81	353773.	1.38	251.56	1.00	93258.59	.53	.00	.00	82996.88	.47	R	1.12

Table 9.9 Pit 1020 - Tonnages, bottom-up

MUTUCA MINE , PIT 1020. - TONNAGES .

(in thousand tons)

	ALL BLOCKS					HEMATITE				WASTE				ORE/WASTE	
		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH		CUMULATIVE		BENCH CUMUL.	
I	LEVEL	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	ABS.	REL.	(T / T)	
1	1020.	23.	.00	23.	.00	251.56	1.00	251.56	1.00	.00	.00	.00	.00	R	R
2	1030.	58.	.00	81.	.00	505.16	.96	836.72	.97	22.50	.04	22.50	.03	26.01	37.19
3	1040.	92.	.00	173.	.01	918.75	.95	1755.47	.96	45.00	.05	67.50	.04	20.42	26.01
4	1050.	147.	.00	320.	.01	1197.66	.85	2953.13	.91	210.94	.15	278.44	.09	5.68	10.61
5	1060.	202.	.00	522.	.02	1476.56	.80	4429.69	.87	376.88	.20	655.31	.13	3.92	6.76
6	1070.	253.	.00	774.	.03	1766.41	.78	6196.09	.84	511.88	.22	1167.19	.16	3.45	5.31
7	1080.	303.	.00	1077.	.05	2056.25	.76	8252.34	.82	646.87	.24	1814.06	.18	3.18	4.55
8	1090.	395.	.00	1472.	.06	2821.88	.79	11074.22	.81	767.81	.21	2581.88	.19	3.68	4.29
9	1100.	466.	.00	1938.	.08	3587.50	.80	14661.72	.81	888.75	.20	3470.63	.19	4.04	4.22
10	1110.	586.	.00	2524.	.11	3554.69	.72	18216.41	.79	1355.63	.28	4826.25	.21	2.62	3.77
11	1120.	645.	.00	3170.	.14	3521.88	.66	21738.28	.77	1822.50	.34	6648.75	.23	1.93	3.27
12	1130.	736.	.00	3907.	.17	3751.56	.63	25489.84	.74	2219.06	.37	8867.81	.26	1.69	2.87
13	1140.	829.	.00	4736.	.20	3981.25	.60	29471.09	.72	2615.63	.40	11483.44	.28	1.52	2.57
14	1150.	920.	.00	5656.	.24	4599.22	.62	34070.31	.70	2809.69	.38	14293.13	.30	1.64	2.38
15	1160.	1011.	.00	6667.	.29	5217.19	.63	39287.50	.69	3003.75	.37	17296.88	.31	1.74	2.27
16	1170.	1110.	.00	7777.	.33	5271.88	.60	44559.38	.68	3532.50	.40	20829.38	.32	1.49	2.14
17	1180.	1209.	.00	8986.	.38	5326.56	.57	49885.94	.67	4061.25	.43	24890.63	.33	1.31	2.00
18	1190.	1308.	.00	10294.	.44	5220.13	.53	55114.06	.65	4668.75	.47	29559.38	.35	1.12	1.86
19	1200.	1407.	.00	11701.	.50	5129.69	.49	60243.75	.63	5276.25	.51	34835.63	.37	.97	1.73
20	1210.	1503.	.00	13204.	.57	5087.50	.51	65931.25	.62	5526.56	.49	40362.19	.38	1.03	1.63
21	1220.	1598.	.00	14802.	.64	6245.31	.52	72176.56	.61	5776.88	.48	46139.06	.39	1.08	1.56
22	1230.	1630.	.00	16432.	.71	6130.47	.50	78307.03	.60	6015.94	.50	52155.00	.40	1.02	1.50
23	1240.	1652.	.00	18084.	.78	6015.63	.49	84322.66	.59	6255.00	.51	58410.00	.41	.96	1.44
24	1250.	1641.	.00	19726.	.84	4142.66	.42	88495.31	.58	5737.50	.58	64147.50	.42	.73	1.38
25	1260.	1630.	.00	21356.	.89	2320.69	.31	90925.00	.57	5220.00	.69	69367.50	.43	.45	1.31
26	1270.	1606.	.00	22962.	.92	1462.03	.26	92307.03	.56	4286.25	.74	73653.75	.44	.35	1.25
27	1280.	1554.	.00	24516.	.95	634.38	.16	92941.41	.55	3352.50	.84	77006.25	.45	.19	1.21
28	1290.	1503.	.00	26019.	.97	317.19	.12	93258.59	.54	2418.75	.88	79425.00	.46	.13	1.17
29	1300.	1484.	.00	27503.	.99	.00	.00	93258.59	.54	1485.00	1.00	80910.00	.46	.00	1.15
30	1310.	1477.	.00	28981.	.99	.00	.00	93258.59	.53	992.81	1.00	81902.81	.47	.00	1.14
31	1320.	1470.	.00	30451.	1.00	.00	.00	93258.59	.53	500.63	1.00	82403.44	.47	.00	1.13
32	1330.	1463.	.00	31914.	1.00	.00	.00	93258.59	.53	323.44	1.00	82726.87	.47	.00	1.13
33	1340.	1456.	.00	33370.	1.00	.00	.00	93258.59	.53	146.25	1.00	82873.12	.47	.00	1.13
34	1350.	1450.	.00	34820.	1.00	.00	.00	93258.59	.53	90.00	1.00	82963.12	.47	.00	1.12
35	1360.	1444.	.00	36264.	1.00	.00	.00	93258.59	.53	33.75	1.00	82996.88	.47	.00	1.12

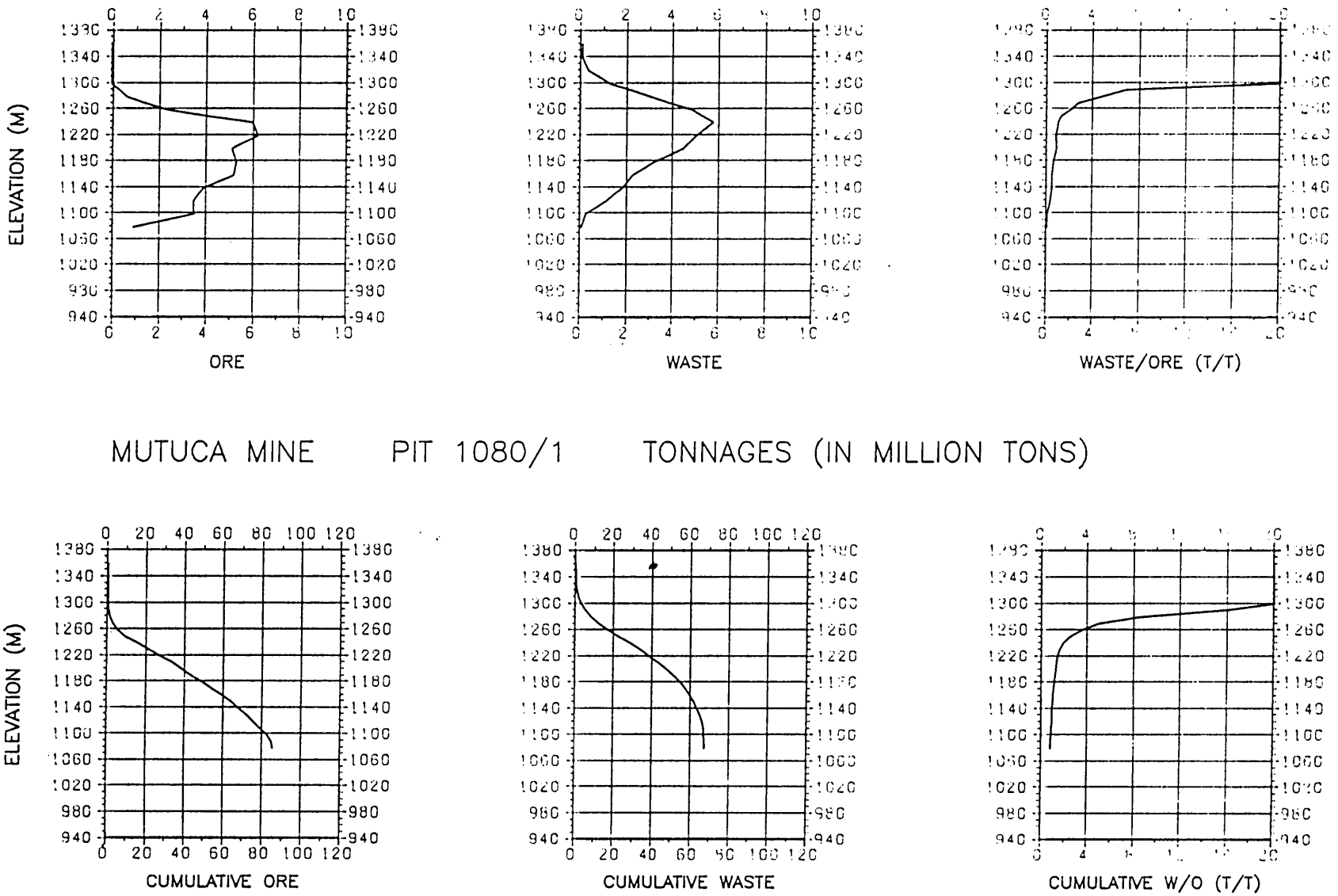
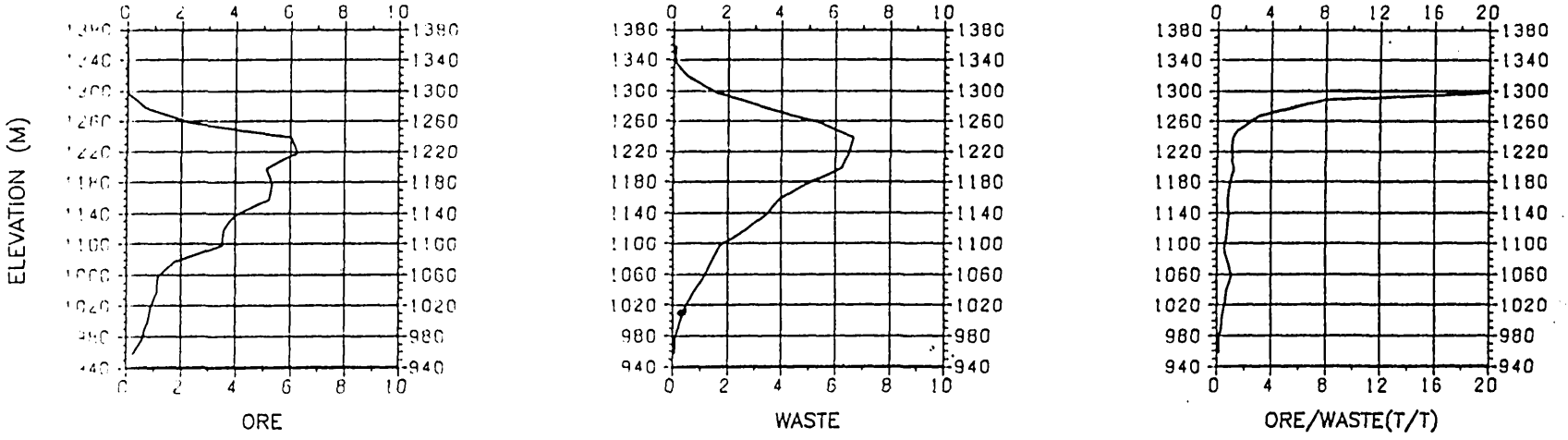


Figure 9.15 Ore and waste tonnage curves - pit 1080 - road on the east wall only

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MUTUCA MINE PIT 960/1 TONNAGES (IN MILLION TONS)

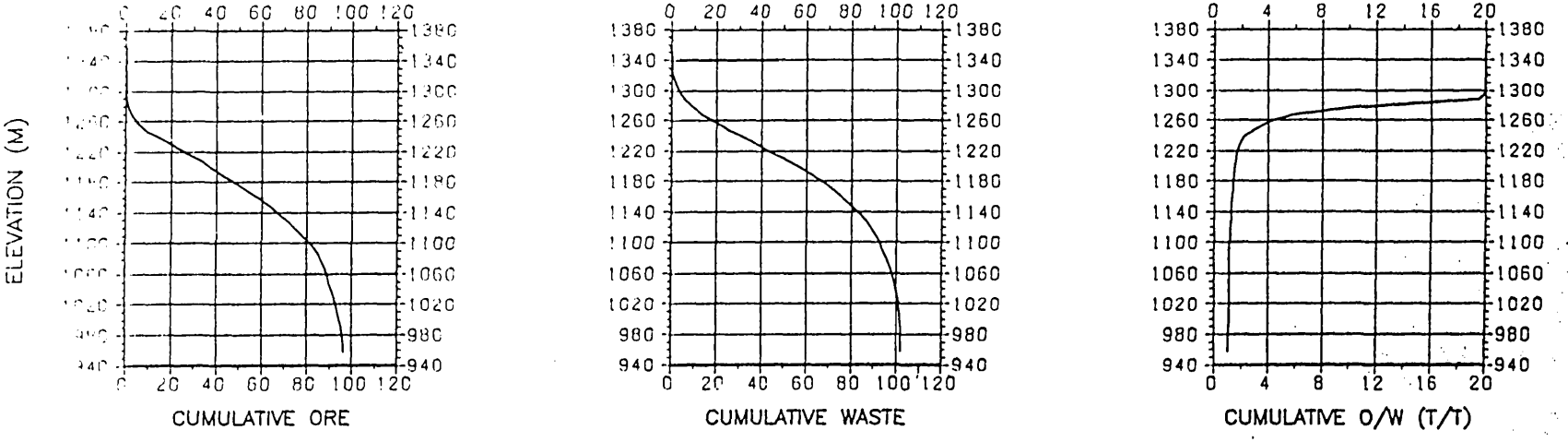


Figure 9.16 Ore and waste tonnage curves - pit 960 - road on the east wall only

Table 9.5 Analysis of total tonnages-road spiralling around the pit

Minimum level (m)	total number of blocks (in thousands)	total tonnage (Mt) Hematite	Waste	Ore to Waste Ratio O/W (t/t)	W/O (t/t)
1080	22.0	85.2	79.7	1.07	0.93
1020	24.8	93.1	91.7	1.02	0.98
1000	25.6	93.5	96.1	0.97	1.03
980	27.4	96.3	104.5	0.92	1.09
960	28.3	97.4	109.3	0.89	1.12

9.5.2 Comparison with hand-calculated results

The results in table 9.10 were generated manually, by MBR personnel in Brazil. In order to compare this figures with those of table 9.5, one should bear in mind that they were computed with the topographic map of May, 1986, whereas the "top map" used for obtaining the table 9.5 results was recorded in November, 1985. Therefore, there is roughly a difference of half a year between them, which is reflected both in tonnages of ore and waste excavated during this period. In addition to that, table 9.10 presents results for pit 990, whereas table 9.5 shows results for pits 980 and 1000, which need to be averaged in order to give an approximate idea of the tonnages for a hypothetical pit 990.

Table 9.10 Hand calculated results - road spiralling around the pit

Pit	total tonnage (Mt)		Striping Ratio		
	Hematite	Waste	Ore to waste (t/t)	(cu m/t)	Waste to ore (t/t)
1080	81.647	66.6	1.23	0.46	0.82
1020	89.600	88.7	1.01	0.52	0.99
990	91.950	95.4	0.96	0.58	1.04

9.5.3 Incremental analysis - preliminary economic appraisal

The five pits with switchbacks on the east wall only were analysed on an incremental basis, pit 1000 being selected as the envelope likely to contain the best pit. This preliminary study was later on reviewed by way of an economic analysis(see chapter 11). Table 9.11 features the aforesaid results, figure 9.17 being the corresponding graphical representation.

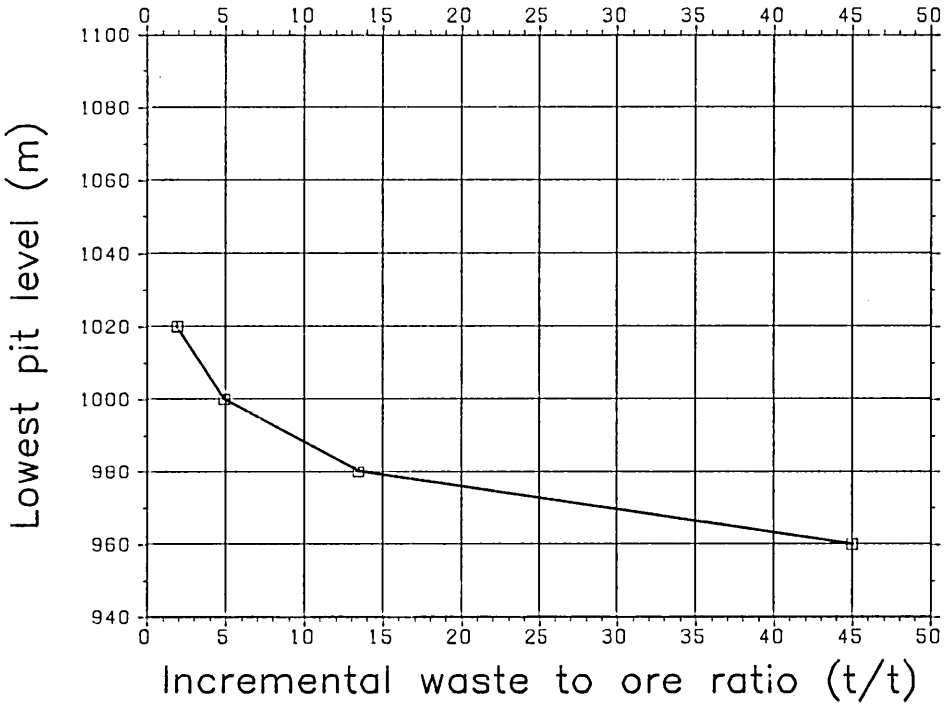


Figure 9.17 Incremental waste to ore ratio

Table 9.11 Incremental analysis-switchbacks on the east wall only

Incremental pit		Incremental tonnage(Mt)		Waste to ore ratio(t/t)	
from	to	Hematite	Waste	Ore/Waste	Waste/Ore
1080	1020	8.3	15.8	0.53	1.90
1020	1000	2.6	12.8	0.20	4.92
1000	980	0.26	3.5	0.07	13.46
980	960	0.06	2.7	0.02	45.00

Note: the break-even stripping ratio is approximately 5.0t/t(waste to ore).

9.6 Graphical outputs

The following items are a sequence of plots showing graphically the results which have been mentioned throughout this chapter, each item having a detailed description of the corresponding graphs, whenever necessary.

9.6.1 Points of the pit surface and cord length

Figure 9.18 is a pointwise representation of a pit, which shows that the maximum cord length restriction is respected.

9.6.2 Cross-sections featuring ore and waste blocks

Figures 9.19 and 9.20 show the same cross-section, NS−500, of pits 1080 and 960. The central portion of the section, between coordinates EW−200 and EW−400, is the present location of the beneficiation plant, which is to be substituted soon. The slope of the west wall, towards EW−900, is steeper than that of the east wall, towards EW 300. The graphical representation of sections of the pit, showing ore and waste blocks, followed suggestions by Allen(1986; Royal School of Mines; personal communication).

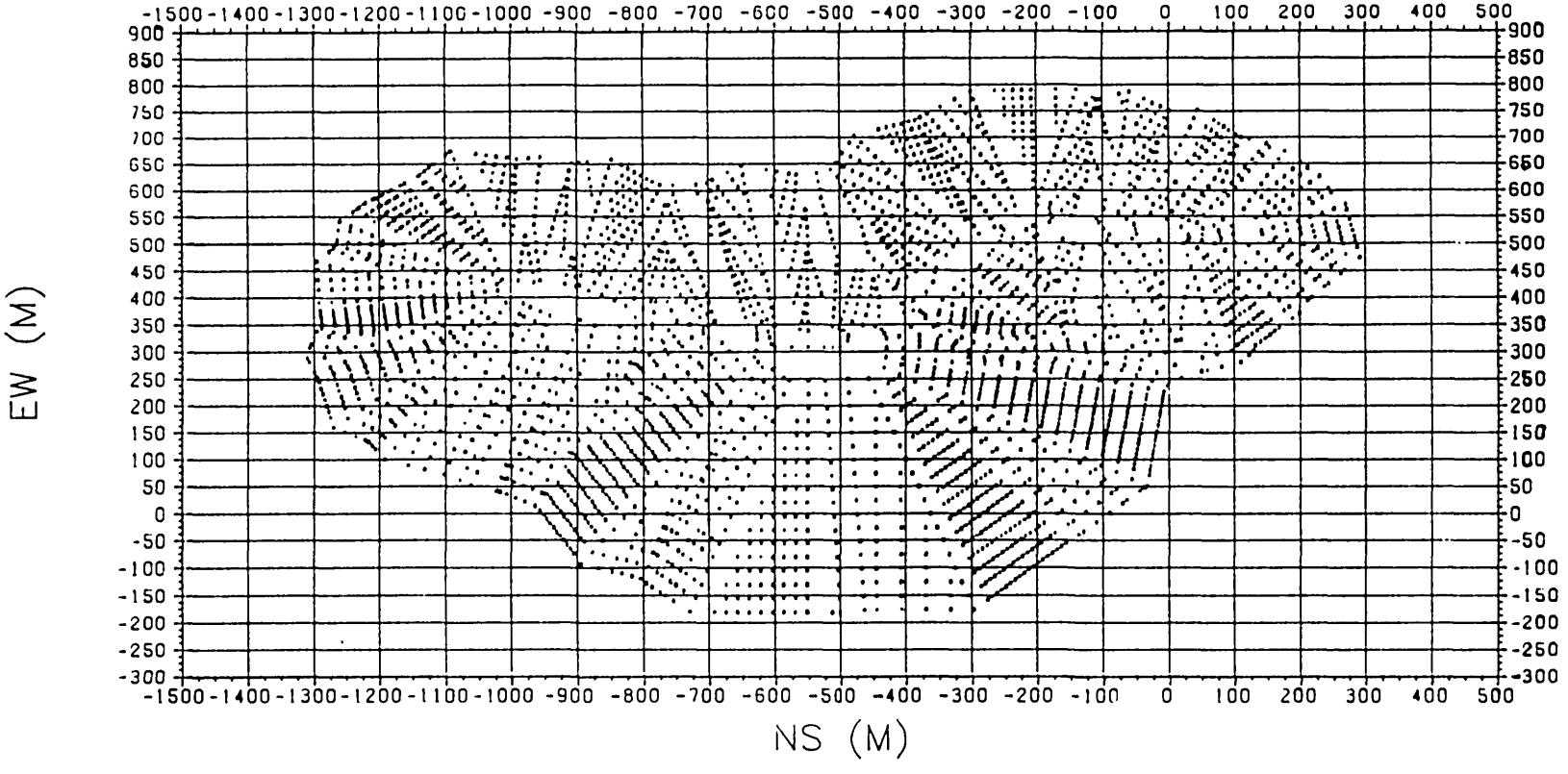
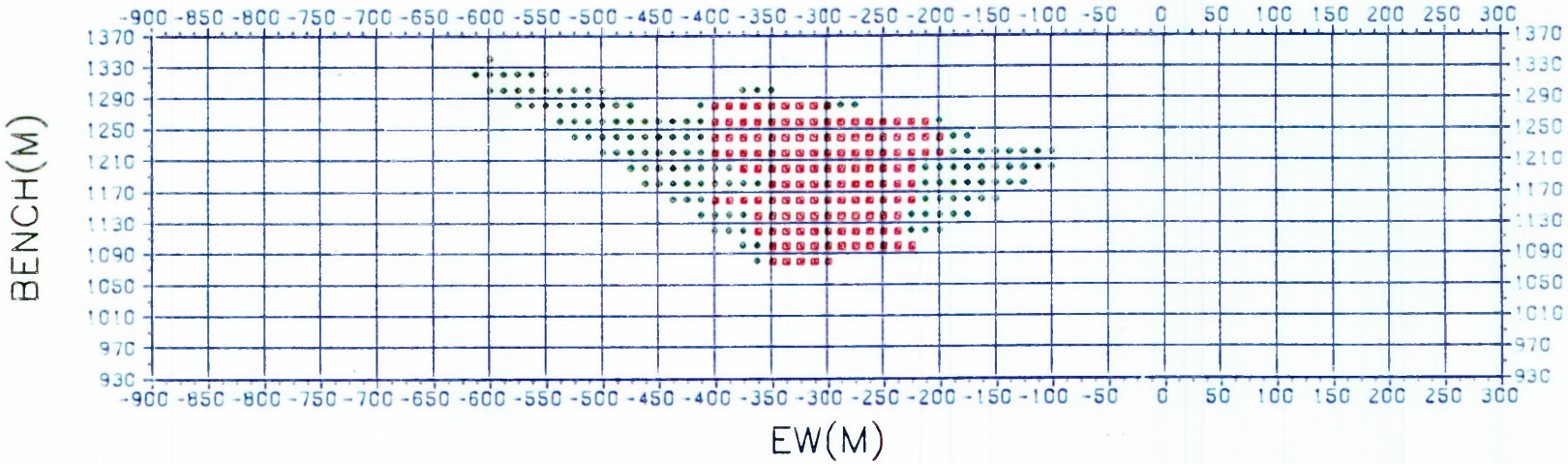


Figure 9.18 Points of the pit surface and cord length

UMCKA02 15.43.16. 86/06/27. ADFT 2 14248

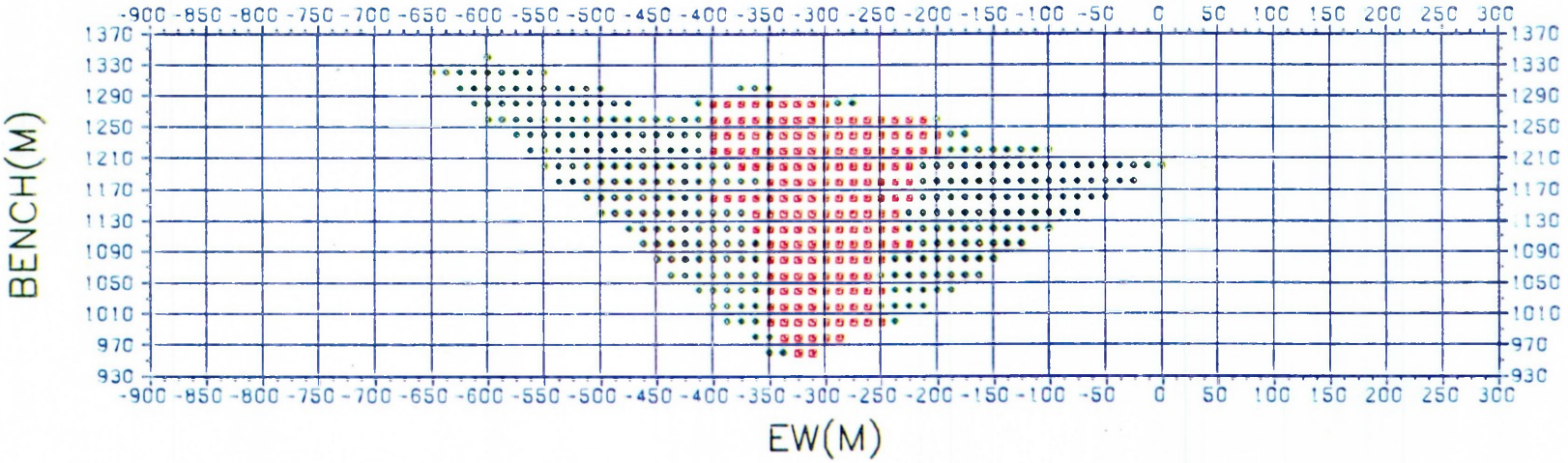


MUTUCA MINE – PIT 1080/1

CROSS–SECTION NS –500

Figure 9.19 Cross-section NS –500 - pit 1080 - road on the east wall only

UMCKA02 15.27.05. 86/06/27. ADFT 2 14239



MUTUCA MINE – PIT 960/1

CROSS-SECTION NS –500

Figure 9.20 Cross-section NS –500 - pit 960 - road on the east wall only

9.6.3 Longitudinal sections

Figures 9.21, 9.22 and 9.23 are different longitudinal sections of the same pit. This kind of visual aid is very useful for the mining engineer to visualize in which directions the ore body spreads, and how the pit limits conform to the shape of the mineralized zone.

9.6.4 Bench outline and ore body shape

Figures 9.24 through 9.27 show different bench maps of the same pit, displaying ore and waste blocks as well as the pit outline. Most blocks in bench 1260 have already been mined out, whereas in bench 1200 all blocks remain untouched.

The importance of this sort of graph is that the waste to ore proportion can be visualized, bench by bench. One possible application for these plots is that the mining engineer, with the aid of a digitiser, can "mine out" selected areas and produce alternative short-term mining plans very easily.

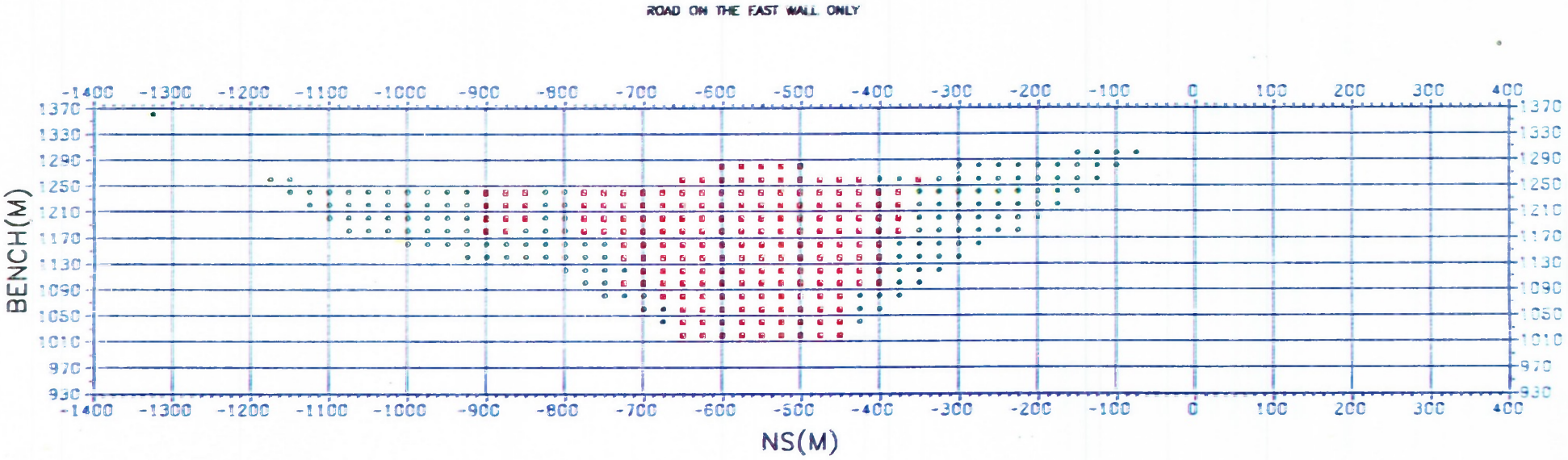
9.6.5 Pit shape and internal angle

Figure 9.28 displays another version for pit 1000, obtained with a permitted range for the internal angles different from the recommended values(see item 9.2.6), which were used for the generation of the version shown in figure 9.14.

The importance of having a permitted range for the internal angle is exemplified in figures 9.29 and 9.30. The "working pit" 1200 featured in the former figure was obtained through an early version of the OPD program. Its central polygon is the bottom level, 1200m, the next one is bench 1220. The ore body shape, in this bench, has been reproduced with too many details. On the other hand, figure 9.30 displays the bench 1220 as generated by the last version of the program. The polygon is much smoother than the aforementioned version, yet the shape of the ore body has been approximated quite reasonably.

Another reason for including figure 9.30 at this point of the chapter is for the sake of comparison with figure 9.26. The former shows bench 1220 of a working pit, containing virtually no waste blocks, the latter displays the ultimate pit's bench 1220, with the waste blocks outnumbering the ore blocks by a large margin. This simple example highlights the importance of working pits, which are the scope of the next chapter.

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MUTUCA MINE – PIT 1000

LONGITUDINAL SECTION EW 300

Figure 9.21 Longitudinal section EW 300 pit 1000 - road on the east wall only

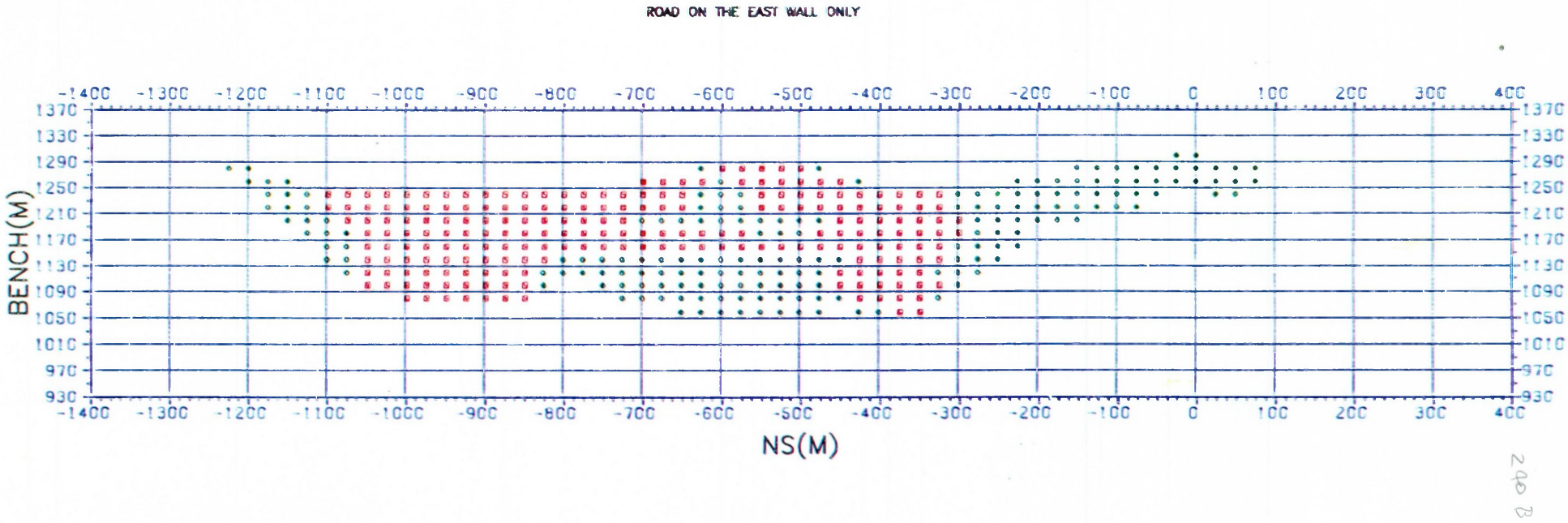


Figure 9.22 Longitudinal section EW 400 pit 1000 - road on the east wall only

UMCKA02 13.12.37. 86/06/30. AAQM 1 2.23 14391

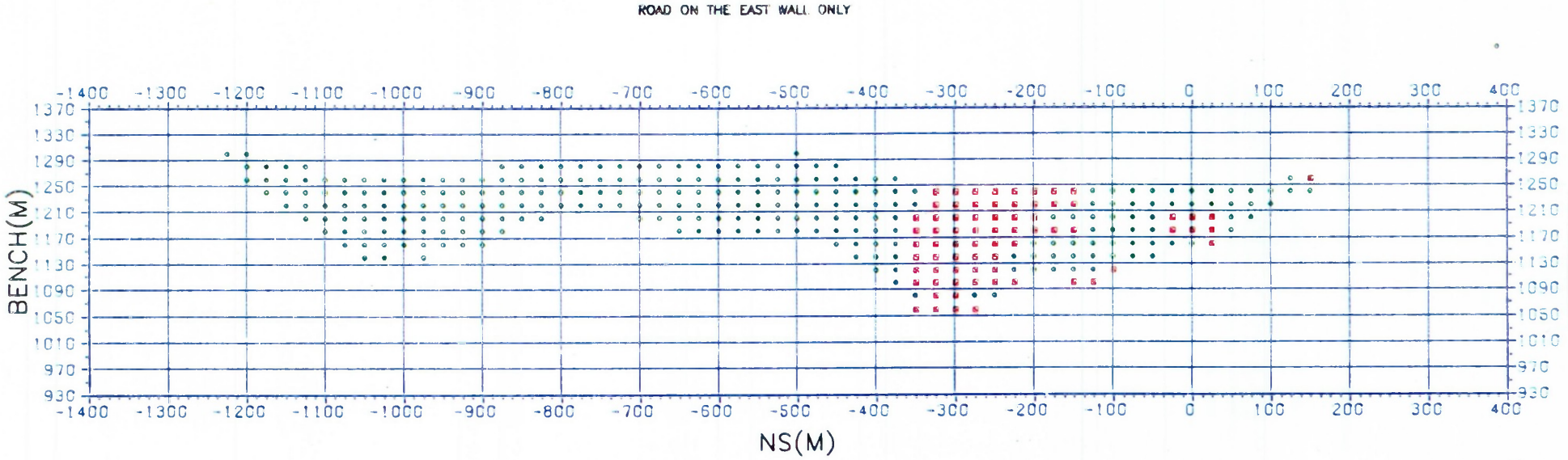


Figure 9.23 Longitudinal section EW 500 pit 1000 - road on the east wall only

UMCKA02 16.04.27 86/07/01 ACLH 1 14543

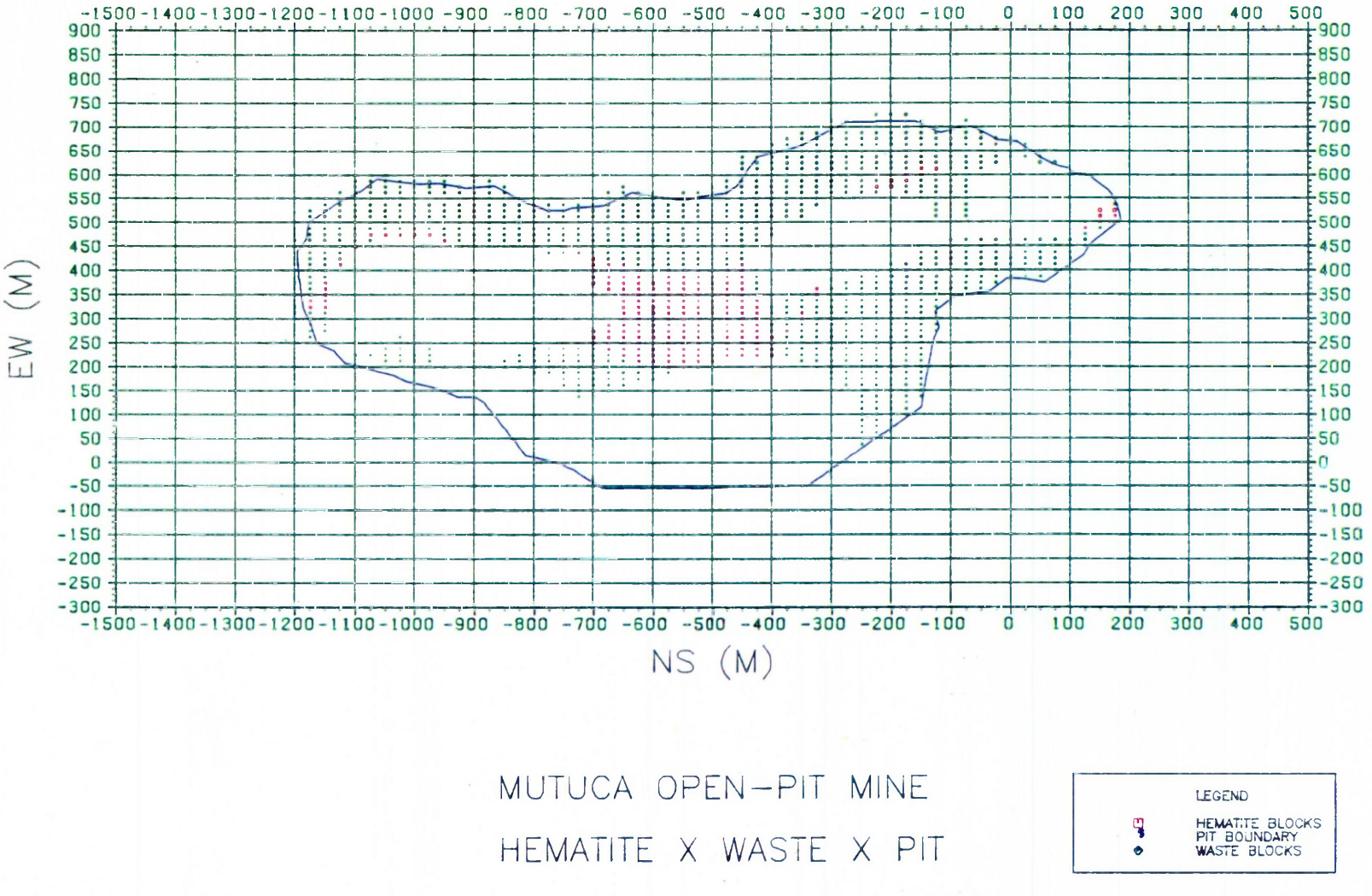


Figure 9.24 Bench map - hematite×waste×pit - bench 1260, pit 1000, road on the east wall only

UMCKA02 16.11.40 86/07/01 ACLH 1 14545

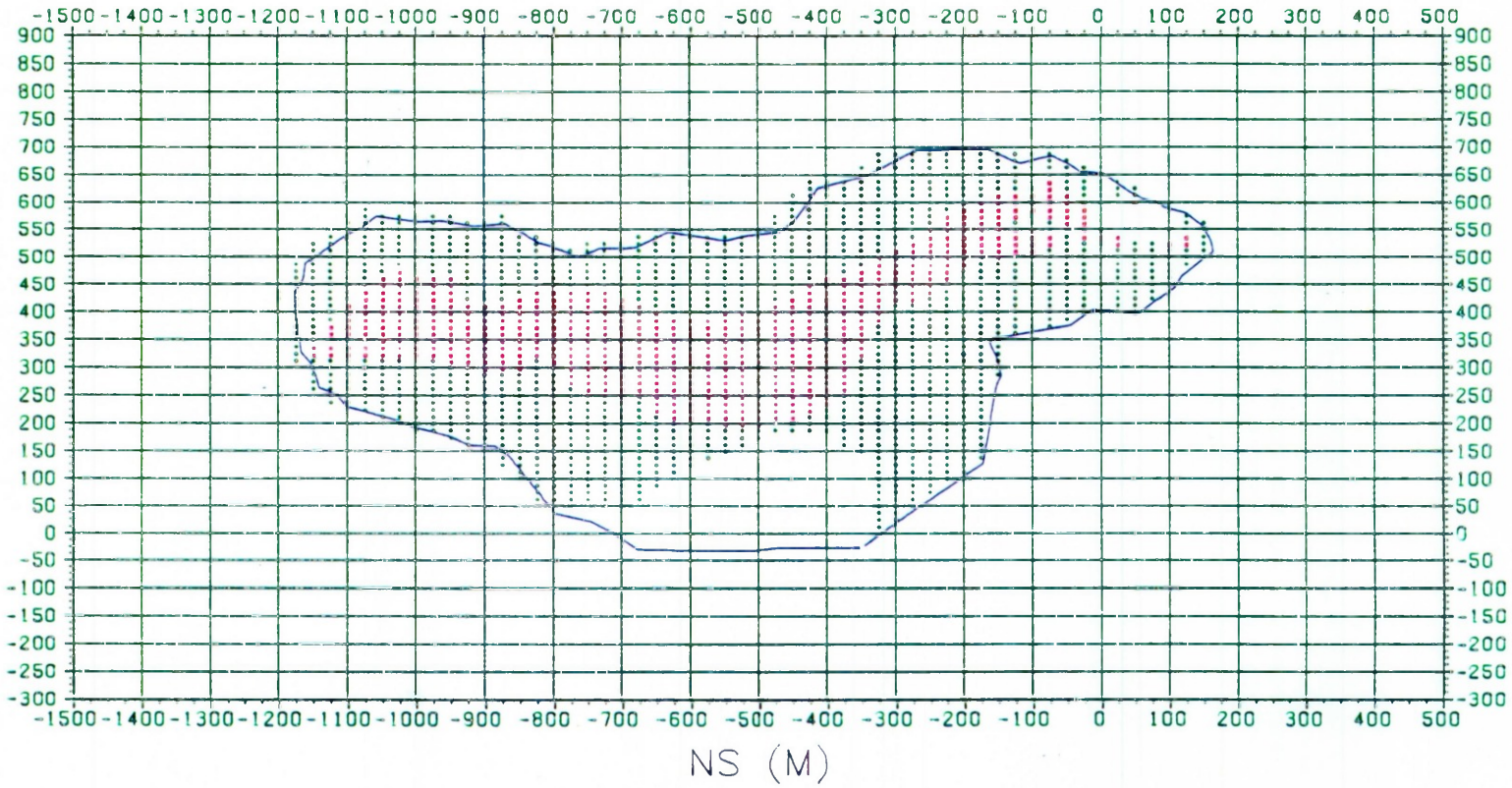


Figure 9.25 Bench map - hematite×waste×pit - bench 1240, pit 1000, road on the east wall only

UMCKA02 16.35.09 86/07/01 ACLH 1 14549

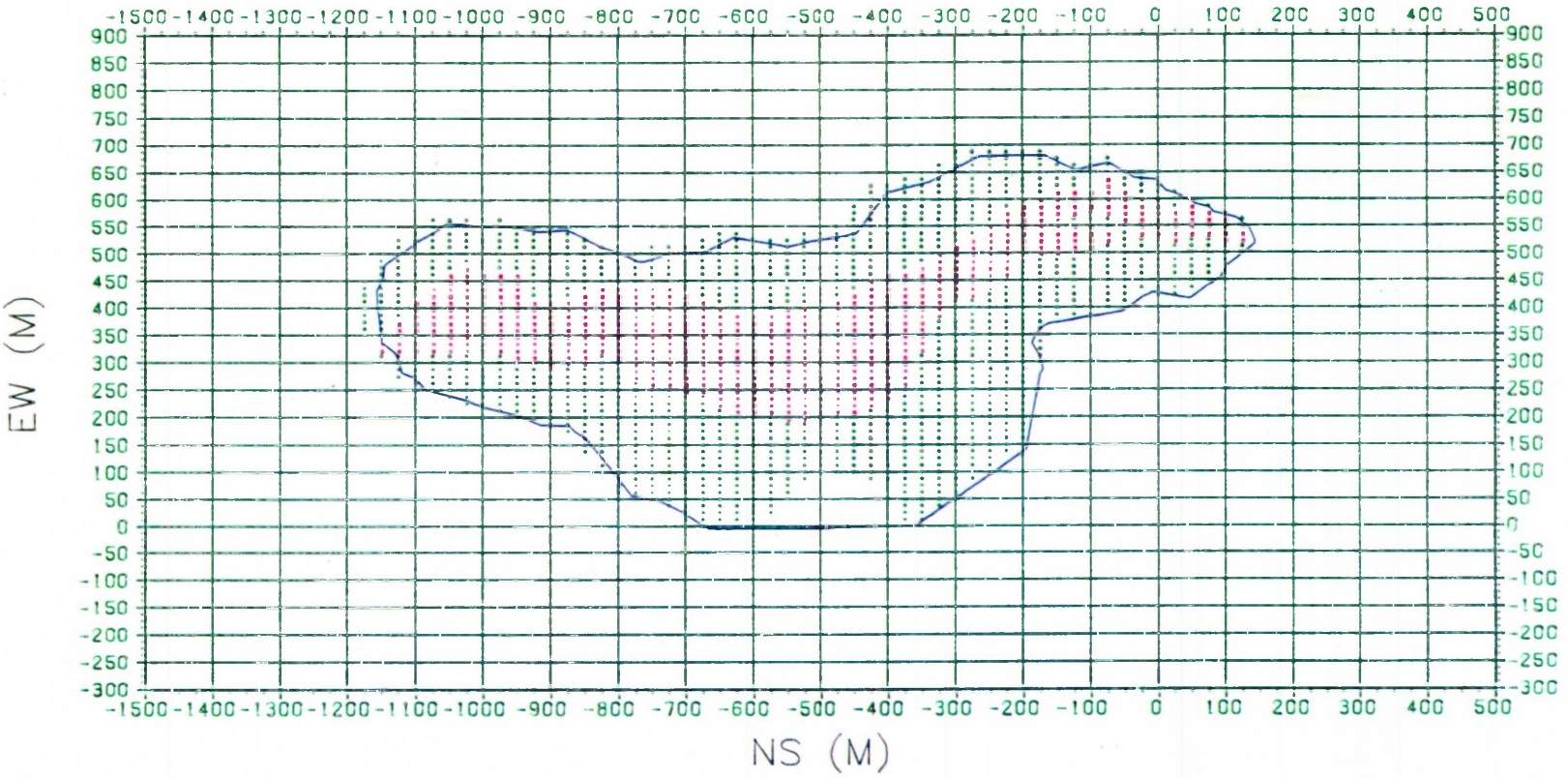


Figure 9.26 Bench map - hematite×waste×pit - bench 1220, pit 1000, road on the east wall only

UMCKA02 15.14.03 86/07/01 ACLH 2 14533

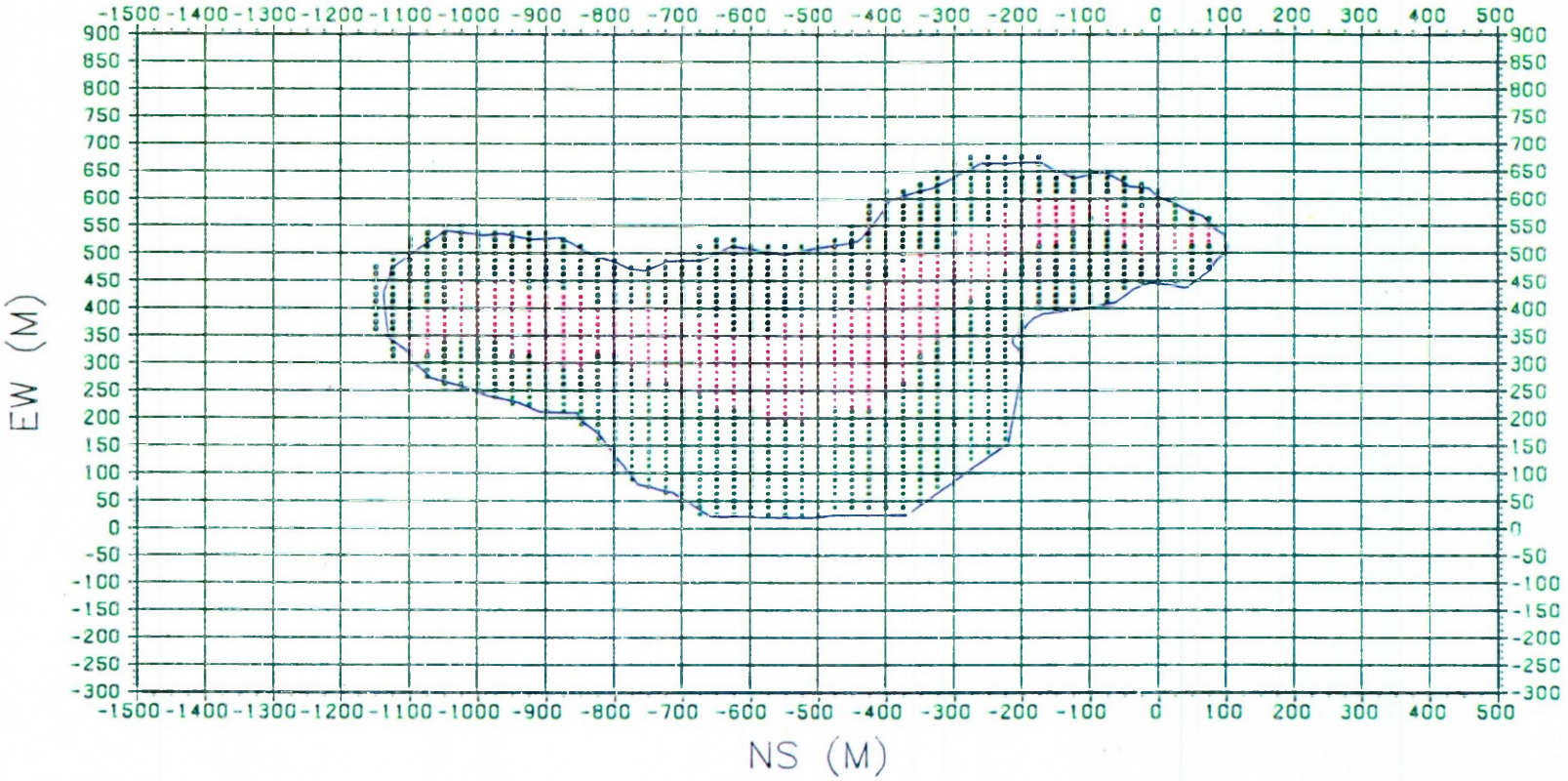


Figure 9.27 Bench map - hematite×waste×pit - bench 1200, pit 1000, road on the east wall only

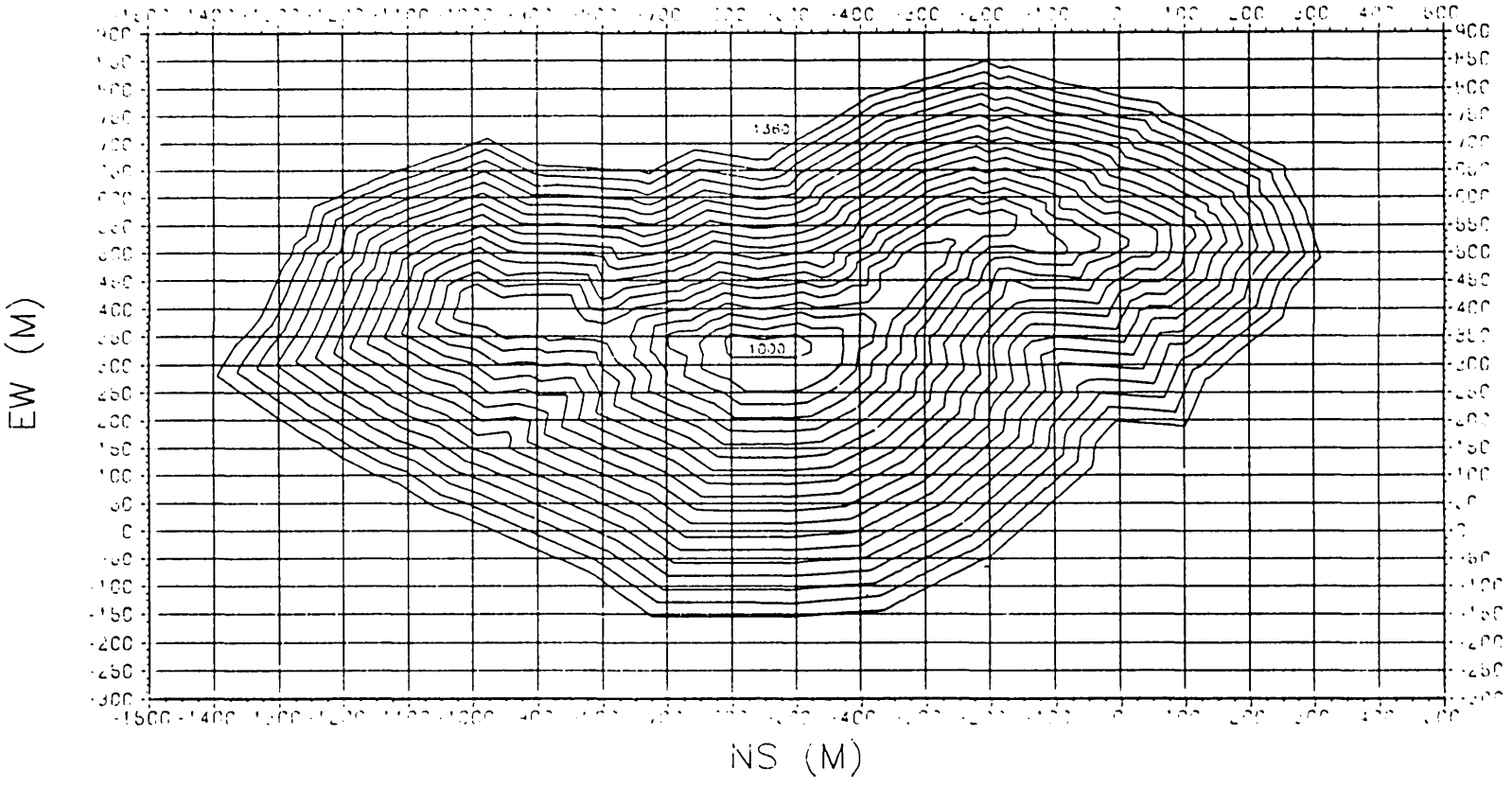


Figure 9.28 Pit shape as a function of the internal angle

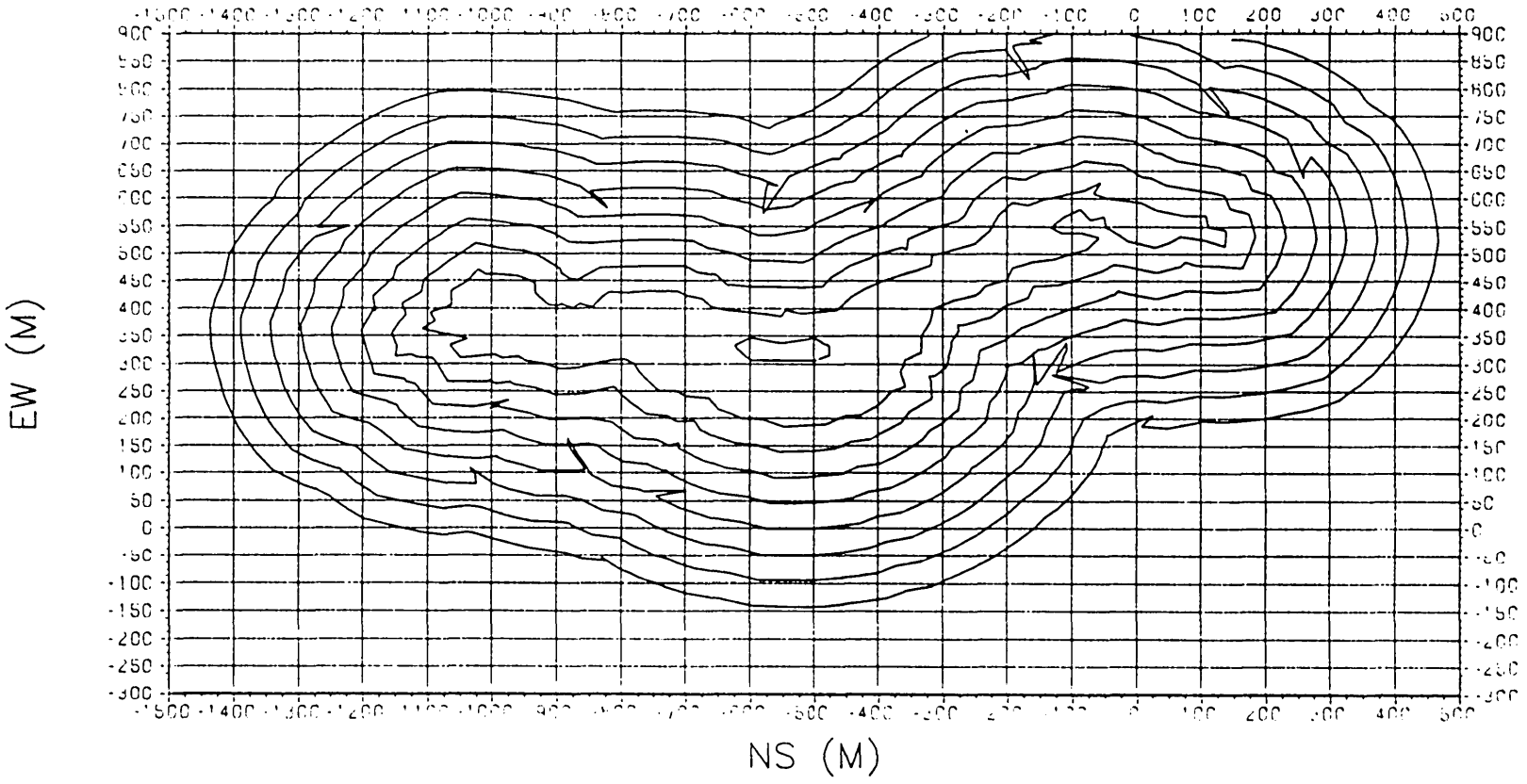
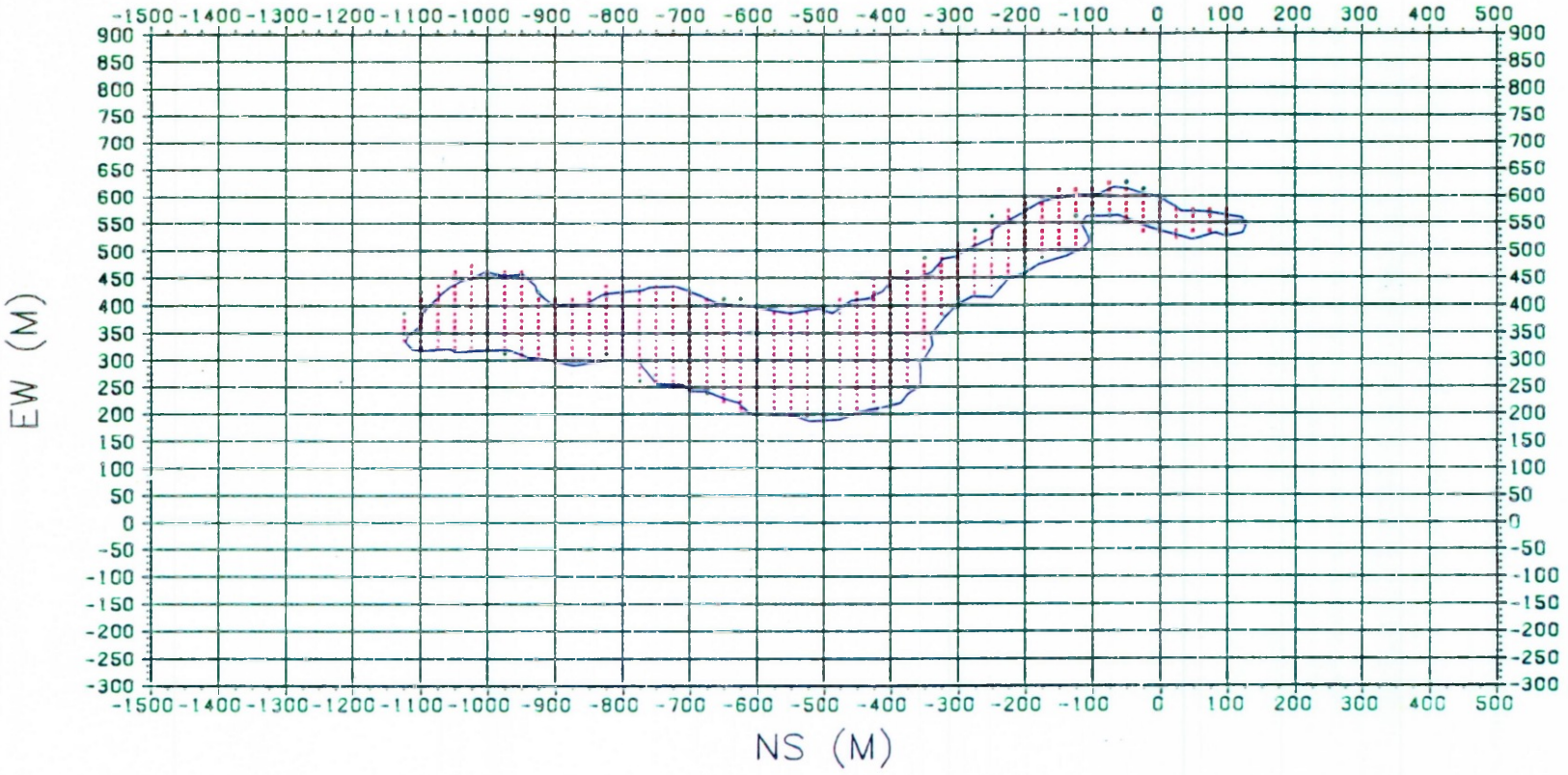


Figure 9.29 Early version of the OPD program - working pit 1200

UMCKA02 16.44.53 86/07/01 ACLH 1 14554



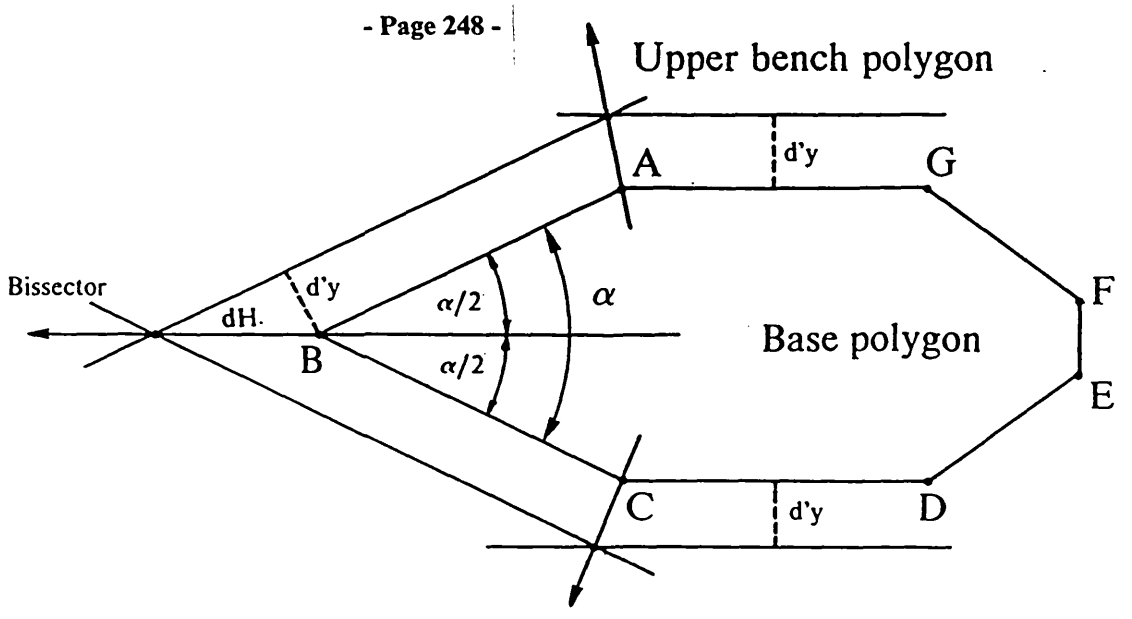
MUTUCA OPEN-PIT MINE

HEMATITE X WASTE X PIT

BENCH 1220/ WORKING PIT 1200

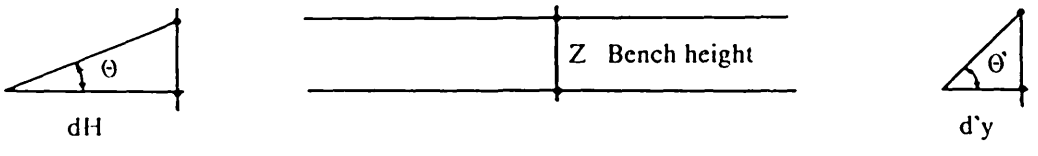
LEGEND
HEMATITE BLOCKS
PIT BOUNDARY
WASTE BLOCKS

Figure 9.30 Present version of the OPD program - bench 1220, working pit 1200



a)Projection of points - along the bissector, slope = Θ

b) Projection of chords - normal to chords, slope = Θ'



d'y / dH = sin (α/2) dH = d'y / sin (α/2) = (Z /tgΘ')/sin(α/2)
but dH = Z / tgΘ
thus Z /tgΘ = dH = (Z/tgΘ') / sin (α/2)
tgΘ' = tgΘ / sin (α/2)
tgΘ = tgΘ' sin(α/2)
therefore Θ = arctg (tgΘ'×sin (α/2))

Examples

α	Θ'	Θ
	chord	point
180°	52°	52°
150°	52°	51°
90°	52°	42°
45°	52°	26.1°
30°	52°	18.3°
15°	52°	9.5°
0°	52°	0.0°

Figure 9.31 Scheme of the "Litzinger" algorithm

10. Working pit sequences

10.1 Summary and conclusions
10.2 Introduction to this chapter
10.2.1 "Working pit sequences"
10.2.2 Pit scheduling
10.2.3 Stripping rates and working pit sequences
10.2.4 Size of the pushbacks
10.2.5 Suggested solution for Mutuca Mine
10.3 Input data
10.3.1 Working slopes
10.3.2 Lowest elevation for each working pit
10.3.3 Bench height
10.4 Brief description of the procedure adopted
10.5 Numerical results-quantitative analysis
10.6 Graphical outputs
10.6.1 Cross-sections
10.6.2 Longitudinal sections
10.6.3 Ore and waste tonnage curves
10.7 Simplified pit scheduling exercise
10.8 Incremental analysis-preliminary economic appraisal

10.1 Summary and conclusions

This chapter starts with an introduction in which an explanation about "working pit sequences" is given and a brief review of publications on pit planning and scheduling is made. Next, the input data needed for the pit phasing exercise is described and the methodology adopted is summarized. Following that, the criteria for the selection of a particular sequence of pushbacks are explained. Working pits are established for the ultimate pits with intermediate depths, namely, pits 1020, 1000 and 980. The resulting numerical and graphical outputs are presented afterwards. A simplified pit scheduling exercise is reported and discussed. Finally, an incremental analysis is made as a preliminary economic appraisal of the sequential working pits.

A conclusion emerging from this chapter is that one can not separate the geometrical, operational, sequential and economic characteristics of open-pit planning. Long-range mining plans, even if they do not take into account all practical details, must prove to be feasible, as far as these characteristics are concerned. The planning exercise can be further detailed by phasing and scheduling each working pit **per se**.

10.2 Introduction to this chapter

The definition of "working pit sequences" and the relevance of this approach are commented on the following topics. Many observations were based on a lecture by Savage (RTZ Technical Services), at the Royal School of Mines, in 1986.

10.2.1 "Working pit sequences"

One of the best explanations on working pit sequences was given by Mathieson (1982):

> "(...) orebodies are normally mined in stages, so as to defer waste stripping of the surface mining ventures. These stages are commonly called sequences, expansions, phases, working pits, slices or pushbacks. They are the basic building block on which more detailed time period planning is subsequently made.
>
> If a given pit design cannot, for example, be practically subdivided into phases, then all waste above the first ore bench must obviously be stripped in the preproduction period. On the other hand, if phasing is possible, then some of this waste stripping and hence negative cash flow can be deferred until the production period".

10.2.2 Pit scheduling

Although computer applications in open-pit design have been published as early as 1961, by Caldwell, only recently has the emphasis been transferred to the planning and scheduling aspects of surface mining. However, proposed algorithms have not always had successful practical applications.

Graham (1983) reported:

"Mining scheduling has been approached using linear programming techniques (...). It appears that the complexitites of many open pit mines result in a very large linear program which is costly to write and run on a computer. Also, because of the rather 'black box' nature of the results, they are viewed with a certain scepticism. However, future improvements will render this technique quite practical."

Mathieson (1982) lists a series of limitations of scheduling systems, and concludes with the following observation:

"In summary, the automatic scheduling systems developed to date have not enjoyed very wide application due to their relatively high computing cost to run, and their inability to properly simulate real open pit mining situations. However, they can provide an initial guideline as to the preferred development sequence for the ore body."

Iles and Perry (1981) mentioned a similar problem:

"The initial intentions were to implement PED [Pushback Extraction System, a scheduling procedure] using linear programming, but they were discouraged by an excessive computer processing time."

10.2.3 Stripping rates and working pit sequences

Different approaches can be used to solve the problem of phasing the exploitation of a ultimate pit. In this respect, the following fundamental questions need to be answered:

- What are the basic criteria for subdividing the ultimate pit in sequential working pits: the need of a certain tonnage of ore, waste to ore ratios, deferred stripping requirements, equipment capacity or other standards?
- How many stages should be considered, that is to say, how large should each pushback be?
- What geometrical constraints should be observed: working slopes, minimum width, equipment operational space, road access, and/or other factors?

The geometrical constraints are very important. The solution has to take into account the ore body shape, the topography, the ultimate pit design and the working slopes,

simultaneously. There is no "rule of thumb", the problems of each mine being approached individually.

As it is impossible to dissociate these three questions, the discussion that follows will not contain separate answers to each one of them. The comments in sub-items **a** to **d**, below, are based on a lecture by Savage at the Royal School of Mines in May, 1986. A constant annual extraction of ore is assumed herein.

Stripping rates and working slopes(see figure 10.1):

a) The simplest choice is to adopt a constant flat working angle. Depending on the ore body shape as related to the ultimate pit design, an annual stripping volume declining with depth, that is to say, with time, might result.

b) Another option is to select a convenient working angle, in general, flatter than the ultimate pit slopes, in order to have a waste to ore ratio increasing with depth, the same occurring with the annual stripping rates. The working slope is to be kept the same for all sequential pushbacks.

c) A third hypothesis is to chose a variable working slope, gradually approaching the slopes of the ultimate pit, as the pushbacks are getting deeper and deeper. By doing this, the annual stripping requirements are likely to be roughly the same throughout the mine life, resulting in a constant waste to ore ratio.

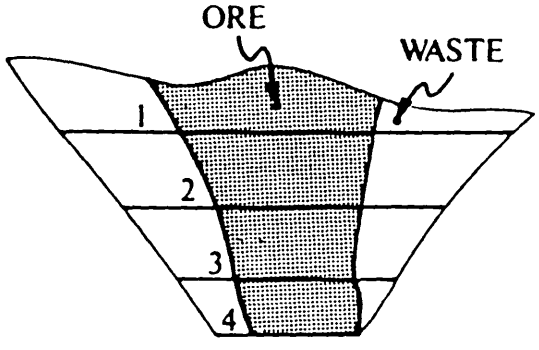
d) A fourth method is the phased mining stripping sequence, which aims at having bigger profits when they are most needed, i.e., at the beginning of any mining operation, when the project is in its "payback" period and the risk exposure is maximum, in financial terms. This is achieved by deferring the stripping requirements as much as possible. The annual extraction of waste will increase up to a maximum, approximately at half the mine's life, decreasing afterwards. The feasibility of this approach is subject to the particular conformation of the deposit, which not always admits this solution.

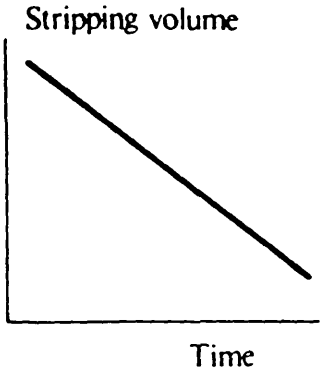
In reality, these approaches might be mixed together, as a function of ore body shape and dip.

Mathieson (1982) lists some preliminary judgements that must be made before the design work is initiated: probable maximum ore and waste mining rate which will be required in a given phase; the size and type of equipment to be used and hence the required minimum operating bench width necessary; and an appropriate inter-ramp slope angle.

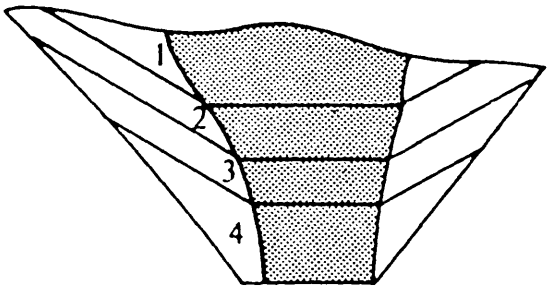
And he continues: "Using these constraints, the mining engineer then proceeds to design a series of phases in some detail, complete with haul roads, (...). He ensures that

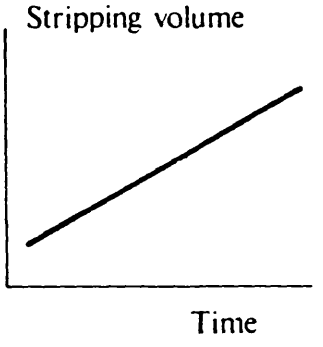
a) Declining stripping ratio



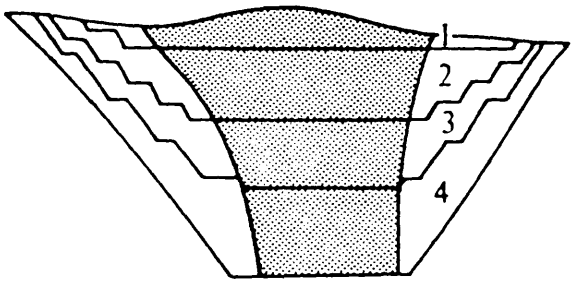


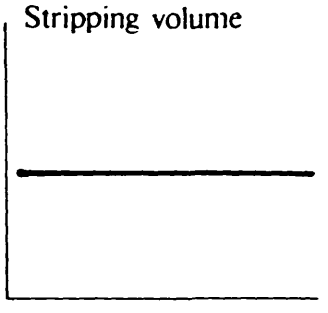
b) Increasing stripping ratio



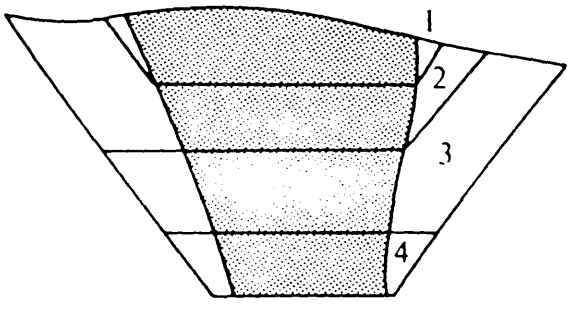


c) Constant stripping ratio





d) Phased mine stripping sequence



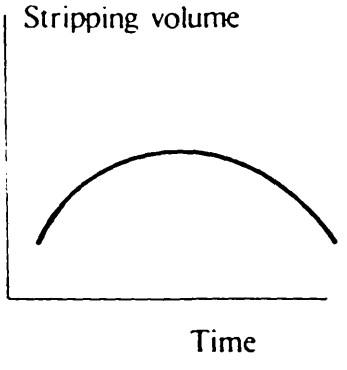


Figure 10.1 Stripping volumes and working slopes

each bench, from one phase to the next, is mineable, and flat ramp access to each active bench is provided. He does not, in this exercise, need to be conscious of, nor constrained by the amount of ore encompassed by each phase, however. In other words, he does not have to juggle phase limits at this stage to include a certain quantity or number of years of ore supply. The variable ore and waste quantities by phase will be subsequently scheduled."

(...)

"During the mine life, typically three or more phases are active at any one time: one perhaps nearing exhaustion on ore supply, another beginning to be mined for ore, and another one or two being stripped."

10.2.4 Size of the pushbacks

Milner (1977) wrote a paper on long-range planning of an open-cast copper mine:

"We know from experience that a pit containing a two year's supply of ore is the minimum size of stage pit that can be mined efficiently. For Gibraltar, this minimum size is approximately thirty million tons. If a stage-2 pit is 200-foot pushback completely surrounding a stage-1 pit, then a two-year pit is too small and a larger stage-2 will have to be optimized. This is an area where good engineering judgement is required."

The article by Blackwell (1979), on evaluation and planning of an open-pit copper mine as well, contains the following remarks :

"Using the open-pit computer program, the material within the final pit limits was divided into increments of a specified size, corresponding to the material to be mined in successive time increments. Generally speaking, the early increments were made to correspond to a year's production; the later increments were much larger."

In the paper by Mathieson(1982), five pushbacks are mentioned. Apart from the preproduction stage, in the subsequent phases the ore tonnages gradually grow from 6% of the total ore up to almost 30%. The percentages of waste tonnages referred to total waste increase approximately at the same pace. However, as the total tonnages of ore and waste are different, the waste to ore ratio curve is U-shaped, and goes down to a minimum around half the mine's life. The last pushback to be mined is the biggest one, with ore and waste tonnages amounting to nearly 400 million tons, the ore life being 5.2 years and the waste to ore ratio, 2.0 t/t for that phase.

10.2.5 Suggested solution for Mutuca Mine

Mutuca has been mined for over 25 years now, therefore its situation is totally different from a completely new mining project. For instance, no pre-production stripping is needed, as ore is already exposed at the bench being mined. In addition to that, a substantial amount of ore will be available, virtually with no waste removal requirements, as soon as the present beneficiation plant is removed from its present location, on top of the ore body (see chapter 2).

An interactive process is needed for the establishment of working pit sequences. With so many factors affecting the outcome, perhaps the trial-and-error approach is the more obvious route.

The principal guidelines used were:

- Ore tonnages evenly balanced between phases;
- Pushback ore life around 5 years' production;
- Stripping requirements as constant as possible, that is to say, peaks in the striping rates should be avoided; and
- Constant working slopes.

The first tentative establishment of working pit sequences for Mutuca was made with an early version of pit 1080, which had been run with the slopes shown in figure 9.1 as input data. Three pushbacks were considered, namely from the top levels to elevation 1200m, from 1200m to 1120m and from 1120m down to 1080m. The results obtained are shown in table 10.1. Waste tonnages are much less than those displayed in table 9.5, because in that preliminary version(shown in table 10.1) the slopes were not flattened to account for the in-pit road.

The analysis of the distribution of ore and waste tonnages between the phases revealed that the intermediate stage (1200m - 1020m) had too much material, whereas the last stage (1120m - 1080m) contained too little. As a consequence of that, the elevation 1140m was selected as the lowest bench of the intermediate pushback, aiming at balancing the three stages.

More recently, the slopes for the ultimate pit have been changed(see figure 9.2) and the pits, redesigned. However, this subdivision has remained acceptable for the working pit sequences.

(in thousand tons)

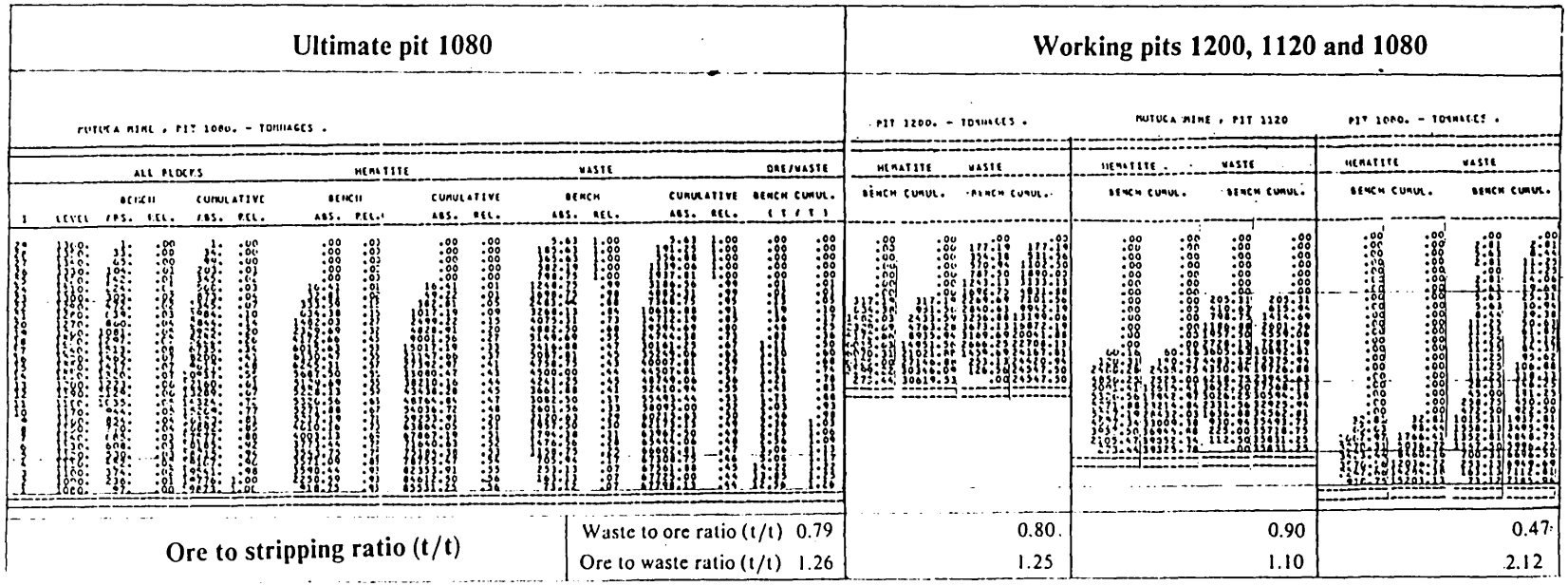


Table 10.1 Working pit sequences for an early version of pit 1080

10.3 Input data

The working pits are no more than subsets of an ultimate pit, hence most of what has been said in the previous chapter applies here, as far as input data are concerned.

10.3.1 Working slopes

A constant value of 23 degrees was adopted for the working slopes, corresponding approximately to half the intermediate slope of the ultimate pit, which has slopes ranging from 43 degrees, on the east wall, to 52 degrees, on the west wall. Figure 10.17 shows the basis for the establishment of the working pit slope.

10.3.2 Lowest elevation for each working pit

The lowest level used for designing the first working pit was 1200m; for the second pushback, elevation 1140m was adopted; and for the third and last stage, the bottom of the ultimate pit was the only choice(1020m, 1000m or 980m, in this case).

10.3.3 Bench height

It is the same as the bench height adopted for designing the pit shells, 20m. However, the actual bench height at the mine site is 10m (see item 9.2.6).

10.4 Brief description of the procedure adopted

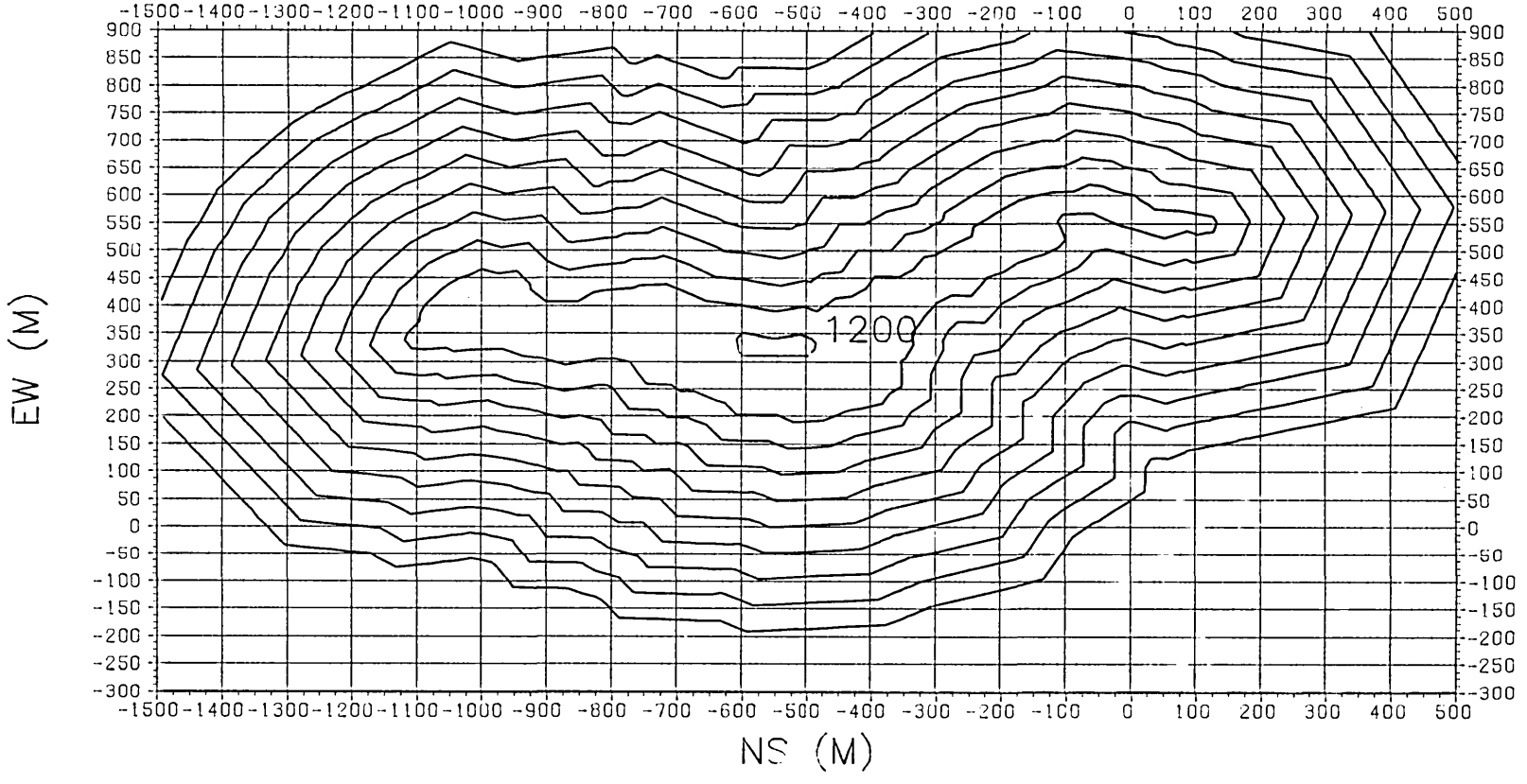
In order to understand the procedure that was adopted, one should bear in mind the existence of a sequence of surfaces, one lying below the other, as follows:

- topographic map;
- working pit 1200;
- working pit 1140; and
- ultimate pit (1120, 1000, or 980).

First, the bottom surface of working pit 1200 (see figure 10.2) was generated, it was merged with the topographic map, and the blocks lying within this working pit were matched with blocks contained by the optional ultimate pit, whatever it was (1020,1000, or 980). An inventory of blocks resulted, as well as a bottom surface for working pit 1200.

The data files with the coordinates of blocks lying within the optional pit shells and the topographical boundary had already been created, indicating whether each block contained ore or waste.

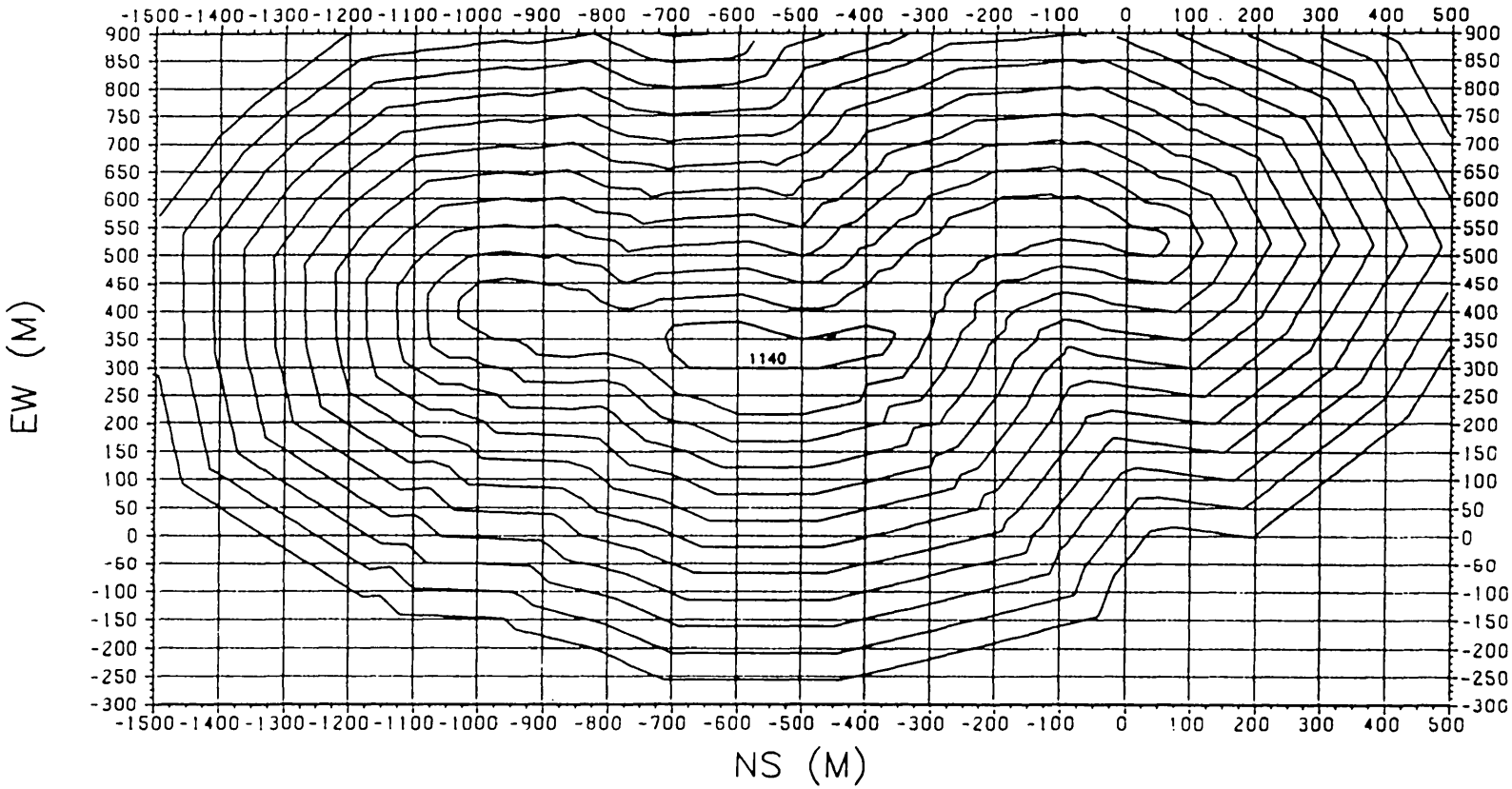
Subsequently, the pit shell of working pit 1140 was generated(see figure 10.3). This time, the pit surface was merged with the surface which had resulted from the previous merging of the ground surface and the pit shell 1200. The outcome of this intermediate



MUTUCA MINE WORKING PIT 1200

WORKING SLOPE 23 DEGREES

Figure 10.2 Working pit 1200



MUTUCA MINE WORKING PIT 1140

WORKING SLOPE 23 DEGREES

BENCH HEIGHT 20 METRES

Figure 10.3 Working pit 1140

stage was a second inventory of blocks, and a second bottom surface, this turn for working pit 1140.

This procedure could be repeated for any number of intermediate working pits, that is to say, the bottom surface of any working pit could be taken as the top surface of the following pushback further down.

Both the lowest level and the bottom surface of the last working pit are coincident with those of the ultimate pit. The resulting inventory of blocks was checked against previous partial inventories, so that no blocks were counted more than once.

The precision of this procedure could be improved, in the future, as no block fractions are taken into account(see item 9.3.3). However, it gives a reasonable idea on the order of magnitude of the tonnages contained in each pushback .

10.5 Numerical results - quantitative analysis

Table 10.2 features the working pit sequences for pit 1000 in detail, and tables 10.3, 10.4 and 10.5 show the summarized version of working pit sequences for pits 1020, 1000 and 980, respectively. Only the optional pits with switchbacks on the east wall were analysed.

Table 10.3 Working pit sequences for pit 1020-summarized
road on the east wall only

Working Pit		Total Tonnage (in million tons)		Waste to Ore Ratio (t/t)	
from	to	Ore	Waste	from-to	cumulative
top	1200	30.5	23.3	0.76	0.76
1200	1140	32.4	35.2	1.09	0.93
1140	1020	29.7	24.2	0.81	0.89
top	1020	92.6	82.7	0.89	0.89

(in thousand tons)

Ultimate pit 1000		Working pits 1200, 1140 and 1000		
		PIT 1200. - TONNAGES	MUTUCA MINE . PIT 1140. - TONNAGES .	PIT 1000. - TONNAGES
		HEMATITE WASTE	HEMATITE WASTE	HEMATITE WASTE
		BENCH CUMUL BENCH CUMUL	BENCH CUMUL BENCH CUMUL	BENCH CUMUL. BENCH CUMUL.
Ore to waste ratio (t/t)	Waste to ore ratio (t/t) 1.00	0.80	1.16	1.03
	Ore to waste ratio (t/t) 1.00	1.25	0.86	0.97

Table 10.2 Working pit sequences for pit 1000 - detailed

Table 10.4 Working pit sequences for pit 1000-summarized road on the east wall only

Working Pit from	to	Total Tonnage (in million tons) Ore	Waste	Waste to Ore Ratio (t/t) from-to	cumulative
top	1200	30.5	24.5	0.80	0.80
1200	1140	32.4	37.6	1.16	0.98
1140	1000	32.3	33.3	1.03	1.00
top	1000	95.2	95.4	1.00	1.00

Table 10.5 Working pit sequences for pit 980-summarized road on the east wall only

Working Pit from	to	Total Tonnage (in million tons) Ore	Waste	Waste to Ore Ratio (t/t) from-to	cumulative
top	1200	30.4	23.8	0.78	0.78
1200	1140	32.4	34.5	1.06	0.92
1140	980	32.6	40.7	1.25	1.03
top	980	95.4	99.0	1.03	1.03

Tables 10.6, 10.7 and 10.8 give additional information on the working pit sequences for pit 1020. One can see how well balanced are the three suggested phases, and that the waste to stripping proportion of each phase is not very different from the ultimate pit's stripping ratio.

Table 10.6 Percentage of total tonnage pushbacks for pit 1020-road on the east wall only

Working Pit from	to	Ore	Waste
top	1200	33	28
1200	1140	35	43
1140	1020	32	29

Table 10.7 Production rates (approximate values)

pushbacks for pit 1020-road on the east wall only

Working Pit		Total Tonnage (in million tons)		Rate (Mtpy)		Life (Years)	
from	to	Ore	Waste	Ore	Waste	Ore	Waste
top	1200	30.5	23.3	6.5	6.5	4.7	3.6
1200	1140	32.4	35.2	6.5	6.5	5.0	5.4
1140	1020	29.7	24.2	6.5	6.5	4.6	3.7
top	1020	92.6	82.7	6.5	6.5	14.2	12.7

Table 10.8 Summary of phase tonnages(in million tons) and lifes pushbacks for pit 1020-road on the east wall only

Working Pit from	to	Waste above first ore bench	Waste on ore benches	Ore	Ore life (years)	Cumulative ore life (years)
top	1200	-	23.3	30.5	4.7	4.7
1200	1140	5.6	29.6	32.4	5.0	9.7
1140	1020	13.3	10.9	29.7	4.6	14.2
TOTAL		18.9	63.8	92.6	14.2	14.2

Although a constant working slope was adopted, the resulting waste to ore ratio is almost constant, for all pushbacks, apparently contradicting what has been said in item 10.2.3 (sub-item "stripping rates and working slopes"-b). Nevertheless, the constant working slope is observed only for the generation of the working pit shells. When these surfaces are merged with the preceding top surfaces, two different slopes might result: the working slopes are only observed as far as the ultimate pit boundaries are not reached, the ultimate pit slopes predominating from then on. Consequently, the situation is more or less as follows:

a) shallowest working pit: virtually only the working slopes are observed;

b) intermediate phase: the slopes of both working pit and ultimate pit occur; and

c) last pushback: the ultimate pit bottom surface is entirely predominant.

The resulting situation is such that it is almost as if a variable working slope were adopted, gradually approaching the ultimate pit slopes. Once a nearly constant stripping rate results, this case is more similar to that described in item 10.2.3, sub-item "stripping rates and working slopes"-c.

10.6 Graphical outputs

10.6.1 Cross-sections

In this thesis, working pits will be referred to as follows: lowest elevation of the working pit/lowest elevation of the ultimate pit. For example, the three working pits for the ultimate pit 1000 are designated: 1200/1000, 1140/1000 and 1000/1000.

Figures 10.4 and 10.5 are the cross sections NS−500 of two different working pits, 1200 and 1000, of the same ultimate pit 1000. These pushbacks are respectively designated by 1200/1000 and 1000/1000. One can see the bottom outline following a very flat working angle, in figure 10.4, and the same applies for the top outline of figure 10.5. The lowest level in figure 10.4 is 1200m, whereas the highest level in figure 10.5, around the coordinate EW−300, is 1120m. The intermediate working pit 1140/1000, which has not been shown, is encompassed by the bottom of the working pit 1200/1000 and the top surface of the working pit 1000/1000.

For the sake of comparison with figure 10.5, the working pit 980/980 is displayed in figure 10.6. One can see that the extreme EW coordinates in the former figure are approximately −540 and −40, whereas in the latter they are, respectively, −580 and 0, a wider range. Also it should be noticed, in those figures, that although the top outline follows the working slope, 23 degrees, the bottom outline features the ultimate pit slopes.

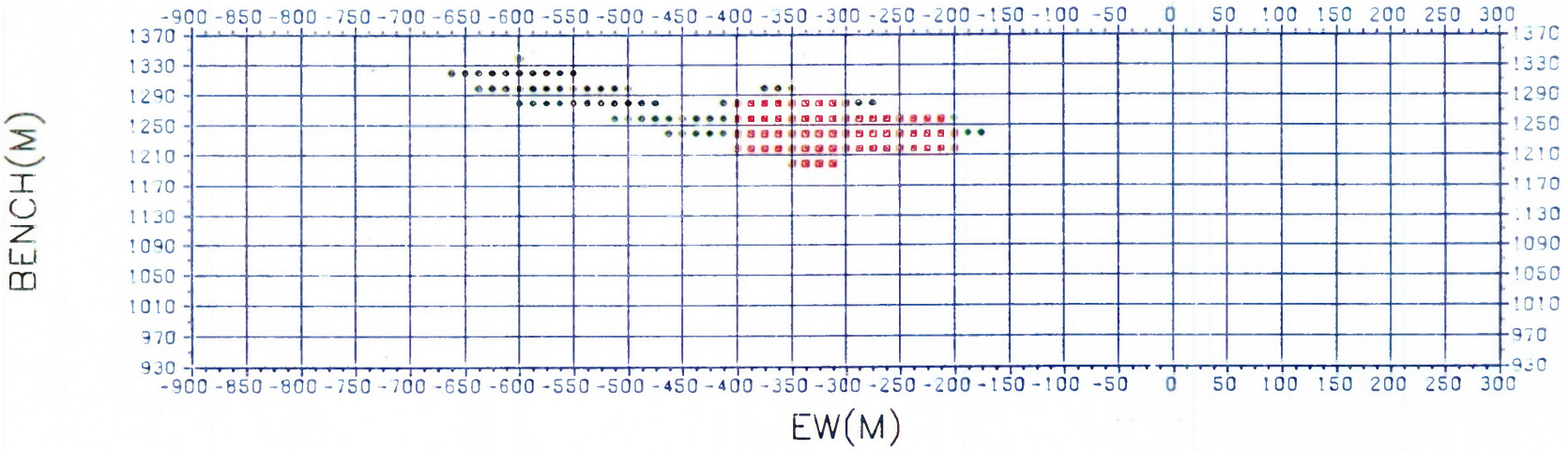
10.6.2 Longitudinal sections

Figures 10.7 and 10.8 are two examples of longitudinal sections of working pits. It is worth observing that waste blocks are largely outnumbered by ore blocks, as it is likely to happen when the concept of working pits is adopted. Again, the intermediate working pit(1200m - 1140m) is implied by the bottom outline of figure 10.6 and the top outline of figure 10.8.

10.6.3 Ore and waste tonnage curves

The numerical results which have been shown in table 10.2 can be visualised with the aid of figures 10.9, 10.10 and 10.11. For the sake of comparison, figure 10.12 shows the corresponding graph for the ultimate pit 1020. One can see how each working pit reproduces the patterns of the ultimate pit's ore and waste tonnage curves. These graphs are quite useful for scheduling the mining operations, which is the object of the folowing item.

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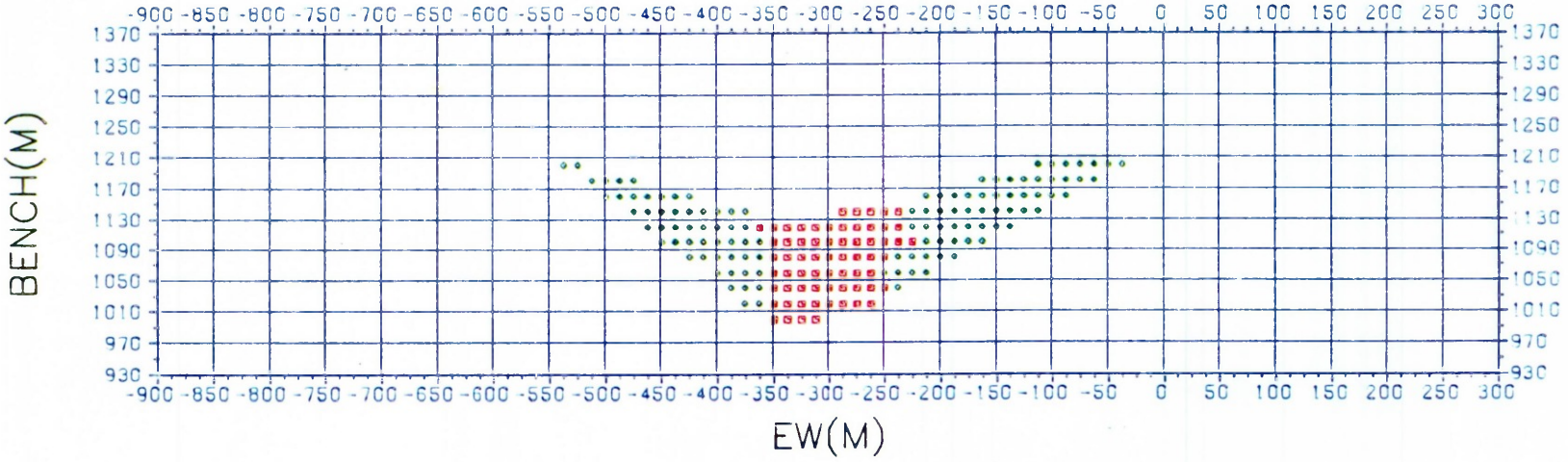


MUTUCA MINE – WORKING PIT 1200/1000

CROSS–SECTION NS –500

Figure 10.4 Cross-section NS – 500 - working pit 1200/1000

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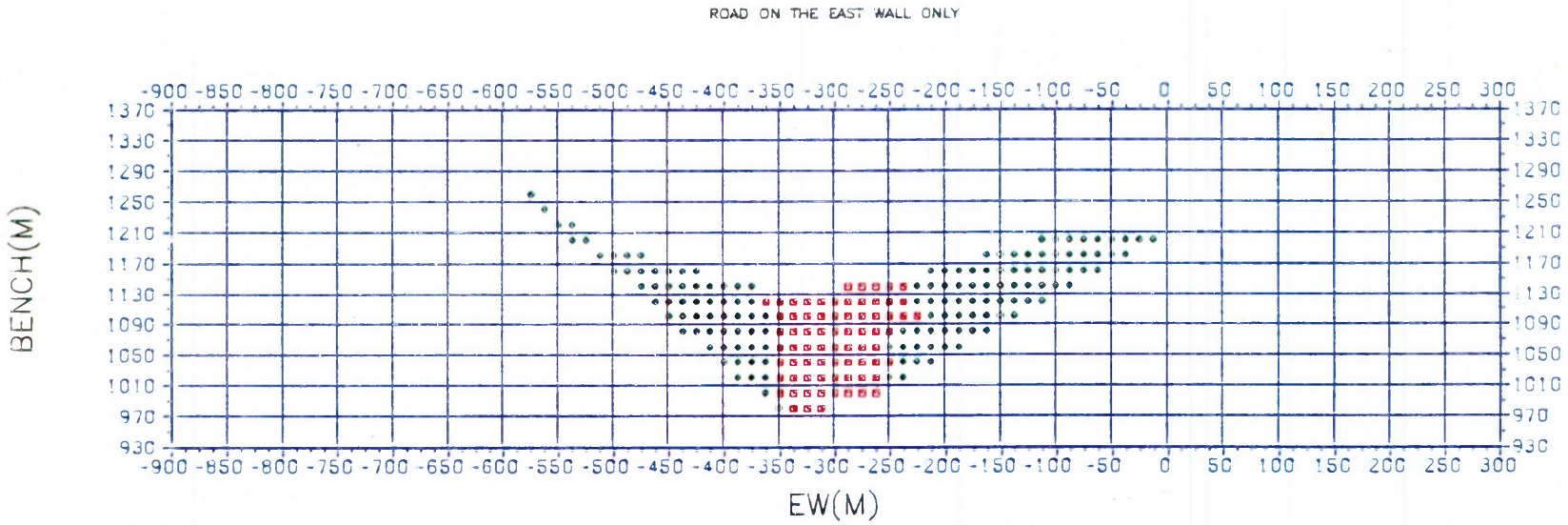


MUTUCA MINE – WORKING PIT 1000/1000

CROSS–SECTION NS –500

Figure 10.5 Cross-section NS – 500 - working pit 1000/1000

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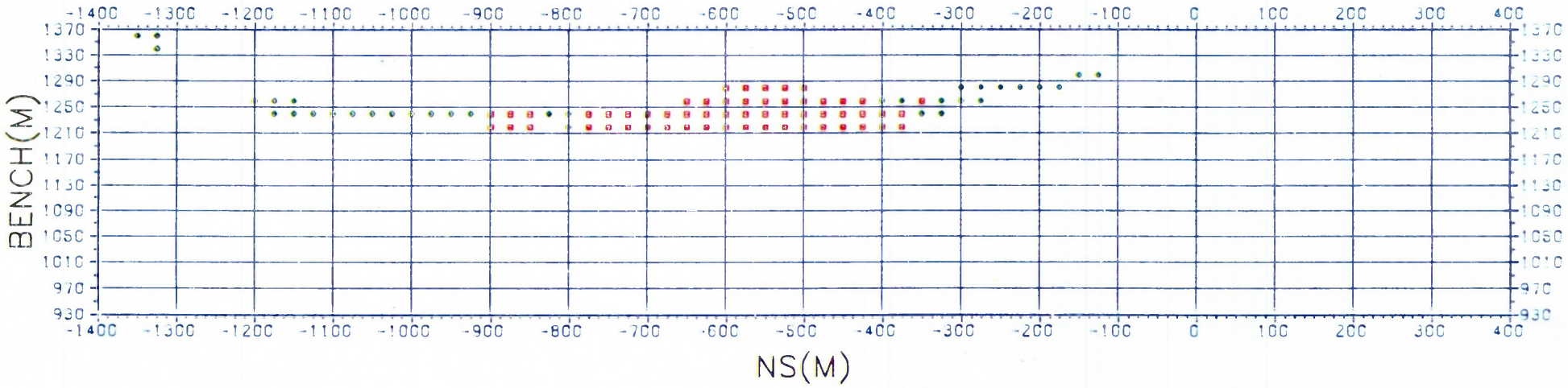


MUTUCA MINE – WORKING PIT 981/981

CROSS–SECTION NS –500

Figure 10.6 Cross-section NS – 500 - working pit 980/980

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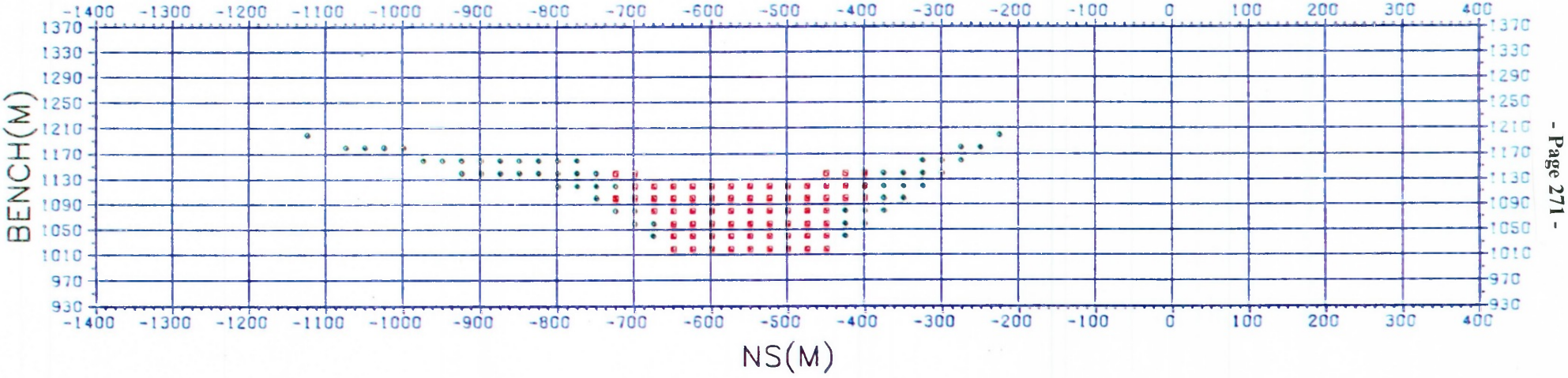


MUTUCA MINE – WORKING PIT 1200/1000

LONGITUDINAL SECTION EW 300

Figure 10.7 Longitudinal section EW 300 - working pit 1200/1000

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MUTUCA MINE – WORKING PIT 1000/1000

LONGITUDINAL SECTION EW 300

Figure 10.8 Longitudinal section EW 300 - working pit 1000/1000

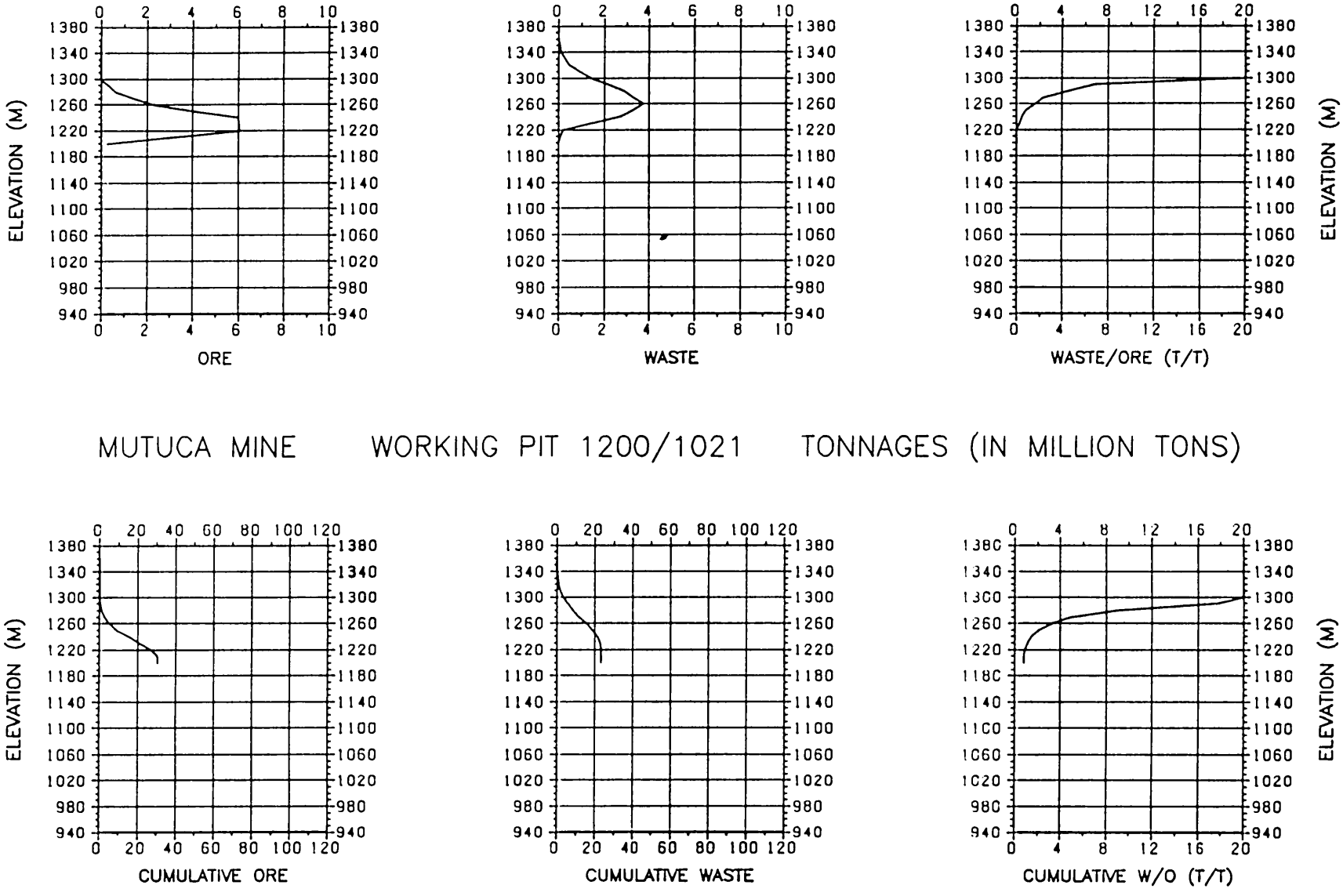


Figure 10.9 Ore and waste tonnage curves - working pit 1200/1020

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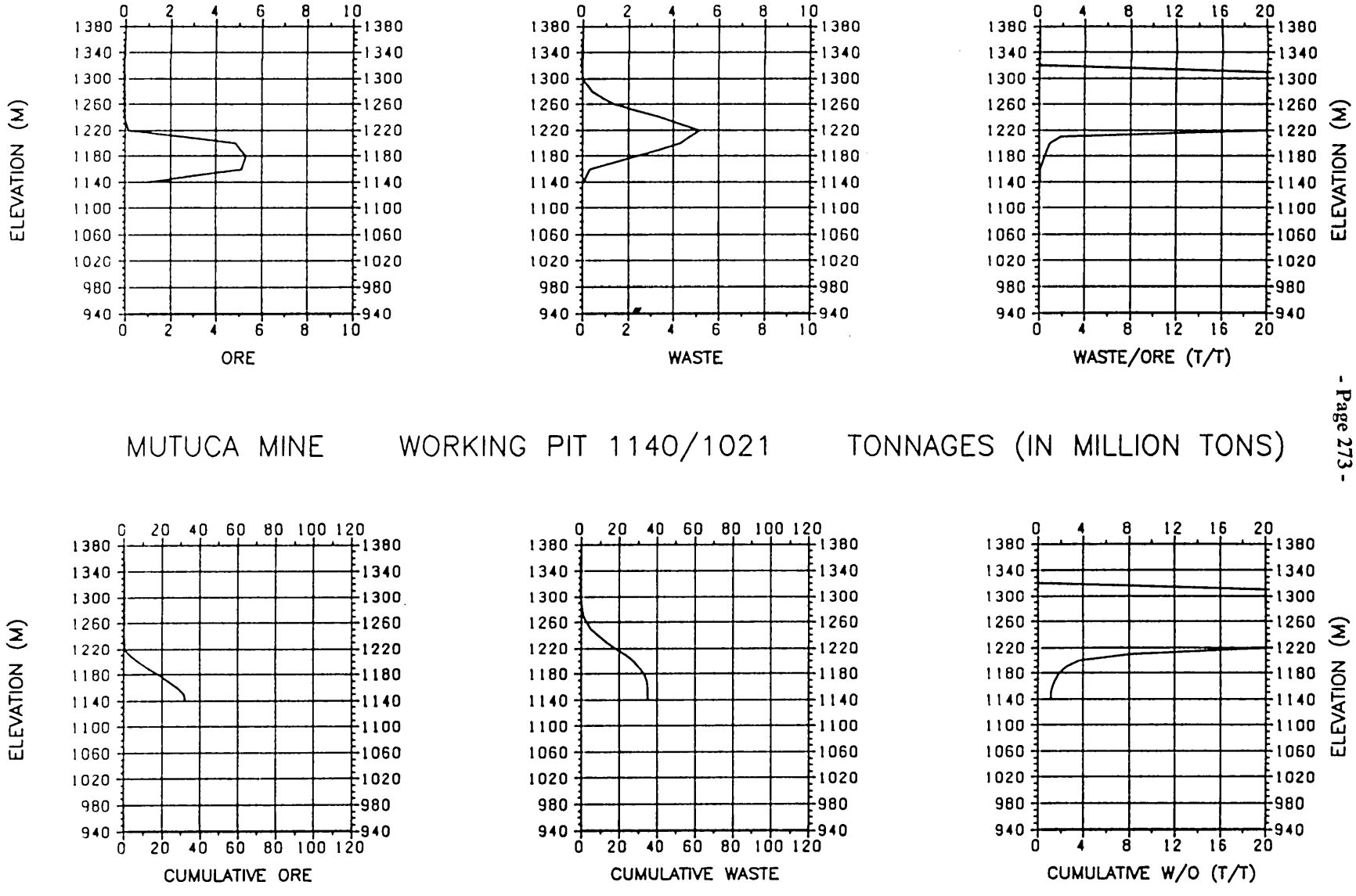


Figure 10.10 Ore and waste tonnage curves - working pit 1140/1020

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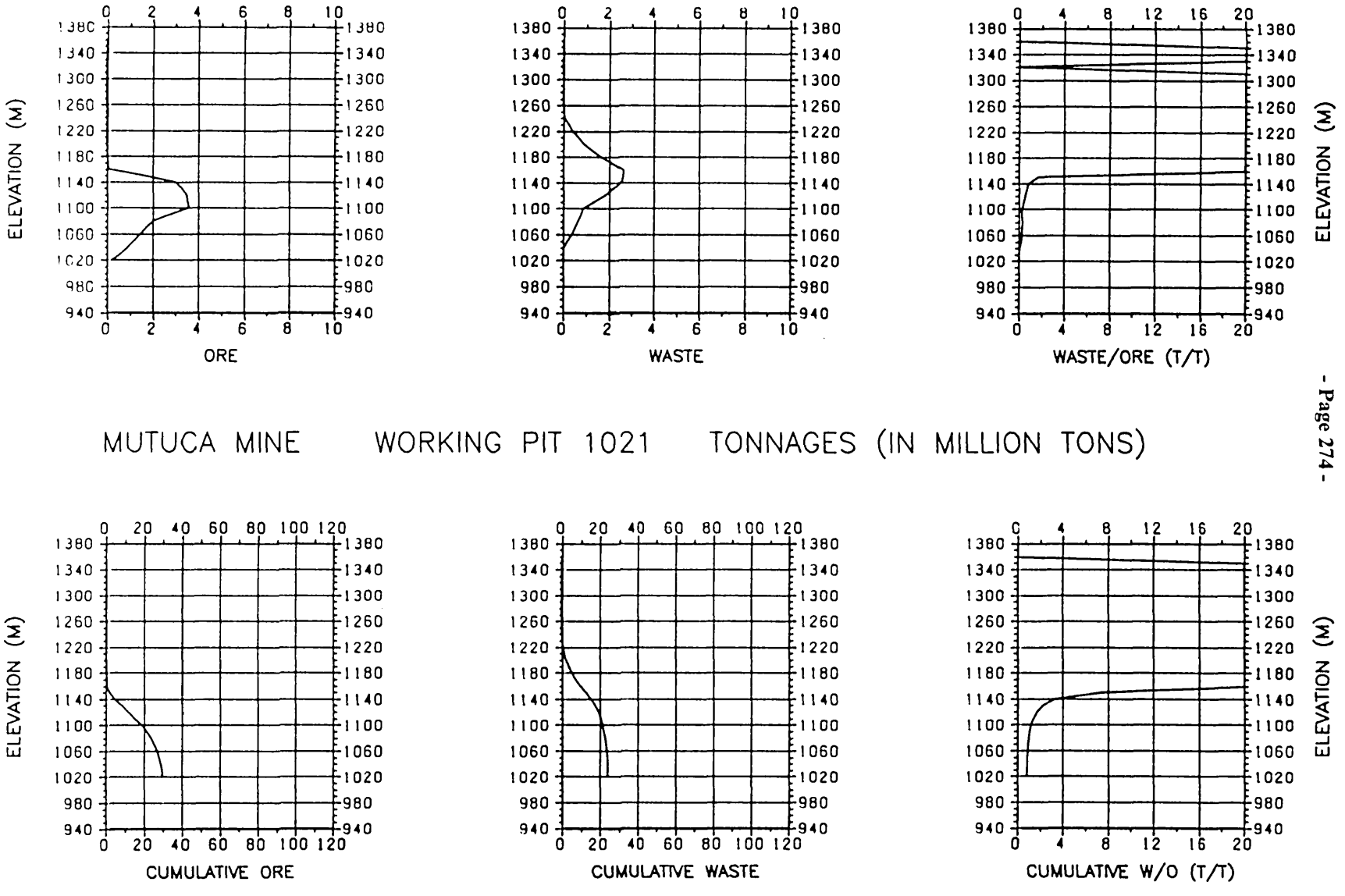


Figure 10.11 Ore and waste tonnage curves - working pit 1020/1020

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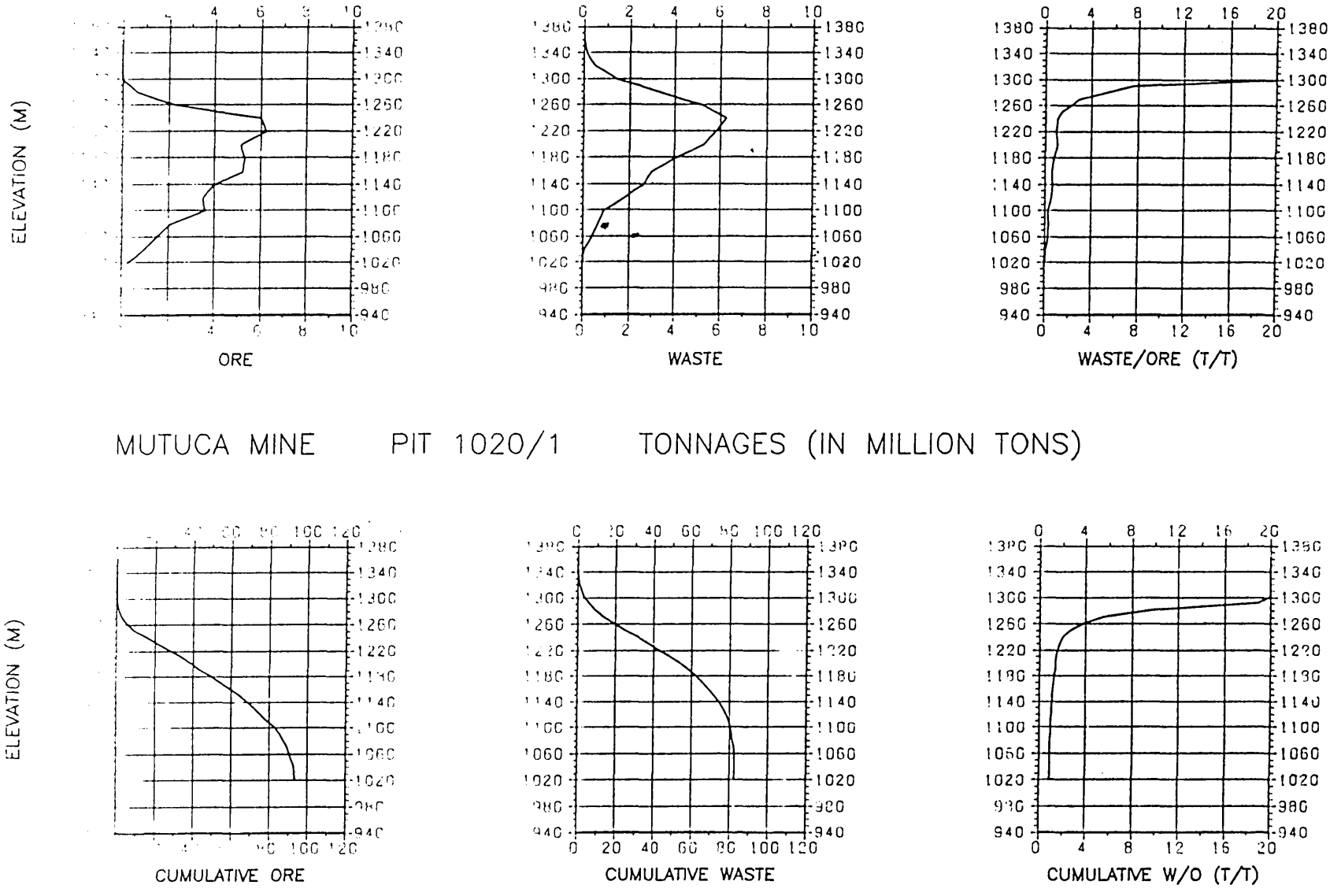


Figure 10.12 Ore and waste tonnage curves - ultimate pit 1020

10.7 Simplified pit scheduling exercise

A straightforward pit scheduling exercise was made with the results of the phasing of pits 1020, 1000 and 980, which have been described in the precedent items.

The ore extraction rate being fixed at 6.5 Mtpy by the market demand, the annual stripping rate is the variable under control. The planner has to select an adequate value for it, aiming at a reasonable time lag for the exposure of ore. At any one time, the development of ore should provide for a minimum ore supply to the beneficiation plant, which is a function of the stripping rate to be chosen.

The hypothesis of not hiring additional stripping equipment was adopted, therefore any peak in the stripping curve is undesirable, that is to say, the yearly rate of waste removal should be kept as constant as possible.

A suggested scheduling for the working pit sequences of pit 1020 is represented in figures 10.13 and 10.14. The stripping rate is constant. On the average, waste stripping is completed one year earlier than ore extraction, for the three pushbacks.

Figures 10.15 and 10.16 show exactly the same analysis applied to pit 980. At first, a variable stripping rate had been adopted, as the waste tonnages increased with the depth of sequential pushbacks. Afterwards, the stripping rate was smoothed out, giving a constant 7.1 Mtpy rate.

10.8 Incremental analysis - preliminary economic appraisal

It is worth mentioning what has been said by Savage(1986;RTZ Technical Services; personal communication): "It is as if one were evaluating a new project each time the pit boundaries are to be advanced". Yet, as metal prices vary, one should not take steps that block future possibilities. Mathieson shares this opinion(see topic 1.3.1 of this dissertation). already been included in the item 1.4.1 of this thesis, under "pit sequence planning and scheduling", hence it will not be repeated.

In order to preserve the analogy with chapter 9, a preliminary economic analysis will be made, using the break-even stripping ratio (BESR) concept, as described by Miller (1982).

The sequential pushbacks are essentially incremental pits, therefore their waste to ore ratios can be directly compared with the BESR, which is approximately 5.0 for Mutuca Mine (see chapter 11). As a consequence of that, all the aforementioned partial pit expansions are feasible, as their stripping ratios are well below the BESR (see tables 10.3,

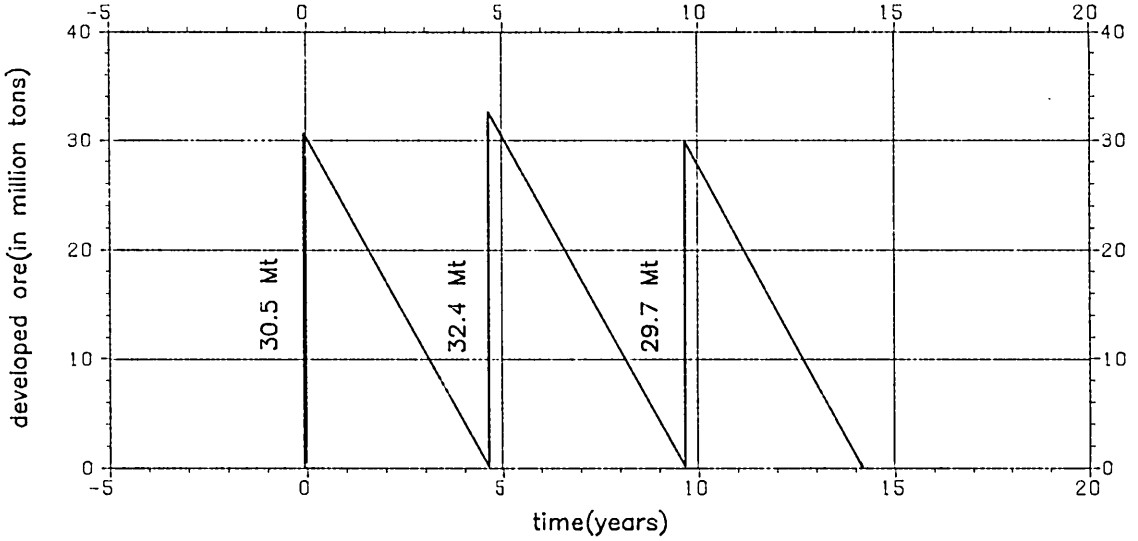


Figure 10.13

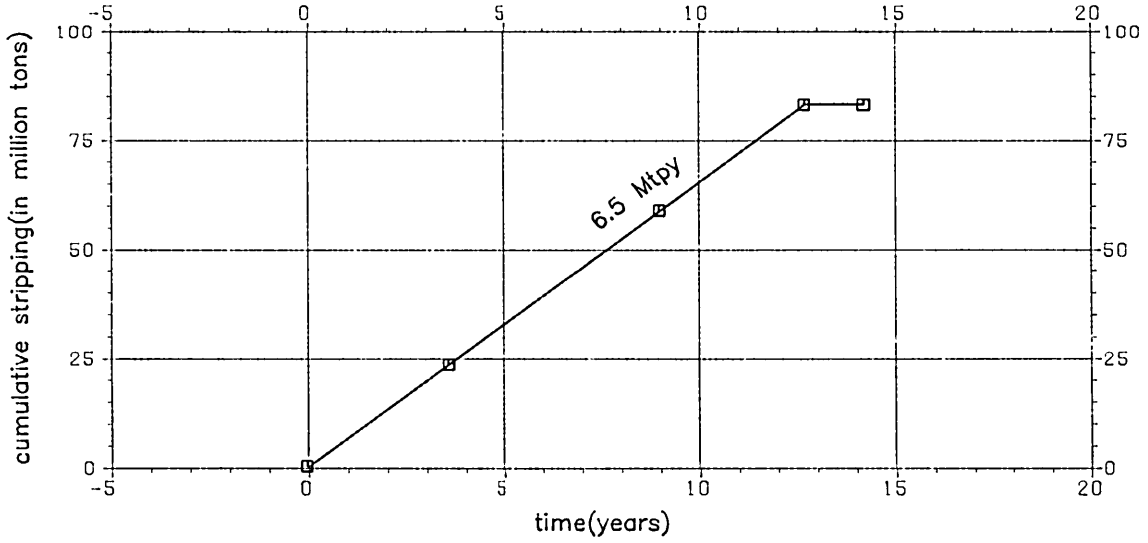


Figure 10.14

Working pit sequences for pit 1020

•Suggested scheduling

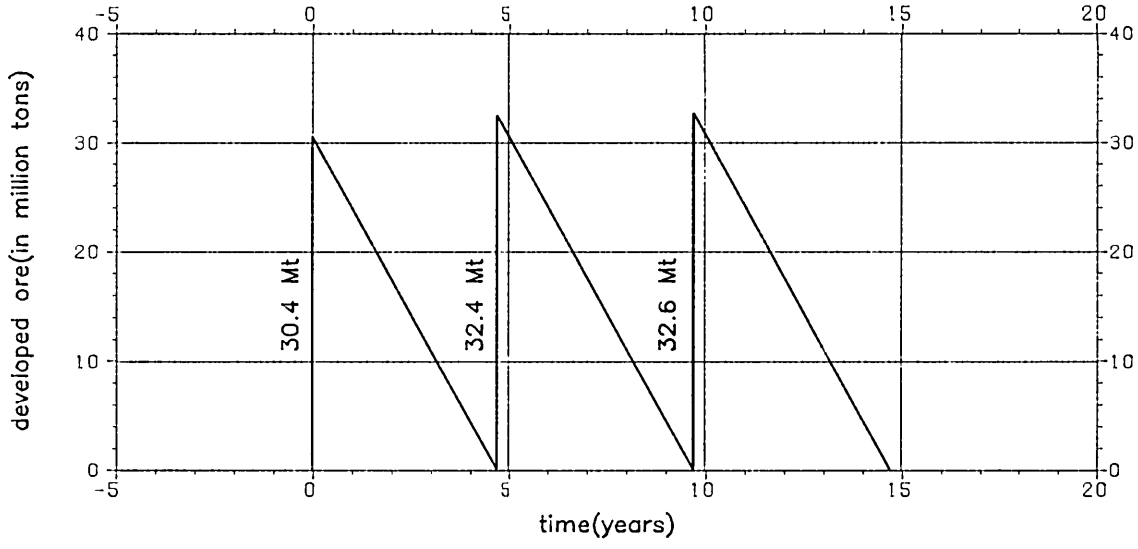


Figure 10.15

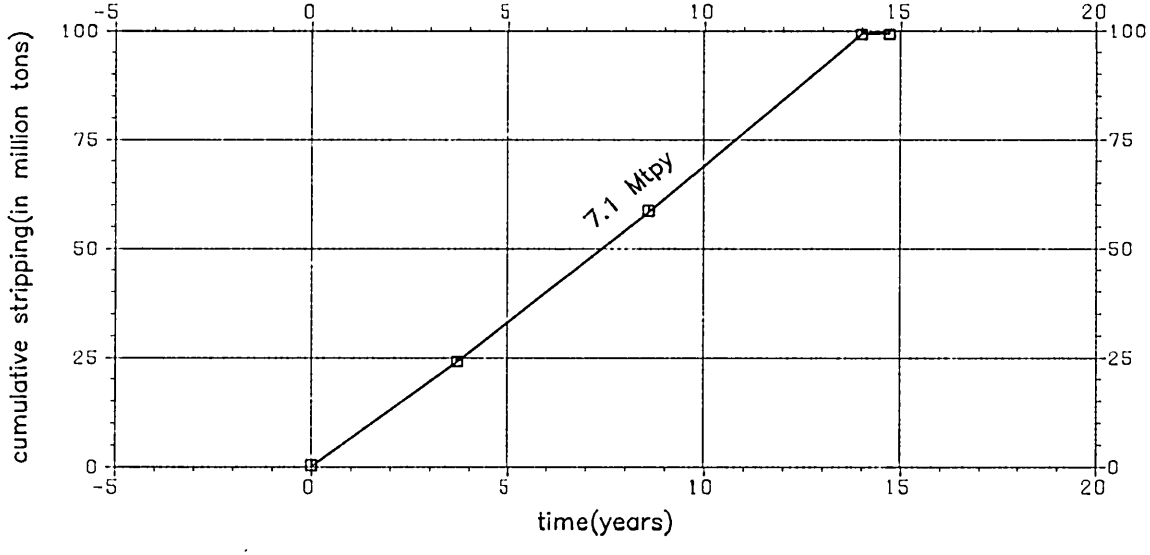


Figure 10.16

Working pit sequences for pit 980

Suggested scheduling

10.4 and 10.5)

A more comprehensive economic analysis, using the Net Present Value (NPV) Method, is presented in the following chapter.

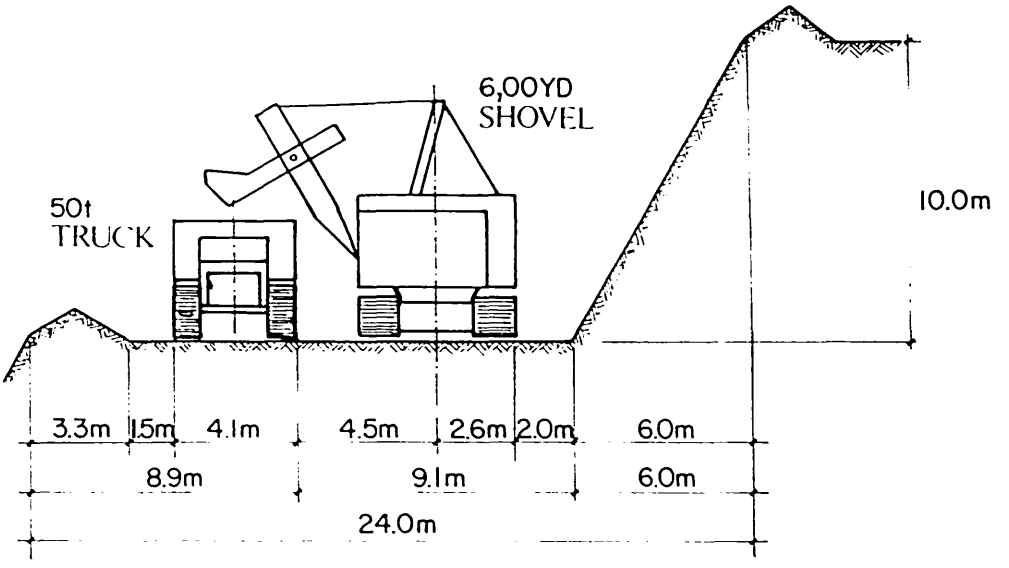


Figure 10.17 Criterion for fixing the working pit angle

Source SODERBERG,A. and RAUSCH,D.(1972) Pit Planning and layout. In: Surface Mining, Pfeider. New York, AIME, pp. 141-66.

11. Economic analysis

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11.1 Summary and conclusions

This chapter starts with a brief discussion of the methods of investment analysis, in general. The economic appraisal of mineral ventures, in particular, is examined afterwards, and the approach adopted for Mutuca Mine is presented. Subsequently, the input data needed for the economic analysis are mentioned. After that, the procedure for cash flow determination is described and the potential variability of certain key variables is assessed in economic terms. Finally, a series of possible improvements to the aforesaid analysis are discussed.

The main conclusions of this chapter are:

- it is worth exploiting the additional reserve which has been discovered recently;
- the former "ultimate" pit 1080 should be abandoned in favour of deeper options;
- the mining exploitation could go down to levels 1020m or 1000m, depending on the trade-off between mining costs and ore prices;
- these conclusions should be re-assessed whenever a significant change occurs in any key variable of the problem; and
- once the mineral inventory offers the possibility of having a more detailed economic analysis, in the future, this hypothesis should be examined.

11.2 Introduction

In the following discussion, the optional methods for analysing investment alternatives will be presented. Whenever needed, the reader is referred to textbooks on basic Investment Analysis. Only a few mathematical expressions are included in this chapter.

11.2.1 Methods of investment analysis

There are three basic methods of investment analysis, namely, Internal Rate of Return(IRR), Equivalent Annual Cost(EAC), and Net Present Value(NPV). The social cost/benefit analysis will not be considered in this text because it is applicable usually to governmental projects(Grant,1964).

The payback period is an index that might be very important for the appraisal of high-risk ventures with uncertain future returns. However, it has its weaknesses as a primary criterion for investment decisions(see Grant, 1970, page 528), hence no further comments will be made on it.

Traditional methods: before going into further details on the three basic methods of investment analysis, a brief comment on methods traditionally used in the mining industry should be given to the reader.

Hoover(1909) mentioned that "before the yield on a mine investment can be determined, a portion of the annual earnings must be set aside in such a manner that when the mine is exhausted the original investment will have been restored. (...) If they be invested in safe securities at about the usual rate of 4%" (...).

And he continued : "Mining business is one where 7% above provision for capital return is an absolute minimum demanded by the risk inherent in mines, even when the profit in sight gives warranty to the return of capital."

For its traditional use in mining enterprises, Hoskold's method will be briefly commented. The remarks made by Grant(1964, page 125) are summarized below:

> "In this method [Hoskold's] it is assumed that uniform annual deposits will be made into a conservatively invested sinking fund that will earn interest at a relatively low rate. The annual deposits are to be just sufficient to replace the original investment at the end of the life of the property. The rate of return is computed by dividing the investment into the annual amount remaining after setting aside the sinking fund deposit".

Grant (1964) assumes 4% interest on the sinking fund, and his example arrives at a 15.2% rate of return, using Hoskold's method:

> "If the same proposal is analysed by correct compound interest methods, the computed rate of return is 20% per year(...)"
>
> "The important point to recognise (...) is that this 15.2% return is the result of two separate investments, one yielding 20% and the other yielding only 4%. Presumably the decision to make the investment with the 20% yield does not require that there also be a decision to make the 4% investment. If not, the 4% investment has no relevance in making the decision about the proposed investment with the 20% yield. It is the 20% figure, not the 15.2% one, that should be used as the index of attractiveness of the proposed investment in the mining property."
>
> "This Hoskold-type viewpoint on computing rate of return is rarely, if ever, appropriate as a basis for decision making on proposed investments. Hoskold seems particularly indefensible in the common case where it is used when no actual sinking fund is comtemplated(...)"

The **Net Present Value (NPV)** Method discounts all cash flows, given a discount rate, to a certain reference date, normally at the start of the project. Although not as usual as the other approach, the reference period of time can be in the future. The essential aspect is that different investment alternatives should be compared at the same point in time. The key variable of this method is the interest rate that should be considered. The rentability offered in the market for the sum involved, as compared with the cost of capital, is the answer.

Groundwater(1967) mentioned that the NPV is one of the important determinants in evaluating a business for purchase or sale.

Taylor(1985) dicusses the shortcomings of the NPV approach under conditions of active operation. He asks the question on whether economic interest is really best served by maximizing a partly hypothetical "present" value that speedily becomes "past" and lapses into irrelevance. Tiny differences in the NPV of cash flows representing different policies extending far into the future can result, because of the insensitivity of this method to time-distant ore. Practical complications of grade uncertainties and metal price fluctuations are generally ignored. It is not difficult to devise different cut-off and output policies that yield identical and plausibly optimum present values, which vary only in their rates of decline of value.

These topics are very important, and perhaps the best approach to them, in the context of this research work, would be to consider each pit expansion as if it were a new project(see item 10.8). On the other hand, one of the conclusions of this chapter is that an economic re-assessment of the optional pits should be made whenever a key variable of the problem is substantially altered. In addition to that, it is suggested the future use of both Monte Carlo simulation of cash flow and risk analysis.

For the specific case of Mutuca Mine, an interest rate of 8% p.a.(**per annum**), in real terms, was used. The study was extended to assess the sensitivity of the NPV to changes in the interest rate, hence rates of 6% and 10% p.a. were adopted, again in real terms. No allowances were made for inflation, in this work.

The **Equivalent Annual Cost (EAC)** Method transforms a non-uniform series of payments into a uniform series. The resulting cash flow might be negative as well as positive. However, the name of the method refers to **cost**, perhaps because of its main characteristic of evaluation of service-producing alternatives, as opposed to income-producing alternatives, which are more usually assessed through NPV and IRR(Stermole,1979).

One subtlety in the application of this method is that, whenever the investment options have different project lives, the method requires a common "planning horizon", which means the alternatives should be repeated, each one a certain number of times, until a common multiple for their lives is reached.

The EAC method is used more oftenly for determining the economic life of equipment, in equipment replacement studies. For example, if two optional equipments are to be analysed by this approach, assuming that any equipment will have to be replaced by a similar one at the end of its economic life, the differences in estimated life are immaterial.

"In general, prospects for price increases and for extra costs incident to replacement are favourable to the selection of longer-lived alternatives; prospects for technological improvements, changes in service requirements, and price reductions are favourable to the selection of shorter-lived alternatives."(Grant, 1970, page 78).

The **Internal Rate of Return (IRR)** Method determines the interest rate that produces a null NPV, after discounting both positive and negative cash flows over a given period of time.

When assessing the rentability of a single investment option, the IRR approach might run into difficulties, if more than one change in sign occurs in the net cash flow. The standard mineral project has negative cash flows (investments - exploration expenditure plus development capital costs) at the pre-production phase, followed by a series of positive cash flows(return on investment - revenue minus production costs), thus presenting only one change in sign(see "Rate of return taxation of minerals", 1977, pages 9 and 16). However, in many cases, additional investments are required along the project life, e.g., for equipment or installation replacement, hence creating the aforementioned situation. For **n** years, the internal rate of return is the root of a polynomial of the n^{th} degree. According to the rule of sign, by Descartes, there are as many roots(solutions) to the polynomial equation as the number of alternations of sign of its terms(Fleischer,1973, page 41).

In addition to the above, if many alternatives are to be analysed, another difficulty arises: the internal rate of return **per se** is only a measure of rentability, but should not be used to decide between investment options, specially when they have different orders of magnitude. The correct analysis should compare the alternatives in pairs, starting from the two options with smaller investments and higher rates of return. The differential cash flow should be computed and the rate of return of the **incremental investment** should be compared with the minimum market rate, or minimum attractive rate of return, or rather with the cost of capital for the company concerned, depending on the type of business, the

scale and life of the project, and the risk involved. If the return on the incremental investment is higher than the minimum envisaged by the firm considered, it is worth investing in that hypothesis, otherwise, it is not. Subsequently, another pair of investment options is compared, and so on. The process stops when either no more alternatives are available or, when the options are not mutually exclusive, there is no more capital to be allocated.

The differential cash flows have to be calculated first, for this method to be applied, when dealing with more than one investment option. The possibility of obtaining multiple solutions for the internal rate of return equation can not be ruled out.

For all that has been said, caution is recommended for using the IRR Method. Nevertheless, when properly applied, it is as good as the other methods of investment decision.

In the standard economic analysis of the optional pits for Mutuca Mine, the IRR Method can not be applied, because the rate of return would be infinite, as no investments were considered, only operational costs and revenues, a positive cash flow resulting for all periods of time.

11.2.2 Economic appraisal of mineral ventures

With regard to the application of investment analysis to computer-based economic evaluation of ore reserves, the work presented at the APCOM II by Parks(1962) deserves mention, as one of the earliest papers in the field.

O'Brian(1969), in the APCOM VIII, gave a panoramic view of applications of investment analysis within mineral exploitation. The advent of the computer made possible the extensive use of sensitivity and risk analyses in the appraisal of mining projects, as these analyses require a great number of variables and intensive data handling.

Stermole(1979) made a general study of economic analysis of open-pit mines, examining the impact of escalation and inflation, income tax, borrowed money and the mining plan. He presented a case study of a copper mine.

Winkle(1979) discussed the cost records, and Davey(1979) presented some criteria for mineral block evaluation, both papers focusing open-cast mines.

Lane(1964) examined the relationship between a selected cut-off grade and the net present value of an open-pit mine. Both ore prices and mining costs were assumed stable throughout his analysis, which was made with the aid of a computer.

Plewman(1970) studied the basic economics of open-pit mining, assuming regular geometric shapes for the ore body. Prices and costs were considered to be constant.

Kim(1977) presented a computerized model for economic analysis of open-pit mines, in which both mining costs and ore prices are treated as stochastic variables. A risk analysis was made using the probability of instability for various slope angles.

Mutuca has been mined for more than 25 years and its particular problems have been well identified, the majority of the aforementioned publications being therefore either rather general or too specific for them to be applicable for that mine. Copper is by far the most conspicuous metal in the literature, publications on economic assessment of open-pit iron ore mines being much more rare.

11.2.3 Suggested solution for Mutuca Mine

The economic analysis that will be presented in the following topics has the following characteristics :

- it is assumed that MBR has already decided to undertake the Mutuca Mine expansion project;
- the annual rate of production of saleable ore has been fixed by MBR in approximately 6Mtpy, that is to say, the new beneficiation plant to be constructed will be fed at 6.5 Mtpy, given its recovery index. Therefore, the capital investment required for the new plant is constant for all optional pits being analysed;
- the economic analysis will be focused strictly in each optional pit **per se**, and not in the Mutuca Mine expansion project as a whole, which has been extensively studied by MBR personnel, and is beyond the scope of this thesis;
- the NPV Method will be used, a basic interest rate of 8% p.a. being adopted.
- subsequently, some variations will be introduced in key variables, such as ore prices, mining and stripping costs, and interest rates, in order to assess the sensitivity of the pits' NPV's to these changes.
- a brief comparison will be made between results obtained by way of three different economic criteria, namely, total profit, break-even stripping ratio and net present value(see table 11.7).
- although it is recognised that the deeper pits, which have substantially more waste to be removed, are likely to require additional mining equipment, e.g., trucks, no incremental investment has been considered in the analysis.

11.3 Input data

11.3.1 Block inventory of the optional pits

Five optional pits were considered for analysis: 1080, 1020, 1000, 980 and 960. For each option, the coordinates of blocks lying within the pit shell and the topographic boundary were used as input data. The blocks were divided into two categories only: ore and waste.

11.3.2 Plant recovery index

The recovery index for the new Mutuca Mine beneficiation plant is likely to be 91.5% (Source: Amaral; 1986; MBR, Planning Department; personal communication).

11.3.3 Production rate

MBR's projected sales are 5.95 Mtpy, therefore the ore dressing plant should be fed at a rate of 6.5 Mtpy, given the plant recovery index of 91.5% (Source: Cadenas; 1986; MBR, Planning Department; personal communication).

11.3.4 Stripping rates

The waste tonnages range from 67.2Mt, for pit 1080, to 102.0Mt, for pit 960 (see table 9.4). The selected stripping rates vary, respectively, from 5.6 Mtpy to 7.4 Mtpy.

11.3.5 Stripping costs

A stripping cost of US$1.20/t has been adopted by MBR for economic studies being made in Brazil, therefore it will also be used in this research work.

Nevertheless, the average stripping unit cost was US$ 1.28/t for the 1985 calendar year, corresponding to approximately 10% of the sum of other mining operational costs. Its standard deviation was US$ 0.71/t for the same period of time(Source: Mineraçõe Brasileiras Reunidas S.A.-MBR; Monthly Report, Mine cost sheet for Mutuca Mine; 1985; unpublished report).

A fixed stripping cost was adopted, for the sake of simplicity. This cost normally tends to increase with pit depth. However, as a semi-mobile in-pit crusher will be installed in Mutuca soon, the stripping cost might not vary substantially with depth.

11.3.6 Cost centres

Table 11.1 shows the cost centres for Mutuca Mine. All costs used in the economic analysis were obtained from MBR on a confidential basis, hence will not be explicitly shown. The cost centres used in the evaluation are displayed in table 11.2.

Table 11.1 Mutuca Mine cost centres

Stripping (Operating and Maintenance)
Drilling and blasting
Loading and hauling
Pit support
Mining (Operating and Maintenance)
Drilling and blasting
Loading and hauling
Pit support
Beneficiation (Operating and Maintenance)
Reprocessing
Reclassification
Sizing and crushing
Product handling
Dam maintenance
Plant support
Other costs
Administration
Taxes
Depreciation

Source: MBR; 1986; Monthly Report, Mine cost sheet for Mutuca Mine; unpublished report.

Table 11.2 Cost centres considered in the economic analysis

Total direct mining costs(see table 11.1)
Transportation of ore from the mine to the train loading station
Truck loading
Transportation
Mineral tax("Imposto Único Sobre Minerais"-IUM)
Train loading
Railway freight
Ore handling at the seaport terminal
Train unloading
Stockpiling
Re-screening
Reclaiming
Sampling
Shiploading
Other taxes("Plano de Integração Social"-PIS)
Sales expenses
Income tax

Source: Mendes;1986;MBR, Financial Department; unpublished report.

11.3.7 Interest rates

The Net Present Value(NPV) method calls for an interest rate for the cash flow to be discounted. There are two different aproaches, as far as investment analysis is concerned, for the selection of this interest rate: **real terms** or **current terms**.

Krige(1978) reported that under conditions of high inflation, the most meaningful criteria for economic viability are those expressed in **real terms**, i.e., measures showing the position after elimination of the effects of the depreciation of the currency.

In real terms, an interest rate of 7 to 8% p.a. is more usual for mining enterprises. In order to assess the impact that variations in the interest rate have over the resulting NPV, the rate can be varied slightly. However, return rates of 10% p.a., in real terms,

are very rare. Projects economically sound could be rejected, if the NPV analysis uses an excessively high discounting rate(Spencer,R.; 1986; RSM; personal communication).

When working in **current terms**, the effects of inflation and escalation should be taken into account. The relationship between the real rate and the current rate is as follows(Neves,1982):

$$(1 + R) = (1 + i) \times (1 + r)$$

Where

i = interest rate in real terms;

r = inflation rate; and

R = resulting interest rate in current terms.

Example : i = 5% p.a. ; r = 8% p.a. ; thus R = 13.4% p.a.

Stermole(1979), in a case study, refers to an escalation rate of 8% p.a. and a discount rate of 15% p.a. . In a "workshop discussion", he answered the question on what is an appropriate interest rate as follows:

> "(...) if we feel that 12 or 15% represents the other opportunities that exist for the use of money, from an economic point of view we are not interested in making investments that won't do at least that well"(Stermole, 1979, page 338).

In their papers, both Lane(1964) and Kim(1977) used a discount rate of 15% p.a.; Dohm(1979) adopted 12% p.a. .

"For an operation of stature and ample reserves, normally the interest rate will not exceed 8 to 12%. Should large risks be involved the present value is simply used as a starting point for negotiations from which additional sums of money are deducted based on the inherent risk factors involved" (Raymond, 1976, page 61).

Blackwell(1970) mentioned that a rate of 8% to 15% p.a. net of tax applies for most industrial contexts. Groundwater(1967) refers to an interest rate of 10% p.a., normally representing the lowest rate acceptable for the company concerned. He reported that the rate of return to investment in the United Kingdom had been around 9% p.a., varying considerably with the nature of the business concerned. However, the preliminary results of **post mortems** of mining enterprises, he wrote, "will surprise both government and owners - the original investments would have done better had they been put in the bank and left there!"

Krige(1980) studied the rates of return realized on investments in the gold mines of the newer goldfields outside the Witwatersrand. He showed that these rates of return cannot be equated with the returns that are expected when investments are made. He suggested the expected levels of threshold returns in real terms for new mining projects was some 7 to 8 % p.a. in real terms, and the corresponding average discount rate for producing mines some 3% p.a.(real). If a 9% p.a. inflation rate is considered, the resulting discount rate in current terms is 12¼% p.a., using this 3% p.a. threshold rate, in real terms, for producing mines.

If a theoretical **final net cash flow** in **real terms** has its NPV calculated by way of a discount rate in **real terms**, the result will be exactly the same as that obtained by first inflating the **final net cash flow**, then discounting it with the aid of an interest rate in **current terms**. Therefore, in this theoretical example, it would be better to make all calculations at constant prices, i.e., **real terms**.

However, in real-life cash flows, which have a series of components, inflation **does** affect the outcome, both in terms of NPV and IRR, according to Spencer(1986; RSM; personal communication), in the following cases:

working capital- in order to meet the inflated working capital costs, the return on investment is decreased each year by the ammount corresponding to the increment in working capital requirements due to inflation. The resulting NPV is consequently reduced, in **real terms**.

fiscal drag- the depreciation allowance does not change, whether inflation is considered or not, for determining the cash flow. Therefore, when working in **current terms**, the effect of the depreciation allowance is smaller than when working in **real terms**, resulting in a greater net profit before tax. As a consequence of that, the total income tax due is higher, and the rate of return in **real terms** is reduced, larger reductions in the IRR in **real terms** corresponding to higher inflation rates.

If inflation does not affect all variables in the same proportion, in a cash flow, specific inflation rates for each variable are needed. This is particularly troublesome in cash flows dealing with different currencies, e.g., in the case of international loans. The only possible solution is to treat these variables by way of Monte Carlo simulation of cash flow(Neves, 1982, page 138).

Addititonal complications arise in the economic analysis for a new mining venture that sells its products on the international metal markets: different inflation rates between

the countries hosting the mines and those hosting the metal markets; and changing pattern of currency exchange rates between these countries(Krige,1978).

The average rate of return on 86 mineral ventures in Canada was reported to be 16.2% p.a. . A base case study was defined covering all discoveries made in that country during the period 1951-1974. The potential social value arising from the 86 economic deposits is Can$ 2.751 million, using an 8% p.a. cost of capital. This 8% p.a. real cost of capital represents a 5 percent risk premium over the 3 percent historical average cost of capital in the Canadian bond market. A riskless discount of 3 percent is assumed for the evaluation of taxation payments from the Government's viewpoint(Centre for Resource Studies, Queen's University, Kingston, Ontario, 1984).

Whitney(1981, page 2.33) presented four methods for treating inflation for purposes of cash flow analysis. The fourth method assumes an explicit inflation rate for each variable, for each year in the project life. The cash flow components are first inflated through the mine's life. Subsequently, they are deflated by an average inflation rate and the resulting cash flows are used to do the DCF Analysis. The resultant IRR and NPV will not have a factor for inflation included in them. This will be similar to the current dollar calculation.

As a reference only, some published values of IRR's are listed: 20% p.a., in a project of iron ore concentrate(Pereira, 1982); 24% p.a., in the financial evaluation of a mine of adamellite, which is the source of the tungsten(Quinlan, 1979); and 10 to 20% p.a. in a new gold mine(Munro, 1977).

11.4 Brief description of the procedure adopted

In order to illustrate the economic effect of deepening the pit, the following example is useful:

- The lowest pit level is changed from 1020m to 960m.
- The total of ore is increased by 3.6Mt.
- The total of waste is increased by 19.3Mt.
- Any additional ore tonnages will only be available at the end of the exploitation of the ore which was contained by pit 1020, whereas the incremental waste tonnages start to be removed immediately, from the top levels of pit 960.
- An "investment" is made(stripping costs) for a number of years, aiming at having a future reward(revenue generated by selling the additional ore).
- This hypothesis could be economically appraised in terms of incremental cash flow, the NPV Method being applicable.

In this research work, the aforesaid "investment" made in stripping costs was considered as operational cost only, for the sake of determining the net cash flow.

The factor that discounts a future sum to a present value, given an interest rate and the corresponding number of interest periods, is called "Present Worth Factor"(PW). For instance, the present value of a future unitary single payment which is due 10 years from now, discounted at a 5% p.a. interest rate, is PW=0.6439(Grant, 1964). Therefore, for **M** monetary units under the same conditions, the resulting NPV will be **M** times the PW(5%;10).

11.4.1 Determination of the cash flow

Table 11.3 shows an example of a cash flow determination for pit 1000.

Table 11.3 Cash flow determination for pit 1000

Year(from - to)	0-14	15
Stripping rate(Mtpy)	6.8	6.8
Total stripping(Mt)	95.2	0.2
Cumulative stripping(Mt)	95.2	95.4
Annual stripping cost(US$M)	8.70	0.26
Production rate(Mtpy)	6.5	6.5
Total production(Mt)	91.0	4.2
Cumulative production(Mt)	91.0	95.2
Annual production cost(US$M)		
unit production cost = upc (US$/t)	6.5 X upc	4.2 X upc
Product sales(Mtpy) (recovery index = 91.5%)	6.0	3.8
Total sales(Mt)	84.0	3.8
Cumulative sales(Mt)	84.0	87.8
Total annual gross revenue(US$M)		
unit ore price = uop (US$/t)	6.0 X uop	3.8 X uop
Net profit before tax (NPBT)		
Revenue-(stripping+production costs)	NPBT	NPBT
Income tax		
(45% of NPBT)	IT	IT
Net profit after tax(NPAT)		
NPBT - (IT)	NPAT	NPAT

11.4.2 Determination of the Net Present Value(NPV)

Table 11.4 shows the Net Present Value of hypothetical cash flows for pit 1080, obtained by varying the ore price.

Table 11.4 Net Present Value of hypothetical cash flows - pit 1080

Year (n)	Net Cash Flow as a function of ore price variations(US$M)		
	Ore price variation -20%	0%	+20%
1	6.62	17.58	28.55
2	6.62	17.58	28.55
3	6.62	17.58	28.55
.	.	.	.
.	.	.	.
.	.	.	.
12	6.62	17.58	28.55
13	0.81	1.66	2.50
Net Present Value(US$M)	59.11	156.70	254.34

Note: the discount rate is 5% p.a.

11.4.3 Variation of ore prices

Three different hypothesis were examined: a) current ore prices; b) 20% increase; and c) 20% decrease. These would be dramatic changes, in real-life terms, however, the objective of this exercise is only to assess the sensitivity of the optional pit's NPV to variations in the ore prices.

11.4.4 Variation of mining costs

The impact of changes in both stripping and other operational costs in the NPV was assessed. Variations of the order of 20% were analysed.

11.4.5 Variation of interest rates

Three different values were considered: 6% p.a.; 8% p.a.; and 10% p.a. (see item 11.3.7). The economic analysis was made at constant prices, i.e., no inflation was considered, therefore, the interest rates are indexes of rentability in **real terms**.

11.4.6 Variation of ore quality

The ore price is established as a function of the iron content(Source: Kahan; 1986; MBR, Commercial Department; personal communication). Fluctuations in the iron ore grades have a direct impact over the NPV. As the grades were evaluated by way of geostatistics, they have an intrinsic error of estimation(kriging error), which can be used as a measure of potential variability.

11.5 Economic analysis

11.5.1 Comparing the NPV of optional pits

Five optional pits were analysed in economic terms, the NPV's being calculated for two different ore price levels as well as two interest rates, in addition to the basic case.

Table 11.5 shows the NPV's of the five optional pits for the current ore price and for +/−20% variations on the price.

Table 11.5 Sensitivity analysis - NPV x ore prices

Pit lowest level	Total tonnage		Production rate		Ore price variation					
	Ore	Waste	Ore	Waste	-20%		0%		+20%	
(m)	(Mt)	(Mt)	(Mtpy)	(Mtpy)	NPV	Rank	NPV	Rank	NPV	Rank
1080	85.0	67.2	6.5	5.6	17.16	1	98.11	2	179.05	5
1020	92.6	82.7	6.5	6.5	13.80	2	98.71	1	183.65	1
1000	95.2	95.4	6.5	6.8	10.92	3	97.17	3	183.34	2
980	95.4	99.0	6.5	7.1	9.46	4	95.80	4	182.16	3
960	96.2	102.0	6.5	7.4	8.31	5	95.05	5	181.80	4

Notes: the interest rate is 8% p.a. in real terms; NPV's are in US$M.

Table 11.6 features the resulting NPV's of the five optional pits for variations of +/−25% in the basic interest rate, therefore three different NPV's are shown for each pit, corresponding to interest rates of 6%, 8% and 10% p.a. .

Table 11.6 Sensitivity analysis - NPV x interest rates

Pit lowest level	Total tonnage		Production rate		Interest rate(% p.a.)					
	Ore	Waste	Ore	Waste	6		8		10	
(m)	(Mt)	(Mt)	(Mtpy)	(Mtpy)	NPV	Rank	NPV	Rank	NPV	Rank
1080	85.0	67.2	6.5	5.6	110.18	3	98.11	2	87.98	1
1020	92.6	82.7	6.5	6.5	111.82	1	98.71	1	87.79	2
1000	95.2	95.4	6.5	6.8	110.18	2	97.17	3	86.40	3
980	95.4	99.0	6.5	7.1	108.69	4	95.80	4	85.13	4
960	96.2	102.0	6.5	7.4	108.02	5	95.05	5	84.36	5

Notes: the interest rate is 8% p.a. in real terms; NPV's are in US$M.

The comments on tables 11.5 and 11.6 will be made under the topic "sensitivity analysis", which follows.

11.5.2 Sensitivity analysis

From the results shown in table 11.5 one can see the impact of variations in ore prices over the NPV's of optional pits. For instance, if the ore price goes down by 20%, from the current value, and an 8% p.a. interest rate is assumed, the NPV of pit 1080 is divided by a factor of 5.7; if the price goes up by the same percentage, the NPV is multiplied by a factor of 1.8. The present worth of the cash flow is, therefore, very sensitive to changes in ore prices.

The same analysis could be extended to other variables, in order to find out the key aspects of the problem and to establish how sensitive the NPV's of the optional pits are to variations in each particular factor.

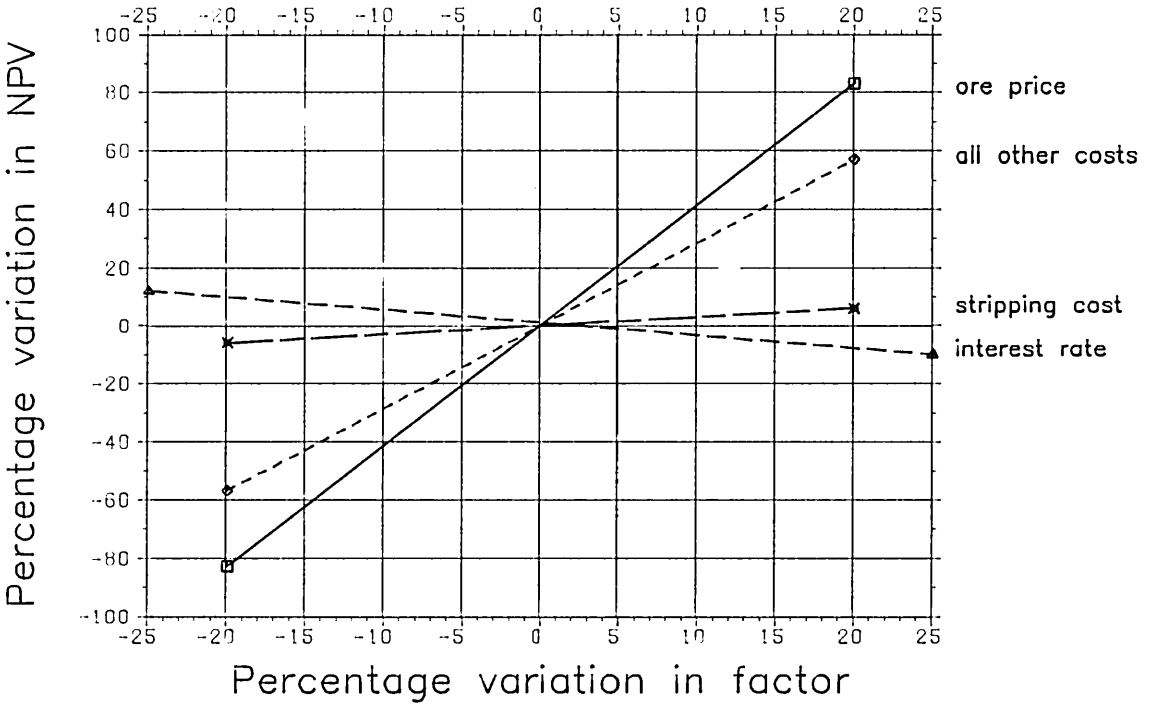
The so-called sensitivity analysis deals with different variables, not always measured in the same unit. Hence, the variations are all expressed in percentage terms. A graph is usually produced, with the relative changes of all variables in the x-axis, and the resulting percentual changes in the NPV represented in the y-axis. For a range of variations, a series of points can be obtained, therefore a curve can be drawn by linking the points. The steepest lines will reveal the factors to which the NPV has greater sensitivity.

Figure 11.1 shows how sensitive the NPV of pit 1080 is to changes in a series of components of the cash flow. As far as the sensitivity of the NPV of pit 1080 is concerned, the factors can be ranked in decreasing order of importance as follows:

- ore prices(more important);
- all other costs;
- interest rates; and
- stripping costs(less important).

The following observations resulted from the economic analysis:

- for the basic 8% p.a. interest rate, the NPV's of all five optional pits are fairly similar, pit 1020 presenting the best result, for a narrow margin, followed by pits 1080 and 1000, in that order.
- if ore prices are increased substantially, pit 1020 still is the best option, yet it is pit 1000 that occupies the second place, that turn, followed by pit 980. It is as if an increase in the ore price were justifying deeper pits. In fact, as there are so many variables in the problem, the differences in NPV are not significant. In practical terms, the hypotheses might be equally attractive.
- now, if ore prices are substantially reduced, pit 1080 turns to be the best option, this time by a larger percentual margin. It is as if a reduction in ore prices were encouraging the selection of shallower pits.
- similarly, if interest rates are reduced from 8% to 6% p.a., the "investment" in waste stripping receives a boost, pits 1020 and 1000 being the most attractive hypotheses, followed closely by pit 1080.



SENSITIVITY ANALYSIS

PIT 1080 – interest rate = 8 percent p.a.

Figure 11.1 Sensitivity analysis - NPV of pit 1080

- on the other hand, an increase in interest rates results in shallower pits being favoured, pit 1080 being the best alternative by a narrow margin.
- if significant changes occur in any key variable of the problem, the economic analysis should be repeated.

11.5.3 Comparing the results of three different economic criteria

Table 11.7 shows the ranking of the five optional pits according to three different economic criteria, namely, total profit, break-even stripping ratio and net present value. One can see that the NPV is the most conservative approach, as far as deepening the mining operations is concerned, exactly because it penalizes profits from ore distant in time by way of a discounting factor.

Table 11.7 Comparison between total profit, BESR and NPV

Pit Minimum level(m)	Ranking according to the economic criterion		
	Total profit	BESR	NPV
1080	5	3	2
1020	1	2	1
1000	2	1	3
980	4	4	4
960	3	5	5

11.6 Suggestions for further improvements

11.6.1 Mining costs as as a function of mine depth

From the mine bench being mined at present, 1240m, down to the lowest pit level, say, 1000m, there are 240m in vertical distance. At an 8% grade, this represents an increase in the trucks haulage distance of 3 kilometres. The unit consumption of Diesel is 0.316 l/t.km (Source: Rabello; 1985; MBR, Mutuca Mine; personal communication). The gradual increase in the hauling distances, as the mine goes deeper, should have an impact on the average stripping unit cost.

To date, this cost has been regarded as constant, in this research work. In the future, with the in-pit semi-mobile primary crusher system in operation, the hauling cost figures will be affected, and the optional pits should be reassessed in economic terms.

11.6.2 Mining costs as a function of type of material

Different materials are present in Mutuca Mine, ranging from hard to soft ore and from Phyllite to Itabirite, their densities being very different(see table 9.3). The unit consumption of explosives required by those materials covers a wide range, as can be seen in table 11.8.

Table 11.8 Unit consumption of explosives for different materials

Material		Unit consumption (g/t)	Unit consumption (g/cu m)
HEMATITE	Hard ore	140	560
	Mixed ore	90	315
	Soft ore	70	210
WASTE	Compact Itabirite	185	555
	Quartzite	125	310
	Schist-Phyllite	115	205
Average per ton of ore fed into the plant		90	
Average per ton of stripped material		20	

Note on explosives: granulated- NITRON, ANFEL;
dynamite, slurries- TOVEX, SPT.

(Source: Rabello; 1986; MBR, Mutuca Mine; personal communication)

Other factors might vary as well, given the distinct materials to be either stripped or mined and processed. For instance, for the period from April, 84, to March, 85, the average energy consumption was 2.807kWh per ton of ore fed into the ore dressing plant. The friability of the ore might affect this cost figure, but for the time being the economic analysis has not made any distinction between different categories of ore(hard, mixed, or soft). Nevertheless, as this information is available in the mineral inventory, it could be used in the future.

11.6.3 Ore price as a function of size fraction(product) and grade

Four different products will result when the new beneficiation plant of Mutuca Mine is in operation, each one having a different market price. Also, as the ore price is a function of the iron content, the grades play an important role. However, the economic analysis which has been made in this thesis as yet adopts an average ore price, because the different ore categories have not been considered in this study.

In the future, the different product prices could be used for a detailed economic evaluation, particularly considering that the geostatistical estimates for the percentage in weight of each size fraction are available for each block.

In short, the details available on a block by block basis in the mineral inventory, as for different types of material and types of ore, grades and tonnages, could be extensively used to produce more detailed and more realistic economic analyses. So far, many simplifications have been introduced, e.g., only one type of ore and waste, constant ore grade and price, and constant production split. For instance, the effect of changes in the percentage of the (−2";+¼") size fraction, bench by bench, could be considered in the cash flow, as Mutuca's most valuable product, Lump Ore for Direct Reduction(LORD) originates from this granulometric bracket(see figure 8.4).

11.6.4 Cash flow simulation and risk analysis

The random component of mining and stripping costs, as well as of ore prices, could be taken into account by way of Monte Carlo simulation (see Michelson, 1969), therefore incorporating in the economic model the possibility of making risk analyses, as described by Krige(1979).

For example, the frequency distributions of mining and stripping costs, as well as of mining and stripping tonnages actually produced, could be easily computed, as they are reported monthly by MBR. On the other hand, using the kriging error and assuming a normal distribution centered around the kriged estimate for each block, the ore quality can be simulated through pseudo-randomic computer routines.

Other random components can be incorporated to the economic model, as railway transportation to the seaport terminal and shiploading rates, which are centered around their planned targets, the stockpiles absorbing local fluctuations. In addition to that, there are the differences in exchanging rates and inflation rates, which contribute to make the picture rather complex.

The final purpose of the simulation exercise would be to generate a distribution of NPV's, as opposed to obtaining just one single deterministic result, therefore offering a

more flexible view of the economic evaluation.

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