A Thesis Entitled

AN APPROACH TO RATIONAL DECISION MAKING IN THE ORIENTATION OF MINERAL EXPLORATION EFFORTS.

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LIST OF ACRONYMS

AI	Available Investment
BCEI	Basic Commodity Exploration Index
CPT	Commodity Profitability Threshold
CS	Chance of Success
CSP	Commodity Source Profile
DA	Deposit Allocation
DCFROR	Discounted Cashflow Rate of Return
DTN	Deposit Type Number
GEP	General Exploration Potential
INEA	Investment in Non-Exploration Alternatives
LME	London Metal Exchange
MBC	Maximum Budget per Commodity
MCB	Minimum Confidence Boundary
MEI	Market Exploration Index
MIC	Modified Investment per Commodity
MJEB	Maximum Justifiable Exploration Budget
MRA	Minimum Reserve Analysis
NJB	Non-Justifiable Budget
NPV	Net Present Value
NSR	Net Smelter Return
OI	Overlap Index
PTI	Price Time Index
RELOI	Relative Overlap Index
RCEI	Relative Commodity Exploration Index
RCS	Relative Chance of Success
RMI	Relative Market Index

RPTI	Relative Price Time Index
RSPI	Relative Socio - Political Index
SD	Standard Deviation
SPI	Socio - Political Index
TCS	Total Chance of Success
TJB	Total Justifiable Budget

ABSTRACT

The problem of strategic decision making in the metalliferous minerals industry has, to date, tended to have been solved by a stochastic process. This thesis describes a new approach to this problem involving rational decision making for the orientation of mineral exploration efforts.

The thesis is composed of two basic parts, the first being the specific statement of the problem, underlaying assumptions and constraints, and its theoretical solution. The second part being an example of the use of the theory by a hypothetical mining company to determine the best exploration strategy, and a review of the status of known deposits in the light of the results of the strategy developed.

Success is defined, in general, as the excess of reality over desire. Using this concept in exploration, reality is expressed as a series of grade-tonnage curves representing the sources of the commodity. Financial desire is initially defined as an internal rate of return, but this is then translated to equivalent grade-tonnage combinations and is then also depicted as a series of grade-tonnage curves. The chances of exploration success are then determined by overlaying the grade-tonnage curve of reality on that of desire.

On the basis of this overlaying specific deductions are made regarding the relative amount of effort that can be rationally justified for each commodity. In addition, specific, attractive deposit types are identified and minimum grade and tonnage criteria are calculated for each deposit type within each commodity.

Finally, these specific conclusions are combined to form the best overall strategy for investment in mineral exploration by a hypothetical company.

PART 1

Derivation of a General Theoretical Approach to the Solution of the Problem of the Rational Orientation of Exploration Efforts.

1.0 INTRODUCTION

1.1 General Review

The basic objective of the work described in this thesis was to try to develop a system of reasoning that would solve the problem of how to best orient the investment of a company in exploration. The logic system developed would be expected to work at the strategic planning level within the management framework.

This study then, was an examination of strategic behaviour as exhibited by an organization. For the purposes of this study the meaning of "strategic behaviour" as defined by Ansoff (1) was used, namely:

"Strategic behaviour is the process of interaction of an organization with its environment, accompanied by a process of changing internal configurations and dynamics"

Such strategic studies have become routine in many industries; but little evidence is available to justify the belief that formalized decision systems are widely used in the mining and exploration industries. The usual level of sophistication in decision making is exemplified by a quotation from the CIM's 1970 conference on Decision Making in the Minerals Industry (2):

"A well known geologist with long experience in metals exploration describes a typical decision by a firm as "let's spend $X in the ZY area to see what's there", giving no explict weight in budgeting expenditures to expectations for mineral prospects and mineralization"

Stermole (3) also has a comment on the state of decision making in exploration management:

"The whims of management should not be the basis for reaching decisions".

Having seen the results of such a stochastic system of decision making Stermole (4) draws the following conclusion:

"If systematic methods are not used to compare the economic considerations of investment alternatives, it seems evident that in certain investment decision making situations the wrong choices may be made from an economic viewpoint".

From the above, it may be argued that much decision making in exploration management is based upon stochastic rather than rational processess, with the consequence that incorrect decisions are made. If wrong decisions are made, it means that a less than optimal strategy is being employed.

At this point it is helpful to consider the nature of decisions themselves. The requirement for, and quality of decisions is succintly expressed by Thuesen (5):

"... the need for action demands decisions in many situations not fully covered by concrete facts. Then decisions must often be based upon qualitative knowledge".

Thus it is clear that decisions must be made in order for an organization to be successful; and, that many of these decisions will be made in conditions of uncertainty. The risk is then that wrong decisions will be made. This problem is compounded by the linkage between decisions, as explained by Thuesen(6):

"... all decisions involve choice between alternative courses of action. We may call each course of action a strategy, so that the task of the decision maker is to choose between a number of alternative strategies".

So, incorrect individual decisions mean an incorrect overall strategy. Moreover, the linkage between decisions means that a stochastic decision making system is biased towards failure.

If using a stochastic decision making process tends to produce failure, then it is desirable to use a rational system to reverse such a tendency. However, is such a rational system possible to construct ? - Hillier (7) concluded it should be possible to identify a rational decision making process, provided that:

"... the decision maker can:

1. Give a consistent preference order for all alternatives or events of interest, and

2. express consistent preferences for combinations of events and stated probabilities".

Such constraints mean that the problem must be limited and a specific goal or set of goals stated explicitly.

From the above examination of the nature of decisions, it would appear that the reason for making a correct decision is to achieve success. It is then, appropriate to consider what is understood by the term "success". According to the Oxford Dictionary, success is:

"accomplishment of an end aimed at"

For the purposes of this study the "end aimed at" is defined as the maximization of the rate of return on an investment. Having said that, comments may be made on the relationship between strategic behaviour and success. According to Pryor (8):

"The risks are spread so that the combine is not dependent on the full success of all its ventures, nor are its interests confined to any one mineral."

Pragmatically, this may be translated as hedging your bets. Thuesen (9) and (10) had some general comments about the relationship between success and strategy:

"Attention may be focused on doing worth-while things or on doing things very well. Economic success depends to an extent on each",

and;

"... it is apparent that the extent of the success of a venture depends upon its potentialities for income less the sum of the costs of finding it and carrying it on",

therefore,

"... it appears warranted to draw the conclusion that the outcome of an understanding is jointly dependent upon the potentialities of the undertaking itself and upon how well it is prosecuted".

Success may therefore be defined as the relative excess of what is actually possible over what is actually required. In short, success is the excess of reality over desire. Such a definition permits not only the determination to be made as to whether a strategy is successful, but also how successful. In other words, it allows for the relative ranking of one strategy compared to others.

Reality in geologic terms may be expressed by the size and quality of a deposit. As was stated above, desire in financial terms is expressed by a rate of return. Clearly, before success could be evaluated common modes of measurement are needed. Part of the work of this thesis was to translate financial desire, as expressed by a rate of return, in to geologic terms of tons and grade. Once such a conversion is made, both reality and desire are expressed in the same terms and an evaluation of success may be made for a variety of choices. The alternatives giving the greatest success can then be put together to form a "best" strategy.

The necessity for determining how much success is associated with an alternative is explained by Thuesen (11), as follows:

"... many economic efforts are unfruitful for the reason that there is not sufficient economic input to pass the threshold of success.",

and,

"The threshold idea should be taken in to consideration in evaluating opportunities."

In other words, there is a threshold which must be passed before success results. In exploration terms that threshold is set by financial desire. If that desire is set too high, then failure will result from all exploration activities. On the other hand, if it is set too low, whilst it will be possible for geologic reality to exceed desire and for success to result; the resultant success will be below that which could have been achieved had the investment been made in other, non-exploration alternatives. A method must, therefore, be found to devise a rational decision process which, in general terms, will answer the following questions:

1. Can investment in exploration be justified in competition with other alternatives ?

2. If so, how much of the potential investment may be reasonably consummed by exploration ?

3. What is the blend of commodities, deposit types, sizes and grades that will yield the most success ?

Again, part of this thesis was devoted to devising such a method.

1.2 The Nature of Previous Work

Exploration has been carried out for millenia, and decisions have clearly had to be made for this process to occur. Decision Theory itself encompasses a whole body of scientific endeavour and some of its techniques such as characteristic analysis (12), decision trees (13), and probabilistic simulation (14) have been widely used and described in the context of mineral exploration. Papers on the methodology of project evaluation in the mining industry and descriptions of the techniques used abound, and are typified by Brown (15), Slavich (16), O'Hara (17), Whitney (18), Baker (19) and Rendu (20), to name but a few. Strategic management, as a concept has also been well developed over many years, and is well described, in general terms by Ansoff (1).

The current methods used in investment analysis are well described by O'Neil (21) & (22). Essentially, they comprise cashflow analysis and the calculation of a variety of indices such as payback, net present value (NPV) and discounted cashflow rate of return (DCFROR). The resultant cashflows are subjectively factored in an attempt to bias the information towards the real world. This process is less than scientific as it is not, by its very nature, independently repeatable, and if the correct strategy is developed, as the result of this decision process, then it is achieved by chance.

In specific terms the items which are of interest to the decision maker in exploration were summarized by Pryor (8), as follows:

"The essential facts which will govern the financing and operation of a prospect which survives the exploratory stages can be summarized thus:-

a. Extent and value of the deposit
b. Long term forecasts of markets for products.
c. Economic rate of depletion
d. Terms proposed for capitalization
e. Political stability of the government issuing title
f. Legal and fiscal conditions to be observed
g. Working conditions likely to influence exploitation"

It will be noted that not all the above criteria are of prime interest during the initial stages of decision making with which this thesis is concerned. Turning to those which are of relevance, the extent and quality of deposits has been well documented over the years both in the mining press (23) and in specific publications such as Dixon (24). The key factors in classifying deposits in engineering terms were identified by Botbol (12) some time ago, as dip of fractures, rock type and mineral type. Using such key factors, Botbol did, in fact, classify copper, lead and zinc deposits. However, Botbol's work was limited to a specific geographic area and was not expanded in to a general theory.

The long term forecasts of markets for products is a subject of ongoing concern for many workers. Three summaries of this are provided by the United States Bureau of Mines (USBM) (25) & (29) and Fischman et al. (26). Current market situations and inflation behaviour patterns are well covered by the American

Metal Market (27). In terms of price forecasting, conventional statistical analysis as described, for example, by Davis (28) is widely used; but no previous examples of the application of regionalized variable analysis, as described by David (14), have come to the author's attention.

The working conditions that effect exploitation depend to a large extent on the type of exploitation methods used. Mining methods have been classified in terms of support systems by Atkinson (30), but this seems to be treating the symptoms rather than the disease. No previous attempt to classify mining methods in terms of the key factors identified by Botbol (12) have been found by the author. Mineral processing systems have been well classified as a function of mineralogy by O'Hara (31).

1.3 Nature of the Resultant Decision System

The decision system resulting from this study must fulfill certain requirements of engineering economy if it is to be of use. These requirements were summarized by Thuesen (32):

"The functions of engineering economy are:

- determination of objectives
- determination of strategic factors and means
- evaluation of engineering alternatives
- interpretation of economic significance of engineering proposals
- assistance in decision making"

The question then arises of which methods are appropriate for the evaluation of economic decisions, Stermole (33) answers this question specifically:

"Only three methods ... consistently lead to correct economic decision making for all situations. (They) are DCFROR, net present value and net future worth."

In accordance with this dictum, DCFROR and NPV are used as the basis for decision making in this thesis.

The problem being considered is complex, therefore, it might be expected that the decision system developed will be complex also. However, Rendu (20) concluded that as far as exploration is concerned that:

"Even very simple models will lead to acceptable results"

But what is "acceptable" in terms of the results yielded ? Aristotle may be used for guidance in this regard:

"It is the mark of the educated man to look for precision in each class of things just so far as the nature of the subject admits; it is evidently foolish to accept probable reasoning from a mathematician and to demand from a rhetorician scientific proofs."

In other words, high precision is not required for strategic decisions, therefore, combining the conclusion of Rendu with the logic of Aristotle, it may be deduced that a simple model will produce an acceptable result.

1.4 Possibly New Ideas

The following paragraphs are intended to briefly highlight potentially new ideas.

The method of classifying mining methods as a function of geologic parameters directly, rather than indirectly as a function of support system type appears to be new; as does the estimate of exploitation difficulty expressed by a bias factor. It is expected that these particular concepts will be contentious; but it is hoped that they will stimulate discussion of a rational classification system.

The idea that orebodies can be classified in engineering terms by consideration of a few simple key parameters is, of itself, not new; it was described by Botbol (12) some time ago. However, it is believed that the extension of this idea from the particular to the general, and its integration with a re-classification of exploitation technology to produce a match for all deposit types is new.

The author has been unable to locate any previous description of the concept of transforming the internal rate of return equation and solving directly for the specific grade-tonnage combination that will produce a pre-defined DCFROR.

The concept of Commodity Source Profiles has been partially addressed by Harris et.al.,(34), but not so named. The grade-tonnage combinations that were produced by that study were expressed in terms of subjective probability tables, rather than as grade-tonnage curves at various levels of confidence.

A multitude of studies investigating the sensitivity of grade and tonnage to changes in required return have been performed over the decades.

However, no specific reference has been found to the concept of translating required rate of return in to grade-tonnage terms and of then formalizing it as a Commodity Profitability Threshold.

Further, evidence of the prior use in the mining industry of the definition of success illustrated by the overlap of the Commodity Source Profile on the Commodity Profitability Threshold, has not been found.

Hence, no published material has been found by the author dealing explicitly with the consequential methods described in this thesis for determining the Chance of Success in exploring for a given commodity, or the Total Chance of Success in exploring for all commodities, or for determining the best scheme of allocating budgets as expressed in the Deposit Allocation Diagram, or determining the minimum grade-tonnage requirements for specific deposit types within a specific commodity based upon the interpretation of the Deposit Allocation Diagram.

No published evidence could be found describing the application of regionalized variable analysis to the prediction of future commodity prices.

No previously published evidence has been found by the author describing the weaving together of all the threads mentioned above, both old and potentially new, in to one whole, repeatable, quantified system of reasoning.

1.5 Nature of the Thesis

In discussion of the ideas and concepts mentioned above it became clear to the author that this thesis was somewhat unusual for a scientific, geologic study. Therefore, it was felt that it would be appropriate to say something about the nature of the study so that the reader may view the subsequent chapters in the correct context. In writing this preamble the author has leaned heavily on the work of Professor Ansoff (1) and his description of strategic thinking and scientific reasoning. The following paragraphs on the character of complexity and the supporting axioms are quoted, virtually intact, from his description of his ideas on these subjects. In the author's opinion Professor Ansoff has expressed clearly and succinctly the philosophy underlying the author's approach to the topic of this thesis, therefore, no apology is made for the somewhat lengthy quotations.

"(This thesis) is an exercise in the comprehension of complexity. There are several ways to achieve this goal. The "scientific method", which may be described as an empirical heuristic which holds that the truth or untruth of any assertion about the real world is established only through a process of expermental verification. This means only empirically testable problems may be solved."

Unfortunately, the author is not in a position of controlling a major investment company, therefore, the use of the scientific method to test the validity of the propositions in this thesis is precluded. Therefore a different philosophical approach is needed. Rather than analysing the problem and arriving at understanding by examination of its detailed constituents, it is necessary to synthesize the complexity from simple ideas until reality is mirrored. According to Ansoff:

"This method was first used by Euclid some 2200 years ago, and is today known as complexity aggregation or complexity compression. This method is based upon the assumption that it is possible to identify a small number of relatively simple axioms which have two properties:

(i) they explain complexity at the highest level

(ii) explanations of lower levels of complexity can be derived from the higher levels by logical inference.

This theory was expanded by Chester Barnard (1), who determined that no matter how complex a management problem, it is usually possible to identify a small number of "strategic variables" which determine the essential shape of the solution."

This thesis is written in the Euclidean complexity aggregation tradition and should be evaluated in that light. Again, according to Ansoff the criteria that should be used for that judgement are:

" 1. Conformity to intuitative experience
2. Clarity of propositional content
3. Internal logical consistency
4. External logical consistency
5. Status of a logical scheme with:
   a. widespread conformity to experience
   b. no discordance with experience
   c. coherence among its categorical notions
   d. methodological consequences "

Required as a starting point for complexity compression is a statement of the basic assumptions or axioms upon which the logic will stand. The following section, therefore, contains suitably modified versions of several original axioms upon which this thesis is based. The original axioms are attributed to the mentioned authors:

Whithead or Maslow's Axiom

1. The behaviour of a company is motivated by an aspiration for security and an aspiration for achievement. Since achievement entails risk, each company makes a different trade-off between two aspirations.

2. The vigour with which a company pursues its aspirations is determined by the strength of its achievement drive and the power at its disposal.

Machiavelli's Axiom

1. Companies seek to attain their aspirations by influencing others to behave in accordance with their preferences.

2. Their influence depends on the degree of control which they possess over allowing and/or denying others the fulfillment of their aspirations.

Emery - Trist Axiom

The environment determines the modes and conditions of behaviour necessary for survival and/or achievement of organizational aspiration.

Chandler's Axiom

The success of an organization depends on the alignment between its behaviour in the environment and the conditions for success defined by the environment.

So, the general scene has been set, the previous work reviewed, some potentially new ideas highlighted and a perspective has been given for the appraisal and understanding of the logic. Finally, a few words of caution before the description of the logic begins.

No theory, however elaborate, can completely eliminate risk from the exploration process. However, such a condition can not, logically, be allowed to prevent the application of quantitative thought to minimize the inherent risk. Moreover, the reliablity of the results obtained from the application of the theory contained in this thesis will not exceed the reliability of the data input to that theory. Consequently, if the mechanics of the theory do not materially alter the reliability of the input data, then the theory may be regarded as useful and preferable to a stochastic decision making process.

There are certain constraints on the basic problem which become apparent, including:

. financial

- level of profit
- level of investment
- level of risk
- time

- commodity characteristics
- market characteristics
- geologic reality
- exploitation technology limitations
- socio-political limitations

To be useful these constraints must be measured and classified in standard, quantitative ways, and their effects translated into exploration target requirements in terms of specific grade and tonnage ranges for particular deposit types. These desires may then be matched against real opportunity and the chances of success in a particular scenario assessed.

The succeeding chapters will put forward an approach to this problem, leading to the development of a unified process.

2.0 FINANCIAL CONSTRAINTS

2.1 Level of Profit

The required level of profit for a company is defined for the purposes of this thesis as the internal rate of return produced by a project on an after tax basis when viewed from the 100% equity point of view. This will vary depending upon the type of project under consideration, it's location, etc., and will be set by the company at some level.

2.2 Level of Risk

Some companies are limited in the size of investment that they can accommodate, this constraint may well preclude certain types of exploitation systems, particularly those requiring high initial capital input. In turn, this restriction will limit the type of deposit, and hence, possibly a commodity that a given company may reasonably include in its exploration portfolio.

2.3 Time

Time appears as a financial constraint because of the time value of money. It is desirable for exploration to take place in a timely manner, that is to say neither too soon nor too late. It is clearly poor strategy to bring on-stream a new project just as the market starts a protracted downturn.

Moreover, even the largest companies have a finite amount of resources at their disposal for exploration, so the question of when precisely this activity should take place must be addressed.

3.0 COMMODITY CHARACTERISTICS

In order to limit the problem so that it may actually be solved, it is necessary to classify commodities with respect to the stated goal, i.e.; making a profit. Certain characteristics may be regarded as beneficial to the aim of achieving that goal, namely:

- high unit value
- common occurrence
- amenable to standard technology
- high, sustained demand
- low supply
- non - monopoly supply
- non - strategic supply
- located near consumption centers
- in an area of political stability
- low ratio of known reserves to future demand
- majority of current supply imported
- located in an area of little or no environmental or other bureaucratic restriction
- other

The fact that some of the above criteria would seem to be mutually contradictory does not preclude them from inclusion as benefits, it merely makes perfection difficult to achieve.

It is not enough simply to say that because a source or deposit is located in , say, a politically unstable area this is bad. It is necessary to quantify what " bad " means. This may be done by ordinal ranking. That is a quantitative value may be assigned, in a qualitative way to political stability. A very stable system may be given a value of 100, and a very unstable system a value of 1. Thus one may classify political stability on a scale of 1 - 100 with regard to exploration. This logic may be applied to other qualitative characteristics, specifically: common occurrence, amenability, location, environmental impact, bureaucratic impact and other. The remaining parameters are commonly measured in terms of percentage, and so all characteristics are now classified on the same scale.

With all significant characteristics of a commodity quantified it is possible to assess both the absolute utility of a specific commodity and the relative utilities of various commodities in terms of satisfying the stated goal.

3.1 Basic Commodity Exploration Index

The way in which this may be achieved can be outlined as follows. The significant characteristics are each assigned a value; these values are combined in some standard way to produce a Basic Commodity Exploration Index, BCEI. The significant characteristics are:-

Characteristic	Scalar Value
Unit value	C(1)
Abundance	C(2)
Amenability	C(3)
Removal	C(4)
Supply	C(5)
Monopoly	C(6)
Strategic significance	C(7)
Location	C(8)
Political stability	C(9)
Reserve/demand ratio	C(10)
Import situation	C(11)
Environmental impact	C(12)
Bureaucratic impact	C(13)
Other	C(14)

$$BCEI = \{[\ C(1)+....C(n)\ ]/[\ n * 50\ ]\}$$

- where, n = number of characteristics.

50 is used to express BCEI on a relative basis with respect to an "average" value of 1.0.

It was decided to express this and other qualitative and quantitative indices with respect to "1", an average condition, in order to convey more meaning to the reader. In general usage it is conventional to express departures from a norm as more or less difficult or attractive. Therefore, if these indices are expressed with respect to "1", then it is possible to ascertain what a rating of, say, 1.5 means. It would mean that conditions would be one and a half times better than normal. This technique has the advantages of being easily translated in to intuitive perception and, therefore, of reinforcing the ability of the reader to modify the course of the strategic planning process by subjective judgement and feedback.

The above equation defines the Basic Commodity Exploration Index as the average value of the sum of the component characteristics. This may not actually be true. However, currently social science is unable to supply a quantitative theory which links these components. Faced with such a situation and needing to produce an answer as to a commodity's basic desirability as an exploration target, the best theoretical solution is to assume that they are independent, random activities and, therefore, the best quantitative measure of their combined significance is the arithmetic mean of their scalar assessments. As was mentioned in the Introduction, this clearly limits the reliability of the results of such an analysis.

The BCEI equation also assigns equal weight to each component. In reality this may not be true, but, currently, no uniform, quantified method of assessing weights exists, and so, under the same logic expounded above, the most reasonable solution is to assign equal, unit weights to each component.

3.2 Relative Commodity Exploration Index

This process may be repeated for each commodity and a Basic Commodity Exploration Index derived for each one. Commodities are real things and exploration is a real process, therefore, BCEI must yield ratio type scalar values, ie: a BCEI can not have a negative value. Given BCEI's are ratio scalar, and that they are measured on the same uniform scale, they may be compared in a relative way with one another. Conclusions also may be drawn regarding the desirability of exploring for one commodity rather than another.

These ideas are explicitly quantified by the Relative Commodity Exploration Index (RCEI), where RCEI is defined as follows:-

RCEI = {[BCEI(i)]/[BCEI(i) + ... BCEI(n)]}

- where, n = number of commodities.

i = the ith commodity.

3.3 General Strategy

For example, assuming the BCEIs' listed below:-

Commodity	BCEI	RCEI
Gold	0.70	0.23
Silver	0.30	0.10
Copper	0.10	0.04
Tin	0.05	0.02
Molybdenum	0.60	0.21
Tungsten	0.80	0.27
Lead	0.15	0.06
Zinc	0.20	0.07
Total	2.90	1.00*

* Note: the sum of the RCEI's must be equal to 1.00 because it repesents the total real effort

available for exploration.

The RCEI's may then be used to rank commodities from an exploration point of view, and also to assign budget expenditure.

For example, suppose a company had a total exploration budget of say $50 million, the expenditure would be best oriented as follows:-

Rank	Commodity	RCEI	$, M
1	Tungsten	0.27	13.5
2	Gold	0.23	11.5
3	Molybdenum	0.21	10.5
4	Silver	0.10	5.0
5	Zinc	0.07	3.5
6	Lead	0.06	3.0
7	Copper	0.04	2.0
8	Tin	0.02	1.0
		1.00	50.0

Doubts about assuming a linear relationship between RCEI and expenditure are addressed in Chapter 9, section 9.2.1.

Such an approach has the benefit of directing effort in to the areas most likely to prove successful in relation to their actual chances of achieving success. By definition, this must increase the chances of successful exploration. On the other hand, it does not eliminate a company's risk of failure altogether, later in this thesis a method of determining this chance of failure will be advanced. However, this approach, even at this qualitative stage, does rationally limit the amount of expenditure that should be channelled into exploration for any particular commodity. Further, given such a ranking a company may choose to eliminate certain commodities from its list, because compared to others they have little potential. This will increase the amount of effort that may be expended in the search for less risky commodities.

Summarizing, the above approach allows a company to allocate the correct amount of effort to a commodity as a direct function of the expected chance of success in exploration for that commodity. This permits a preliminary screening of commodities and sets-up a rational strategy for exploration. As this is a quantified approach, the effects of changes to the scalar values of the input characteristics may be measured using both deterministic sensitivity and probabilistic techniques to assess how, and under what conditions , initial ideas about strategy may be affected.

4.0 MARKET CHARACTERISTICS

Rational exploration decisions require a reasonable, quantified understanding of the various commodity markets. In the same way that beneficial characteristics could be identified for a commodity, useful characteristics can be defined for the markets themselves. Such a list would include:-

- market size
- ratio of domestic market to total market
- significance of recycling
- size of tariff barrier
- bureaucratic impact
- environmental impact
- political impact
- monopoly share
- cartel impact
- potential substitutability
- alternate potential
- price - time cycle
- other

Similarly these characteristics all may be described on a scale of 1 - 100, either on the basis of the units in which they are actually measured, or by subjective rating in the range "very good" to "very bad".

4.1 Market Exploration Index

Consequently, a measure may be derived of the potential of a given market in terms of successful exploration. Such a measure could be called the Market Exploration Index, (MEI), and would be defined as follows:-

$$MEI = \{[\ M(1) + \dots M(i) + \dots M(n)\ ]\}/(\ n * 50\ )$$

- where, i = ith characteristic
  n = number of characteristics

50 is used to produce an MEI value expressed relative to an "average" value of 1.0.

Once again equal weight is given to each component.

It can, of course, be argued that a "correct" price - time cycle is of more significance than, say, recycling. This may well be true, but, again, there is no quantified theory for assessing the relative significance of these market characteristics. However, as the classification system is quantified, the effect of changing the relative weights on the final outcome may easily be measured by performing a sensitivity analysis.

For the purposes of this thesis all characteristics have been assumed to have equal significance.

4.2 Relative Market Index

This approach may be repeated for a variety of commodities and a Relative Market Index, (RMI), derived as follows:-

$$RMI = \{[\ MEI(i)\ ]/[\ MEI(i) + \ldots MEI(n)\ ]\}$$

- where, i = ith commodity
  n = number of commodities.

The explorationist may now rank markets in terms of exploration success. Moreover, by joint use of the Relative Commodity Index and the Relative Market Index together, he is able simultaneously to evaluate not only the technical desirabilty of a commodity, but also its market potential.

4.3 General Exploration Potential

The combination of BCEI and RMI finds quantitative expression in the General Exploration Potential, (GEP), where:-

$$GEP(i) = [\ RECI(i) + RMI(i)\ ]$$

- where, i = ith commodity.

The above approach utilizes a simple linear model and it is arguable how precisely it simulates reality. The requirement for precise simulation is addressed later in section 9.2.1.

5.0 GEOLOGIC REALITY

In order to make the general problem of how to orient exploration in a profitable way tractable, it is necessary to limit the scope of possible geologic scenarios to some finite number. This means that a method of classifying geology in a way which reflects profit potential is required. Conventional geologic classifications were not devised with a view to satisfying this requirement.

A new method of classification is therefore needed. If geology is to be profitable it must be exploited. The current range of feasible technical solutions to the problem of exploitation is limited. It then becomes merely necessary to determine the characteristics of the constraining parameters in exploitation technology and to apply these standards to the classification of geology.

It will be shown later, in Chapter 6 ,that the main characteristics needed for the classification of geology with respect to profit are:-

- spatial location
- structure
- geometry

This type of real or hypothetical information is available for a deposit type, even at a conceptual level. Moreover, having classified geology in terms of these parameters it becomes possible not only to match a deposit to an exploitation system, but also to decide if it is technically feasible to exploit such a deposit; and if so, with what degree of difficulty.

The three parameters mentioned above enable mining systems to be matched to geology. The addition of some real or hypothetical knowledge about the mineralogy of the deposit, also permits a suitable mineral processing system to be selected.

Such a classification, aids the successful orientation of an exploration programs, because it will identify a set of deposits which may not be exploited by currently proven technology, irrespective of economic considerations. Such deposit types are, by definition, not candidates for exploration.

6.0 EXPLOITATION TECHNOLOGY CLASSIFICATION

Exploitation technology is a constantly evolving field; however, from a pragmatic point of view, the number of ways in which a given deposit may be exploited in a standard and reasonable manner is limited. Splitting exploitation in to two sections:-

- mining
- mineral processing.

The practical alternatives may be defined as follows:-

- mining
  - open pit systems
  - natural caving systems
  - artificial caving systems
  - self-supporting systems
  - artificially supported systems
  - other
    - borehole slurry mining
    - in-situ leaching
    - alluvial mining

- mineral processing
  - gravity concentration
  - selective flotation
  - basic flotation
  - cyanidation

It should be noted that the system limits of the problem have been drawn at the point when the concentrate is loaded ready for shipment from the mineral processing plant. The reasoning for this was that virtually all deposits have associated with them, some kind of concentration activity, but not all either have or need their own smelter and refinery. It therefore seemed reasonable that the end product should be defined as the concentrate rather than the finished metal. This means that the value assigned to each commodity must be its net smelter return rather than the price quoted on the London Metal Exchange or any similar place.

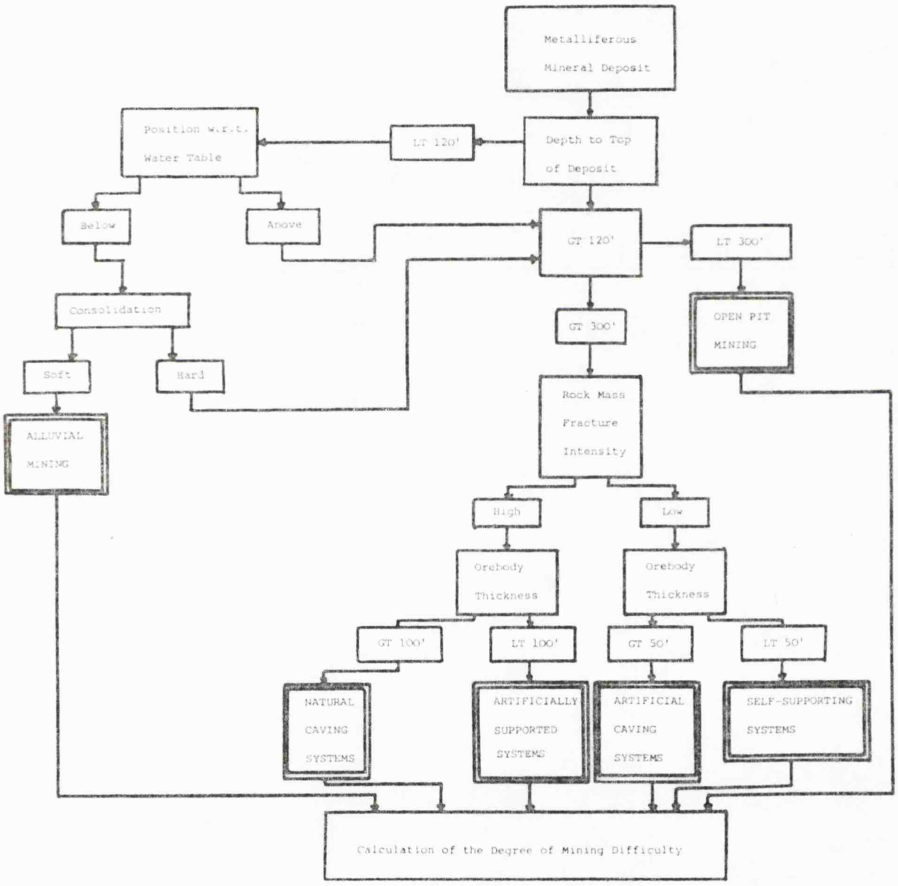
In terms of standard mining systems the technology classification is limited to currently standard systems, therefore, the new systems such as borehole slurry mining and in-situ leaching are excluded from consideration in this thesis because they have not yet attained the same status of standard proven technology.

Similarly, for mineral processing, the list above encompasses technology classifications that account for the great majority of concentration systems. Again, for the purposes of this thesis, special systems which fall beyond the above defined scope will be ignored.

6.1 Selection Logic

Having defined exploitation possiblities, the general logic flow inherent in technology selection is illustrated in Figure 1. The basic information needed to classify the geologic concept

Fig. 1 Mining Method Selection Logic



of a deposit being tested as a potential exploration target comprises:-

- consolidation
- depth to base of deposit
- depth to top of deposit
- relative water table position
- rock mass fracture intensity
- orebody thickness

Such information is available to a geologist, even at the hypothetical stage, because in order to begin considering an exploration target the geologist must have some category of deposit in mind. For these categories the above information is known, or may be inferred.

6.2 Consolidation

A knowledge of the absolute consolidation of both the deposit and the overburden is needed in order to determine the technical feasibility of applying alluvial mining for the exploitation of the deposit. If the material is well consolidated, then current technology limits will preclude the use of alluvial mining. By the term "consolidation", in this context, is meant the degree to which the material possesses shear strength. In alluvial mining excavation of material depends upon failure of the mass in shear due to the action of water jet impingement or dredge bucket impact. It is unreasonable to expect a geologist to have knowledge of such engineering characteristics of material, therefore, the classification shown in Table 1 may be used as a basis for decision making.

From Table 1 it will be seen that alluvial mining applies to "soft" rocks, that is material with a shear strength below 350 psi. Naturally, it is debatable precisely where to draw the line, however, suffice it to say that the 350 psi figure is within the bounds of currently available technology.

Table 1 CONSOLIDATION CLASSIFICATION

Rock type	Class	Uniaxial Str. psi	Friction Angle	Shear Str. psi
1	soft	0-500	0-10	0 - 88
2	soft	0-500	10-20	88-182
3	soft	0-500	20-25	182-233
4	soft	0-500	25-30	233-289
5	soft	0-500	30-35	289-350
6	very weak	500-2000	35-42	350-1800
7	very weak	500-2000	35-42	350-1800
8	weak	2000-4000	42-46	1800-4142
9	medium	2000-4000	42-46	1800-4142
10	strong	4000-8000	46-55	4142-11425
11	medium-strong	8000-4000	46-55	4142-11425
12	strong	8000-16000	55-90	11425-up
13	very strong	16000-32000	55-90	11425-up

Rock Type Description

1. Saturated clays
2. Partially saturated clays
3. Clay gouge
4. Slick fractures
5. Disintegrated rock & sand
6. 3" - 6" blocks
7. Poorly compacted sedimentary rock
8. Poorly cemented sedimentary rock
9. 1 ft. - 2 ft. blocks, competant low density sed. rock
10. Coarse igneous rock
11. 2 ft. - 4 ft. blocks
12. Competant igneous & meta. rock & some fine grain sandst.
13. Quartzities, dense, fine grained igneous rock

The above table was generated after reference to Jaeger & Cook (36) and Attewell & Farmer (42).

Using the classification shown in Table 1, all the geologist needs is some general description which will fit in to one of the 13 categories above. Given this information, not only may a start be made on selecting appropriate mining methods, but an assessment of the degree of mining difficulty may also be started. This point will be amplified later.

6.3 Depth to the Base of the Deposit

Mining technology has limits to what it can achieve. These limits are not fixed for all time, but will move with advances in knowledge. However, all possible deposits may not be mined. In general the limit on mining may be regarded as depth. Today this depth limit is about 10,000 feet below the surface. For the purposes of this thesis then, 10,000 feet will be regarded as the practical limit to exploitation and no deposits deeper than this will be considered.

In the case of alluvial mining, current equipment limits preclude excavation at depths greater than about 120 feet. This value is used in the analysis as one test of the technical viability of alluvial mining.

6.4 Depth to the Top of the Deposit

The basic objective of mining is to make a profit. In order to achieve this goal, capital and operating costs must be minimized. Revenue will only be generated when ore is produced. It is also desirable to produce this revenue as soon as possible after the investment of capital. Further it is desirable to minimize the pre-production development cost within the constraints of maximizing ore grade mined and recovery of resource achieved.

The rigorous solution of such a problem is difficult, and requires a knowledge of the deposit that is not available at the exploration stage. Indeed, if such information were available, the exploration process would not be needed. It is therefore hard to say how a deposit should be exploited before it has even been found. However, we need to have some general rule that will allow for such a decision to be made in a way that will probably be correct, because the type of mining method chosen to exploit a particular deposit will to a large extent determine its economic viability. Therefore, in the absence of a rigorous solution an empirical guide is needed.

The best expression of the mining industry's solutions to profit making are the currently existing mines. The most obvious distinction that may be made between existing mines is whether they are open-pit or underground operations.

In order for open pit mines to make the maximum profit it is necessary to minimize the pre-strip volume. This volume is a function of pit geometry and is driven by one variable, depth. Hence, in order to maximize profit pits must minimize the distance from the surface to the top of the orebody.

Theoretically then all that is required is to examine the data linking pre-strip depth to subsequent profit and to derive a general rule for determining, at this early stage, whether a deposit will be mined by open-pit or underground methods. Unfortunately, such data is not available. So we must then assume that all pits that have been started have been profitable. This we know is false. However, it may reasonably be assumed that pits which have been in production for a reasonable length of time, have been profitable or they would not have been continued. Examining existing long-term pits shows that about 300 feet appears to be the limiting depth to the top of the orebody that can be tolerated by current technology. At Twin Buttes in Arizona the pre-strip depth was 320 feet, and the pit has never made a profit. So maybe 300 feet is being a little over generous; however, consultation with senior design engineers in the mining industry would seem to indicate that 300 feet of pre-strip depth is a reasonable assumption for the cutoff depth between open-pit and underground mining, certainly for the purposes of exploration decision making. It will, therefore, be used in this thesis.

6.5 Relative Water Table Position

The position of the water table with respect to the orebody is of significance in determining the technical feasibility of alluvial mining. Clearly, if the orebody is above the water table it becomes very difficult to mine it with a dredge.

In the case of hydraulicing, it is not a technical requirement that the orebody be below the water table. Pumps could be used. It is more a question of efficiency. The necessary hydraulic head for the monitors is usually developed, at least in part, by a gravity potential as this reduces the cost. The cost of generating high water pressure for monitoring is significant, as the grade of these deposits is usually low. Moreover, if the deposit is below the water table, the pore water pressure will reduce the effective shear strength of the material, making it more amenable to exploitation by alluvial methods.

So, for the purposes of this thesis, it will be assumed that in order to be exploited by alluival methods the deposit must be below the water table.

6.6 Rock Mass Fracture Intensity

Mining requires that the rock mass is subjected to a set of mechanical processes. Therefore, in order to determine which of these sets is the most appropriate to a given geologic scenario it is necessary to know something about the mechanical properties of the deposit.

Mining is a large scale activity; therefore, it is not of direct interest to know about the small scale properties of intact rock specimens. This leads directly to an unsolved problem, how to assign mechanical properties to large rock volumes such as orebodies. As the problem remains currently intractable, a way must be found around this dilemma.

Mining is a relatively low energy activity, which functions through the inherent discontinuities, fractures and joints in the rock mass. It follows, then, that, if a rock mass is characterized by its fracture intensity, it should be possible to identify broad types of mining schemes which are applicable to its exploitation.

Using the same class descriptions as in Table 1 above, the following classification may be made:-

Class	Fracture Intensity	Block Size,in.
Soft	10.00	minus sand
Very weak	8.00	3" - 6"
Weak	6.50	1ft.- 2ft.
Medium strong	3.25	2ft - 4ft.
Strong	1.70	large
Very strong	1.00	intact

The fracture intensity list above is not on any absolute scale, as no generally accepted scale for the calibration of fracture intensity from the strength point of view exists. The scale used is a qualitative one, loosely based upon the shear strength values given in Table 1. In terms of the mining method selection logic shown in Figure 1, highly fractured rock is regarded as having a fracture intensity of equal to or greater than 6.5 on the above scale. That is to say, the deposit is composed of material having a typical size range of 1 foot to 2 foot blocks.

Therefore, rock masses may be divided in to two major relative classes; highly fractured and sparsely fractured. Underground mining systems may also be divided in to corresponding groups; those which rely upon highly fractured rock masses for their implementation, and those which require a relatively intact rock mass.

If the ground is relatively intact, then either it must be artificially fractured in order to allow for its physical removal, or use can be made of its natural "strength" to reduce the amount of ground support required.

If the ground is highly fractured, then either it must be artificially supported or use can be made of this weakness in order to allow failure by self-induced caving to occur.

6.7 Orebody Thickness

The final differentiation between underground mining methods may be made by consideration of the specific geologic concept itself, characterized by the expected thickness of the deposit.

In the case of those mining systems applicable to highly fractured rock masses: natural caving, systems such as block caving, require a certain minimum undercut width before spontaneous and continuous caving can be induced. This undercut width is a function of the tendency to arching in the material, and will vary according to the resistance generated in the rock mass by internal friction.

In section 6.6 above, highly fractured rock masses were defined in terms of the classes in Table 1, as weak, very weak and soft. The specific undercut width depends upon the combination of rock mass strength, depth, density and the general state of stress. However, because mining is a real process, its mechanical limits may be applied to define a minimum orebody thickness which will allow for access of men and equipment to generate the undercut. Current practice would set this minimum dimension at about 30 metres. So, for this analysis, any deposit which is highly fractured and has an average thickness of greater than 30 metres, will be regarded as a suitable candidate for exploitation by natural caving systems. Any deposit in this group which has an average width of less than 30 metres, will be assumed to be exploited by artificially supported mining, such as cut-and-fill.

For the group of deposits that are regarded as relatively intact; ie, have fracture intensities of less than 6.5, thickness is again the final deciding factor in choosing between the two sets of methods, artificial caving and self-supporting. Self-supporting systems depend, largely, for their efficiency on the strength of the rock mass when acting like a beam supported at each end. This, in turn, is largely a function of the tensile strength of the rock. At this stage of exploration planning it is hard to have any accurate knowledge of the tensile strength of the rock, so once again, it is necessary to adopt an empirical solution to the problem. It would seem in today's industry that the maximum span used in self-supporting mining systems is about 15 metres. Therefore, it will be assumed that, if a deposit in this group has an average thickness of less than 15 metres, it will be exploited by self-supporting methods, such as room-and-pillar. If the average thickness is greater than 15 metres, then it will be assumed exploited by an artificial caving system, such as sub-level caving.

6.8 Strategic Implications of Mining Method Selection

Using the logic described above, all deposits may be classified into a set of exploitation categories. Each category has, inherently, certain levels of required capital investment and operating cost. For instance, cut-and-fill tends to be applied to small, high grade deposits; as the deposit is small the initial capital required will probably also be small when compared to the capital needed to start a block caving operation. Further, because it is an underground operation, the chances are that the pre-production time will be longer than for an open-pit. It is also quite likely that the characteristics of the deposit will be known with less precision than for a shallower deposit; this will increase the risk associated with the investment. Again, as it is a small deposit then it is likely that the amount of actual cash generated by the operation will be relatively small.

small.

Most companies set levels of risk, payback time, available capital and revenue requirement that may be used to decide that deposits requiring exploitation by artificially supported mining methods are not attractive primary exploration targets. Such a philosophical approach allows for a better orientation of an exploration program, as it will more nearly fit the company's basic situation.

So it is now, conceptually, possible for a company to decide not to seek for deposit types X, Y & Z because they are most unlikely to be exploitable within the company's financial constraints. This then provides a direct, easily understood link between the field geologist and the company's fundamental goal.

6.9 Degree of Mining Difficulty

In order to calculate capital and operating costs for mining it is not sufficient merely to decide upon an appropriate mining technique. The degree of difficulty likely to be encountered in implementing that technique must also be assessed.

For standard mining technology, defined above this assessment may be made on the basis of a limited number of additional deposit parameters:-

- rock strength
- deposit dip
- water conditions
- depth to the base of the deposit
- other factors

6.9.1 Rock Strength

There is no body of theory currently available linking directly rock strength and ease of exploitation. Therefore, it is necessary to rely on inductive logic to produce an empirical relationship that will quantify their interaction.

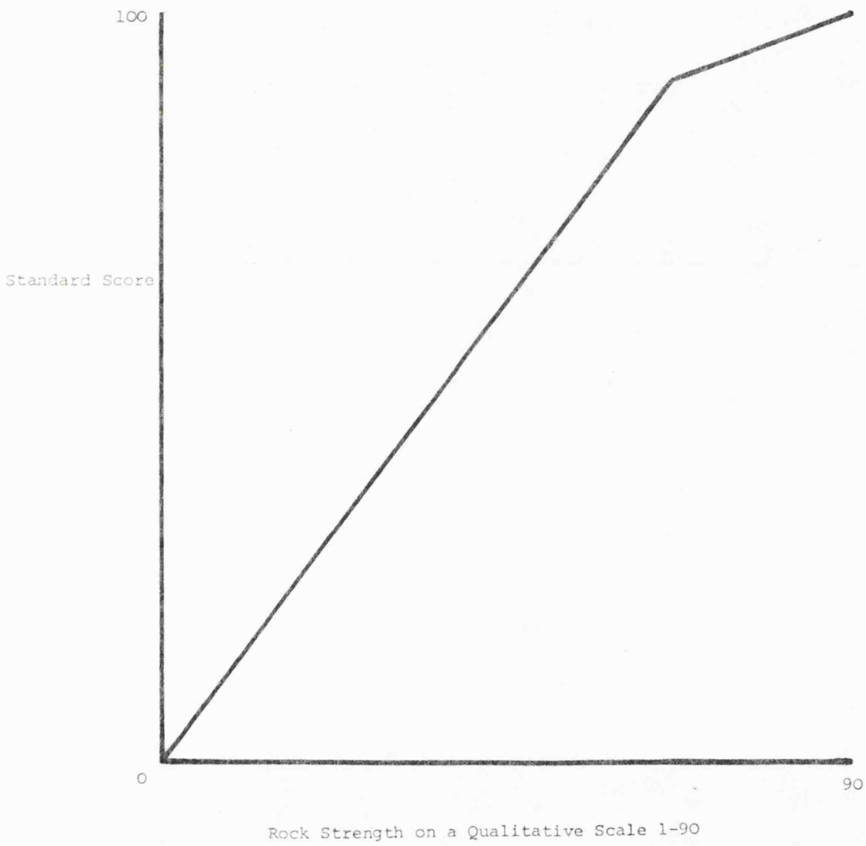
A further problem arises in assigning units to both rock strength and the degree of mining difficulty. One way around this problem is to rank rock strength on some scale, for example:-

Very weak	1 - 10
Weak	10 - 30
Medium	30 - 50
Fairly strong	50 - 65
Very strong	65 - 90

This scale may then be linked to a standard score expressing ease of exploitation on a scale of, say, 1 - 100. This is illustrated in Figure 2.

The logic behind this particular curve is as follows. In terms of ease of exploitation stronger rock masses make for easier mining, but, once the mass starts to become very strong, the benefits associated with increasing strength begin to accumulate at a slower rate. Translating this to real life, as the

Fig. 2. Rock Strength - Standard Score Relationship



rock gets stronger stability increases, but beyond a certain point the work needed for the actual excavation of the rock pushes standard technology very close to its limits.

Obviously, the precision of such a relationship is not high, but it does allow for a first approximation of a quantitative comparison between different rock types to be made.

6.9.2 Deposit Dip

The dip of the deposit affects two aspects of mining:-

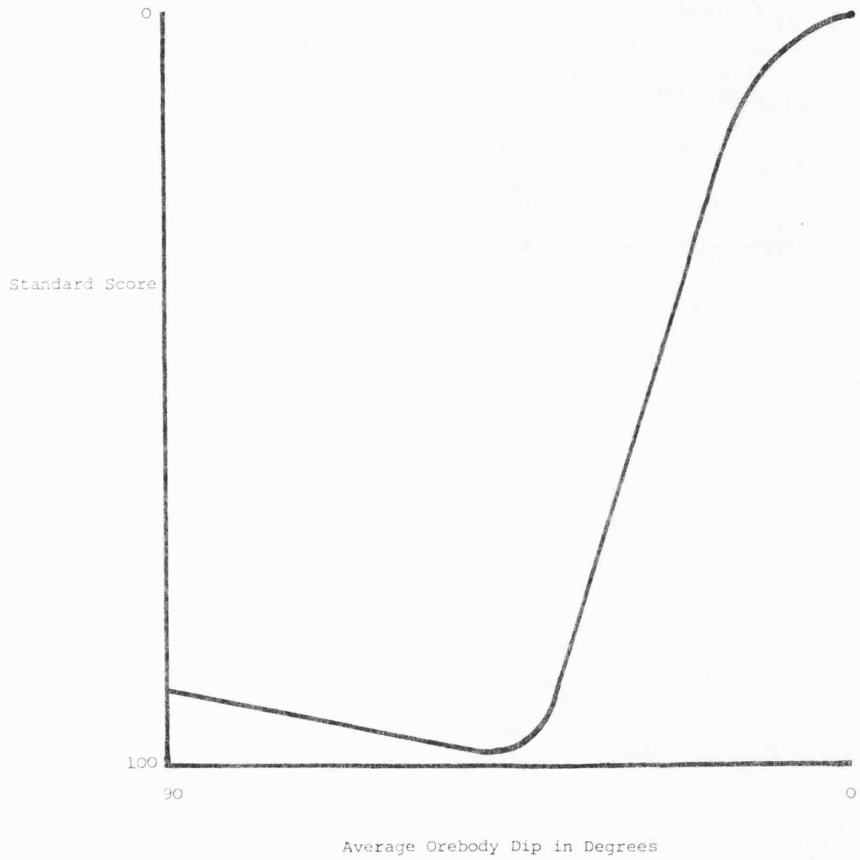
- ease of material transport
- mining loss and dilution

6.9.2.1 Ease of Material Transport

Using the same technique for assigning quantitative values to a qualitative assessment, a relationship between dip and transport difficulty may be derived. Such a relationship is illustrated in Figure 3.

The justification of the relationship is as follows. The best situation, from the transport point of view, would be to have a horizontal dip, because then cheap, high capacity haulage systems like belts and rail can be used. Once the dip has passed beyond about 5 degrees, then productivity declines

Fig. 3 Transportation - Standard Score Relationship



fairly rapidly. When the orebody is vertical the situation is bad, because a high infrastructure cost is needed to allow for the extraction of the orebody. This adverse situation is to some extent mitigated by the fact that gravity may be used in collecting the broken rock in some central point for subsequent transport to the surface.

The worst case occurs at a dip of about 45 degrees, because this provides for the maximum horizontal and vertical dispersion of the transport system, but does not allow for the use of gravity for the collection of rock.

6.9.2.2 Mining Loss and Dilution

Mining loss and dilution are controlled by several poorly understood processes. However, loss and dilution are significant factors in determining the final profitability of a project, and so some way has to be found to assess their influence.

The relative geometry and physical characteristics of the deposit and the surrounding rock mass are the main factors governing the amount of loss and dilution that may be expected. Clearly small deposits are more sensitive to the impact of waste infiltration than large ones. Similarly, if there is a large difference in particle size and density between the deposit rock and the surrounding rock, percolation of waste into the ore may become severe under gravity flow conditions.

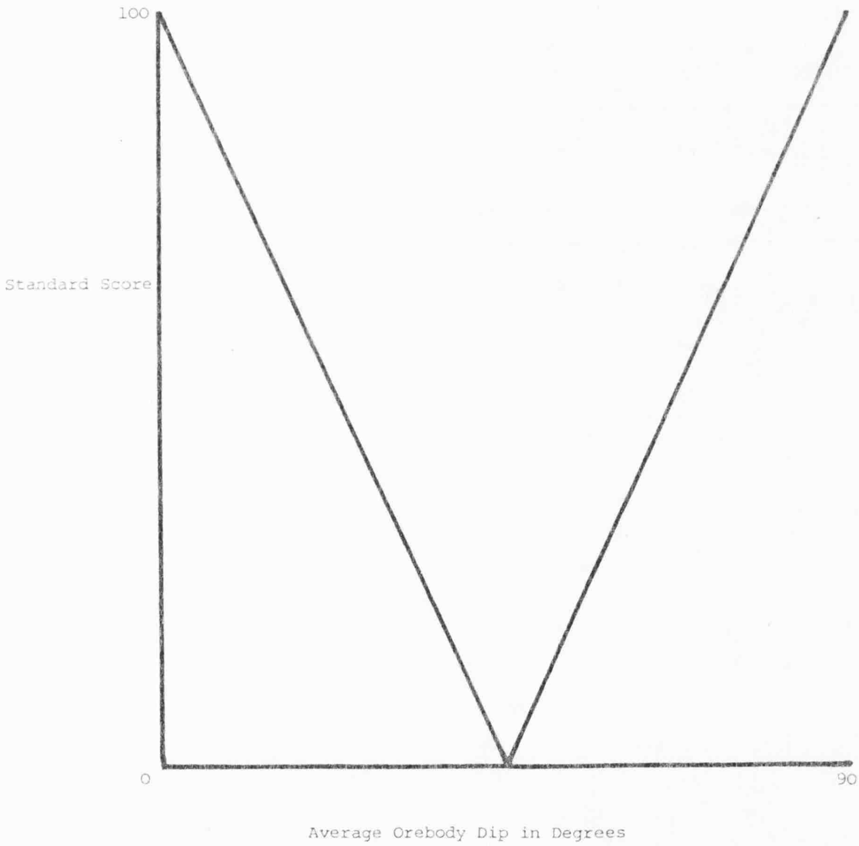
The size of deposits may be typified by the ratio of surface area to volume. The larger the surface area the greater the opportunity for dilution to occur. Unfortunately, the same type of deposit may have a wide variety of shapes, and hence surface area/volume ratios. At the strategic planning stage it is unlikely that the shape of the target deposit will be known. So this factor is of little use for this type of analysis.

Dilution and loss are dynamic processes that is to say, they take place as the result of the relative motion of ore and waste particles under conditions that may be broadly described as gravity flow. This statement contains within it the implicit assumption that the orebody is of sufficient width to allow for its removal by standard real equipment. The relative motion of the ore and waste is governed by the geometry of the deposit. Since gravity flow acts downwards, it is sufficient to consider a one dimensional index of its action; ie, dip. At the strategic stage it is reasonable to assume some knowledge of the deposit dip, because the exploration geologist may be expected to have some general environment in mind when planning an exploration program.
Any doubts about this assumption

may be calmed once some quantitative relationship between dip and loss/dilution has been derived. One such relationship is suggested in Figure 4. There are two "best" geometries in terms of loss/dilution; horizontal and vertical. If the deposit is vertical only the top will be subjected to significant dilution/loss, and this will tend to take place either during initial production or at the end of the mine life. As the mining moves down, the top of the ore shields the rest of the deposit from the effects of dilution. The major dilution and loss will take place at the end of the life of the mine when crown pillar robbing is undertaken. Similarly, for a horizontal dip the major effects of loss and dilution will be felt at the top of the deposit. Therefore, both geometries are regarded as "good" by this analysis.

The worst geometry from the loss/dilution point of view is that which exposes the deposit to the maximum amount of exposure. That is dips of around 45 degrees. This dip is then defined as "bad" from the point of view loss/dilution. A simple first order relationship has been assumed in this analysis and is illustrated in Figure 4.

Fig. 4 Mining Loss and Dilution - Standard Score Relationship



6.9.3 Depth to the Base of the Deposit

The depth to the base of the deposit clearly influences the ease with which the deposit may be exploited. The ideal situation would be to find the orebody lying fully exposed on the surface. The worst possible case would be to find it 10,000 feet below ground.

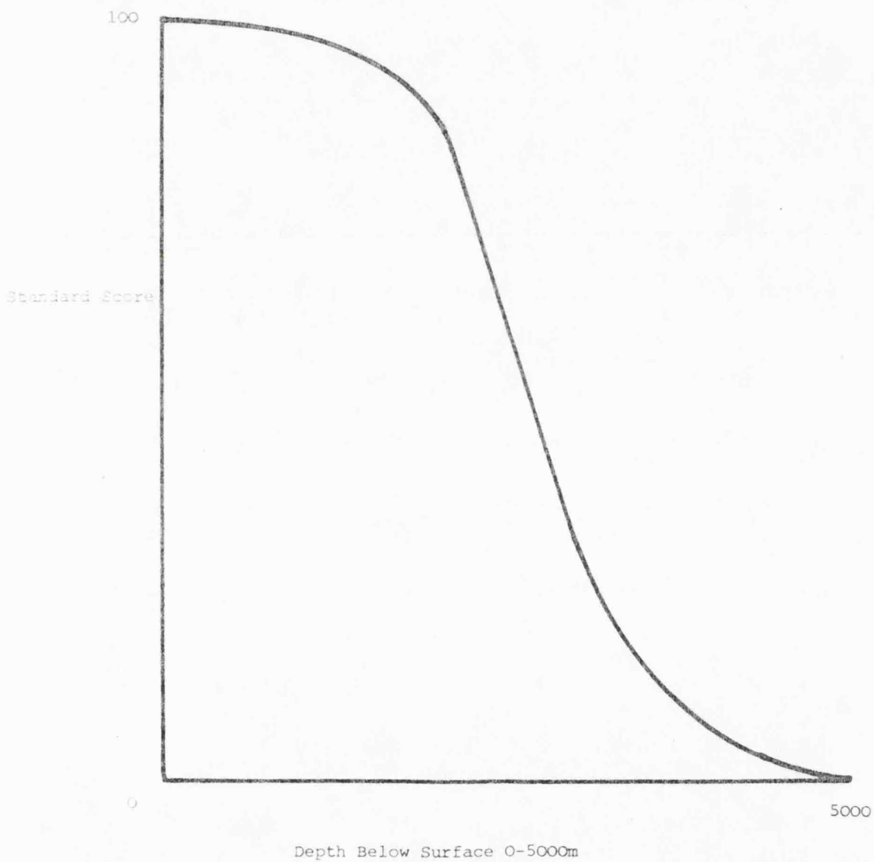
Splitting mining depths in to three categories - shallow, medium and deep - allows for a general non-linear relationship to be developed. Shallow deposits are basically categorized as "good", and deep deposits as "bad", with an approximately first order graduation between the two limits. This relationship is illustrated in Figure 5.

6.9.4 Water Conditions

The amount of water encountered will affect mining ease. On the one hand, high water inflows, such as those found in Zambia or New Guinea, make mining almost impossible, whereas complete absence of water, such as in some coal or uranium mines, make equipment availablity very poor.

However, in the case of a dry mine, water may be imported to alleviate the situation. Clearly then some water is highly desirable but too much is very bad. In the extreme, too much water can prevent

Fig. 5 Depth Below Surface - Standard Score Relationship



mining completely, whereas too little can be overcome, and some is just right.

This qualitative reasoning is reflected in the relationship shown in Figure 6.

6.9.5 Other Conditions

The above relationships cover the main factors affecting the degree of difficulty of mining a deposit. However, there are others such as the presence of gas, a particularly bad footwall, etc., which will affect mining ease. Their presence is not always assured and their impact not always major. Individual consideration of these non-standard factors would make the input to this analysis both unnecessarily difficult and tedious, therefore, a general catch-all category of "other" may be used to compensate for these minor factors. The simple first order of such a relationship is shown in Figure 7.

6.9.6 Bias

The degree of mining difficulty for a given mining method, and for a given deposit may be called "bias". This bias is the compound expression of the above described factors. The difficulty is now presented as to how to combine these factors quantitatively to produce an index of mining ease. There is no general theoretical framework of mining to

Fig. 6 Water Conditions - Standard Score Relationship

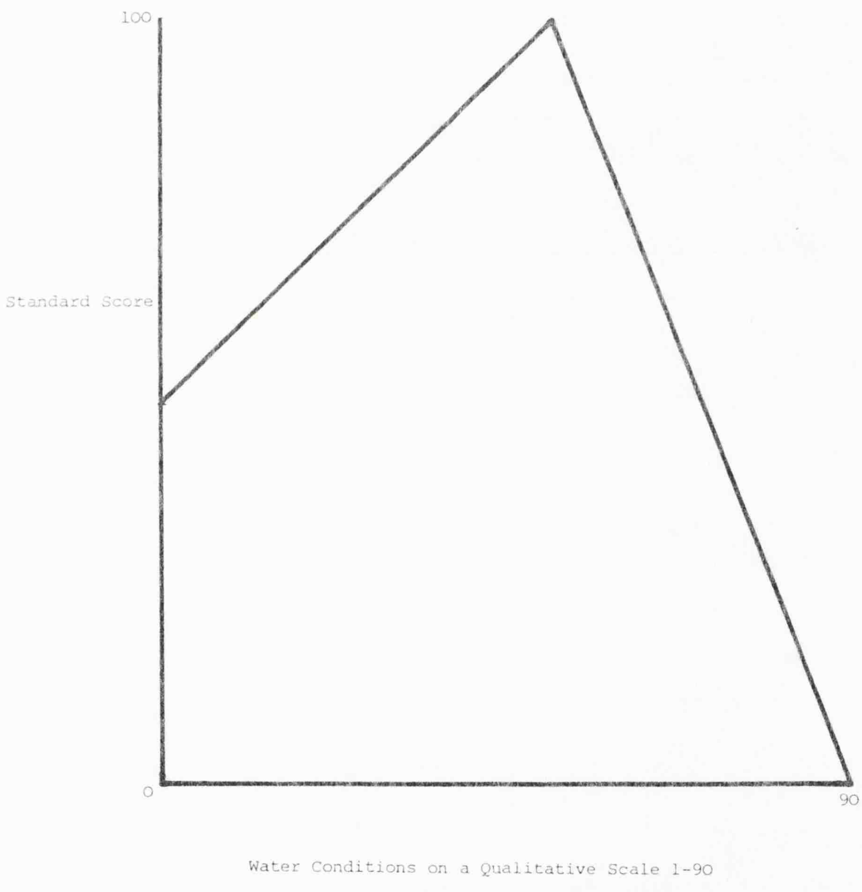
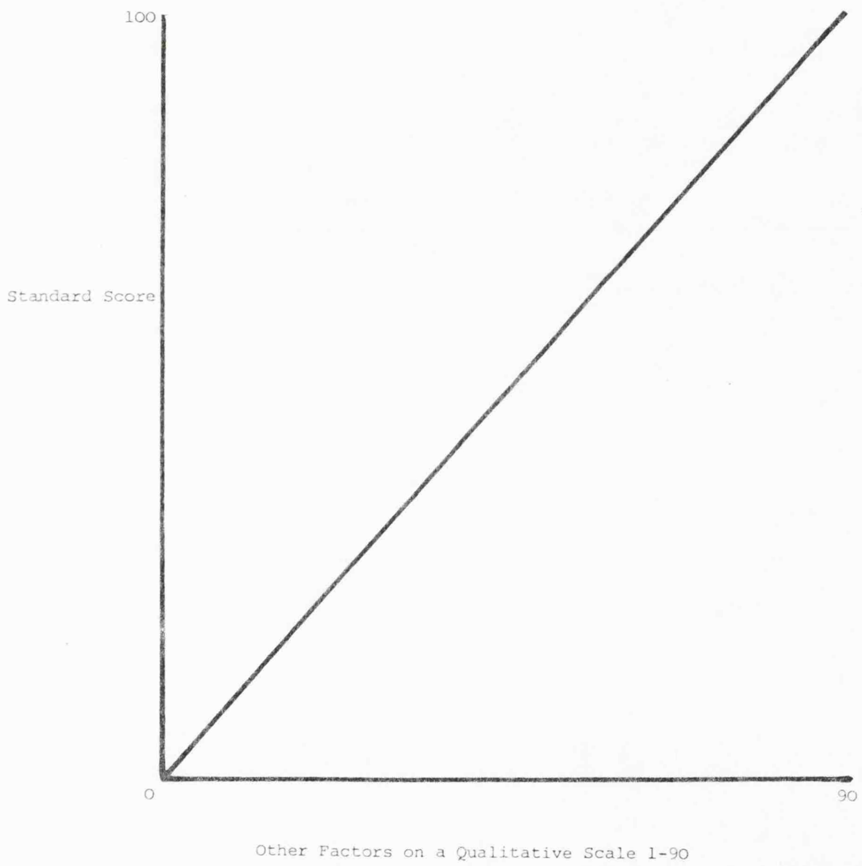


Fig. 7 Other Factors - Standard Score Relationship



assist in this problem. The factors under consideration are:-

1. Rock strength
2. Transport ease
3. Mining loss/dilution
4. Depth to deposit base
5. Water conditions
6. Other conditions

All these factors may be assigned standard scores, all measured to the same scale, and to the bases of the relationships described in the preceeding sections. Let these scores be symbolized as follows:-

Factors	Standard Score
Rock strength	S1
Transport	S2
Loss/dilution	S3
Depth	S4
Water	S5
Other	S6

In the case of open-pit mining it may be argued that the degree of dilution and recovery of a deposit is not determined by gravity flow, because the waste material is being physically removed before the ore is extracted, thus eliminating loss/dilution as a significant factor. This being the case, the bias must be calculated in one of two different ways depending on whether the deposit would be exploited by surface or underground methods.

The question still remains as to what relative significance, or weight, may be given to each of the individual factors. The bias will affect both the capital and operating costs of a project. If mining conditions are poor, productivity of individual equipment will be low, therefore, in order to maintain a given level of production more items of equipment will be needed than for a good, high productivity situation. Hence, capital costs will be higher. Similarly, under adverse conditions, operating costs will increase.

Pragmatically, capital and operating costs will vary within finite ranges for all sets of conditions. For this type of strategic analysis, high precision is not required. It may, therefore, be concluded that it is not necessary to agonize for too long on the question of the relative weights of individual factors. As no general theory exists to assist in the assignment of relative weights, all factors will be given equal significance. Moreover, bias is simply an expression, on average, of the likely difficulty that might be expected in a qualitatively defined scenario. The following definitions are therefore used for this analysis:-

Underground bias = 1.0/[(S1+S2+S3+S4+S5+S6)/300]

Surface bias = 1.0/[(S1+S2+S3+S4+S6)/250]

The resulting bias will tend to increase costs in poor conditions and decrease them in good ones, with respect to some mean value.

These average values of capital and operating cost may be obtained from published data, or from an empirical approach such as that described by O'Hara (31) or Hoskins & Green (37) or Straam (38).

6.10 Mineral Processing

The method chosen for processing ore to produce a saleable concentrate depends upon the mineralogy of the deposit being considered. For the major base metals O'Hara's (31) paper covers how metallurgy may be determined and costed. Using the empirical relationships described therein, metallurgical recovery may be related to the average mined grade, and resultant revenue thus calculated. Hence, revenue and costs may be derived and thus the economic desirability of potential targets determined.

7.0 SOCIO - POLITICAL CLASSIFICATION

At first glance it may seem that there is little connection between politics and geology, however, exploration takes place in the real world, and so potential targets must be classified in terms of socio - political attractiveness. In many cases this may well be the over-riding consideration. The main factors may be summarized as follows:-

Factor	Index Symbol
Attitude of government to capitalism	P1
Long term political stability	P2
Short term political stability	P3
Environmental impact	P4
Ecological sensitivity	P5
Employment generation	P6
Land use conflict	P7
Infrastructure status	P8
Tax policy	P9
Royalty policy	P10
Legal climate	P11
Indigenous labour skills	P12
Relative social development	P13

As was described in earlier sections these factors may be assigned values on a scale of 1 - 100, where 100 is very good and 1 is very poor, with respect to their impact upon a potential mining complex. Thus a Socio - Political Index (SPI) may be derived:-

$$SPI = (P(1) + P(2) + \ldots P(n))/(n * 50)$$

n = number of characteristics

50 is used to express SPI as a value relative to an "average" state of 1.0

This will result in an absolute number for the deposit whose significance is not at first obvious. So, again, a Relative Socio - Political Index, RSPI, may be defined by considering all potential deposits for all commodities, whose value is more descriptive. In fact, it may be useful as a first approximation, simply to derive a RSPI for each country and to assume that all deposits that lie within the borders of that state will have the same RSPI. Whichever assumption is taken the RSPI is defined as follows:-

$$RSPI(i) = SPI(i)/[SPI(i) + \ldots SPI(n)]$$

- where: i = ith commodity

n = the total number of commodities.

Thus commodities may be ranked according to their socio - political risk, by proportionally adding either on the basis of a deposit - by - deposit basis, or by adding the relative contribution made to the potential production of a given commodity by a given country.

Further the definition of General Exploration Potential may be expanded to include this socio - political component:

$$GEP(i) = [\ RECI(i) + RMI(i) + RSPI(i)\ ]$$

- where: i = ith commodity
- all other terms defined as before.

Hence, a quantitative, repeatable approach to the ranking of risk with respect to somewhat elusive phenomena is now available. Consequently, general corporate strategy can now be outlined in a broad way, and the affects of changes in assumptions on the final strategy may be measured. Thus giving greater confidence to management in the decision making process.

8.0 PRICE CONSIDERATIONS

At several points in the preceeding argument the question of price has been touched upon, but not examined in any detail. The way in which price varies has a major impact on the risk associated with making the choice to exploit commodity "A" rather than commodity "B". So it is clearly necessary to establish some way not merely to predict prices, but, more significantly, to quantify the amount of risk associated with the prediction. Commodities may then be ranked according to this risk in a useful way.

Currently, there are two main ways in which price predictions may be made:

- use of statistical techniques such as time-series analysis and regression.
- use of some form of qualitative ranking as described for socio - political risk above.

There are limitations associated with both approaches. The first alternative, normal price forecasting, is known to be inaccurate; moreover, it does not provide a useful measure of risk. The second alternative may produce a correct result, but only by chance.

There is, however, another possibility - geostatistics. So far, the theory of variograms and regionalized variable analysis has been applied largely to grade distributions. In principle, however, there is no reason why it should not be applied to any dependent variables, including price and time.

In the strict sense of the word, there is no dependency between price and time. Price does not vary just because time passes. Price varies due to the interplay of a whole host of factors that are conventionally indexed to time. Therefore, it may be deduced at this stage that, if we are treating the symptoms rather than the disease, the precision of our answer is likely to be low. However, as social science is unable to provide a quantitative theory linking price and time, we are obliged to fall back on statistical approaches. The regionalized variable technique produces the best estimator of likely grade distributions that is currently available to us. Given that price and time may be assumed dependent, then analagously it would seem reasonable to use it to predict future prices.

From a variogram we may obtain three significant parameters; the sill, the nugget and the range. In terms of price - time dependency, the sill provides an estimate of the maximum error we may expect in making predictions - all things being equal. The range gives an idea of the maximum time span beyond which we may not reasonably make predictions. The nugget value indicates how much inherent error we may expect in even our most accurate analyses. None of this information is given by conventional statistical analysis.

The analysis being descibed in this thesis should be regarded as a "steady - state" type of study. This being the case it is necessary to remove the effects of inflation from historical price data. An example of this approach will be given later.

The significance of price predictions decreases as time recedes in to the future; this is because of the influence of discounting. The revenue generated next year has less value than that same revenue generated today. Therefore, the significance of the error of estimation of future price is also decreasing with time. The rate of discounting depends upon the profit demanded from the project.

8.1 Price Ranking

Suppose that variograms have been calculated for a group of n commodity prices, and the the values of sill, nugget and range are known for each commodity.

Let sill values be, $S(i)$ = maximum risk for i

nugget values, $N(i)$ = minimum risk for i

range values, $R(i)$.

predicted average price value be $P(i)$

- where, "i" is the ith commodity.

It is first necessary to define "good" in planning terms. Ideally, a price should be stable for long periods. It should be emphasized that this is an ideal from a planning rather than a speculative point of view.

The limits of price variability are given by the sill and nugget values, in absolute terms, and may usefully be re-expressed in terms relative to the predicted average price. They are then expressed as percentages in conformity with the definitions of the other qualitative indices.

Relative Sill, $RS(i) = [\ 1.0 - (\ S(i)/P(i)\ )] * 100$

Relative Nugget, $RN(i) = [\ 1.0 - (\ N(i)/P(i)\ )] * 100$

Relative Range, $RR(i) = R(i)\ /\ R'$

-where, R' = the average range over all commodities.

Thus a Price - Time Index, for each commodity may be derived as follows:

$$PTI(i) = [\ RS(i) + RN(i) + RR(i)\ ]/[\ 3 * 50\ ]$$

- where: $PTI(i)$ = relative price - time index for commodity i.

50 expresses the PTI with respect to an "average" condition of 1.0

The relative price time index may be quantified thus:-

$$RPTI(i) = PTI(i)/[PTI(i)\ ...\ + PTI(n)]$$

- for commodities i to n

So, following the above procedure, the commodities may be ranked in terms of price as exploration alternatives. This particular ranking is useful as it makes some quantified statements about the future which may be checked as the exploration effort progresses. Such feedback may be used for subsequent modification or re-orientation of exploration activity in such a way as to reduce risk.

The General Exploration Potential, GEP(i), may be modified to include the RPTI(i) as follows:

$$GEP(i) = (\ RPTI(i) + RCEI(i) + RMI(I) + RSPI(i)\ )$$

- all definitions as previously described.

9.0 COMMODITY PROFITABILITY THRESHOLD

Most of the elements needed to start a strategic analysis have now been assembled, with two major exceptions: the expression of desire and real possibility.

For the purposes of this argument "desire" means financial requirement, specifically the achievement of a defined DCFROR. "Real possibility" means the actual deposits that are available for discovery and/or acquisition.

Actual deposits are normally characterized in terms of tons of ore at some grade, or grades. The financial desire is defined in terms of DCFROR. Clearly, in order to match the two it is necessary to express them both in the same terms.

Bearing in mind that these actual deposits must be found, and that this work will be carried out by geologists, it would seem logical to express the financial constraint in terms of tons and grade.

There are two ways in which the translation of financial units into geologic terms may be achieved:

- by conventional indirect solution

. by direct solution - a new technique.

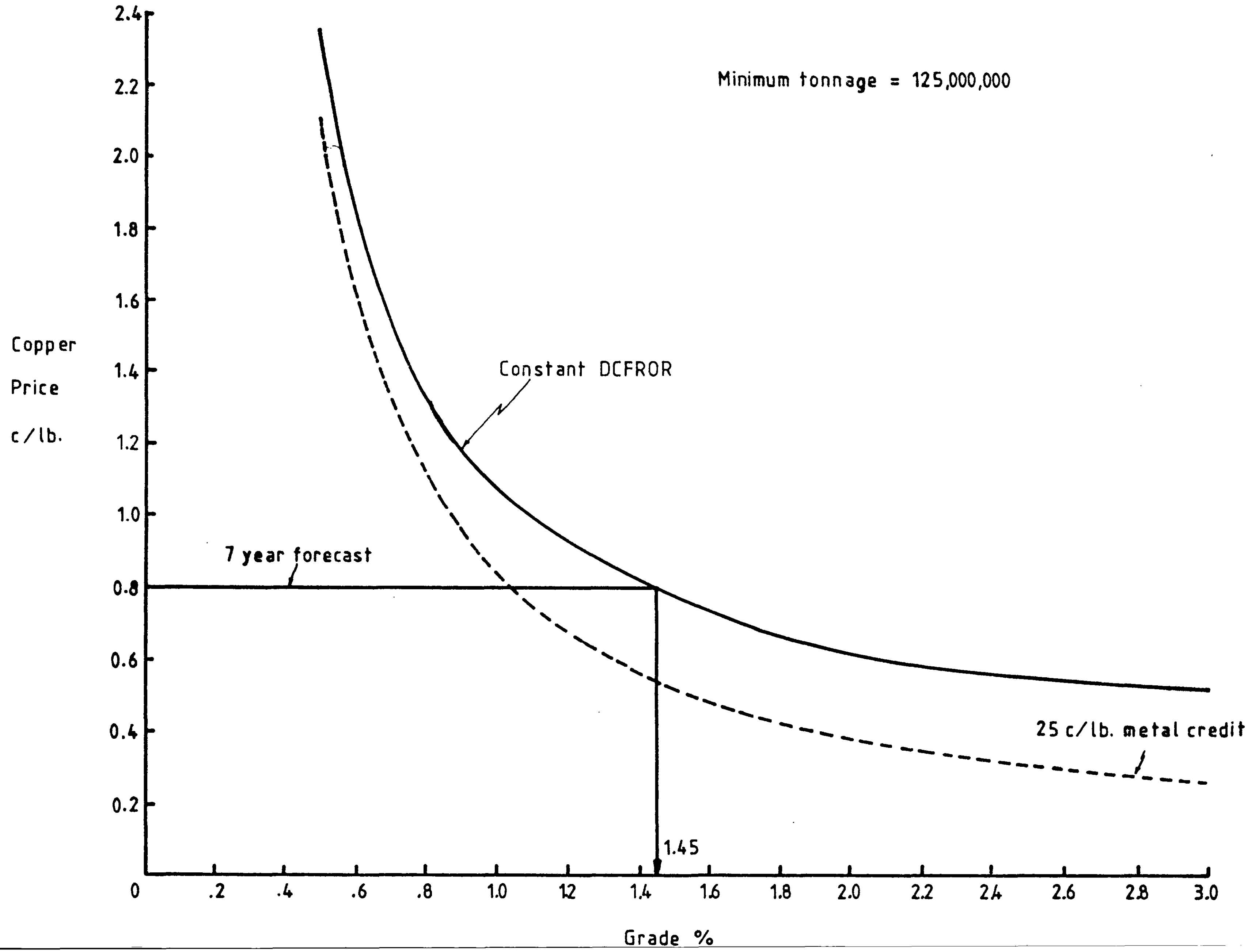
The conventional solution will be examined first as this will throw light on its shortcomings, and set the scene for the development of a new, direct solution.

9.1 Indirect Solution

The usual way rates of return are translated in to tons and grade is by performing a cashflow analysis on a wide range of real or hypothetical deposits, including measuring the sensitivity of a project to changes in deposit size, grade, net smelter return, capital cost, operating costs, and determining the DCFROR of each resultant cashflow. These DCFROR's may then be contoured in a variety of ways to show the effect of changes of each of the several variables mentioned above.

As different types of deposit are exploited in different ways, it is useful to categorize the results in terms of mining method. Figure 8 illustrates, for an open - pit copper mine, for a particular size of 125M tons, the relationship between copper price and grade for a given rate of return. Should the size of the deposit change, naturally , the characteristics

GRADE PRICE RELATIONSHIP FOR A TYPICAL USA COPPER MINE



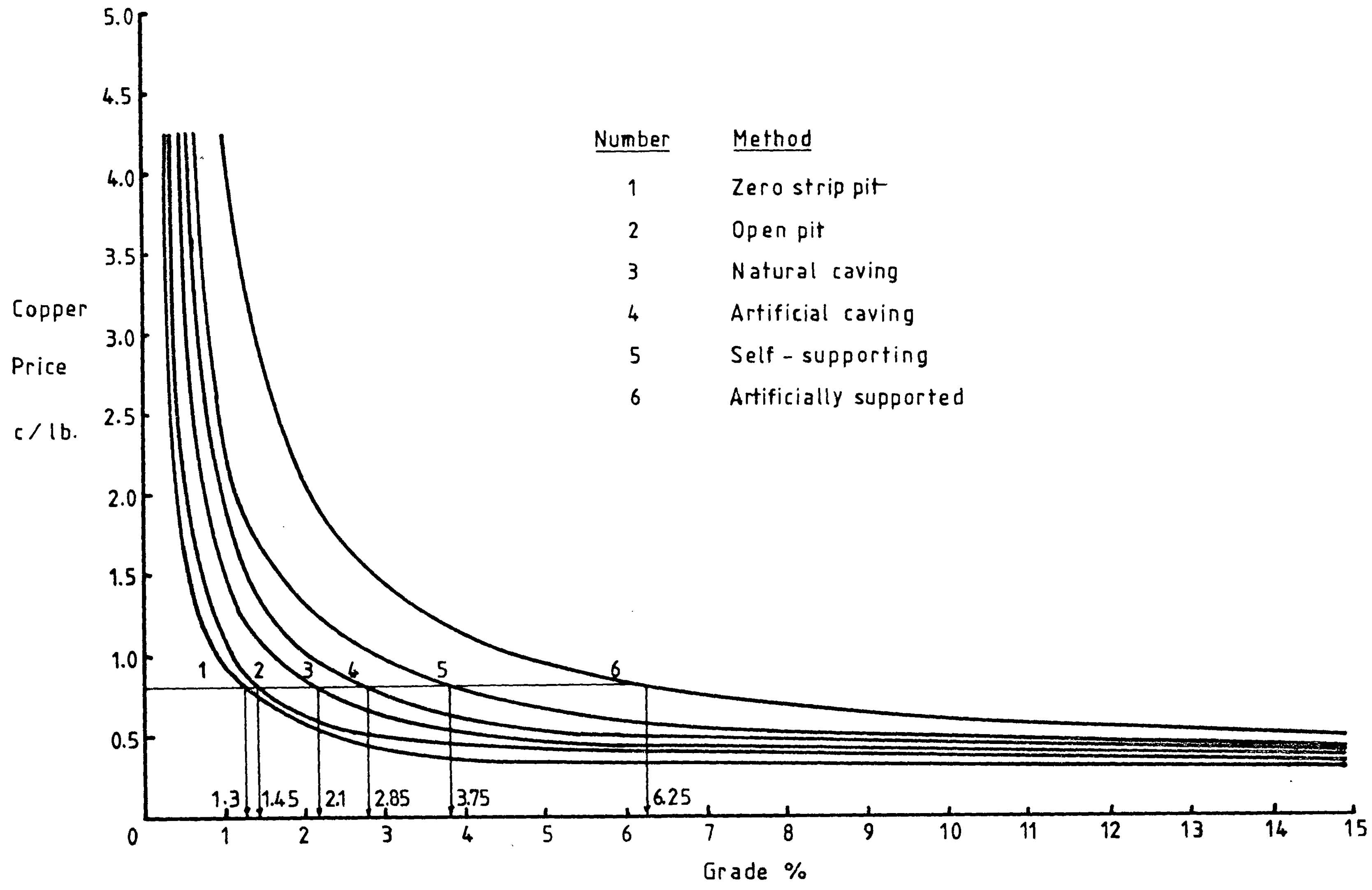
will be of a somewhat different shape; and a whole suite of such curves may be developed for various sizes and rates of return, etc.

However, the point is that certain significant information may be gained from such a relationship. If the price of the commodity for the life of the deposit is defined, we may define the average in-situ grade that is required in the deposit in order to obtain the required rate of return. So now, rather than setting an exploration target as "find an open - pit copper deposit that will achieve 15% DCFROR", we may say "in order to achieve 15% DCFROR you must find a deposit of copper amenable to open - pit exploitation that contains at least 125M tons of ore at an average grade of not less than 1.25% Cu.". This makes the life of the explorationist a good deal easier and , hence, his chances of success a good deal higher.

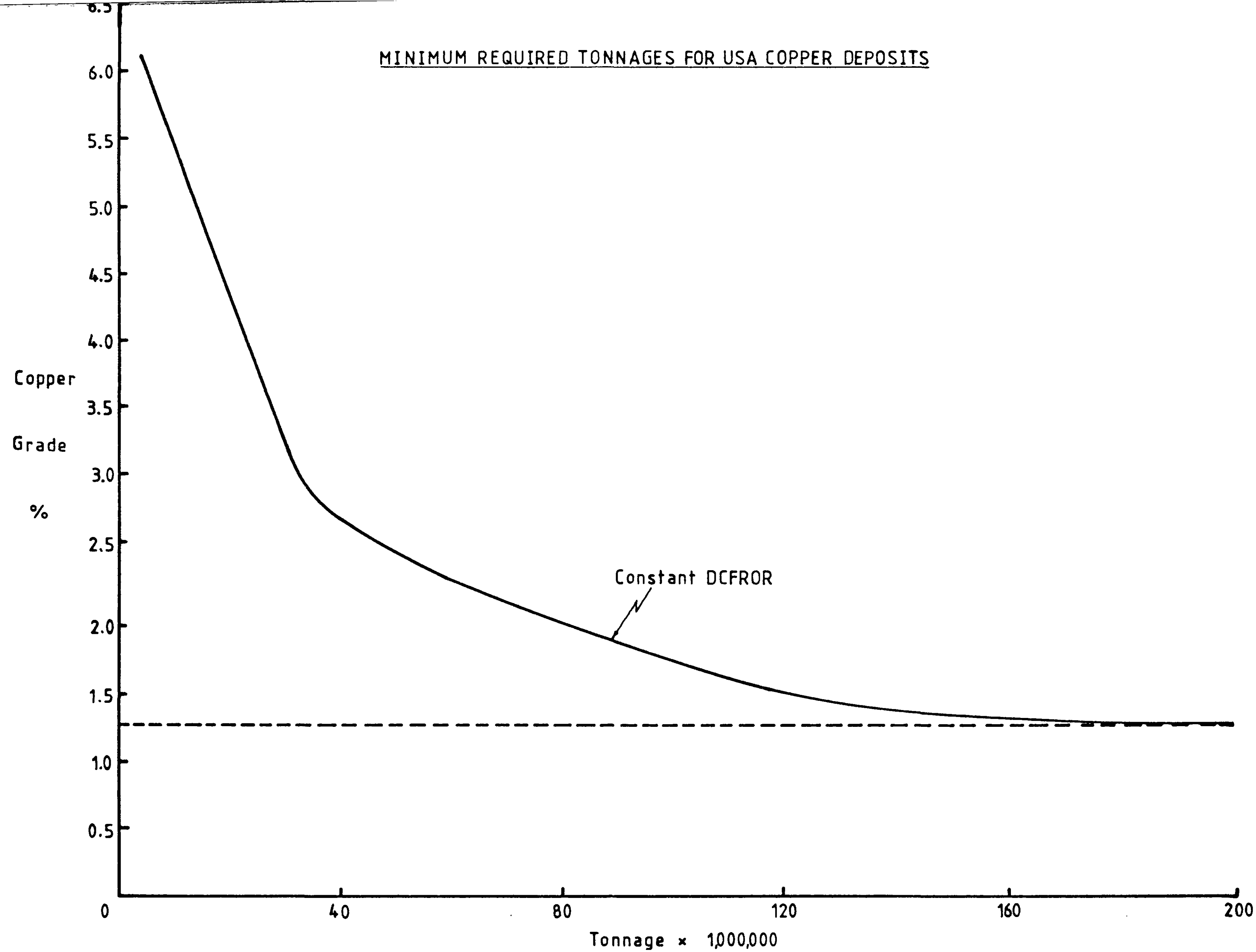
Similarly for the other standard mining methods. The results of this are illustrated in Figure 9. Given a price for the commodity, the characteristic defined in Figure 10 may be derived. This characteristic defines the relationship between grade and tons that will achieve some specific rate of return for a given commodity. Deposits having a size and quality which fall below the line on Figure 10 will not meet the stated financial goal, and would not be deemed

Figure 9

GRADE PRICE RELATIOSHIPS FOR USA COPPER MINES



MINIMUM REQUIRED TONNAGES FOR USA COPPER DEPOSITS



suitable as exploration targets. The line on Figure 10 represents a barrier which must be exceeded in order for a deposit to be acceptable. Therefore, this relationship may be defined as the Commodity Profitability Threshold, CPT.

From Figure 10, it can be seen that if a company wishes to make a certain DCFROR, it must seek deposits of copper having a grade greater that 1.25% Cu, regardless of size. Such information is fundamental in a rational decision making process.

9.2 Direct Solution

It is apparent from the above discussion that a great deal of work is involved in a conventional approach to the problem of translating financial units in to geologic units. This is because the mechanics of the indirect solution are inefficient. In order for a rational decision making theory to be viable, it must be relatively easy to use, or its utility becomes sub-marginal. The clear necessity is, therefore, to simplify the mechanics of the process.

How far is it reasonable to go in simplifying the process ? Obviously, over - simplification will produce useless results, whereas, over - elaboration is already the hallmark of the conventional solution.

9.2.1 Decision Making in Perspective

The decision theory being developed is aimed at defining an exploration strategy that will increase a company's chances of success.

At any point in the life of a project risk is inherent. Figure 11 illustrates the stages in the life of a typical project in terms of risk. The general form of the graph in Figure 11 is well known for projects in general, and has been described by Kennedy (40) for the mining industry in particular. At the moment of concept, the risk of not actually putting a mine into production that will produce an acceptable rate of return is maximum, and total. The function of the exploration process is to reduce that risk to a point where a decision may be made as to whether to turn this prospect in to an actual mine.

During the development process much detailed engineering and construction work is carried out, and the risk is being continuously reduced. However, even during the actual production process there is still a significant risk associated with the project as uncertainty exists about precisely what grade, tonnage, operating conditions, price, etc. will be encountered by the mine.

From Figure 11 it is clear that it is unreasonable to look for greater precision in the theoretical approach to the problem than is demanded by real life. The phase to which the theory elaborated in this thesis applies requires a precision of between 25 - 100 %. The position of the lower boundary is somewhat debatable, it could be argued that exploration takes place in the 50 - 100 % range.

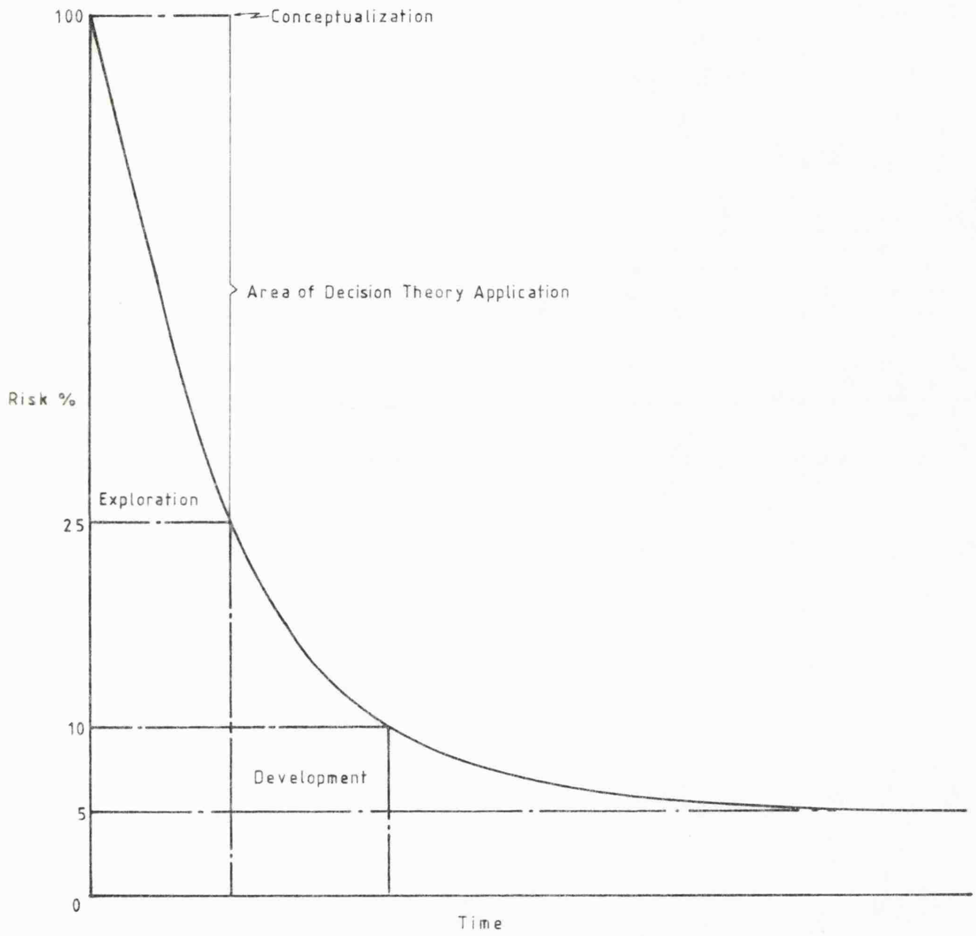
Hence, it may be concluded that first order, linear assumptions are quite adequate for the task to be undertaken.

9.2.2 The Principles of the Direct Solution

In conventional cashflow analysis the DCFROR is defined as that discount rate at which the cumulative net present value of the cashflow is zero. Hence, for that discount rate the cumulative NPV of the capital expenditure during the pre-production phase of the project is equal to the cumulative NPV of the profit made during the production life of the project.

Figure 11

RISK IN THE LIFE OF A PROJECT



That is:

CPNV"X" Capital = CNPV"X" Profit ... (1)

- where: X = discount rate

CPNV = cumulative NPV.

Considering the terms in equation (1) individually:

- . capital - may be defined or calculated
- . X - may be defined
- . profit = revenue - cost ... (2)

Considering the terms in equation (2) individually:

- . cost = all cost charged to the project in any given year.
- . revenue = net revenue at the mine calculated as follows:

Revenue = (In-situ grade - dilution) * Mining Recovery * Price * Tons * Processing Recovery ...(3)

Considering the items in equation (3) individually:

- grade is known, it must fall in the range 0 - 100 %
- dilution is known, again it must fall in the range 0 - 100 %
- mining recovery is known, it too must fall in the range of 0 - 100 %
- processing recovery is known, 0 - 100 %
- price, or net smelter return, may be defined in any range depending upon predictions.
- Tons, unknown.

Hence, it can be seen that "tons", or deposit size is the only unknown. So for a specific profit level and for defined ranges of capital, pre-production life, grade, etc., it is possible to solve for the deposit size.

Fundamentally, the direct solution is simple; everything but the size of the deposit is known or may be estimated quite readily. The details of the workings of the direct solution are given in Appendix A.

Computationally the direct solution is easy because the iterative determination of DCFROR is not required. Thus the simplicity of the direct approach makes the derivation of a Commodity Profitability Threshold a relatively trivial matter. In turn, this makes the whole philosophical approach described in this thesis not merely acaedemically interesting, but, practically, viable.

9.3 Price - Time Definition

In order to perform either a direct or an indirect solution to the problem of the drivation of a CPT a price - time forcast is needed. As stated in Chapter 8 this analysis will be performed on a steady - state basis.

For a commodity of interest a maximum time span for the projection must first be defined. This may be achieved either by picking some number or by using a rule-of-thumb such as the one suggested by the Northwest Mining Association (37):

LIFE = 20% * [(SIZE)**0.25]

- where: LIFE = operating life of the mine

SIZE = size of the largest deposit of a given commodity in tons.

Given the operating life and adding say three years for a pre-production period, will produce a total, maximum project life for that commodity which defines the period for which a price projection is required. This process is repeated for all commodities of interest, and price projections may then be made for periods which have some geologic meaning.

10.0 COMMODITY SOURCE PROFILE

Up to this point the arguments presented have been oriented towards defining what it is desired to achieve in geologic terms. Merely defining what is necessary is not really very helpful in terms of making a decision about exploration targets. In order to make rational decisions it is also necessary to define what is actually available, so that the two sides of the problem may be balanced.

In order to compare two quantities; desire and reality, they must first be measured in the same units. Financial desire is now measured in terms of grade and tons, specifically with a grade - tonnage curve. Clearly then, geologic reality must be expressed in the same way.

The problem becomes to produce, for each commodity, a characteristic curve which describes in grade - tonnage terms the available sources of that commodity. Such a curve may be called a Commodity Source Profile, CSP. This profile will, of course, be independent of any technical or financial contstraints.

Fortunately, grade - tonnage curves are commonly used to describe deposits, so the construction of a

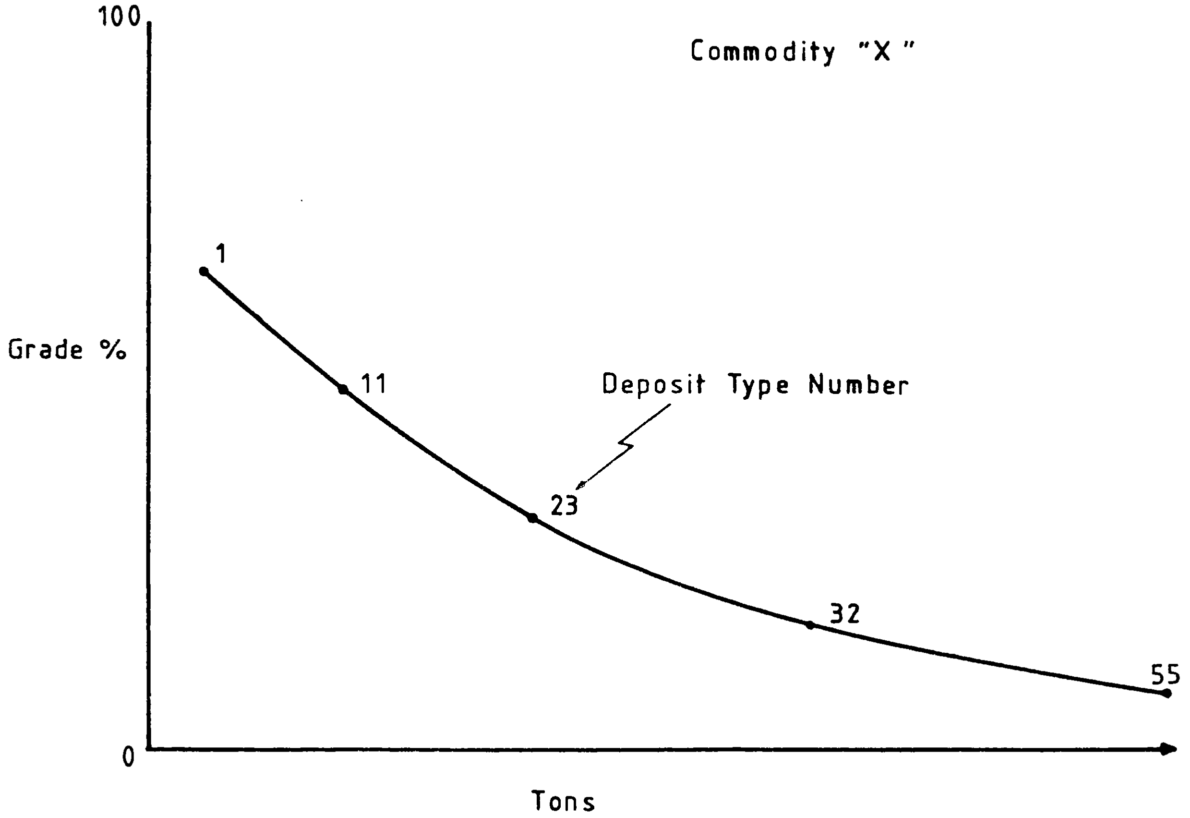
CSP is not overly taxing.

Not all commodities occur in all grade - size combinations of all possible deposit types, so it is reasonable to expect fairly distinctive CSP's for different commodities. In order to build a CSP for a given commodity the types of deposit in which that commodity occurs as a primary component must first be identified, and then grade - tonnage values assigned to each of these types. The general shape that may be expected in a CSP is shown in Figure 12.

The Deposit Type Numbers, DTN, represent which particular type of deposit gives rise to the specific commodity "X". By implication , referring to Figure 12 ,"X" does not occur in DTN 2 - 10, etc.

Copper, for instance, may occur as a porphyry, contact metamorphic or stratiform type of deposit. In the form of a porphyry, it may average 500M tons @ 0.30% Cu. Such an estimate would define say point 55 in Figure 12. Obviously, grade - tonnage estimates for a given deposit type will vary, and the way in which this may be dealt with is described later. For the time being establishing the concept of a CSP, and describing it's derivation is sufficient.

Figure 12 COMMODITY SOURCE PROFILE



11.0 FUNDAMENTALS OF DECISION MAKING

We have now derived, in the same units, the two basic elements needed in decision making - reality and desire. All that is now necessary is compare these two, and a decision may be made as to what is a reasonable exploration target, and hence, an overall strategy developed.

This process is illustrated in Figure 13.

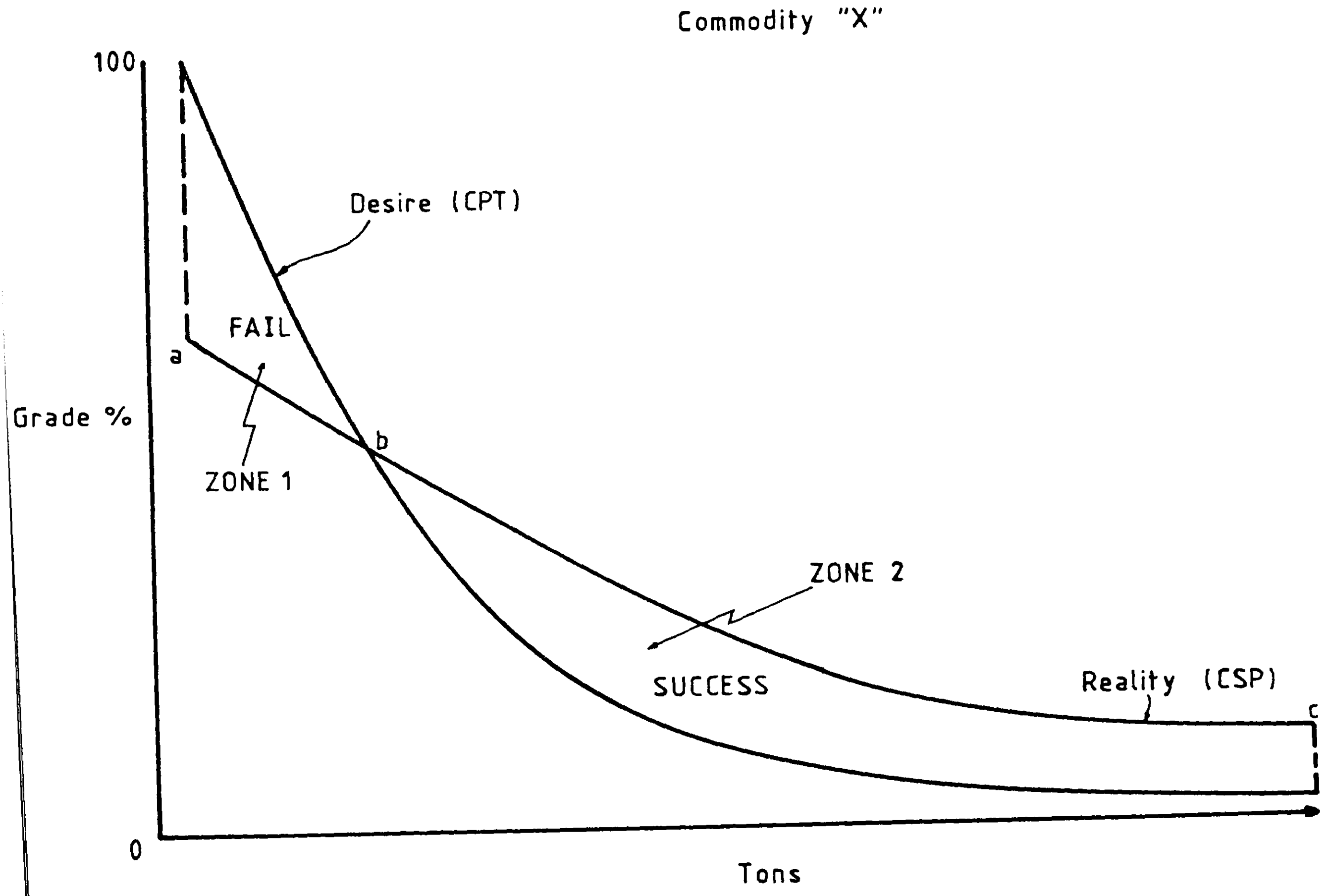
For successful exploration, or indeed any other activity, the results which are really possible must be equal to, or exceed, those results which are actually desired. Plotting the CSP and CPT on the same basis and applying this definition of success produces the decision process shown in Figure 13.

In zone 1, desire exceeds reality, therefore, by definition failure must ensue. Conversely, success is assured in zone 2.

Therefore, given that a new prospect is typified by grade and tonnage, a decision can be made immediately as to its utility as an exploration proposition. Conversely, a set of characteristics may be defined from this graph which can be used as minimum target constraints for all possible grade -

Figure 13

DECISION PROCESS



tonnage combinations on the line bc.

11.1 Overlap Index

In addition, the chance of being successful, should it be decided to explore for commodity "X", is equal to the proportion of the characteristic for which reality exceeds desire. Referring to Figure 13, one way in whch this may be measured is by considering the overlap of the CSP on the CPT. This may be characterized as an Overlap Index, OI. Where:

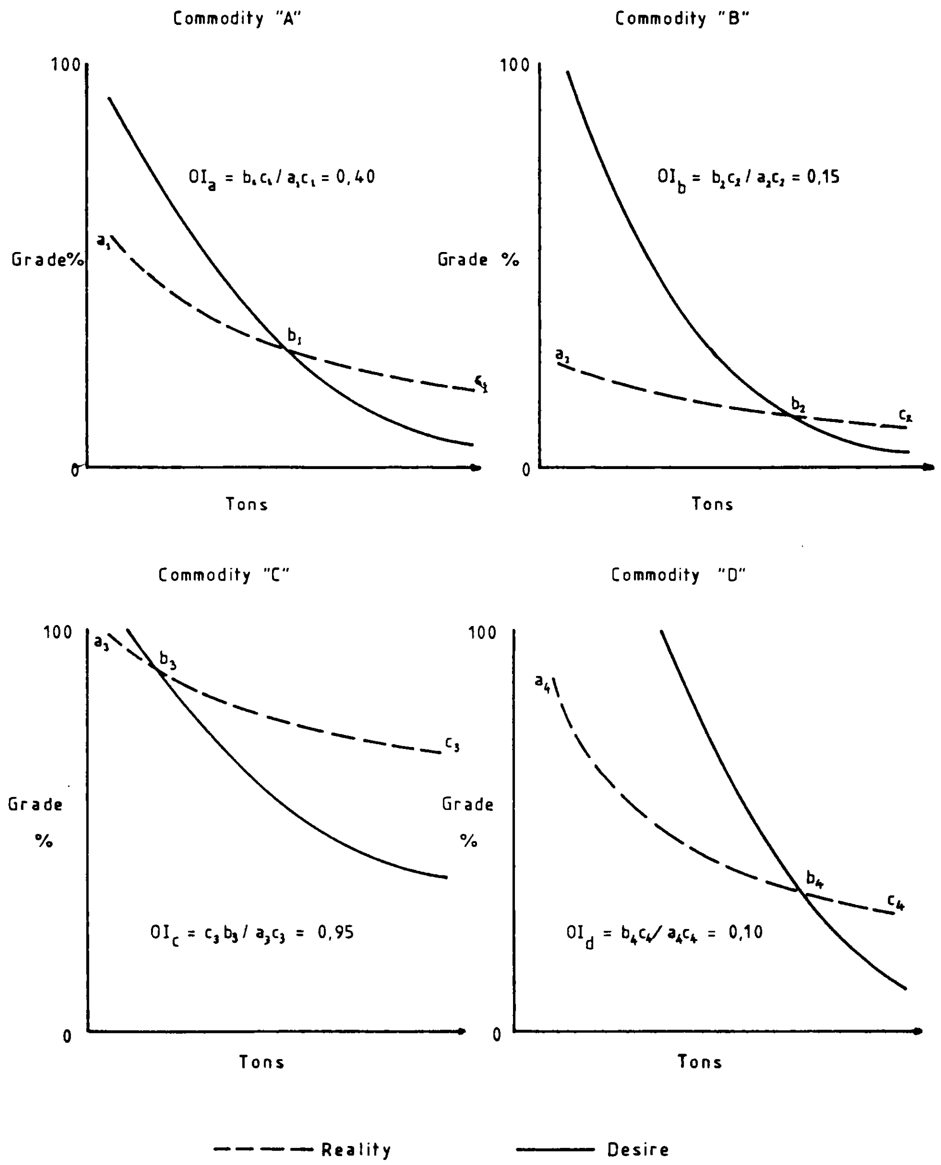
$$\text{Overlap Index} = bc / ac$$

11.2 Commodity Comparison

If this evaluation is repeated for several commodities then strategic comparisons may be made of the likely relative chances of success of one commodity compared with another. This process is illustrated in Figure 14.

Using the information from Figure 14 the utility of each commodity may be used as a basis for ranking.

Figure 14 COMMODITY COMPARISON



Commodity	Overlap Index
D	0.95
A	0.40
B	0.15
C	0.10

This ranking means that exploration for commodity "D" is more likely to produce success than exploration for commodity "C". The next question is "how much more likely ?". As a common index has been used it is possible to answer this merely by recalculating these probabilities on a relative scale.

11.3 Relative Overlap Index

The Relative Overlap Index, RELOI, may be defined as follows:

$$RELOI(i) = OI(i) \;/\; [\; OI(i) + \ldots + OI(n) \;]$$

- where: i = ith commodity

n = total number of commodities.

So in this example:

Commodity	Relative Overlap Index
D	0.59
A	0.25
B	0.09
C	0.07
Total	1.00

This relative ranking then provides an explicit, quantified assessment of how much effort should be expended in looking for each of the different commodities under consideration, not just in terms of the chances of finding a deposit , but in terms of actually being successful in generating a minimum acceptable profit form an eventual operation.

This then is the information that makes rational decision making possible, because it answers directly the question fundamental to the existance of the organization.

This approach also has the property of not over emphasizing the value of a specifically attractive commodity, whilst at the same time not eliminating commodities which "on average" do not show promise but which do have the potential, albeit limited, to produce the occasional bonanza. In other words, to each its due, but only to the extent of its relative promise.

The RELOI may be used to divide an exploration budget. For example, if a budget of $10M were available, then a rational division of this money in terms of RELOI would be:

Commodity	RELOI	Budget, $
A	0.25	2,500,000.
B	0.09	900,000.
C	0.07	700,000.
D	0.59	5,900,000.
	Total	10,000,000.

So, in principle, it may be stated that it is worth spending $2.5M looking for commodity "A", but only $0.7M looking for "C".

While this is better than no knowledge at all, it still does not help the people who actually have to find "A". The information generated so far is too general, expressed in unhelpful terms and based upon single point estimates of the input values and assumptions. What would really be useful would be a refinement of this technique so that specific deposit types at known grade and size ranges necessary for success are identified, for expected variations in grade, price, etc. These refinements will now be addressed

12.0 DEVIATION IN THE COMMODITY PROFITABILITY THRESHOLD

To this point only single valued estimates of input variables have been used. This has been done deliberately in order to allow for the explanation of concepts in a clear and simple manner. However, for this approach to have relavance to the real world a way to deal with uncertainty must be found.

The uncertainty arises because of lack of knowledge about the values of many inputs, specifically; capital cost, deposit grade, commodity price, operating parameters and operating costs. In addition, varying the financial requirements of the corporation will cause a redefinition of acceptable targets.

A two step approach is taken to the solution of these problems:

- first, a qualitative description of the affects of these changes.
- second, a quantitative analysis of the changes.

12.1 Changes in the Required Rate of Return

The initial selection of a suitable rate of return will depend upon two factors - the amount of risk associated with the project, and the nature of the analysis being performed.

If the analysis is using inflated values for cost and price, then a higher rate of return will be demanded than if a constant value analysis is being run. An idea of just how low a constant value rate of return might be can be obtained by consideration of the interest rates in the West at the moment (1982).

The rate of interest for long term lending is currently about 16%. On the other hand inflation is running at about 14%. This indicates, that for a long term project like a mine, a reasonable constant value rate of return would be about 2%, whereas an inflated value analysis would demand at least 16%.

The risk associated with an exploration or mining venture is of course greater than that a bank exposes itself to when accepting a long term loan: therefore, this extra risk would be reflected in a higher than minimum demanded rate of return before investment in a mining or exploration project could be justified. Precisely what the demanded DCFROR should be is not an easy matter for the company to decide. Therefore, the decision making logic used, must allow for the analysis to be easily repeated at a variety of rates so that managment can find out just what the maximum potential DCFROR will be, and to measure the effects of changing the demanded DCFROR on the overall strategy.

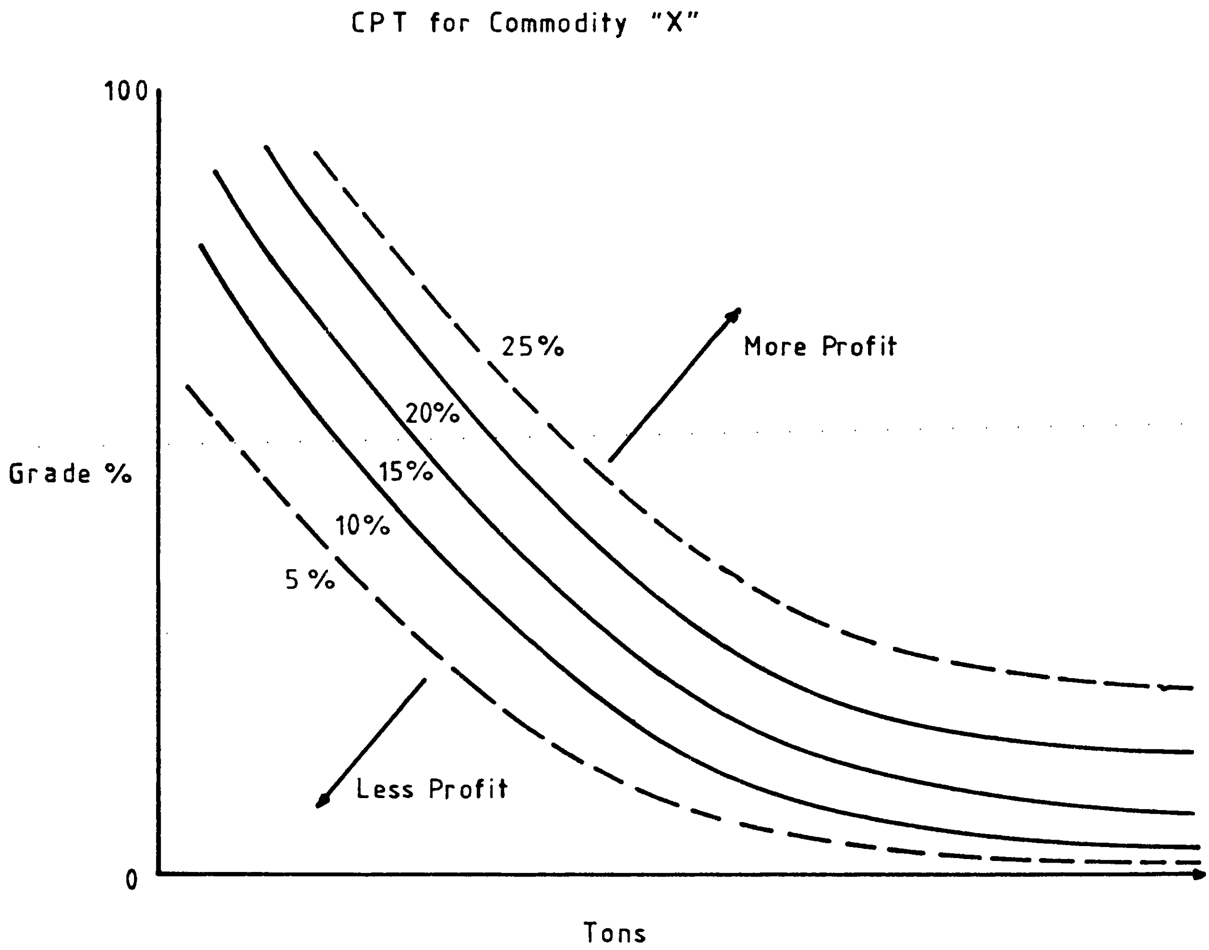
The effect of changes in the desired return is shown in Figure 15. The shape of the graphs on Figure 15 may be generated intuitively. Clearly, higher profits demand higher grades and tonnages; the converse is equally true.

12.2 <u>Changes in Capital Cost</u>

It will be appreciated that certain capital costs will be a function of the size of the deposit, whereas some will depend upon the depth to the deposit from surface. So a three - dimensional plot is really needed to visualize profit changes accurately. Such a plot is given in Figure 16.

In Figure 16 the line "AB" is not parallel to the depth axis because increased depth will mean higher capital costs, which in turn will require a higher grade - tonnage combination to repay.

Figure 15 EFFECT OF CHANGES IN REQUIRED DCFROR ON CPT

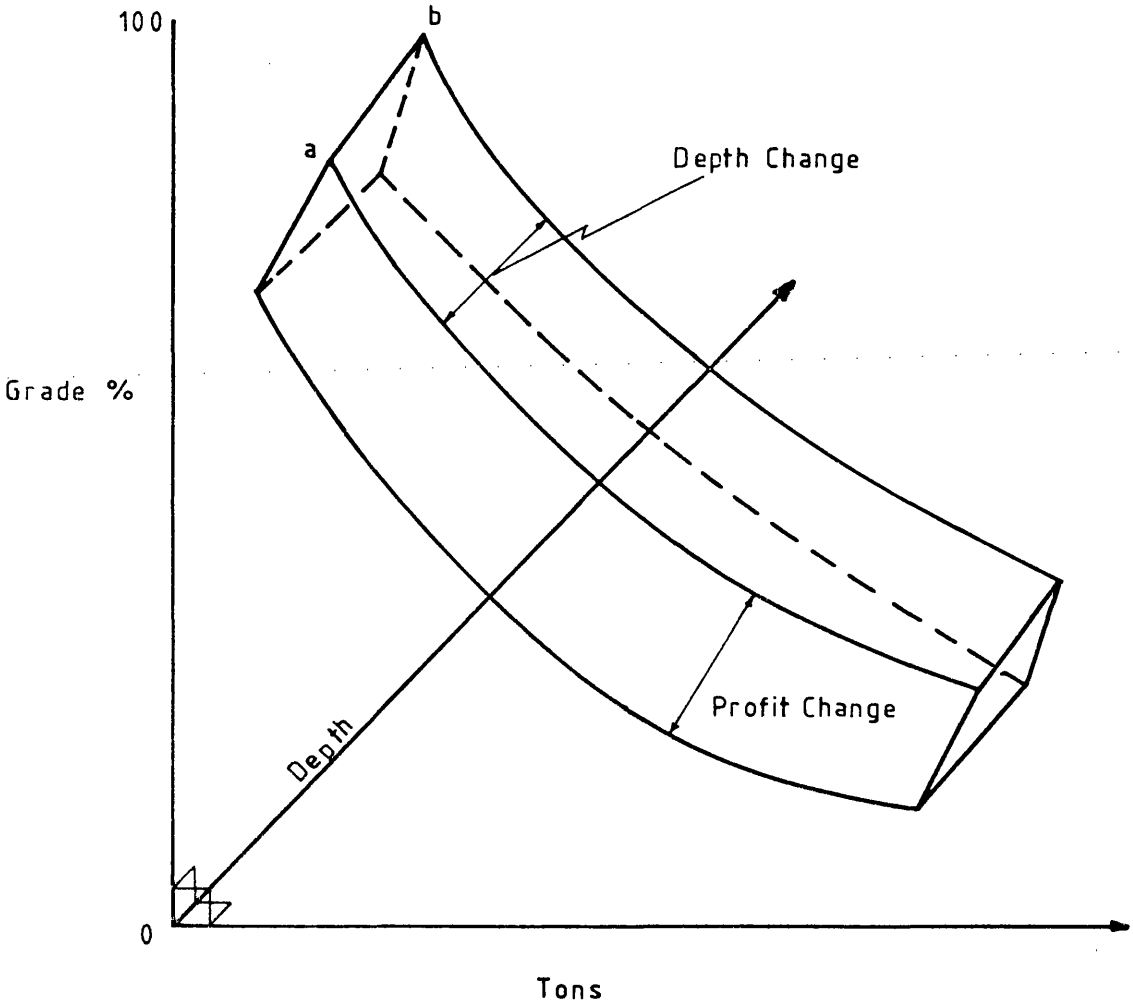


Note: Net Smelter Return Constant

Operating Costs Constant

Operating Parameters Constant

Figure 16 EFFECT OF CHANGES IN DEPTH ON CPT



Note: Net Smelter Return Constant

Operating Costs Constant

Operating Parameters Constant

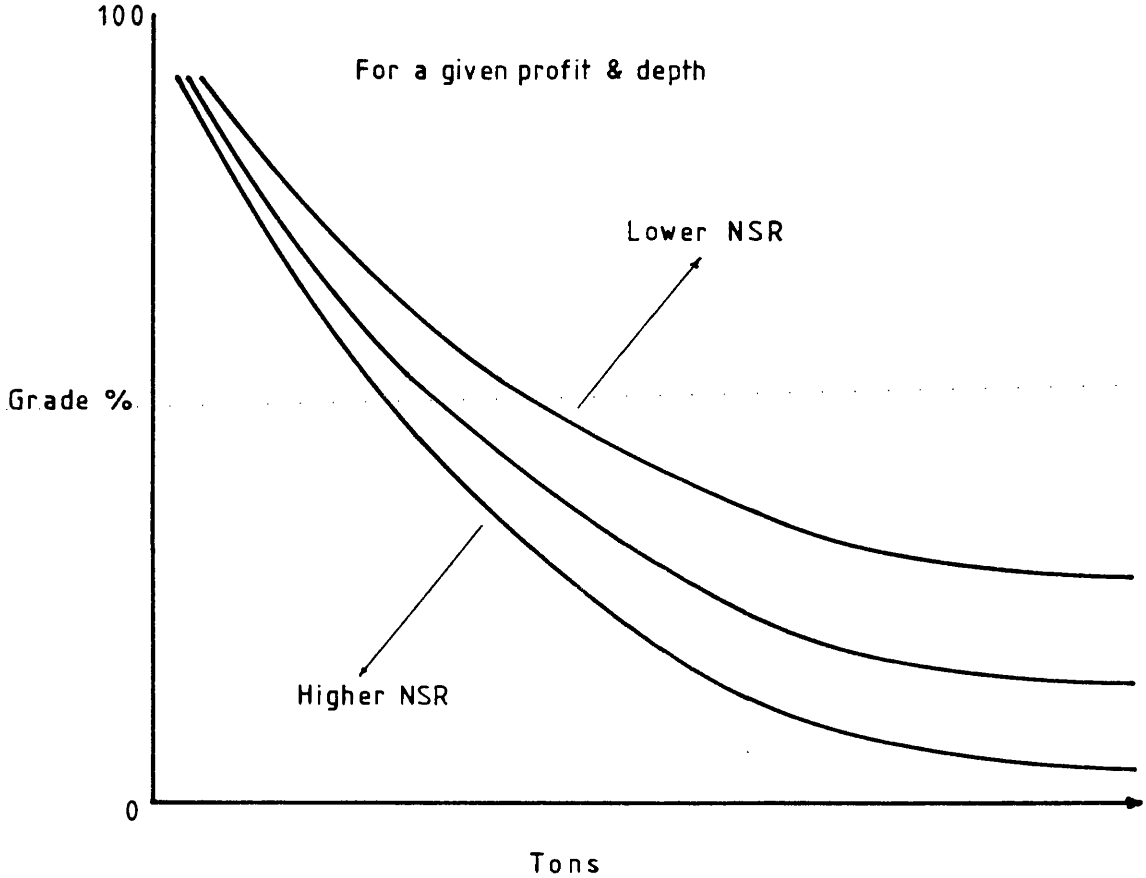
So far variations in only two of the constraints - profit and capital - have been considered, and all three graphic dimensions have been used. Not only that, but a rather difficult complex graph has resulted. It becomes, therefore, necessary to fix the variations of these constraints in a different manner. For the sake of clarity, profit and capital cost will be fixed for the moment. This will eliminate the third dimension of the graph. The resulting two dimensional graph will represent the situation at a given profit level and depth. Similar characteristics, of course, could be generated for other depth - profit combinations.

12.3 Changes in Net Smelter Return

Variations in the net smelter return result from two main causes - geographic location and market volatility. If the deposit is in a remote location with respect to the smelter, higher transportation charges will accrue for the conncentrates, which will thus reduce the net smelter return. Obviously, any changes in the market will show directly in the NSR. The effects of such changes are shown in Figure 17.

Again the derivation of Figure 17 is fairly obvious, the lower the NSR, the higher must be the

Figure 17 EFFECT OF CHANGES IN NET SMELTER RETURN ON CPT



Note: Operating Costs Constant

Operating Parameters Constant

grade - tonnage combination to offset this, and vice-versa.

12.4 Changes in Operating Cost

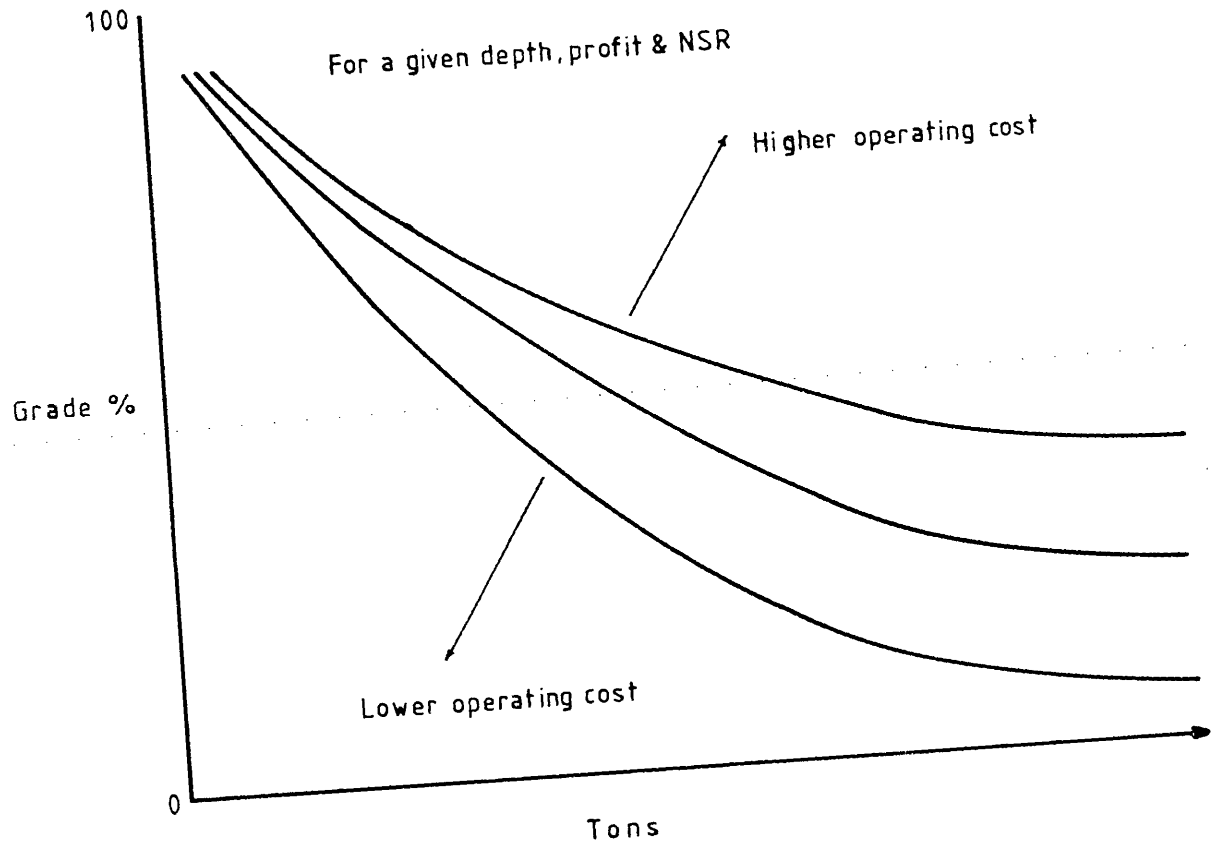
Changes in operating cost will occur due to increased cost of labour and supplies, changes in royalty and taxes, and variations in actual operating conditions themselves. The effects of these changes are shown in Figure 18, note also in this case that net smelter return too is fixed. So there will be similar characteristics for each depth - profit - NSR combination.

Considering Figure 18, higher operating costs will have to be offset by higher grade - tonnage combinations, and vice-versa.

12.5 Changes in Operating Parameters

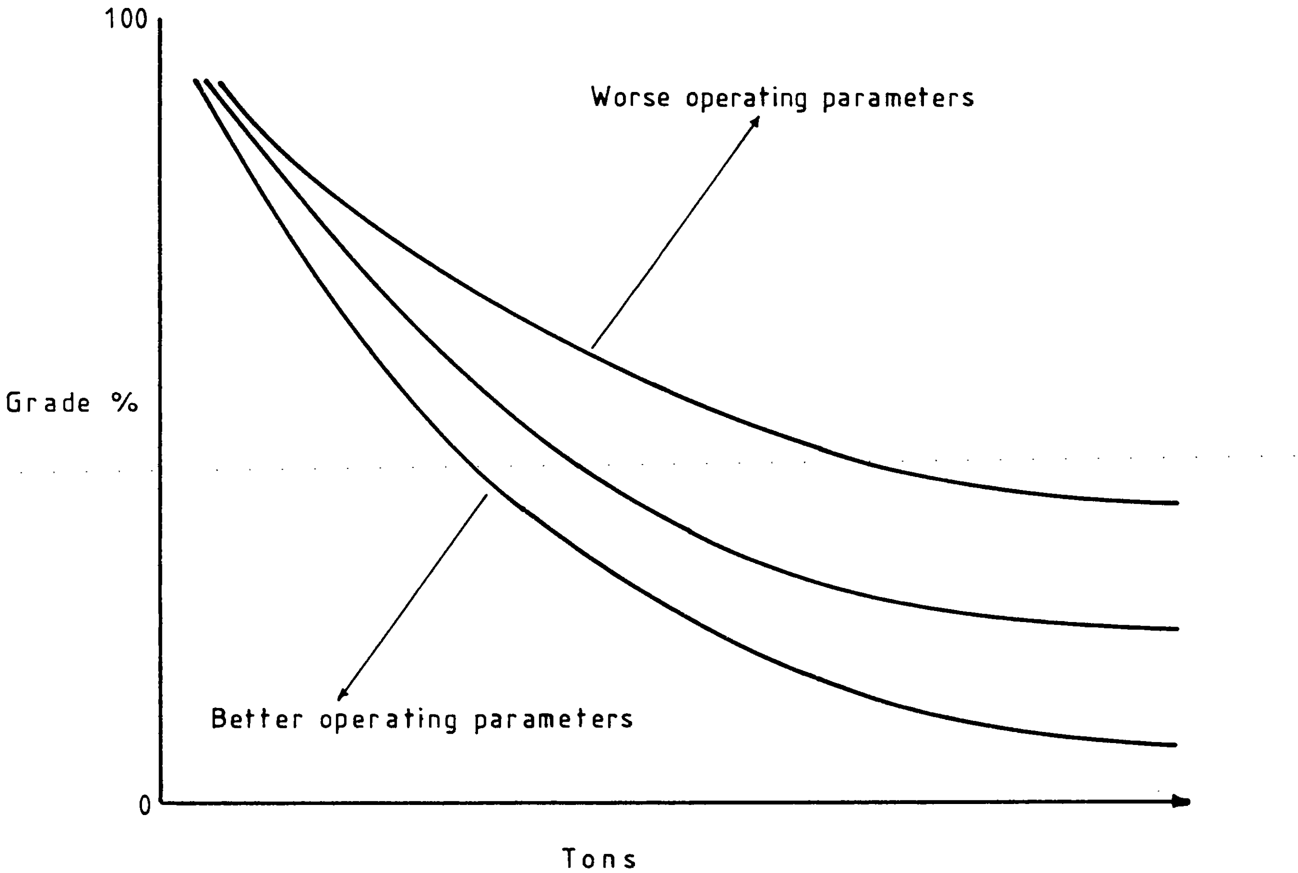
These will occur because machinery may not always perform at a constant level of efficiency due to wear, change in ground conditions, operator skill and so forth. The affects of such changes are shown in Figure 19. In this case operating costs have also been fixed.

Figure 18 EFFECT OF CHANGES IN OPERATING COSTS ON CPT



Note: Operating Parameters Constant

Figure 19 EFFECT OF CHANGES IN OPERATING PARAMETERS ON CPT



For a given depth, profit, NSR, & operating cost

12.6 Net Effect

As may be appreciated from the preceeding descriptions, the number of possible combinations of all factors is large. To reduce these to a managable number consideration must be given to the actual decision making process itself. In this process the required profit is defined, therefore, this element of variability is removed.

From conventional cashflow analysis it is known that the remaining variable groups - capital, operating cost, operating parameters and price - do not have equal impact upon the results. Their order of impact may be listed as follows:

- net smelter return
- capital cost
- operating cost & parameters.

Considering capital cost, part of this cost is a function of size, and part of depth, hence both become dependant rather than independant variables. This is futher explained in Appendix A.

Regarding operating parameters, mining recovery only appears in the calculation when translating from mineable to in-situ reserves as a linear function of tons. So it too becomes a dependant variable. Moreover, for a given mining method, applied to a particular deposit type, it may be assumed that the likely mining recovery will vary within a relatively narrow range, and hence the significance of the uncertainty associated with the estimate of mining recovery is small.

Processing recovery does have a relatively dramatic affect on the CPT as it directly affects net grade. However, for a given mineralogy it is possible quantitatively to link processing recovery to head grade, and thus turn it in to a dependent variable. These linking functions are usually empirically generated, but if the sample set is reasonably large then the precision of these relationship should be sufficient for the needs of this analysis.

Similarly, assuming that mining bias has been calculated in the way outlined above, the operating cost and recovery for mining should also have been estimated with sufficient accuracy for this requirement.

For processing cost, this will also be linked to the operating parameters via head grade and capacity, and again empirical relationships exists that will turn mineral processing operating cost in to a dependant variable.

So, of the constraints identified above, this leaves only net smelter return as an independent variable. Methods for determining reasonable values for this variable were discussed in detail above.

Summarizing, the proposed approach will be similar to the conventional sensitivity analysis commonly carried out in conjunction with cashflow modelling. The least significant independent variables are initially fixed, and investigation made in to changes in the more significant variables. Once their behaviour has been understood, the significance of variations in the lesser variables is studied. Thus in a stepwise process, a full understanding of the characteristics of the CPT of a particular commodity is built - up.

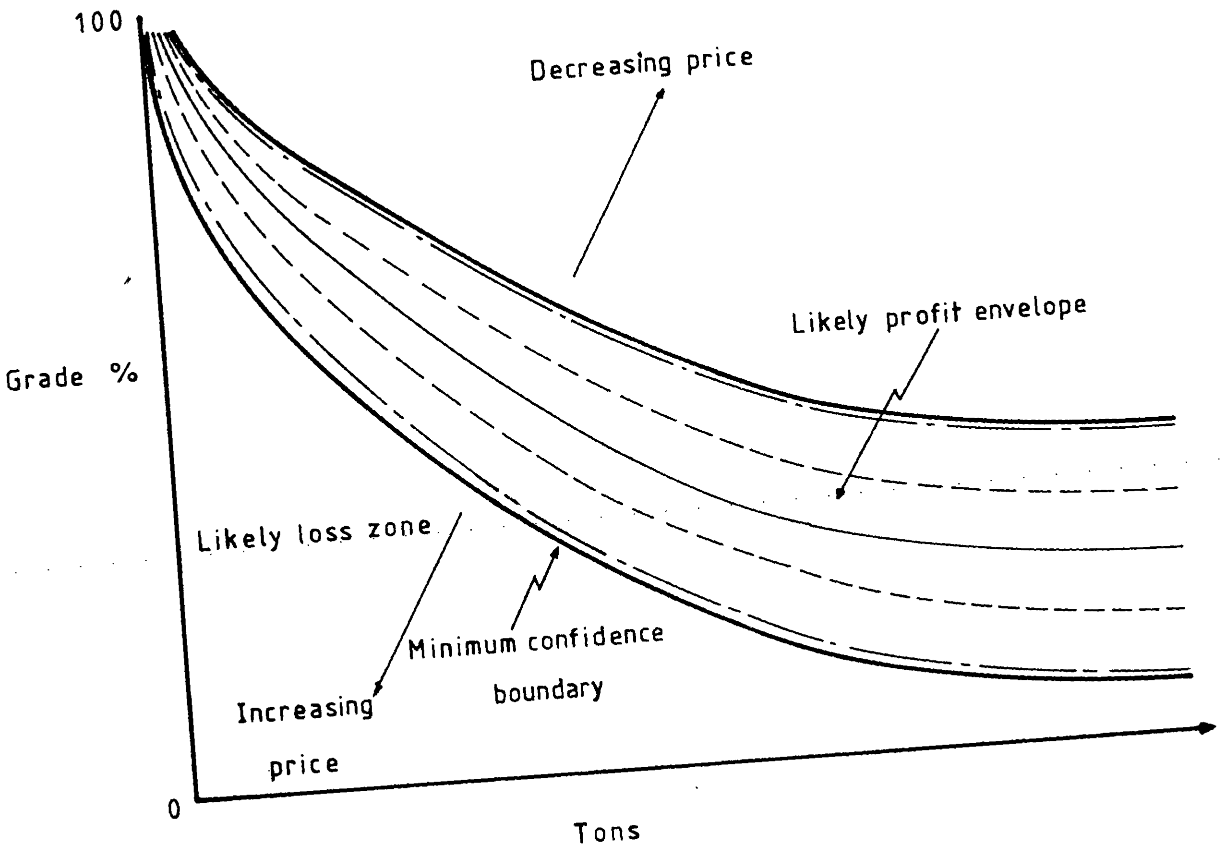
12.7 Quantified Significance

Under the terms of the above argument, NSR or price, remains the only independent variable. Moreover, it is a variable whose value is subject to constant change. There is, therefore, significant uncertainty associated with any estimate of its value. The easiest way to deal quantitatively with this uncertainty is to index it in units of standard deviation about the mean value. Then decisions made on the basis of results generated by variable input price, may be taken at some known confidence level. This allows management to plan rationally as the uncertainty associated with a decision is known. For the purposes of this thesis a range of +/- two standard deviations will be used.

The CPT that will result from this approach is illustrated in Figure 20. This profit envelope will meet corporate goals for 97.73% of the time under normal circumstances.

The lower limit of this envelope forms a Minimum Confidence Boundary, MCB, which gives the minimum grade - tonnage combinations that can be tolerated. A function can be fitted to this line and used as a general corporate guide to provide a simple screen for submittals. The position of the MCB will change with changes in demanded return and confidence levels.

Figure 20 UNCERTAINTY AND THE CPT



——— Mean value

- - - - +/- 1 standard deviation of price

—·— +/- 2 standard deviations of price

Note:

At constant required DCFROR, depth, operating costs & parameters

13.0 DEVIATION IN THE COMMODITY SOURCE PROFILE

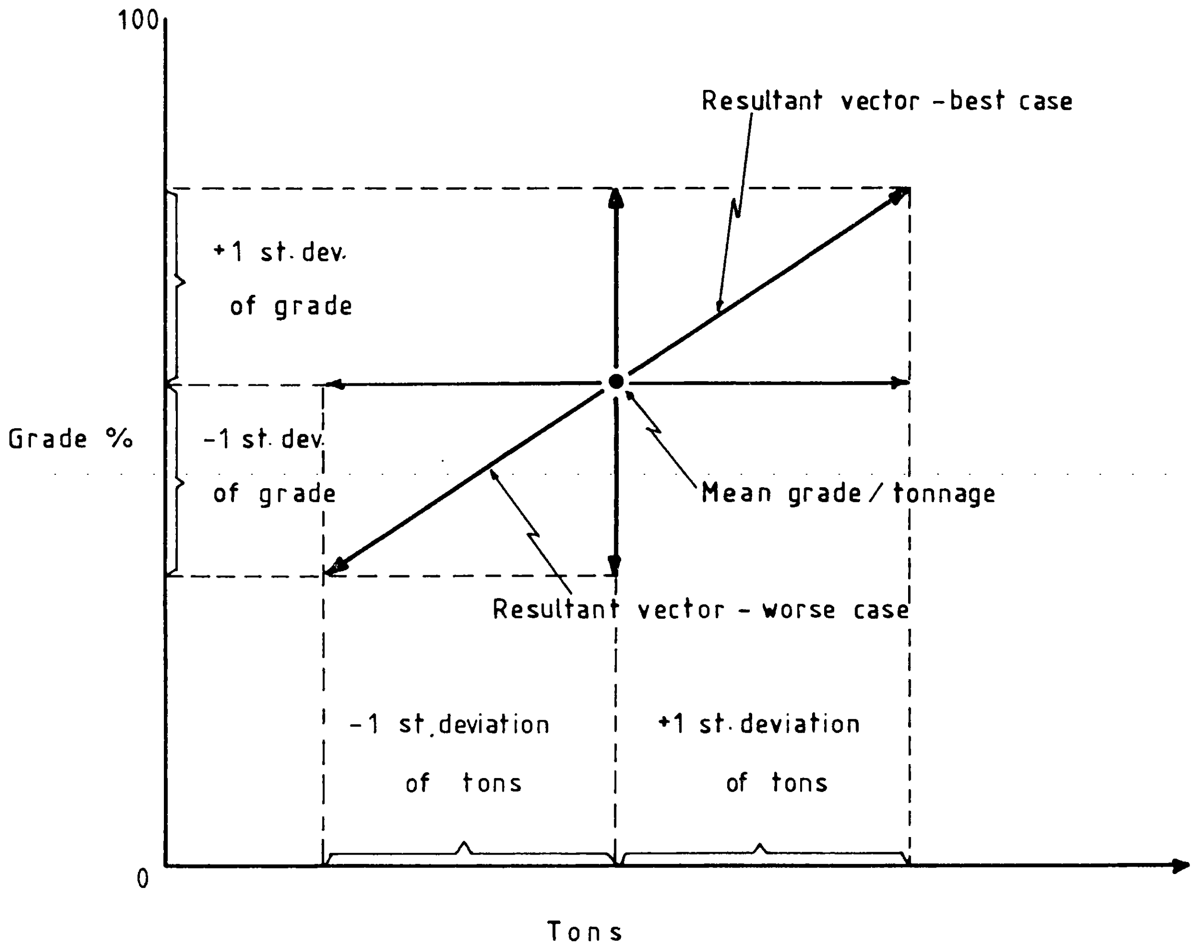
In the original definition of CSP, single - point values of grade and tonnage were used to typify deposit types. In reality such estimates are unreliable as both the grade and tonnage of a particular type classification will vary.

These variations will be in a finite range, and the mode of variation will change from type to type. Furthermore, grade - tonnage variations will not be independant, but will interact in some complex manner. However, in order to determine the maximum extent of uncertainty associated with the CSP, it is only necessary to know the maximum and minimum values of grade and tonnage bounding the ranges of variation, at some defined confidence level. Therefore, the mean and standard deviation of grade and tonnage ranges can be calculated in the normal way, and setting confidence at the same level as for the CPT, bounds can be drawn at +/- two standard deviations about the mean.

The grade and tonnage will vary simultaneously, reflecting the degree of dependance between them, the net result of this simultaneous variation can be represented by the resultant grade tonnage probability vector. This is illustrated in Figure 21.

This particular diagram happens to be for a

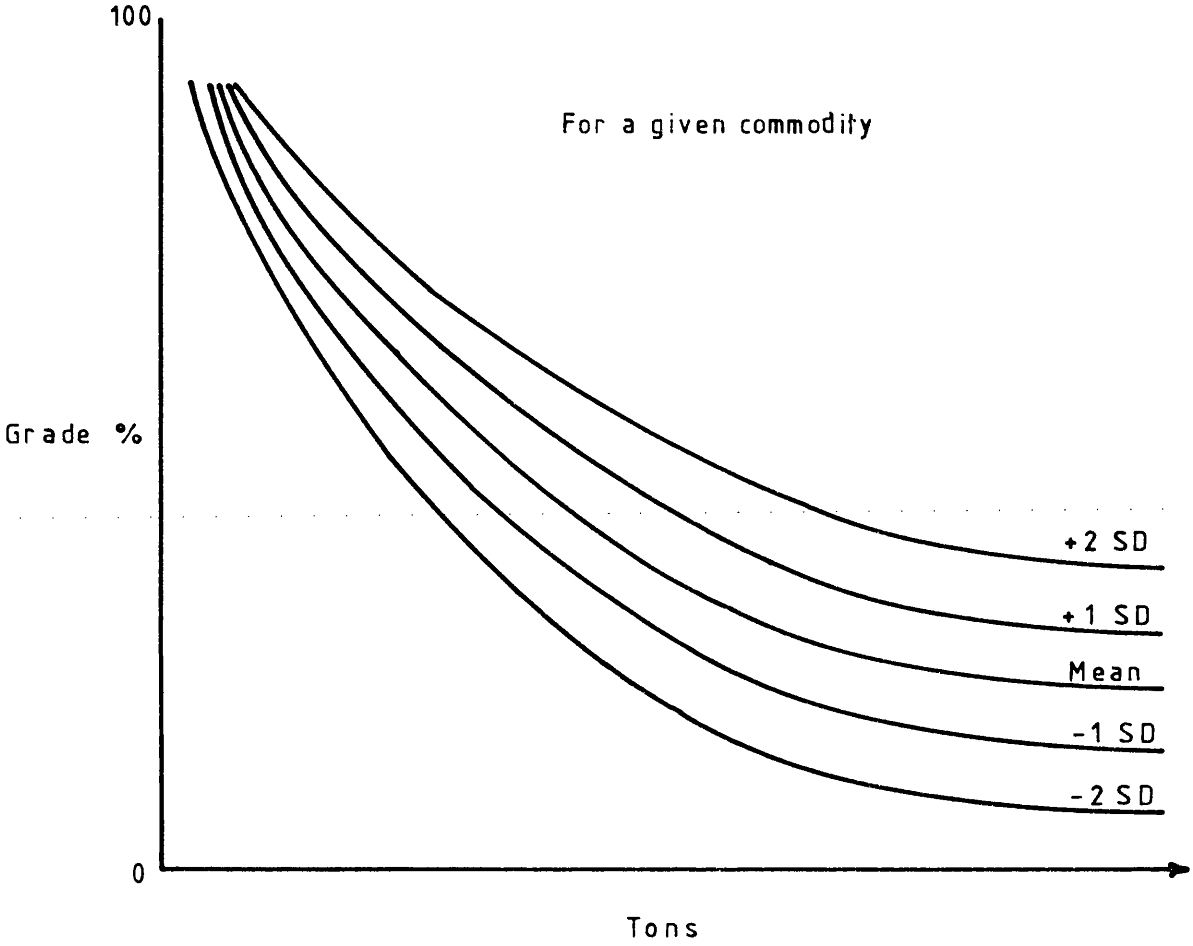
Figure 21 VECTOR DIAGRAM FOR A SPECIFIC DEPOSIT TYPE



probabilty range of +/- one standard deviation, obviously similar diagrams can be produced for any level of confidence desired. To reflect the simultaneous variation in grade and tonnage, the vertex of the resultant vector is contoured, rather than the vertices of the grade and tonnage vectors. The result of following such a procedure for all the deposit types for a given commodity is shown in Figure 22.

Assuming normal models, the chance of the grade being higher than + 2SD above the mean would be 2.28%, and similarly the chance of the tonnage being greater than + 2SD above the mean would also be 2.28%. Thus the chance of finding a deposit that had both a grade at + 2SD above the mean and a tonnage of + 2SD above the mean, would be the product of the two probabilites, about 0.05%. At one standard deviation the same chance would be about 3%. Hence, normally, the lower limit of the CSP, the - 2SD line, would capture about 99.95% of all source deposits of that commodity.

Figure 22 PROBABLE COMMODITY SOURCE PROFILE



14.0 PROBABLE SUCCESS

Now uncertainty has been addressed for both the Commodity Source Profile and the Commodity Profitability Threshold, the probably successful targets may be identified by overlaying the two sets of characteristics. This is illustrated in Figure 23.

The shaded portion, "b" , of the CSP envelope represents the probable success region, and the other fraction of the CSP envelope, "a" , represents the probable failure zone.

Hence, it will be readily appreciated that, at a given confidence level, and given all the input assumptions are correct, then the total chance of finding a profitable deposit of a given commodity, "i" , CS(i), will be given as follows:

$$CS(i) = \{(b)/(a + b)\} * 100.0\%$$

This is, of course, an absolute chance. In order to use this information to formulate a strategy it must be transformed to a relative base:

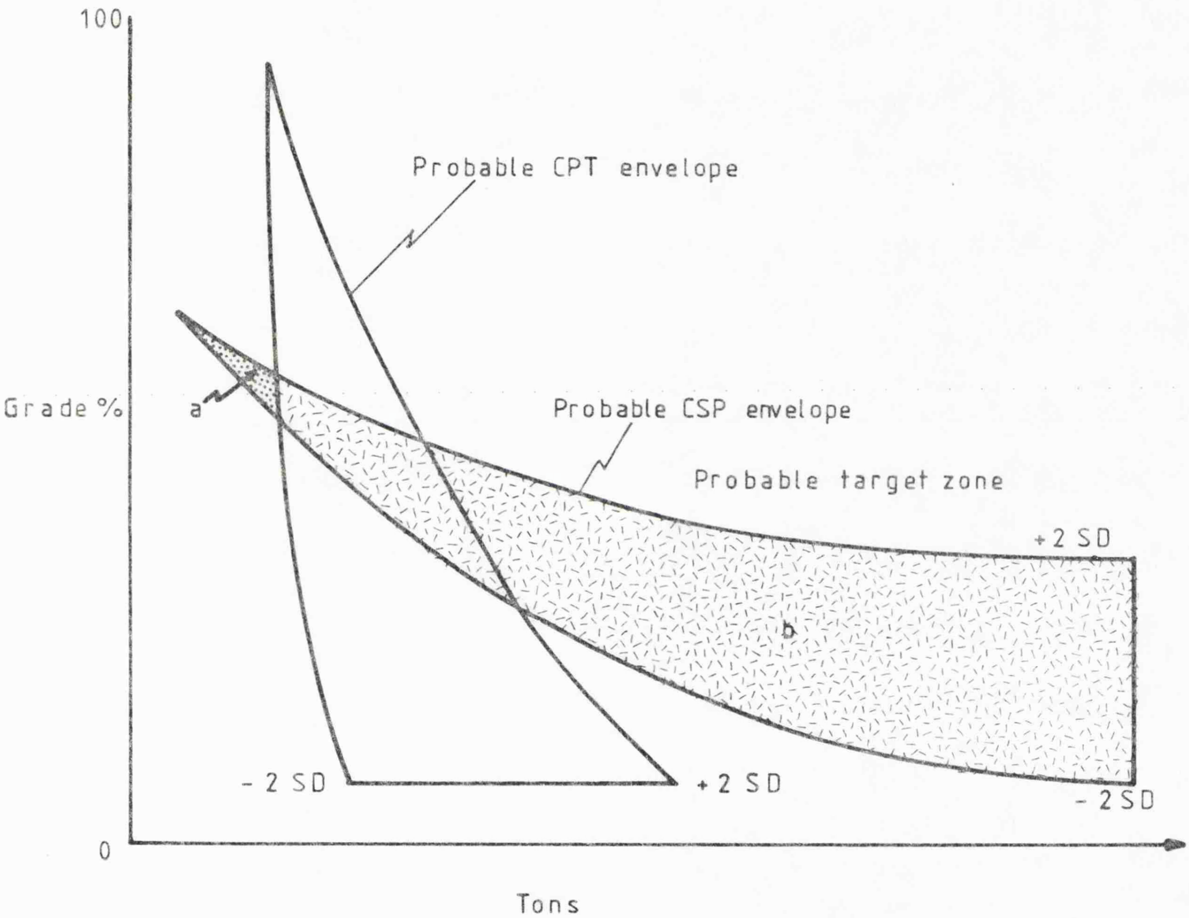
$$RCS(i) = \{[CS(i)]/[CS(1)+CS(2)+....CS(n)]\}*100\%$$

- where, RCS(i) = relative chance of success for commodity i

and n = number of commodities.

Thus commodities may be ranked in order of their

Figure 23 PROBABLE TARGET DEFINITION



chances of making a profit, and the exploration budget may be split accordingly.

Furthermore, the sum of all absolute CS(i) values is the total chance of finding any profitable deposits at all. This chance may then be used to judge the attractiveness of investment in mineral exploration as opposed to some other competitive oppurtunity.

Thus, Total Chance of Success, TCS, is defined as follows:

$$TCS = [\ CS(i) + \ldots + CS(n)\ ]/(\ n * 100\ )$$

15.0 DEPOSIT TYPE SPECIFICATION

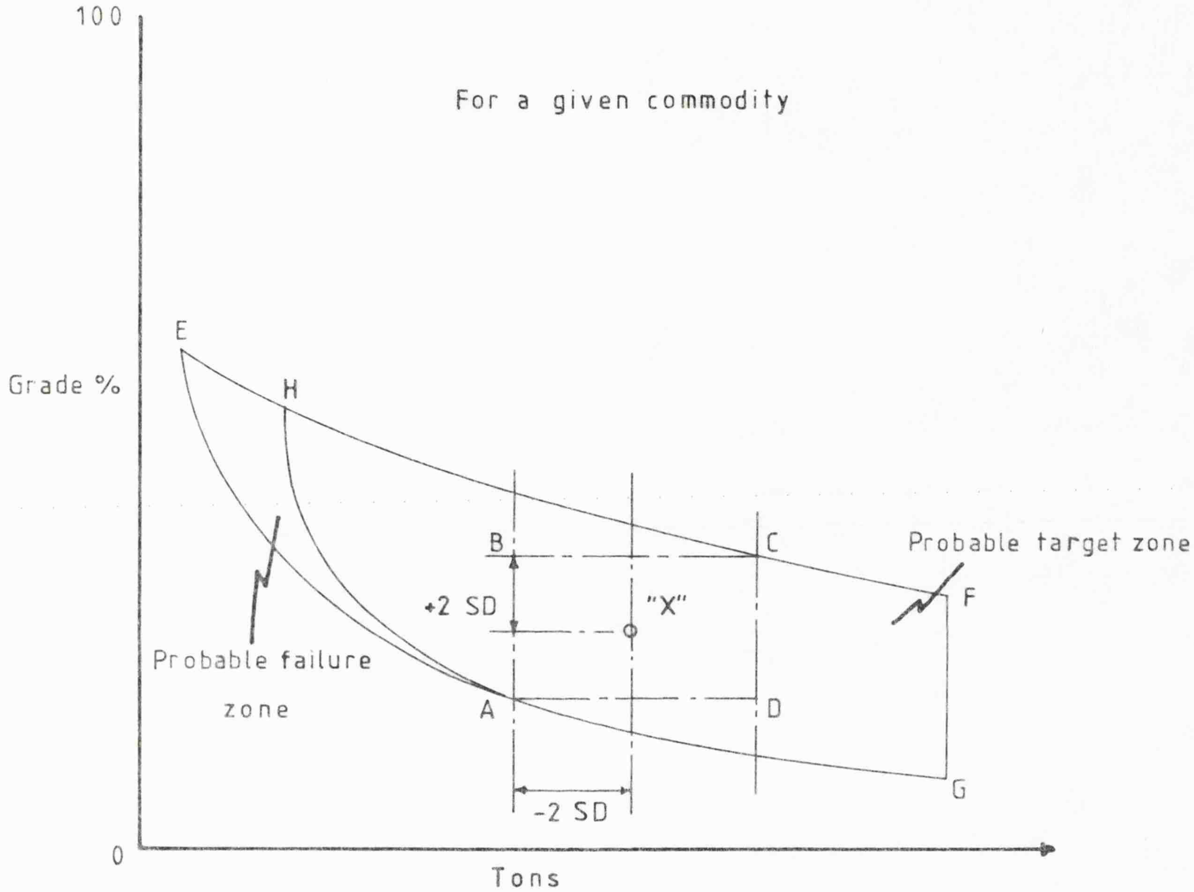
So far the method for determining in general the chances of exploration success has been described. However, it is necessary to provide more specific guidance to field exploration teams. In order for an efficient program to be run specific deposit types must be identified, and not merely general grade - tonnage guidelines. This chapter will explain how this may be accomplished.

In order to make the logic clear it is necessary to consider the Commodity Source Profile; this is illustrated in Figure 24.

Suppose deposit type "X" had a mean grade and tonnage which plotted as shown on Figure 24. The variability of this type is expressed in terms of a zone around the mean bounded at +/- two standard deviations of grade and tonnage.

The polygon EFG represents the limits of the CSP at a 97.73% confidence level. Of that, only the portion bounded by polygon HFG exceeds the Minimum Confidence Boundary and is therefore a potential target zone. The chance of a deposit of type "X", if found, satisfying corporate financial requirements is given by the proportion of the bounding area ABCD that exceeds the MCB. In this case that excess or chance,

Figure 24 DEPOSIT ALLOCATION DIAGRAM



called XS(i,j), is 100%. Where "i" is the deposit type within the commodity "j". The proportion of exploration effort for the given commodity that should be expended in looking for deposit type "X" is equal to the proportion of the area ABCD that exceeds the MCB as a percentage of the Probable Target Zone, HFG. This percentage is defined as the Deposit Allocation, DA(i).

So, DA(i) = [ABCD * XS(i,j)] / HFG

If the total area occupied by the different deposit types is not equal to that of the probable target zone, then the deposit allocations can be recast on a relative basis, and the expenditure apportioned accordingly.

The geologic search constraints have thus been defined, it now only remains to pull all the threads together to form one unfied strategy.

16.0 GENERAL SUMMARY

In the preceeding chapters various indices have been identified, in an attempt to make the qualitative aspects of the decision - making process more systematic, namely:-

- Relative Socio - Political Index, RSPI
- Relative Price - Time Index, RPTI
- Relative Market Index, RMI
- Relative Commodity Exploration Index, RCEI

- all of which find combined expression in the General Exploration Potential, GEP(i), for a given commodity.

The General Exploration Potential of a commodity attempts to quantify in a simple, logical way the intangibles.

On the other hand, from a purely technical point of view, the chance of actually achieving a desired return from investment in exploration has been derived and is expressed in the Total Chance of Success, TCS. This may be used to determine what is the maximum proportion of the total capital available for investment that should be allocated to exploration, as follows:

$$\text{Maximum Justifiable Exploration Budget, } MJEB = TCS * AI$$

- where: AI is Available Investment.

and hence:

$$\text{Investment in Non-Exploration Alternatives, } INEA = (1.0 - TCS) * AI$$

Once the Maximum Justifiable Exploration Budget has been determined, then this may be divided appropriately between commodities by use of the Relative Chances of Success , RCS(i), for each commodity, as follows:

$$\text{Maximum Budget/Commodity, } MBC(i) = MJEB * RCS(i)$$

At this point the intangible feelings on each commodity can be appropriately introduced to modify the theoretical solution to fit with the real world, by use of the General Exploration Potential as follows:

$$\text{Modified Investment per Commodity, } MIC(i) = MBC(i) * GEP(i)$$

The sum of all the MIC's produces the Total Justifiable Exploration Budget. This will only rarely be equal to the Maximum Justifiable Exploration Budget, usually it will be less, so that there will be a Non - Justified Exploration Budget, which should be added to the Investment in Non - Exploration Alternatives, thus:

Total Justifiable Budget, TJB = MIC(i) + ... + MIC(n)

Non - Justified Budget, NJB = MJEB - TJB

Investment in Non -
Exploration Alternatives, INEA = INEA + NJB

In the case where TJB is greater than MJEB, either additional capital must be provided from the Available Investment at the expense of other alternatives, or the GEP's must be re-assessed less optimistically, or both.

Given the Modified Investment per Commodity, the expenditure on given deposit types may be determined using the Deposit Alloctaions, DA(j), as described in Chapter 15:

Investment / deposit type
within a commodity = MIC(i) *DA(j)

- where: i = ith commodity

j = jth deposit type.

The amount of investment justified for a specific prospect within a deposit type, within a commodity, may then be determined according to the relative position of the expected grade - tonnage potential of the prospect in the success envelope.

The limits of the envelope are set at +/- 2SD; therefore, to eliminate negatives add 2SD to all prospect positions (expressed in co-ordinates of standard deviation), then express the resultant values on a relative basis, and hence determine investment. Numerically, for example:

Suppose we have four prospects; A, B, C, D at various standard deviations from the mean:

Prospect	SD
A	+ 1.65
B	+ 1.02
C	− 0.10
D	− 0.85

Add two standard deviations to determine their distances above the threshold, and express on a relative basis:

Prospect	Excess	Relative Excess
A	3.65	0.34
B	3.02	0.28
C	2.90	0.27
D	1.15	0.11
	10.72	1.00

Hence, expenditure per prospect may be determined

Assume: MIC(i) = $1.0M, then:

Prospect	Relative Excess	Expenditure
A	0.34	340,000
B	0.28	280,000
C	0.27	270,000
D	0.11	110,000

So, the rational process can be seen to run logically from initial hypothetical concept, to detailed disbursement of funds for each prospect. This then provides a unified, integrated approach to rational decision making in the orientation of mineral exploration efforts.

PART 2

A Numerical Example to Illustrate the Application of the General Theory.

17.0 DERIVATION OF COMMODITY SOURCE PROFILES

A literature search was undertaken in journals 23) & (41) and reference books (24) & (25) to collect data on several hundred prospects and operating mines. The following information was collected on each property:-

- name
- commodity
- deposit type
- tonnage
- grade

The list was then sorted by commodity and by deposit type. The results of this sort are given in Appendix C. The list was then resorted so that all deposit types having only one representative were eliminated from the list. The mean and standard deviation was then calculated for each deposit type. In addition a chi-squared test was performed on each group to test for normalcy. The resorted list, together with the relevant statistical information is given in Appendix D.

At this point a note on the definitions of the above categories will be helpful.

17.1 Commodity

In many cases a deposit contained more than one commodity. Bearing in mind the object of this numerical example is to illustrate the use of the theory, rather than to produce a universally applicable solution, it was decided to eliminate the concept of multi-commodity orebodies. That is to say, if a deposit contained say, Au, Ag, and Cu, it would be listed three times with the same tonnage and deposit types, but with different commodity and grade classifications. Were this analysis to be performed in earnest, decisions would have to made concerning the classification of element groups, and a primary commodity from within each group would have to be chosen. The other associated commodities would then be expressed as equivalent grade additions to the primary. However, for the purposes of this illustrative example, such complications were regarded as a diversion, and the study was performed on a single commodity per orebody basis. Indeed, it could be argued that the single commodity approach is better, in that it is easier, and any secondary commodities within a deposit could be expressed as smelter credits, thus increasing the net smelter return used for the primary. Whichever approach is used for multi-commodity deposits, if correctly applied, will produce the same answer.

17.2 Deposit Type

Based upon the information revealed by the literature search of all possible types of deposit, only ten seem to be in common usage. Therefore, the classification used for this study was limited to these common ones, namely:-

- porphyry
- sedimentary
- contact metamorphic
- stratiform
- oxide
- volcanogenic massive sulphide
- complex
- hydrothermal
- tri-state
- laterite

17.3 Tonnage

The largest published tonnage was always used in this study.

17.4 Grade

The highest published average grade was always used in this study for each commodity in a deposit.

17.5 Curve Fitting

Given the grade - tonnage data for all deposits within a given commodity, curves were fitted to the mean grade - tonnage values; and also to the values at +/- two standard deviations about the mean.

The results of this process are presented in Figures 25 to 30, for copper, lead, zinc, gold, silver and nickel. Insufficient data was avaiable on other commodities to allow for Commodity Source Profiles to be generated. Details of the fitted curves are given in Appendix E, together with the correlation coefficient showing the goodness of fit between the models and the actual data. It should be noted that in all subsequent calculations the model values will be used rather than the actual data.

Figure 25 COPPER - COMMODITY SOURCE PROFILE

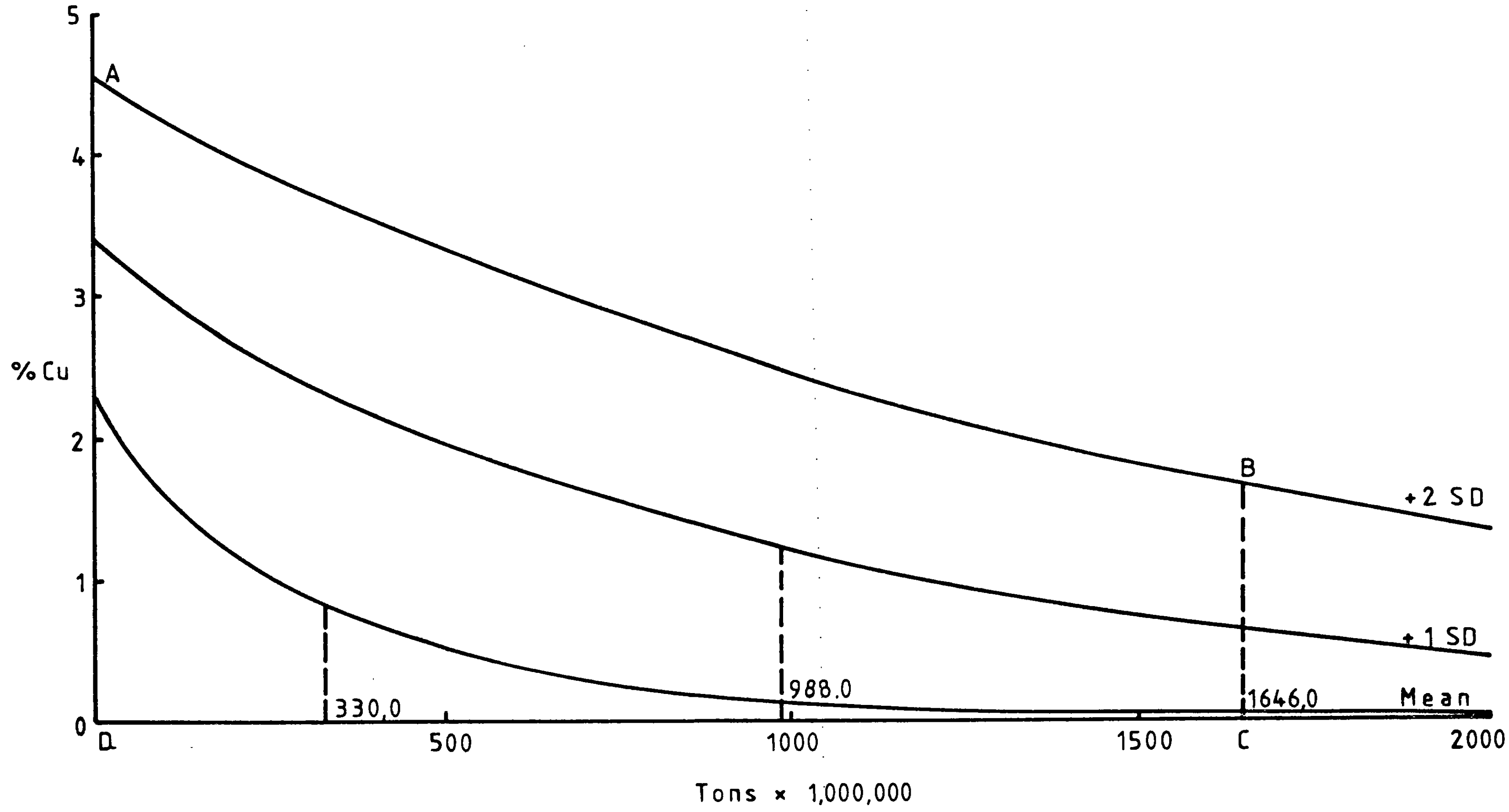


Figure 26 LEAD - COMMODITY SOURCE PROFILE

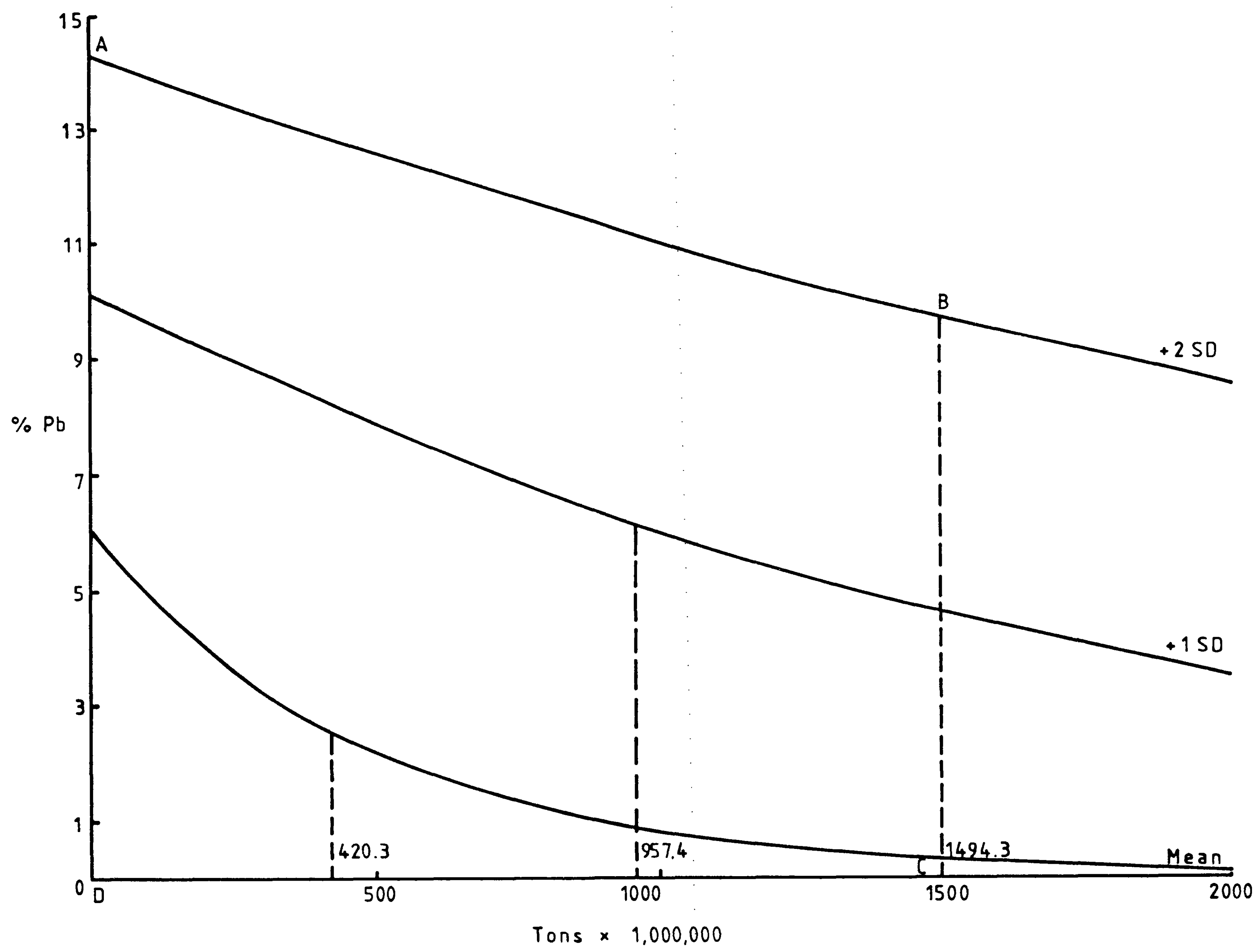


Figure 27 ZINC - COMMODITY SOURCE PROFILE

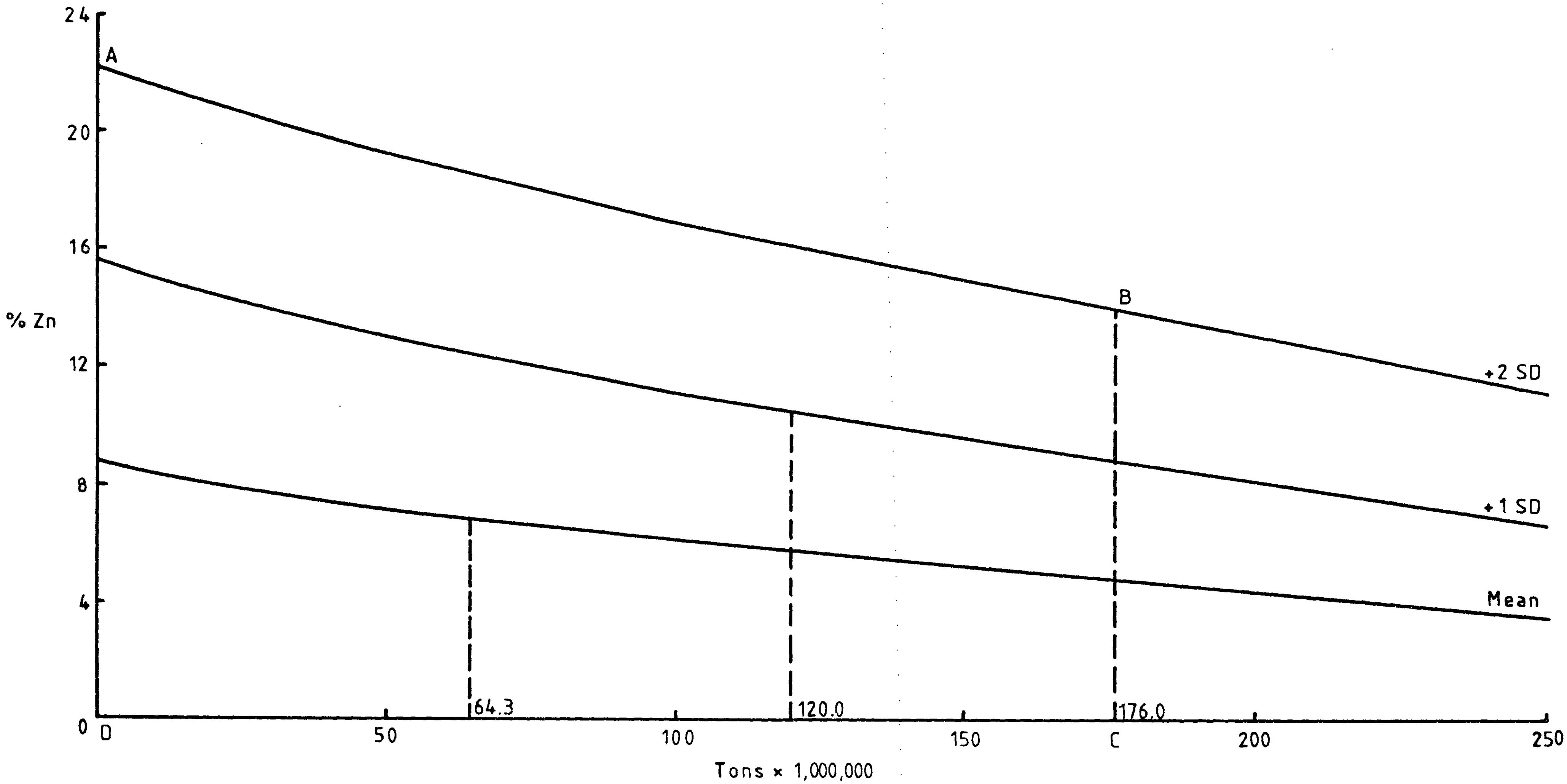


Figure 28 GOLD - COMMODITY SOURCE PROFILE

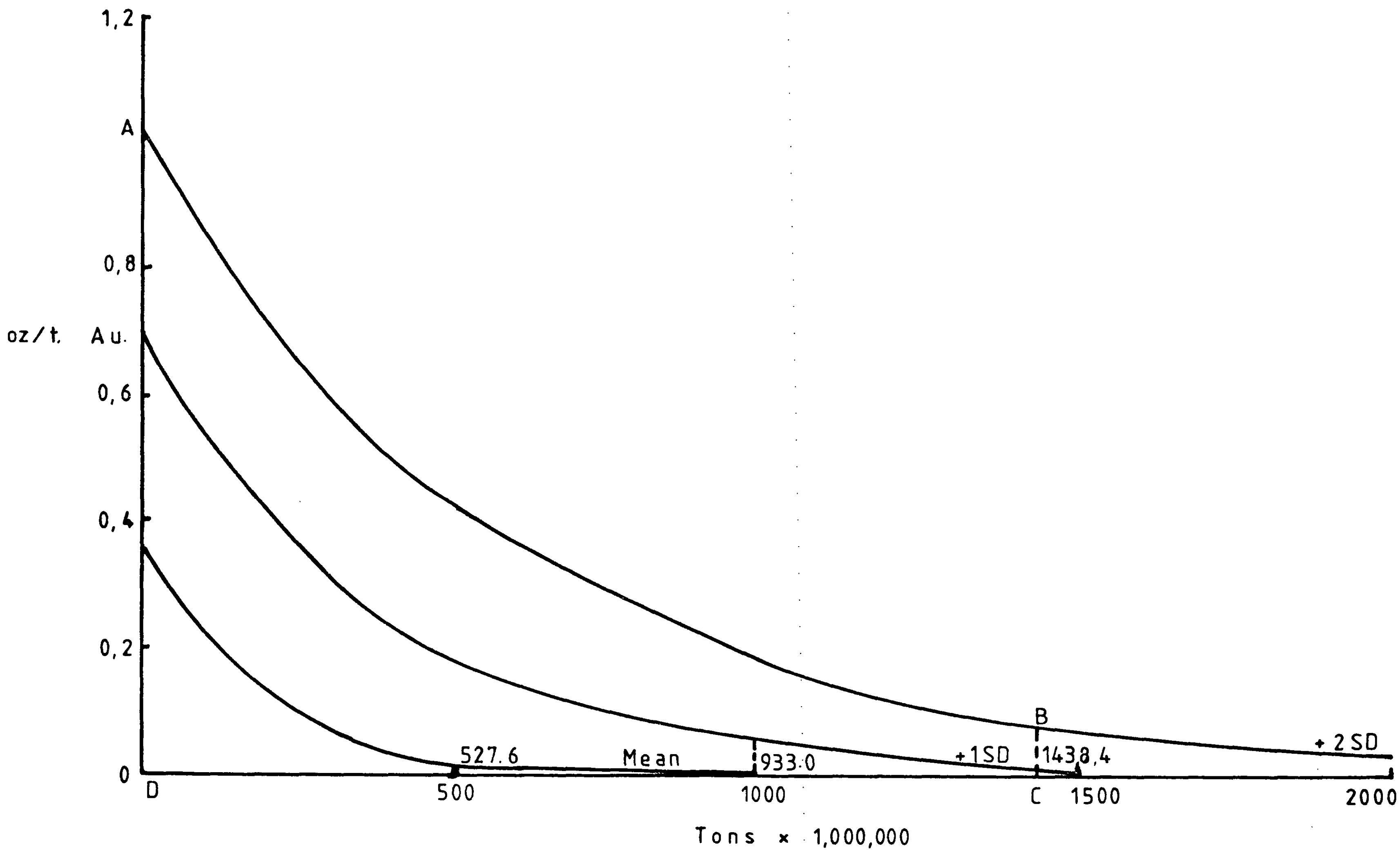


Figure 29 SILVER - COMMODITY SOURCE PROFILE

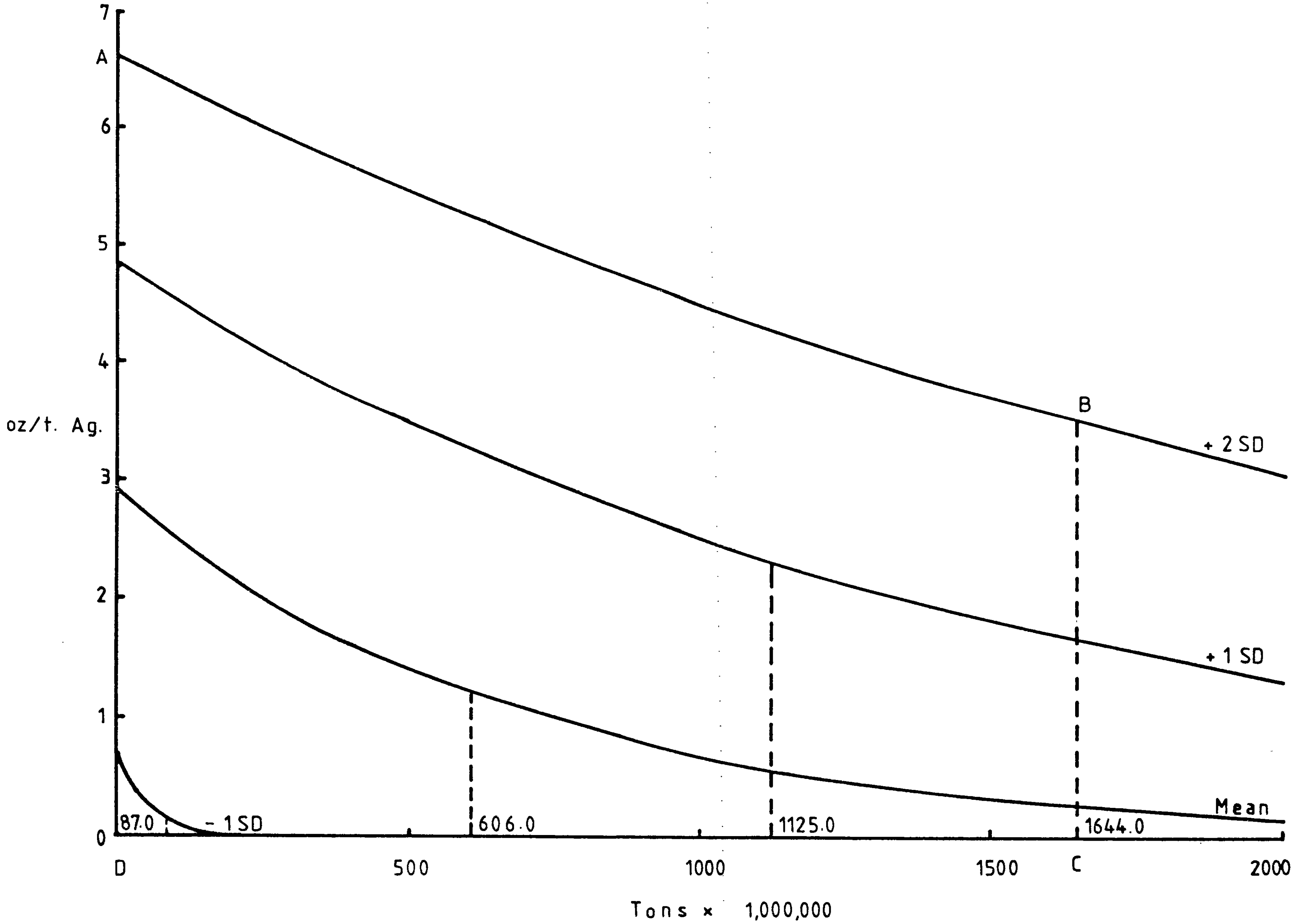
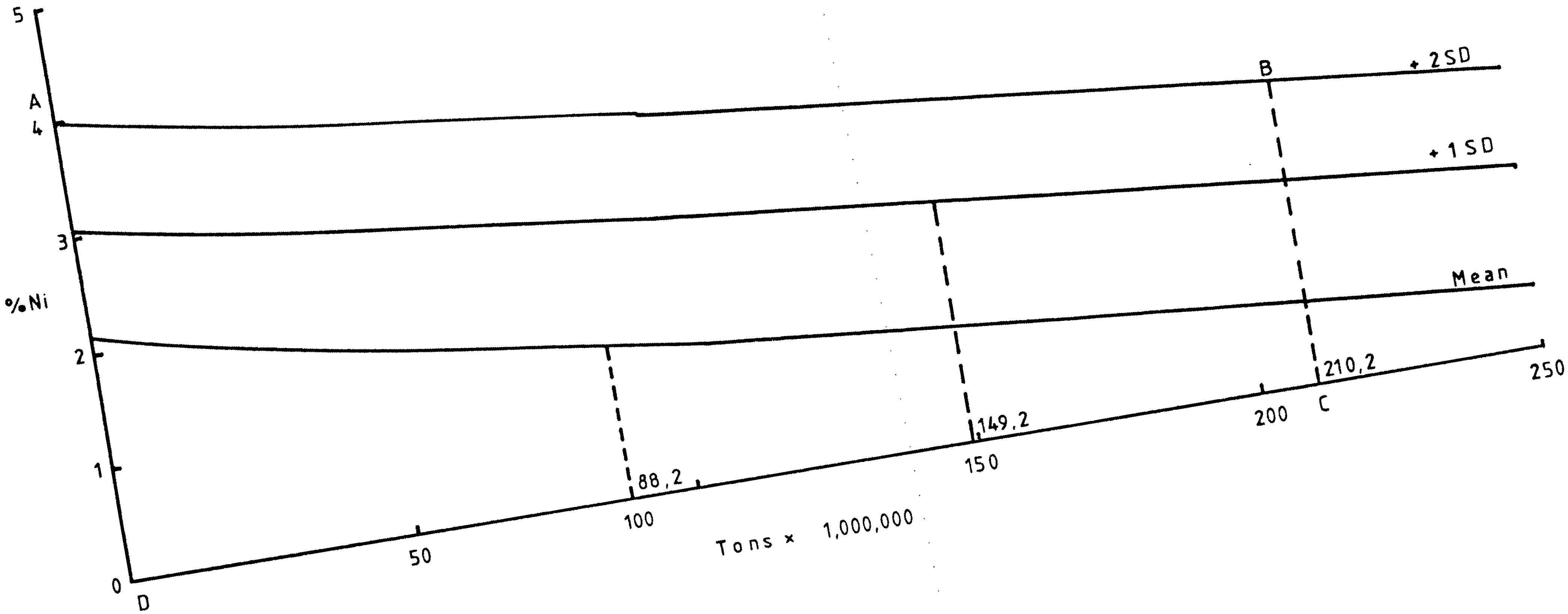


Figure 30

NICKEL - COMMODITY SOURCE PROFILE



18.0 PRICE PREDICTIONS

Geologic reality having been defined, in terms of the Commodity Source Profiles, it was next necessary to define desire. That is to say, the Commodity Profitability Thresholds must be generated for each commodity. The first step in such a process is to generate a price prediction for each commodity over the period of interest. As was suggested in Part 1, regionalized variable analysis was used to define the range of predictability, future prices and error of estimation.

18.1 Basic Data

The basic data for any price prediction exercise is the history of price movement with time. The question is, how far is it reasonable to look back in time ? In the case of gold, data can be obtained at least as far back as the 14th century, but how much of this information is relevant to the future ?

The price of a commodity is to a large extent controlled by demand. Demand in turn is a function of complex, non-quantified, socio-political processes. It could then be argued that so long as the socio-political state in the past seems to be directly related to the present and forseeable future, then prices in that time frame should be considered. Applying this logic, it would seem that the end of World War II marks the start of the present socio-political state. Therefore, prices from 1945 - 1982 have been used as base data for this study. The prices quoted are average annual values on the following bases:-

- copper - US producer price $/lb.
- lead - US producer price $/lb.
- zinc - US producer price $/lb.
- gold - LME cash $/oz.
- silver - LME cash $/oz.

Suitable information for nickel was not readily available, consequently this commodity was dropped from further consideration in this study.

18.2 Inflation

As was also stated in Part 1, it is preferable to perform a constant value analysis, therefore, the quoted prices had to be deflated. Moody's Average Commodity Price Index for each year from 1945 to 1982 was used to adjust all values to a 1945 basis. Details of the adjusted prices are given for each commodity, together with the Moody Index values in Appendix F. The same information is presented graphically in Figures 31 to 36. It should

Figure 31

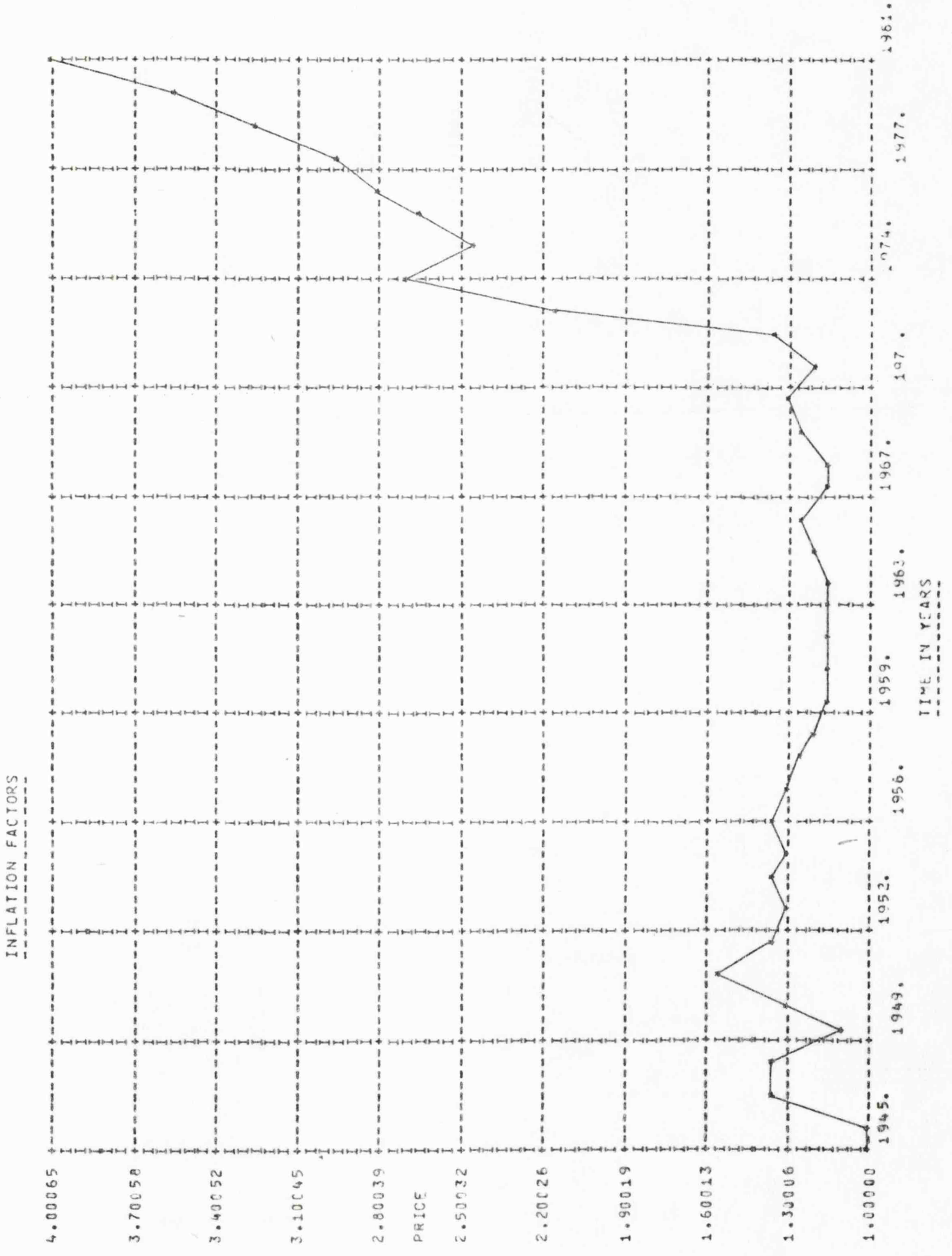

INFLATION FACTORS
PRICE
4.00065
3.70058
3.40052
3.10045
2.80039
2.50032
2.20026
1.90019
1.60013
1.30006
1.00000
1945.
1949.
1952.
1956.
1959.
1963.
1967.
1977.
TIME IN YEARS

Figure 32

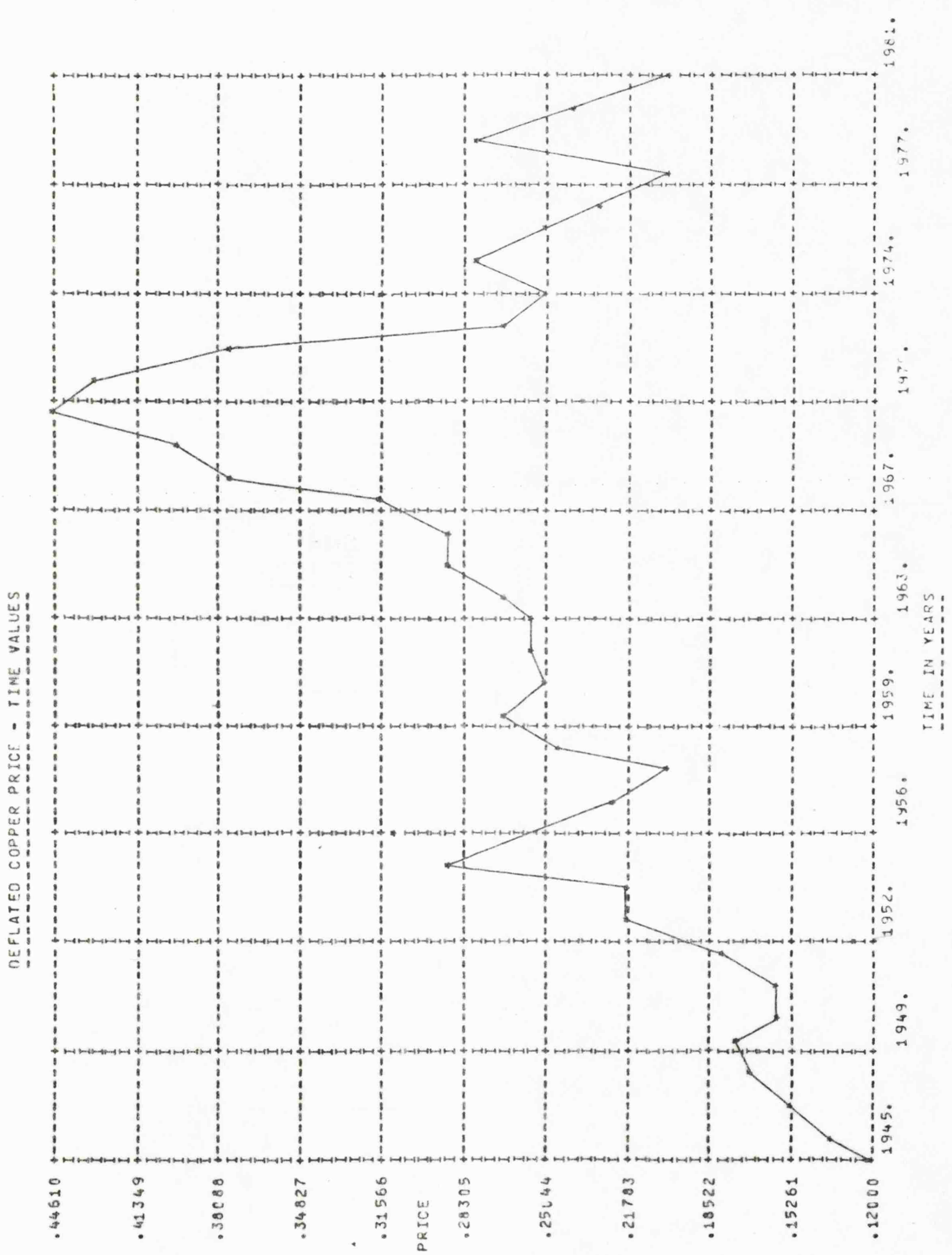

DEFLATED COPPER PRICE - TIME VALUES
PRICE
.44610
.41349
.38088
.34827
.31566
.28305
.25044
.21783
.18522
.15261
.12000
1945.
1949.
1952.
1956.
1959.
1963.
1967.
1974.
1977.
TIME IN YEARS

Figure 33

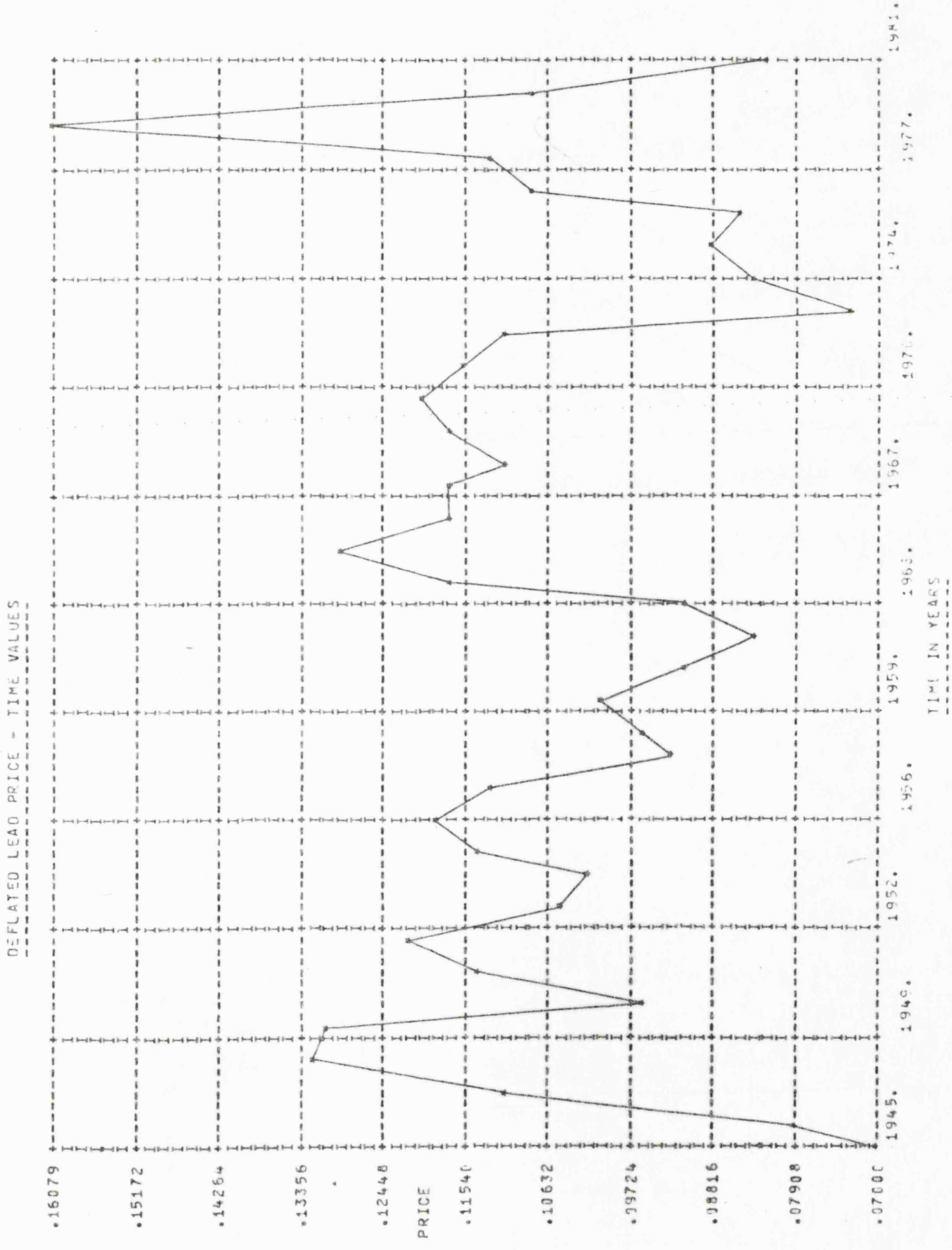

DEFLATED LEAD PRICE - TIME VALUES
PRICE
.16079
.15172
.14264
.13356
.12448
.11540
.10632
.09724
.08816
.07908
.07000
1945.
1949.
1952.
1956.
1959.
1963.
1967.
1970.
1974.
1977.
1981.
TIME IN YEARS

Figure 34

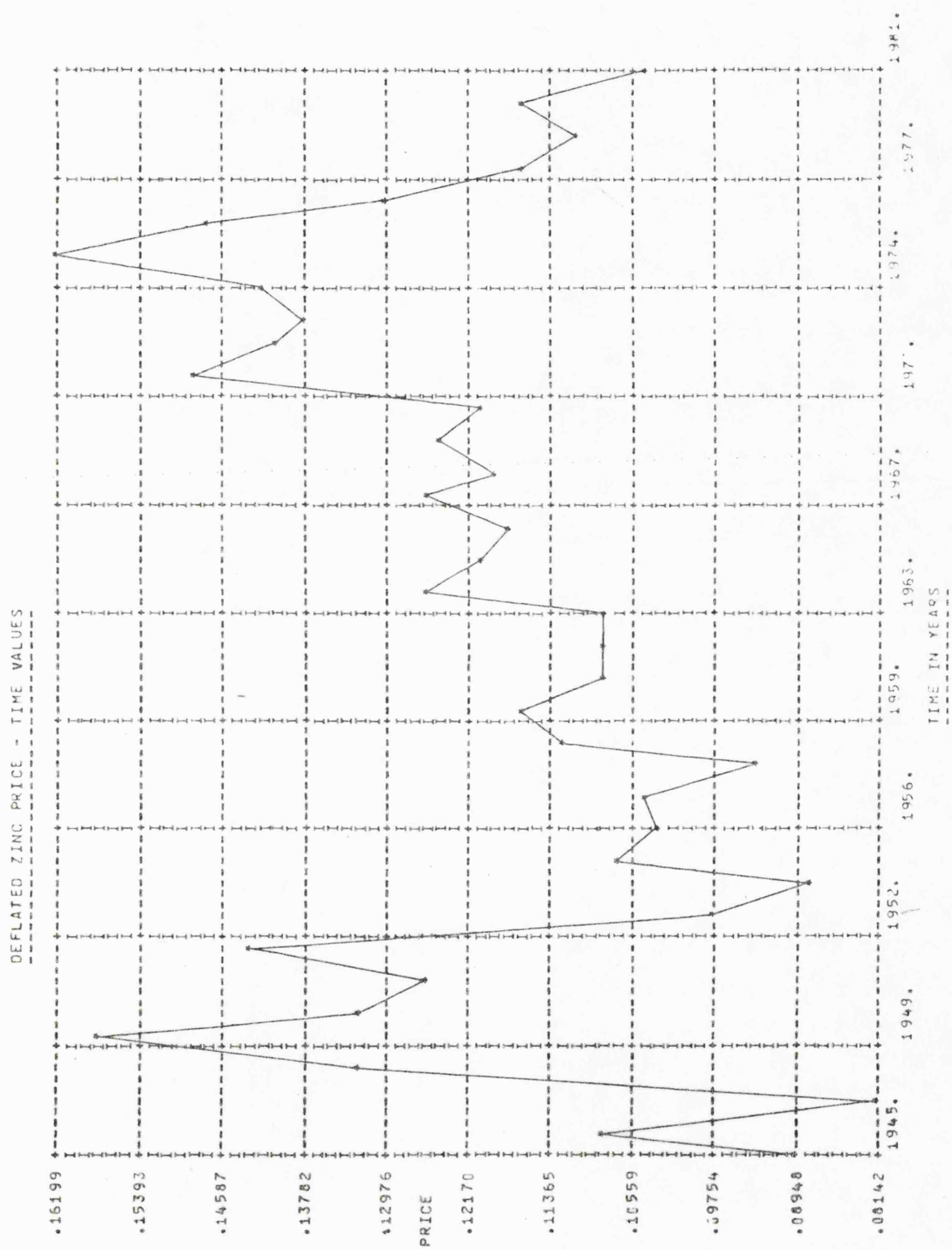

DEFLATED ZINC PRICE - TIME VALUES
PRICE
.16199
.15393
.14587
.13782
.12976
.12170
.11365
.10559
.09754
.08948
.08142
TIME IN YEARS
1945.
1949.
1952.
1956.
1959.
1963.
1967.
1974.

Figure 35

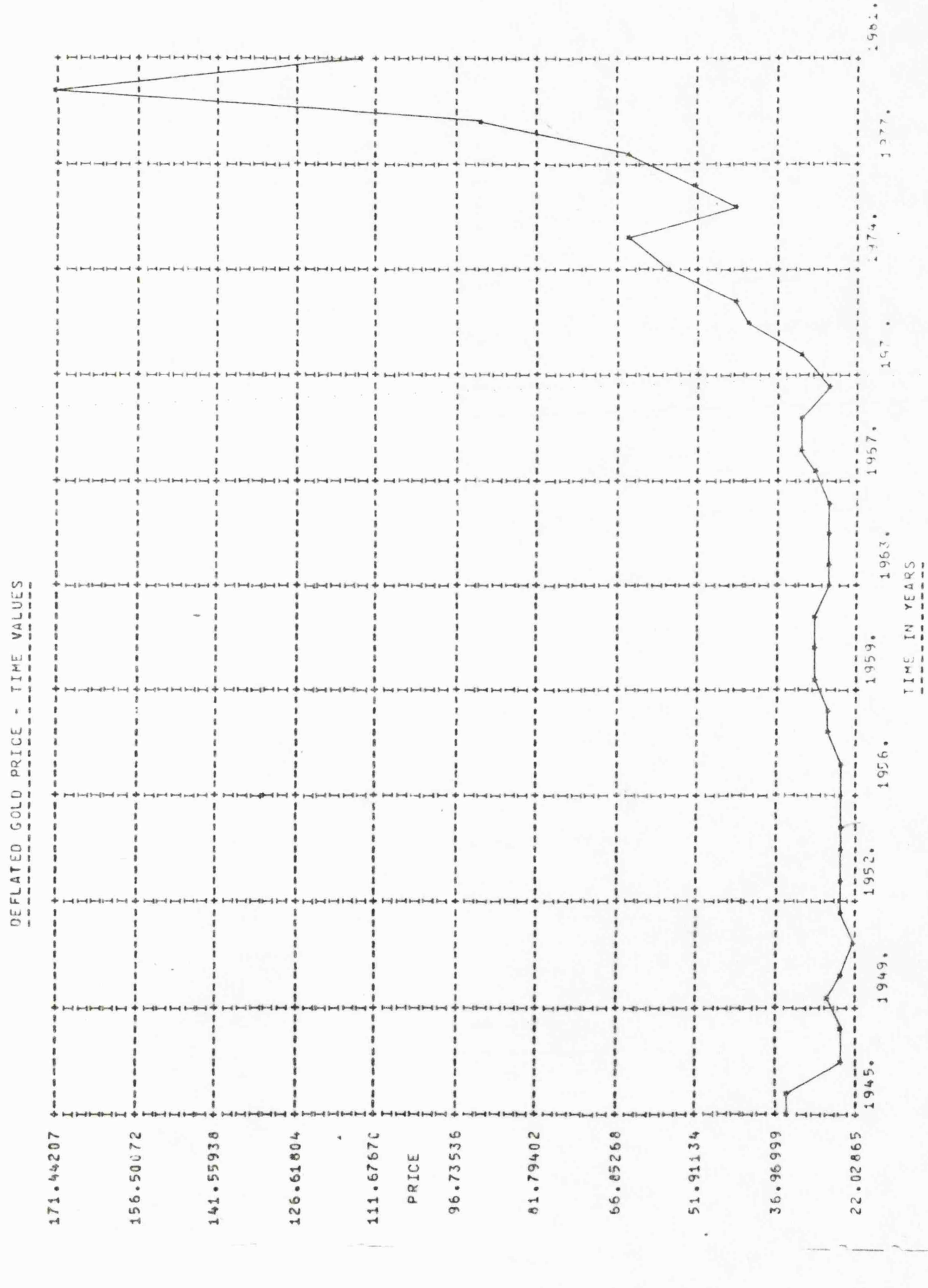
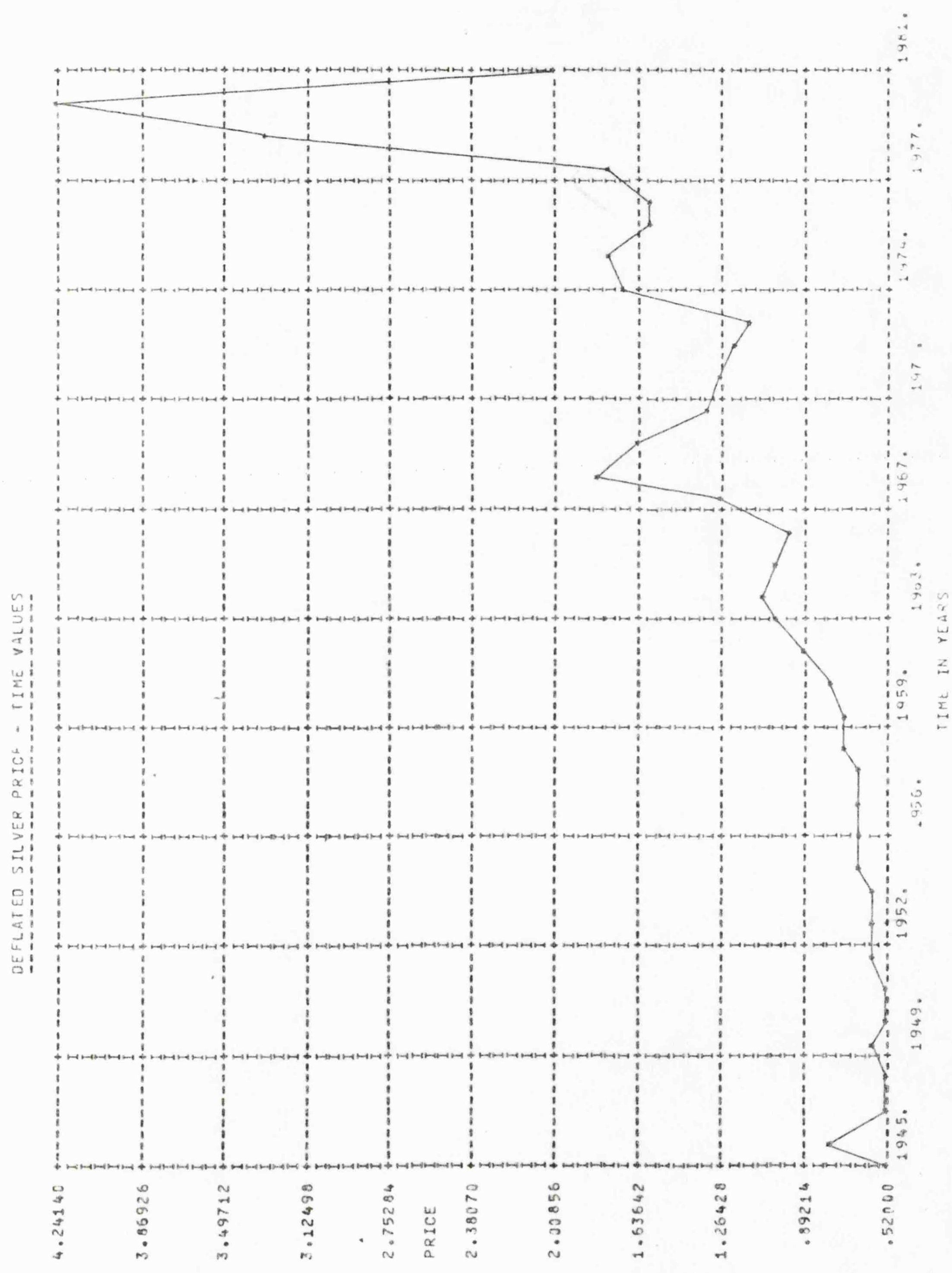

DEFLATED GOLD PRICE - TIME VALUES
171.44207
156.50072
141.55938
126.61804
111.67670
PRICE
96.73536
81.79402
66.85268
51.91134
36.96999
22.02865
1945.
1949.
1952.
1956.
1959.
1963.
1967.
1974.
1977.
TIME IN YEARS

Figure 36


DEFLATED SILVER PRICE - TIME VALUES
PRICE
4.24140
3.86926
3.49712
3.12498
2.75284
2.38070
2.00856
1.63642
1.26428
.89214
.52000
TIME IN YEARS
1945.
1949.
1952.
1959.
1963.
1967.
1977.

be noted that hereafter, the word "inflation" will be used to mean "Moody's Commodity Price Index". These data were taken from the American Metal Market (27).

18.3 Variograms of Price Change vs. Time

Using the data in Appendix F variograms were produced for the change in commodity price against time for copper, lead, zinc, gold, silver and inflation. The results are shown in Figures 37 to 42. On the basis of these variograms the following parameters were obtained:-

Data Set	Range, years	Nugget	Sill
Inflation	14	0.02680	0.51900
Copper	10	0.00060	0.00454
Lead	5	0.00018	0.00021
Zinc	6	0.00015	0.00019
Gold	13	180.83510	526.12010
Silver	22	0.11821	0.43780

Referring to Figures 37 to 42 it can be seen that the fit of a spherical variogram model to inflation, copper, gold and silver is reasonable. However the fit to lead and zinc is very poor. Indeed it could be argued that a spherical model will not fit to these data sets at all. In order not to divert attention away from the main thrust of the thesis it was decided to fit a spherical model to these data rather than to search for a different model that produced a better fit. To this end the average semi-variance was used to

Figure 37

Value - Time Variogram for Inflation

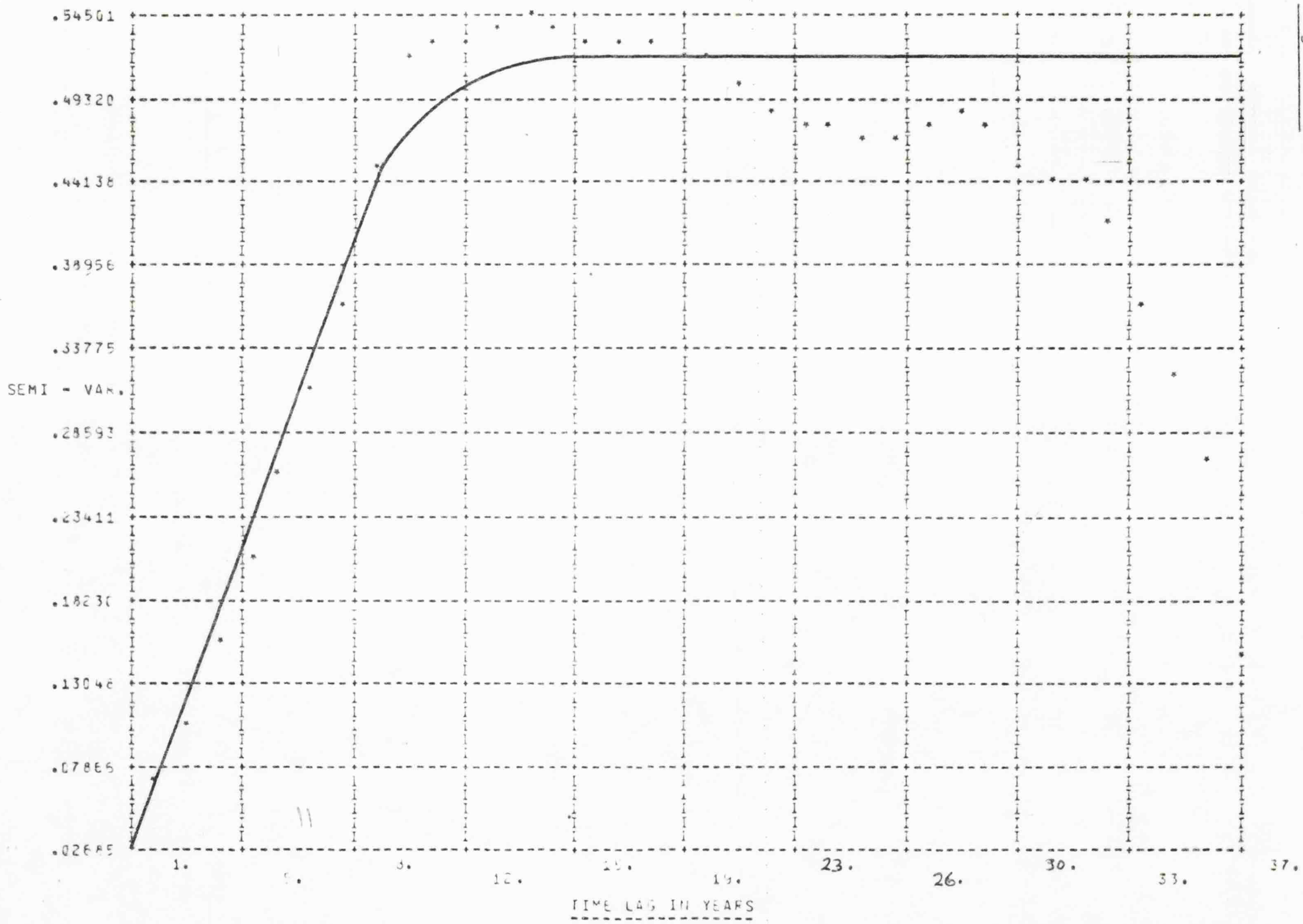


Figure 38

Price - Time Variogram for Copper

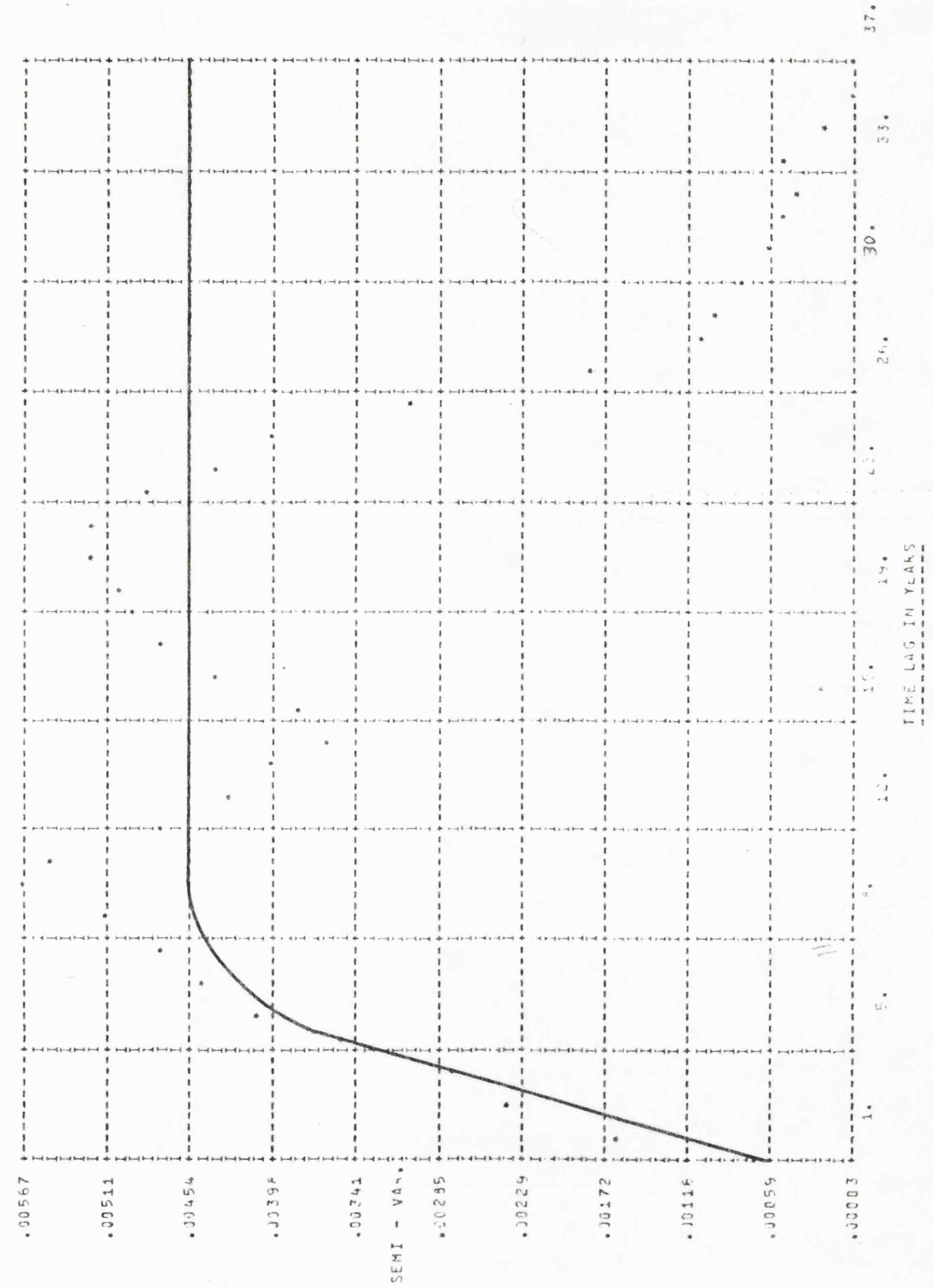
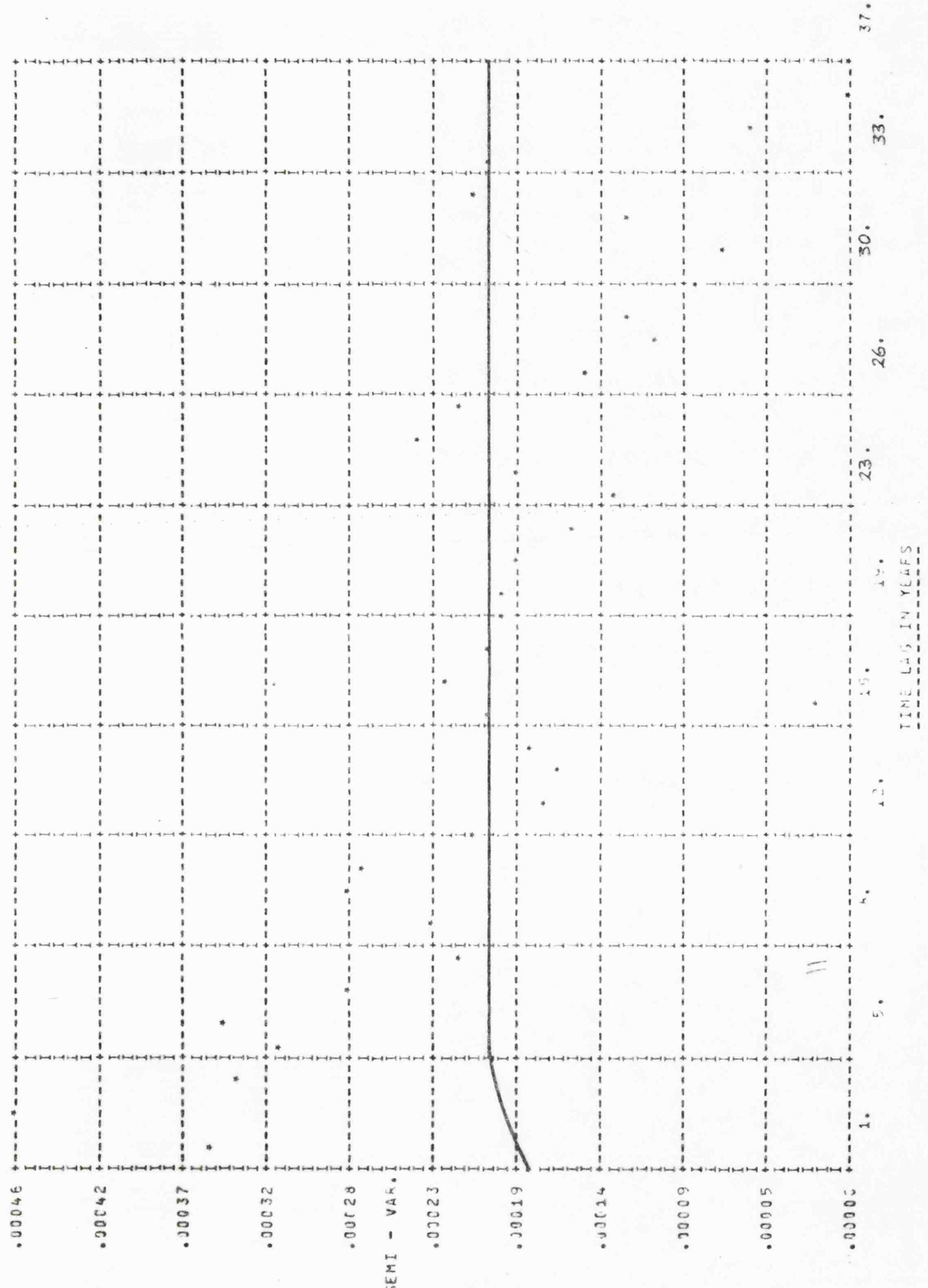


Figure 39



Price - Time Variogram for Lead

Figure 40

Price - Time Variogram for Zinc

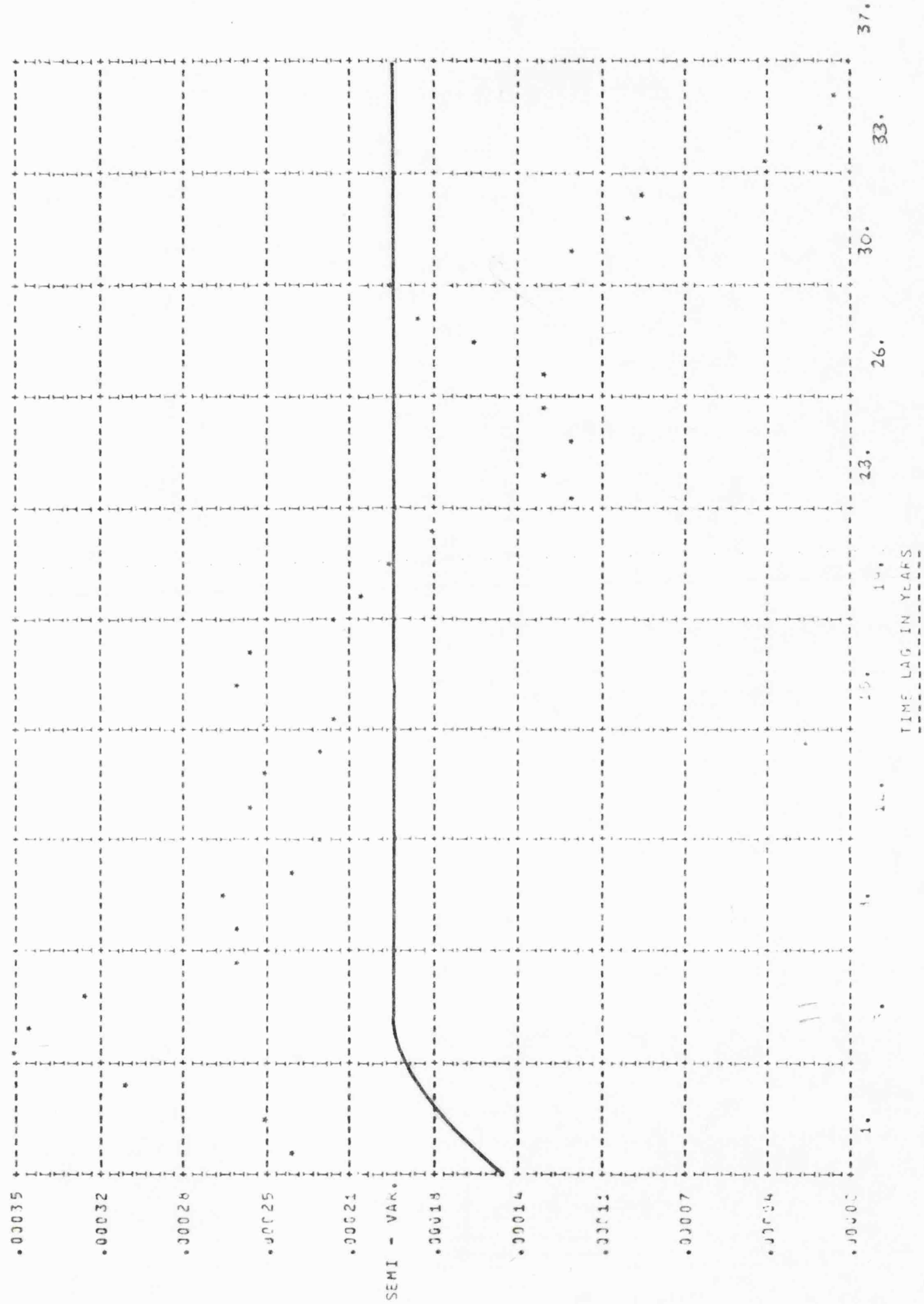
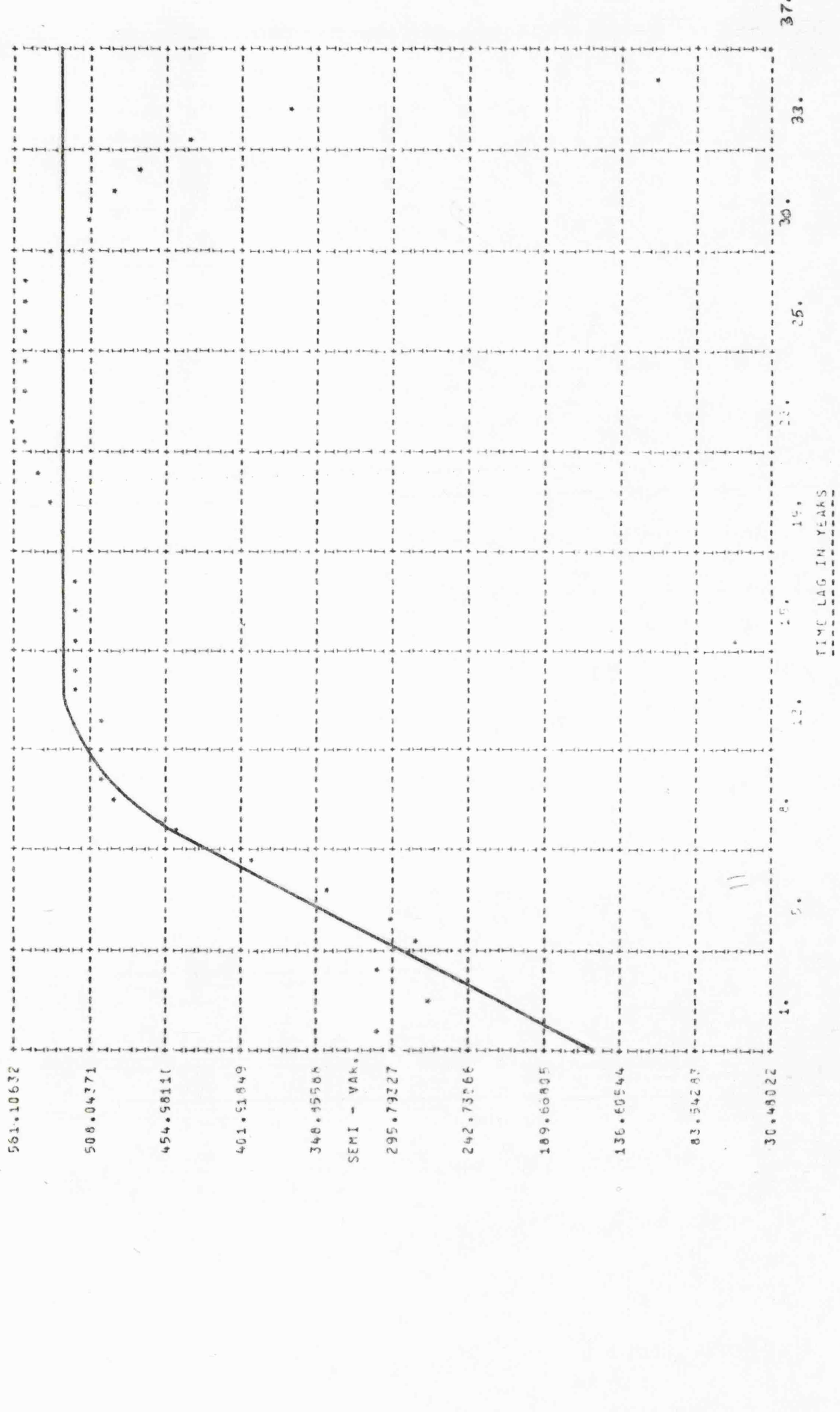


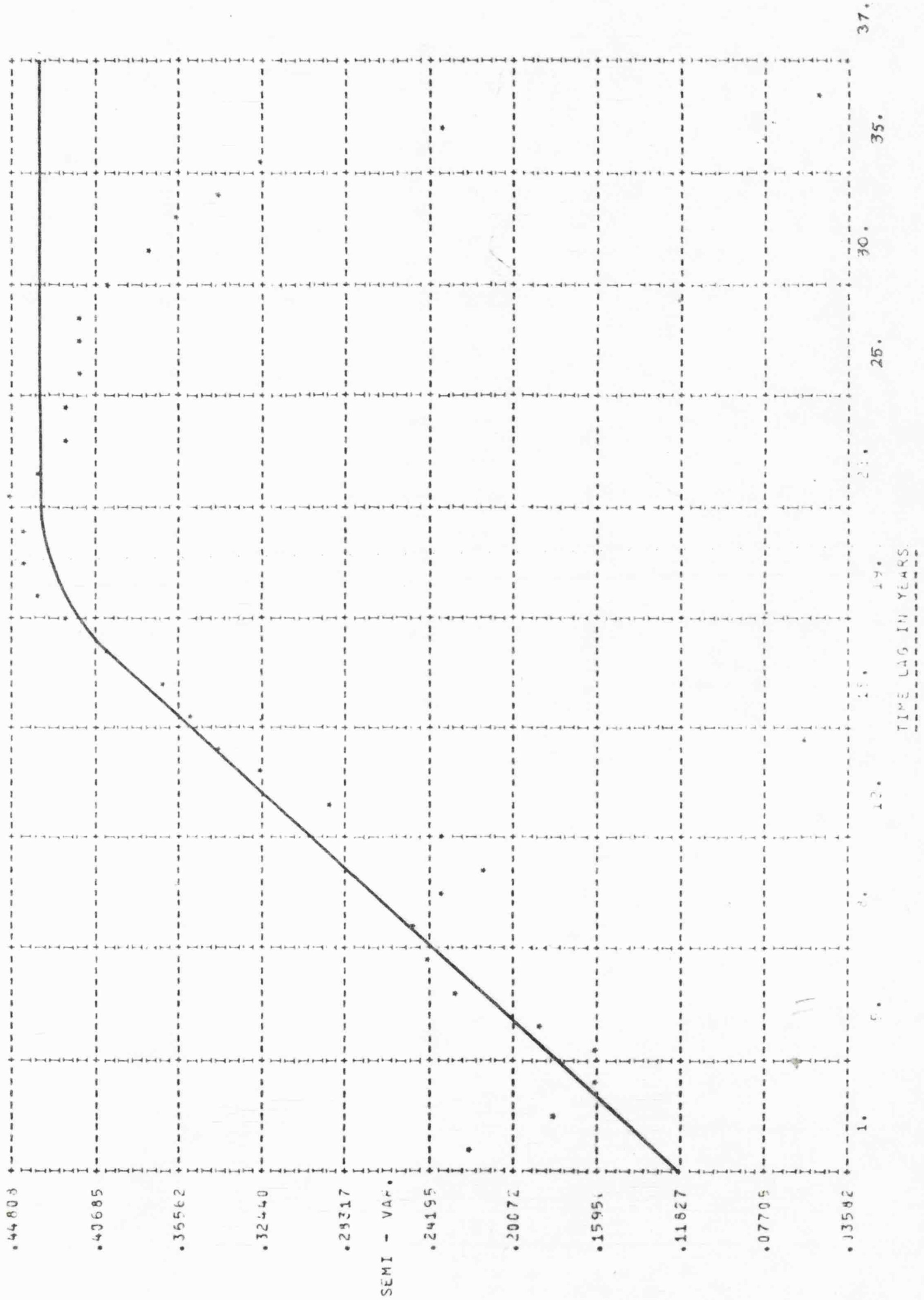
Figure 41



Price - Time Variogram for Gold

Figure 42

Price - Time Variogram for Silver



define the sill value, and the semi-variance at zero lag was taken as the nugget value. It is clear from this limited exercise that considerable future work remains to investigate the applicability of variogram modelling to price forecasting. However, such work is beyond the scope of this study, therefore, no futher effort was expended in this direction.

18.4 Price Kriging Procedure

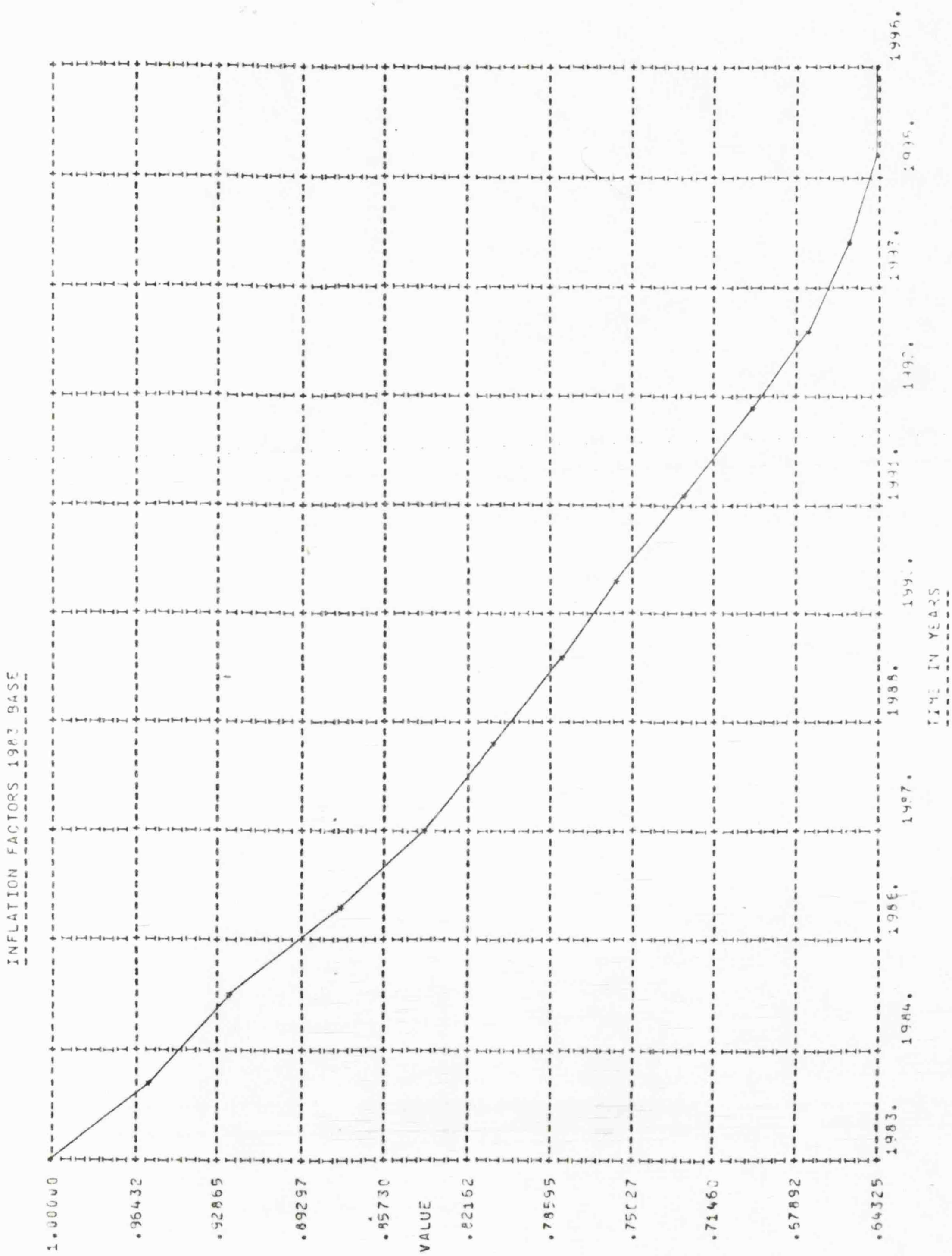
At this stage in the analysis all prices were on a 1945 basis, clearly a more relevant base was required. It was decided to base all calculations on 1983 US dollars. Hence, it was necessary to predict inflation forward and then recalculate it with respect to the 1983 predicted value. Subsequent price predictions could then be adjusted from a 1945 to a 1983 base. Once time extended beyond the range of any data set the predicted value was set equal to the mean value of the values within the range. Standard point kriging was used, with the solution of the simultaneous equation set being achieved by use of the augmented matrix method, rather than by matrix inversion, Davis (28).

The results of this process are given in Appendix F, together with the relevant statistical information on the results. The precision of the estimate was defined as the ratio of the average error of estimation to the average undiscounted predicted value, expressed as a percentage.

As can be seen from the results, the kriging consistently produced conservative results at levels of precision which appear to relate to the sill value. These predicted values are illustrated in Figures 43 to 48.

The predicted price values were discounted at a variety of rates to illustrate the effect of varying levels of profit demand on effective prices. This data is also presented in Appendix F. The price values used in subsequent parts of this study, were the mean undiscounted predicted price and plus and minus one and two standard deviations about this mean.

Figure 43



INFLATION FACTORS 1983 BASE
VALUE
TIME IN YEARS
1.00000
.96432
.92865
.89297
.85730
.82162
.78595
.75027
.71460
.67892
.64325
1983.
1984.
1986.
1987.
1988.
1996.

Figure 44

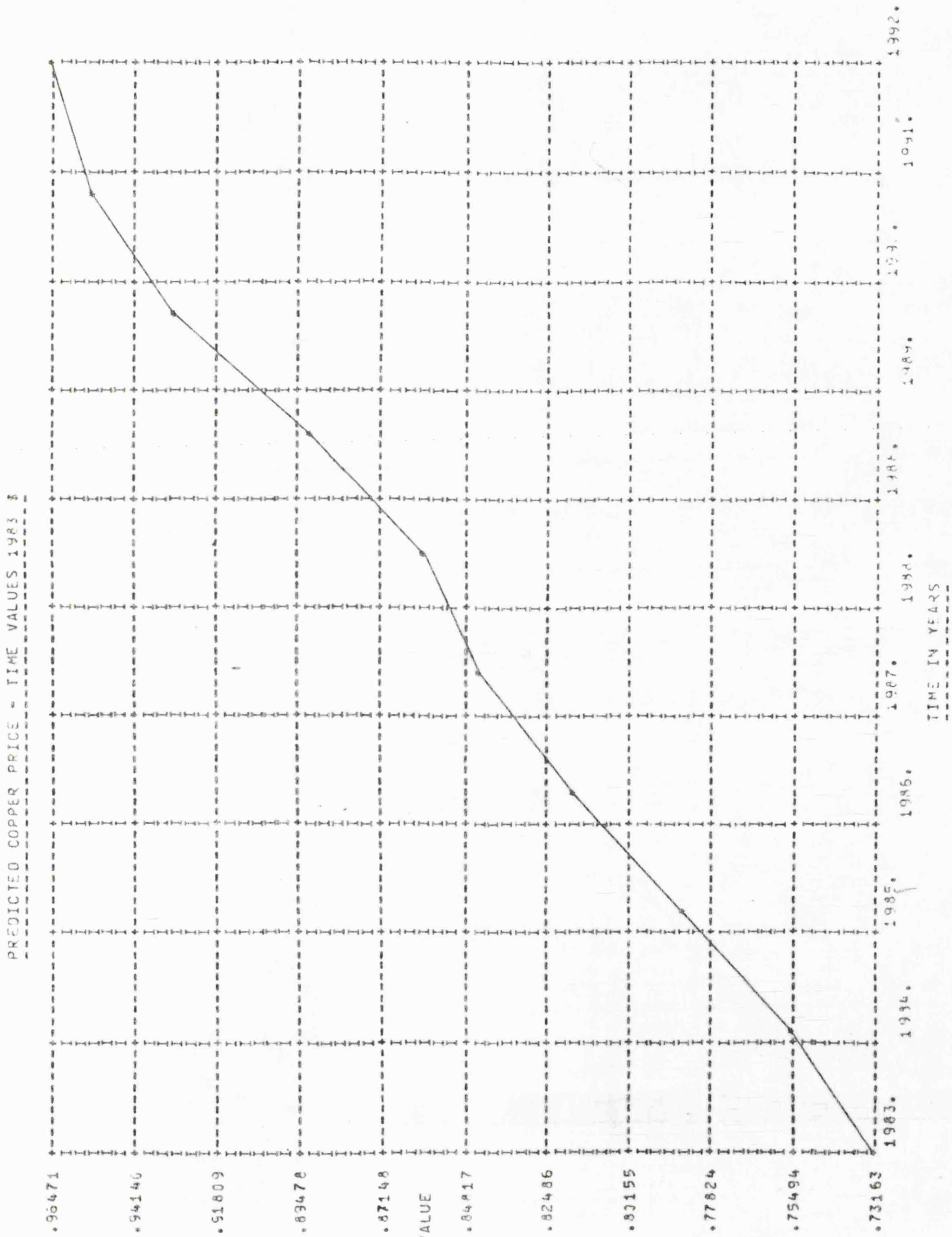
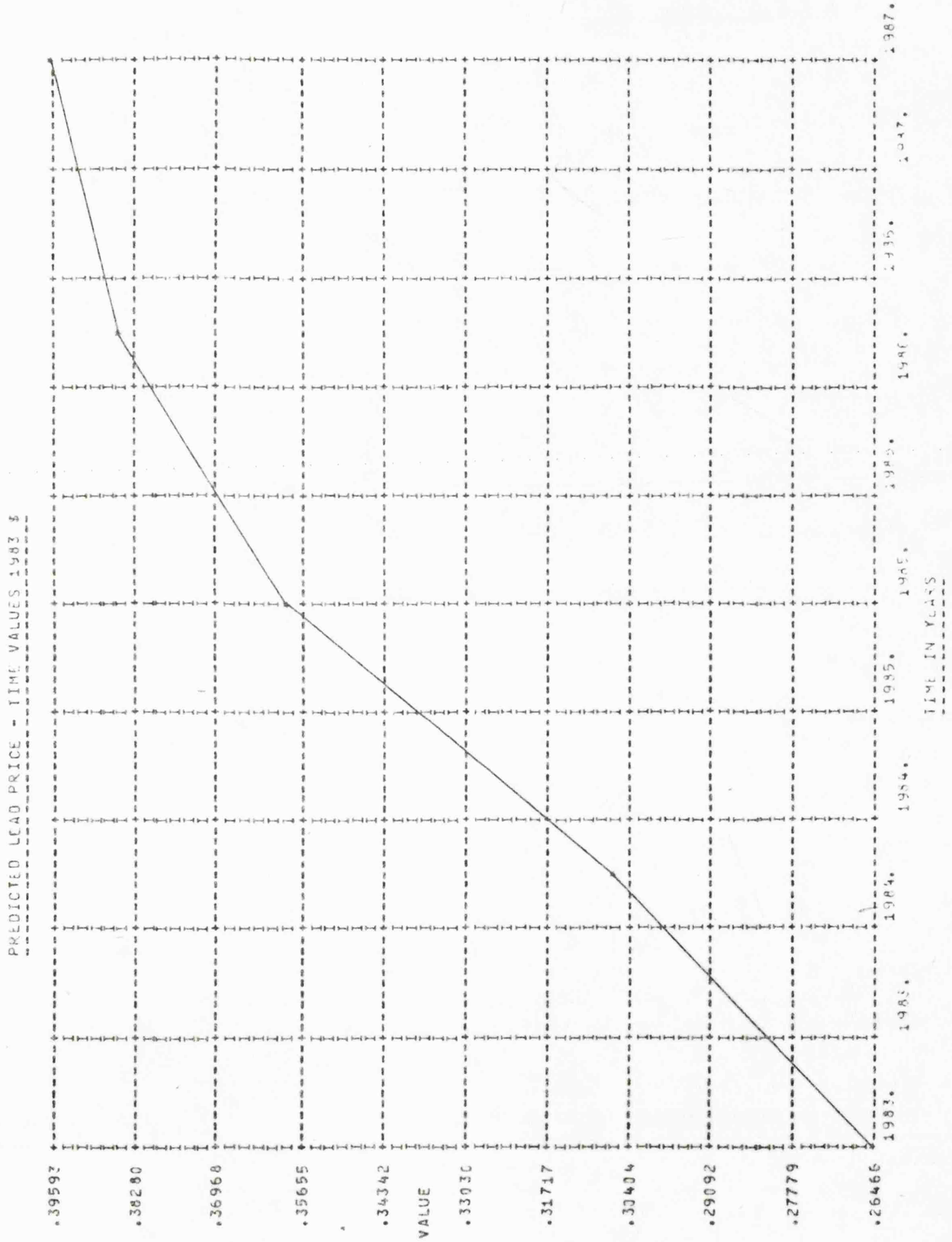

PREDICTED COPPER PRICE - TIME VALUES 1983 $
VALUE
TIME IN YEARS
1983.
1992.

Figure 45



PREDICTED LEAD PRICE - TIME VALUES 1983 $
.39592
.38280
.36968
.35655
.34342
VALUE
.33030
.31717
.30404
.29092
.27779
.26466
TIME IN YEARS

Figure 46

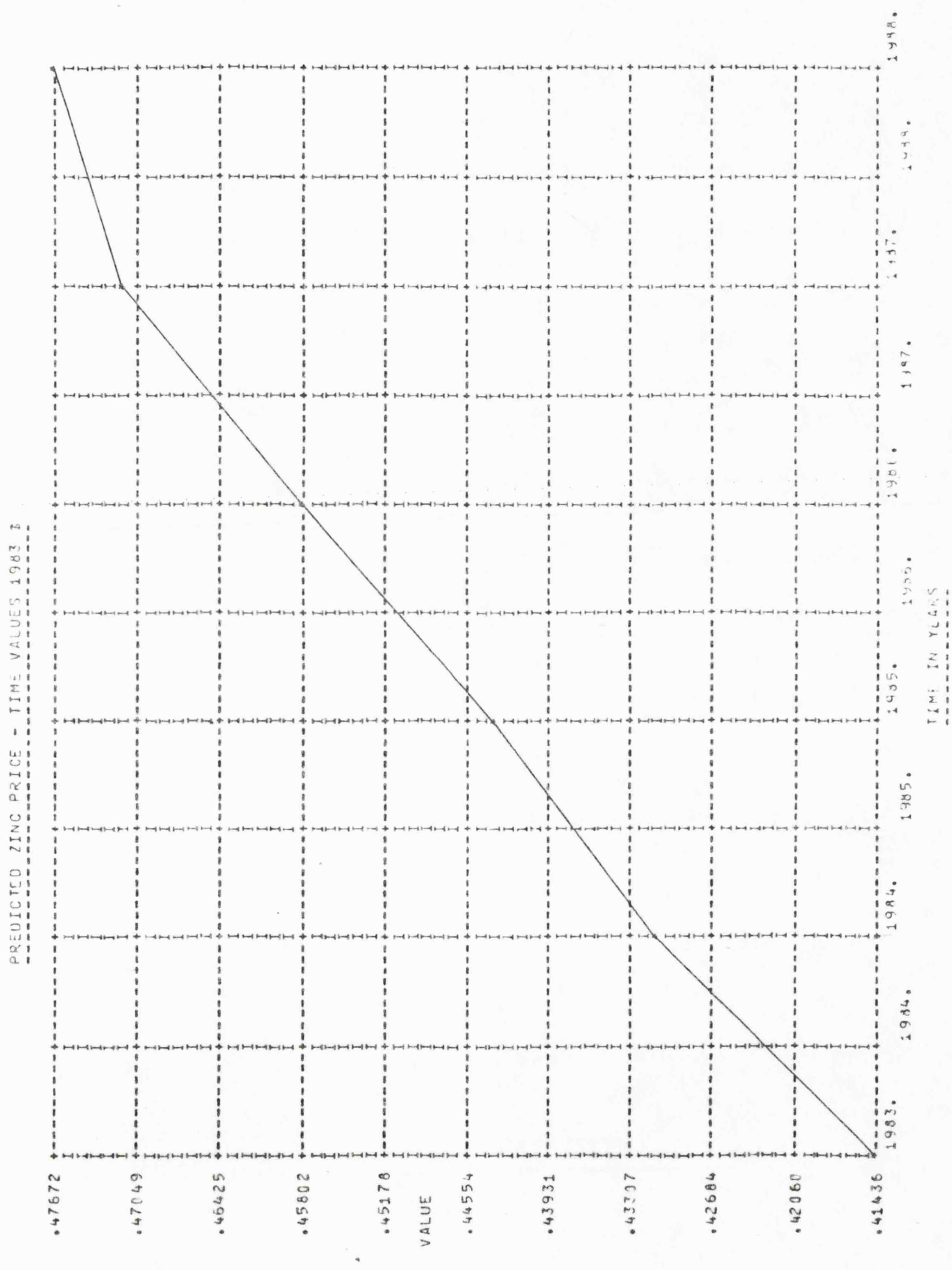
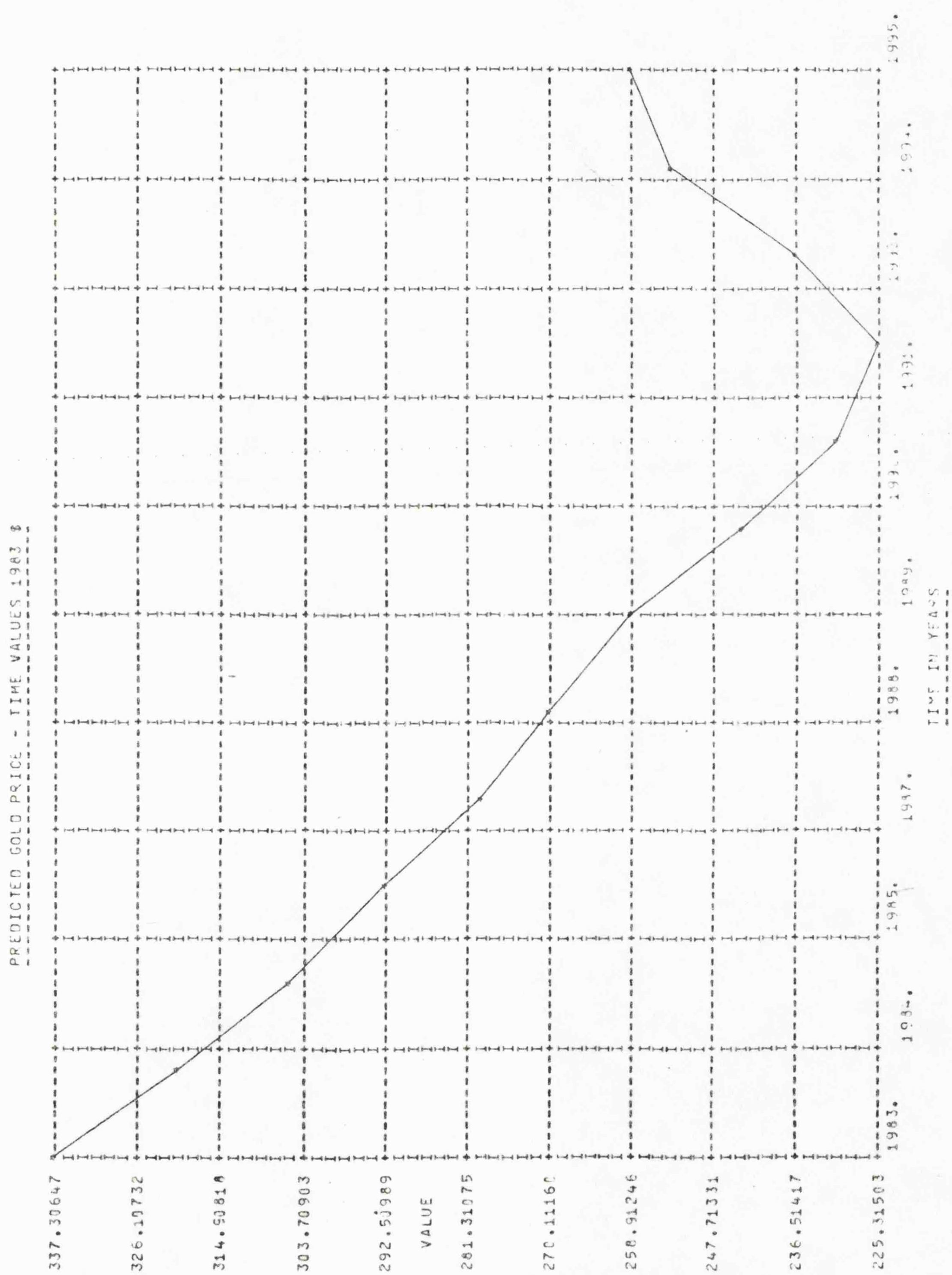
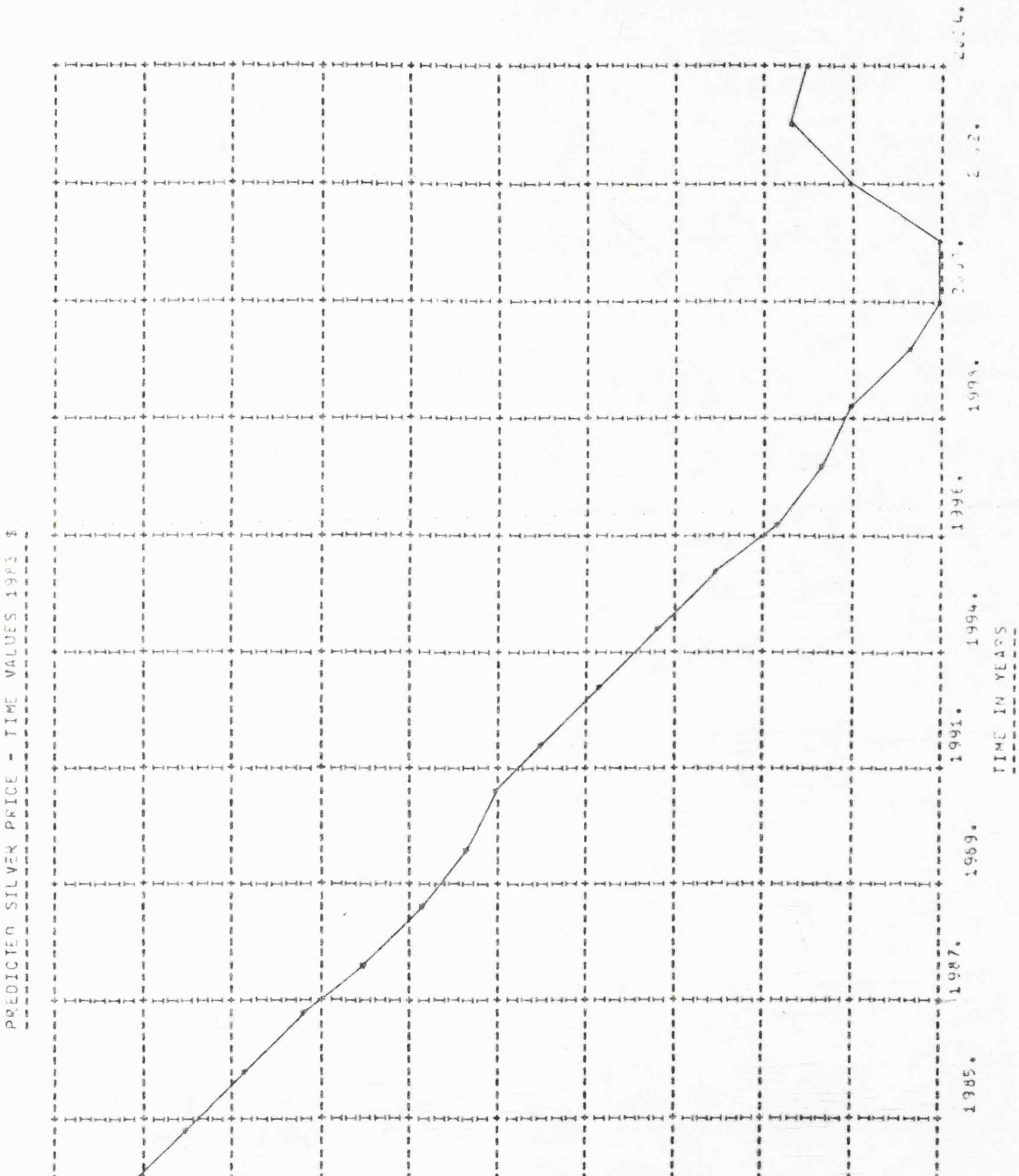

PREDICTED ZINC PRICE - TIME VALUES 1983 $
VALUE
.47672
.47049
.46425
.45802
.45178
.44554
.43931
.42684
.42060
.41436
1983.
1984.
1985.
TIME IN YEARS

Figure 47



PREDICTED GOLD PRICE - TIME VALUES 1983 $
VALUE
337.30647
326.10732
314.90818
303.70903
292.50989
281.31075
270.11160
258.91246
247.71331
236.51417
225.31503
TIME IN YEARS
1983.
1985.
1988.
1995.

Figure 48


PREDICTED SILVER PRICE - TIME VALUES 1983 $
VALUE
8.73843
8.44871
7.86925
7.57953
7.00007
6.42062
6.13089
1983.
1985.
1987.
1989.
1991.
TIME IN YEARS

19.0 CAPITAL AND OPERATING COST CALCULATION

The next step in producing a CPT for each commodity was to derive the operating costs and parameters, capital costs and financial factors required by the minimum reserve analysis described in Appendix A.

Lack of accurate published data on the parameters meant that estimates had to be used for the derivation of the various values. A variety of methods are available to perform such estimates, including those described by O'Hara (31) and Straam (38); however, it was decided to use the generally available computer based "MINING" system (43) because it produced repeatable results in a convenient format.

The results of this analysis for copper, lead, zinc, gold and silver are given in Appendix G. Six mining methods were used for each commodity, namely:-

- zero strip pit
- open pit mining
- natural caving
- artificical caving
- self-supporting
- artificially supported

These were matched to appropriate average deposit types. Porphyry was matched to the first three methods, volcanogenic massive sulphide to the fourth, sedimentary to the fifth and contact metamorphic to the last.

20.0 COMMODITY PROFITABILITY THRESHOLD CALCULATION

For the purposes of this illustrative example it was arbitrarily decided to demand a constant rate of return of 5% DCFROR on long term mineral investments. This figure is approximately 2.5 times the interest demanded, in constant terms, by banks for long term, low risk investments. The higher rate demanded for mining reflects the risky nature of the investment.

Using the information generated by the procedures described in the previous chapters, minimum reserve analyses were run for each commodity for each type of mining method. In each case the average grade +/- one and two standard deviations, together with the average price +/- one and two standard deviations were used as grade and price input ranges. For depth, open pit mines were assumed exploited down to 1000 feet from surface, with 10 * 100 foot increments being investigated. For underground mines, the operating limit was assumed to be 10,000 feet, with 10 *1000 foot increments in this range being used.

This generated 250 cases for each mining method for each commodity. So each commodity was subjected to 1500 iterations, producing a total of 7500 cases.

No other input parameters were varied.

Justification for this lies in the fact that this exercise is by way of being an illustrative example rather than a universal solution. In each case it was assumed that standard sulphide flotation was the processing method used. A summary of the results of is given in Appendix H.

In order to reduce the amount of work involved in this example rather than take all possible cases, values were averaged over depth for both surface and underground mines. The resultant data was plotted overlaying the CSP's and is presented in Figures 49 to 53.

The lower grade limit on the CPT was derived from a calculation of the operating cutoff grade. That is to say, the grade below which an operating loss would result. Details of this calculation and a summary of the results are given in Appendix I.

Figure 49 COPPER - COMMODITY PROFITABILITY THRESHOLD

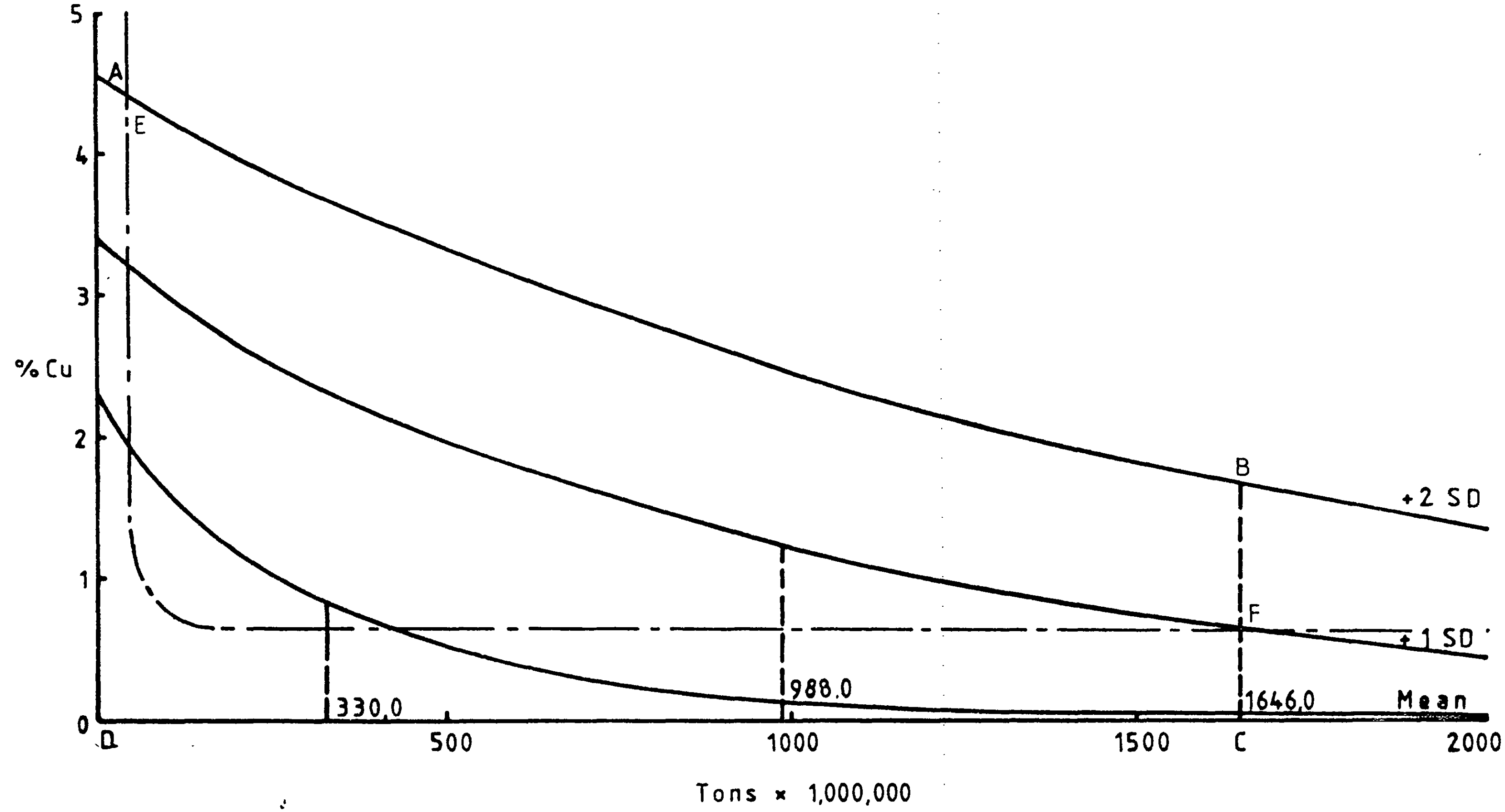


Figure 50 LEAD - COMMODITY PROFITABILITY THRESHOLD

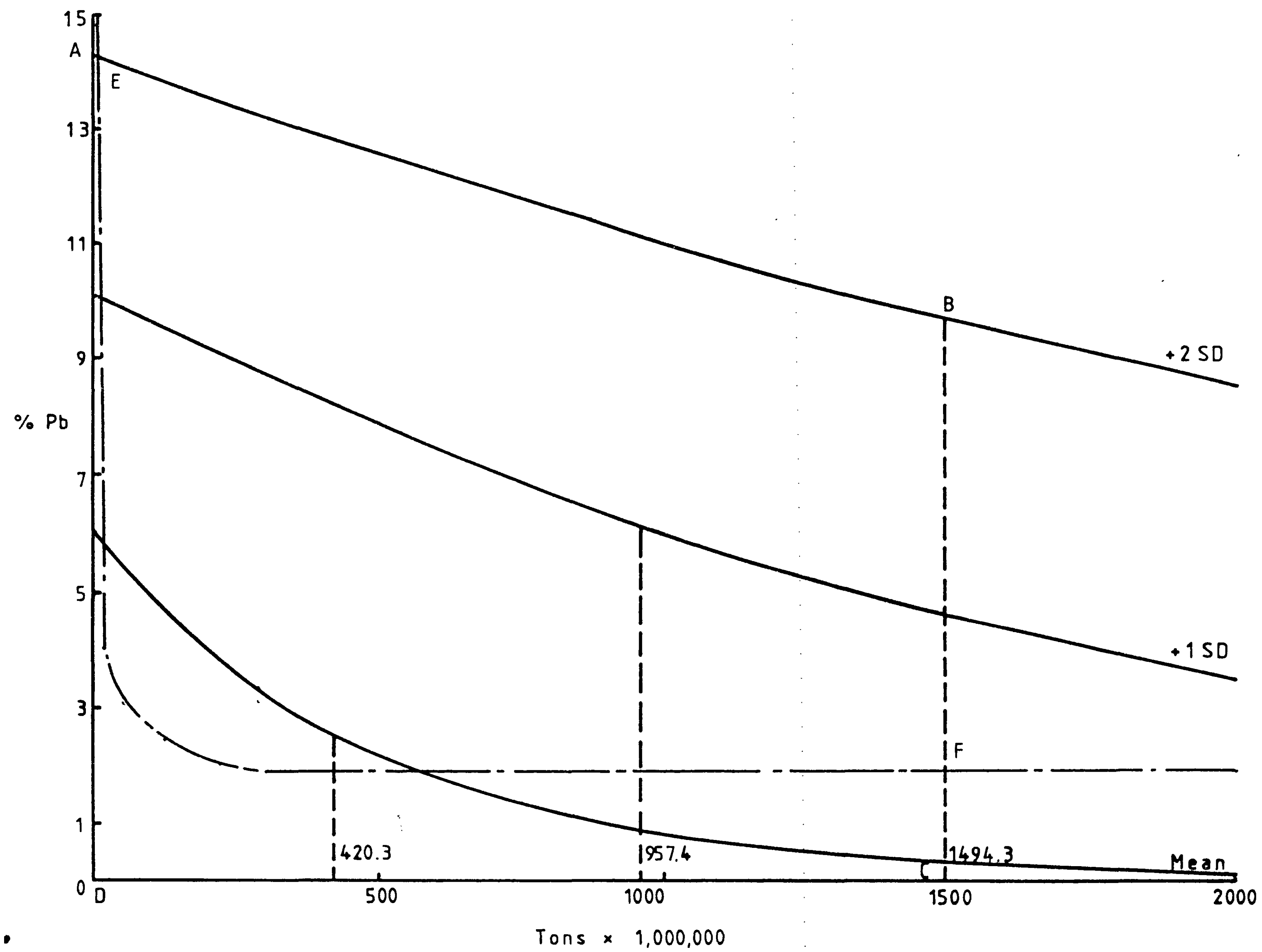


Figure 51 **ZINC - COMMODITY PROFITABILITY THRESHOLD**

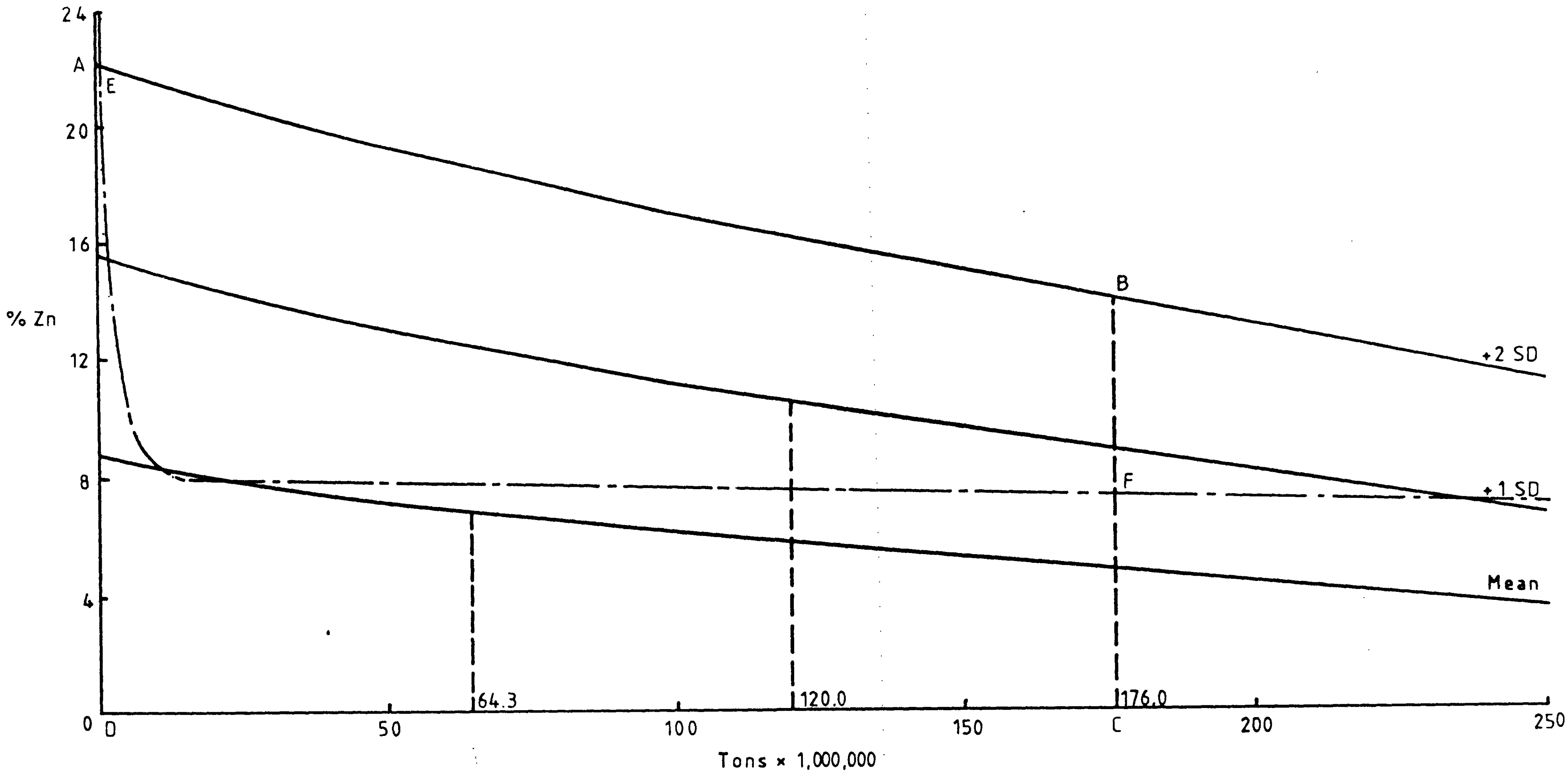


Figure 52 GOLD - COMMODITY PROFITABILITY THRESHOLD

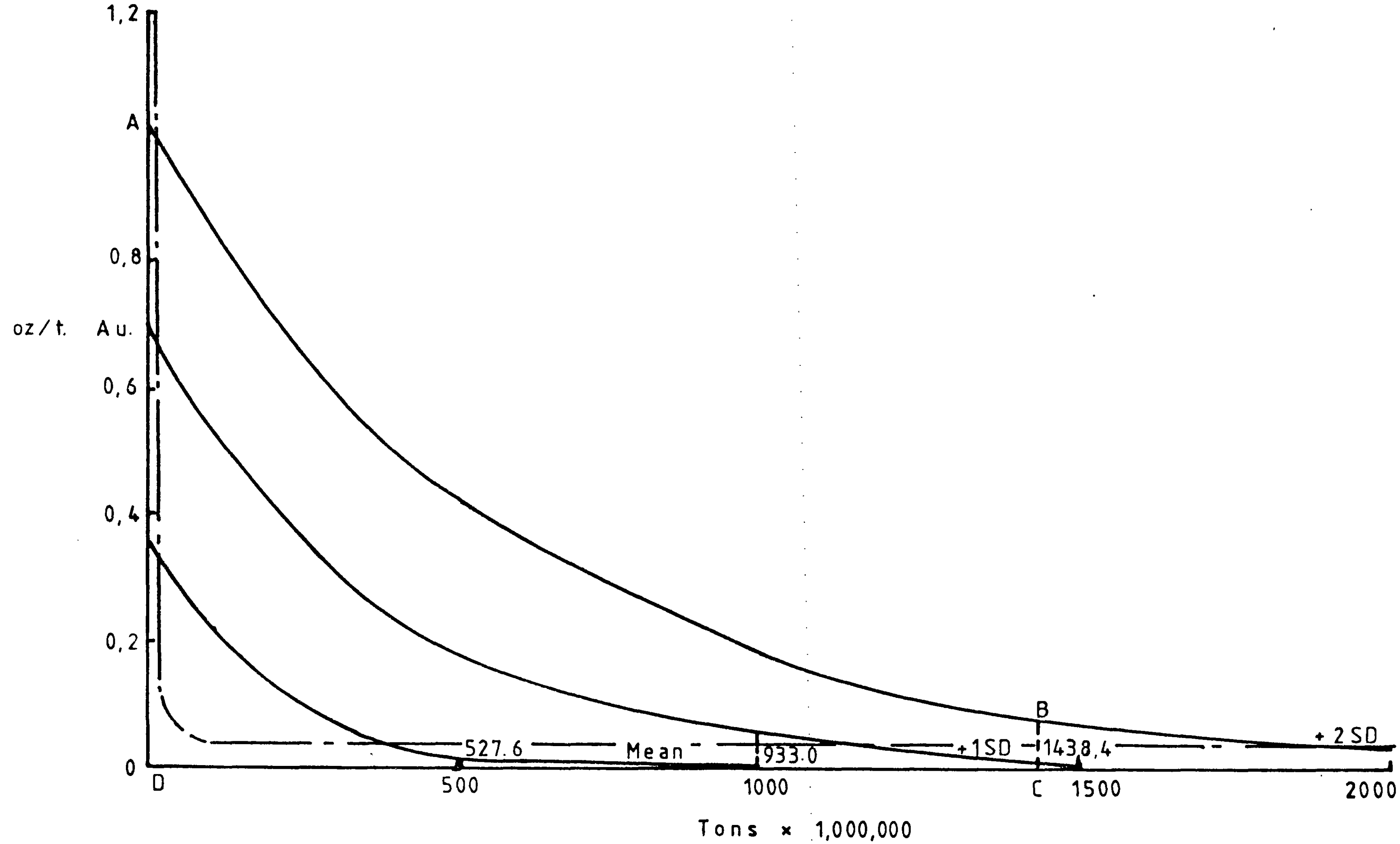
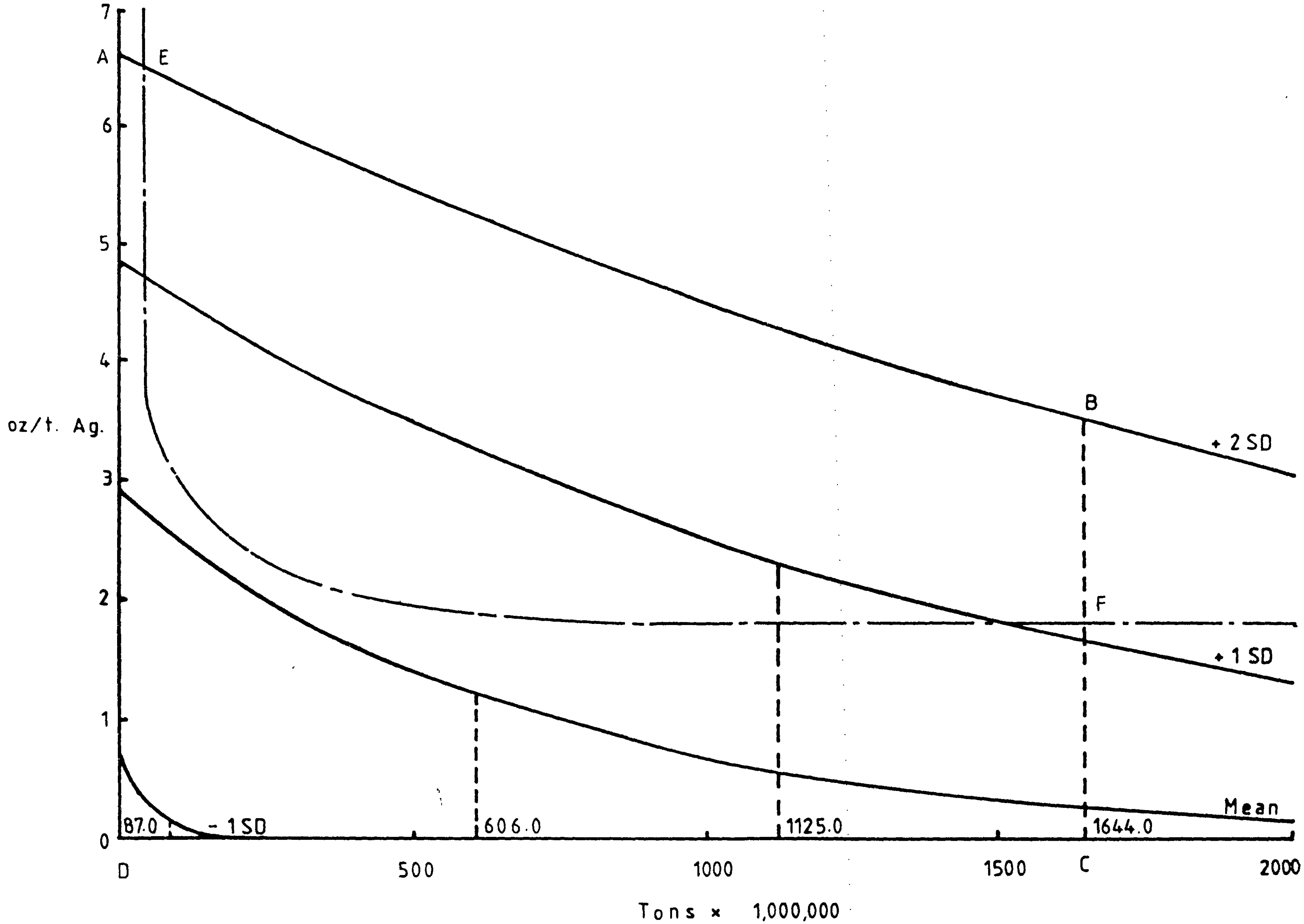


Figure 53 **SILVER** - COMMODITY PROFITABILITY THRESHOLD



21.0 CALCULATION OF PROBABLE SUCCESS

Referring to Figures 25 to 30, it will be appreciated that the realms of possible source of the various commodities are governed by the confidence level used. At the upper limit in this study, the grade and tonnage range is bounded by plus two standard deviations. This produces the polygon ABCD on each CSP, the area of this polygon was measured for each commodity, and is tabulated below.

Referring to the CPT's given in Figures 49 to 53, the region of probable success for each commodity lies between the minimum confidence boundary and the plus two standard deviations limit. That is to say the area bounded by polygon EBF. The region of probable failure is defined by polygon AEFCD. The area of each of these polygons was also measured for each commodity and the Chance of Success, Relative Chance of Success and Total Chance of Success were calculated according to the logic described in Chapter 16. The results are given below.

Commodity	ABCD	EBF	AEFCD	CS(i)%	RCS(i)%
Copper	96.78	71.05	25.73	73.41	20.79
Lead	186.66	155.21	31.45	83.15	23.55
Zinc	157.32	84.82	72.50	53.92	15.27
Gold	54.13	47.27	6.86	87.34	24.74
Silver	164.51	90.86	73.65	55.23	15.65
Total:				353.05	

Total Chance of Success = 353.05 / (5 * 100)

= 70.16 %

Summarizing, therefore, it may be stated that assuming all models and inputs to be correct, under normal circumstances at a confidence level of 97.73%, for the commodites cosidered there is no more than a 70.16% chance of discovered deposit yielding, on exploitation a DCFROR of 5.0%.

22.0 CALCULATION OF GENERAL EXPLORATION POTENTIALS

In order to complete this illustrative example it is now necessary to calculate the indices as described in Chapter 16, for each commodity under consideration.

22.1 Relative Socio-Political Index Calculation

Referring to the original definition in Chapter 7:

Factor	Region						
	East	Europe	N.Am	Australia	Africa	USSR	S.Am
Capital	50	50	100	60	20	1	30
Long term	10	100	100	100	1	100	10
Short term	50	100	100	90	10	100	20
Environm.	60	10	1	50	100	100	100
Ecology	70	1	1	60	100	100	100
Land use	50	10	50	100	100	100	70
Infrastr.	10	100	100	30	1	20	10
Taxes	20	50	100	50	50	1	50
Royalty	70	10	100	70	20	1	50
Legal	40	100	40	100	1	1	20
Labour	10	100	100	100	1	50	30
Social	10	100	100	80	1	30	30
Total:	450	731	892	890	405	584	520
SPI	0.69	1.13	1.37	1.37	0.62	0.90	0.80

The above SPI's, Socio-Political Indices, have been devloped on a regional basis, they were then converted to a commodity basis by factoring the above SPI's as a function of the reserves of that commodity in that region. Details of this calculation are given in Appendix J. The results are summarized as follows:

Commodity	SPI	RSPI
Copper	0.94	0.18
Lead	1.15	0.22
Zinc	1.22	0.23
Gold	0.96	0.18
Silver	1.06	0.19

22.2 Relative Price Time Index

Referring to the original definition in Chapter 8 above, and using the data generated in Chapter 18, the following calculation was made:

Commodity	Sill	Nugget	Range	Av. Price
Copper	0.00454	0.00060	10	0.8549
Lead	0.00021	0.00018	5	0.3426
Zinc	0.00019	0.00015	6	0.4494
Gold	526.12010	180.83510	13	271.0279
Silver	0.43780	0.11821	22	7.3123

Average range = 11.2 years

Commodity	RS	RN	RR	PTI	RPTI
Copper	94.69	99.93	89.29	1.89	0.23
Lead	99.39	99.95	44.64	1.63	0.20
Zinc	99.96	99.97	53.57	1.69	0.21
Gold	-94.12	33.28	116.07	0.37	0.05
Silver	94.01	98.38	196.43	2.59	0.31
Total					1.00

22.3 Relative Market Index Calculation

Using the definition from Chapter 4, the RMI's were calculated as follows:

Factor	Commodity				
	CU	PB	ZN	AU	AG
Market size	100	10	50	100	50
Domestic market size	80	10	50	100	50
Recycling	50	10	70	1	1
Tariffs	50	50	50	50	50
Bureaucratic impact	50	10	50	1	50
Political impact	50	50	50	1	50
Monopoly	50	50	50	50	50
Cartel	50	50	50	50	50
Substitution	50	5	50	100	100
Alternate potential	5	1	80	100	70
Price time cycle	50	10	50	50	50
Other	50	1	50	100	50
Market Explore Index	0.98	0.40	1.00	1.08	0.96
RMI	0.22	0.09	0.23	0.24	0.22

The above scalar values were determined qualitatively.

22.4 Relative Commodity Exploration Index Calc.

Using the definitions of Chapter 5, and assigning values qualitatively, the following results were produced:

Factor	Commodity				
	CU	PB	ZN	AU	AG
Unit value	10	2	5	100	90
Abundance	1	10	10	100	70
Ameanability	90	90	90	100	90
Removal	50	50	50	50	50
Supply	50	50	50	100	50
Monopoly	50	50	50	50	50
Strategic signif.	50	50	50	1	50
Location	50	50	50	10	50
Political stability	50	50	50	10	50
Reserve/demand ratio	20	20	30	100	50
Import situation	100	50	50	100	50
Environmental impact	50	1	50	50	50
Bureaucratic impact	50	5	50	1	50
Other	50	10	50	100	50
Basic Comm. Ex. Ind.	0.96	0.70	0.91	1.25	1.14
RCEI	0.19	0.14	0.18	0.25	0.24

22.5 General Exploration Potential Calculation

All the elements are now in - place for the calculation of the General Exploration Potential for each commodity, according to the definitions made in Chapter 8. The following were the results:

Commodity	RSPI	RPTI	RMI	RCEI	GEP
Copper	0.18	0.23	0.22	0.19	0.82
Lead	0.22	0.20	0.09	0.14	0.65
Zinc	0.23	0.21	0.23	0.18	0.85
Gold	0.18	0.05	0.24	0.25	0.82
Silver	0.19	0.31	0.22	0.24	0.96

As will be appreciated, the sensitivity of the individual elements of these indices may be investigated.

23.0 CALCULATION OF EXPLORATION BUDGET ALLOCATION

Thus far in the example it has been demonstrated that investment in mineral exploration does in fact have the potential to generate a real rate of return of 5% . Moreover, for the commodities considered this required rate of return will only be satisfied for 70.16% of discovered deposits. Using the logic developed in Chapter 16, the following allocation of the exploration budget was made.

Assuming, the Available Investment = $ 50.0M

Maximum Justifiable Exploration Budget = TCS * AI

= $ 35.08M

and, Investment in Non-exploration Alternatives = $ 14.92M

Maximum Justifiable Budget per Commodity may be calculated as follows:

Commodity	RCS(i)%	MJEB(i),$M	MBC(i),$M
Copper	20.79	35.08	7.29
Lead	23.55	35.08	8.26
Zinc	15.27	35.08	5.36
Gold	24.74	35.08	8.68
Silver	15.65	35.08	5.49

The intangibles are now introduced by use of the General Exploration Potential for each commodity to produce a Modified Investment per Commodity as follows:

Commodity	MBC(i),$M	GEP(i)%	MIC(i),$M
Copper	7.29	82.0	5.98
Lead	8.26	65.0	5.34
Zinc	5.36	85.0	4.56
Gold	8.68	82.0	7.12
Silver	5.49	96.0	5.27
Total :			28.27

From which it was found that the Total Justifiable Budget was $28.27M, compared to the Maximum Justifiable Exploration Budget of $35.08M. The difference, $6.81M is non - justifiable exploration expense. Thus on the basis of quantified analysis and qualified "hunches", it has been found that only $28.27M of the available $50.00M can actually be justified for investment in mineral exploration, for the considered commodities, ie, some 57% of the original. So, in this case, it could be argued that a rational decision making approach has saved $21.73M of investment in mineral exploration, which, at a 97.73% confidence level, would have failed to yield the required return on investment.

24.0 CALCULATION OF DEPOSIT TYPE ALLOCTAION

Using the logic described in Chapter 15, and the Commodity Source Profiles generated in Chapter 20, the Deposit Allocation Diagrams, presented in Figures 54 to 58 were produced. From these diagrams the following Deposit Allocations were calculated. The details of these calculations are presented in Appendix K.

Deposit	Budget / Deposit Type, $M.					
Type	CU	PB	ZN	AU	AG	Total
Porphyry	3.53	3.95	0.00	2.14	3.80	13.42
Sediment.	0.00	0.00	0.00	1.78	0.00	1.78
Cont. Met.	0.00	0.16	0.50	0.00	0.00	0.66
Stratiform	1.91	0.21	1.09	0.00	0.00	3.21
VMS	0.30	0.69	1.19	0.21	0.95	3.34
Hydrotherm	0.00	0.06	0.05	2.70	0.26	3.07
Complex	0.24	0.27	1.73	0.29	0.26	2.79
Total,$M.	5.98	5.34	4.56	7.12	5.27	28.27

From the above distribution of funds, the orientation of exploration effort becomes clear in detail. The remaining step required to complete this example is the specification of guidelines for the minimum size and grade of the various deposit types within each commodity. This is discussed in the next chapter.

Figure 54 COPPER - DEPOSIT ALLOCATION DIAGRAM

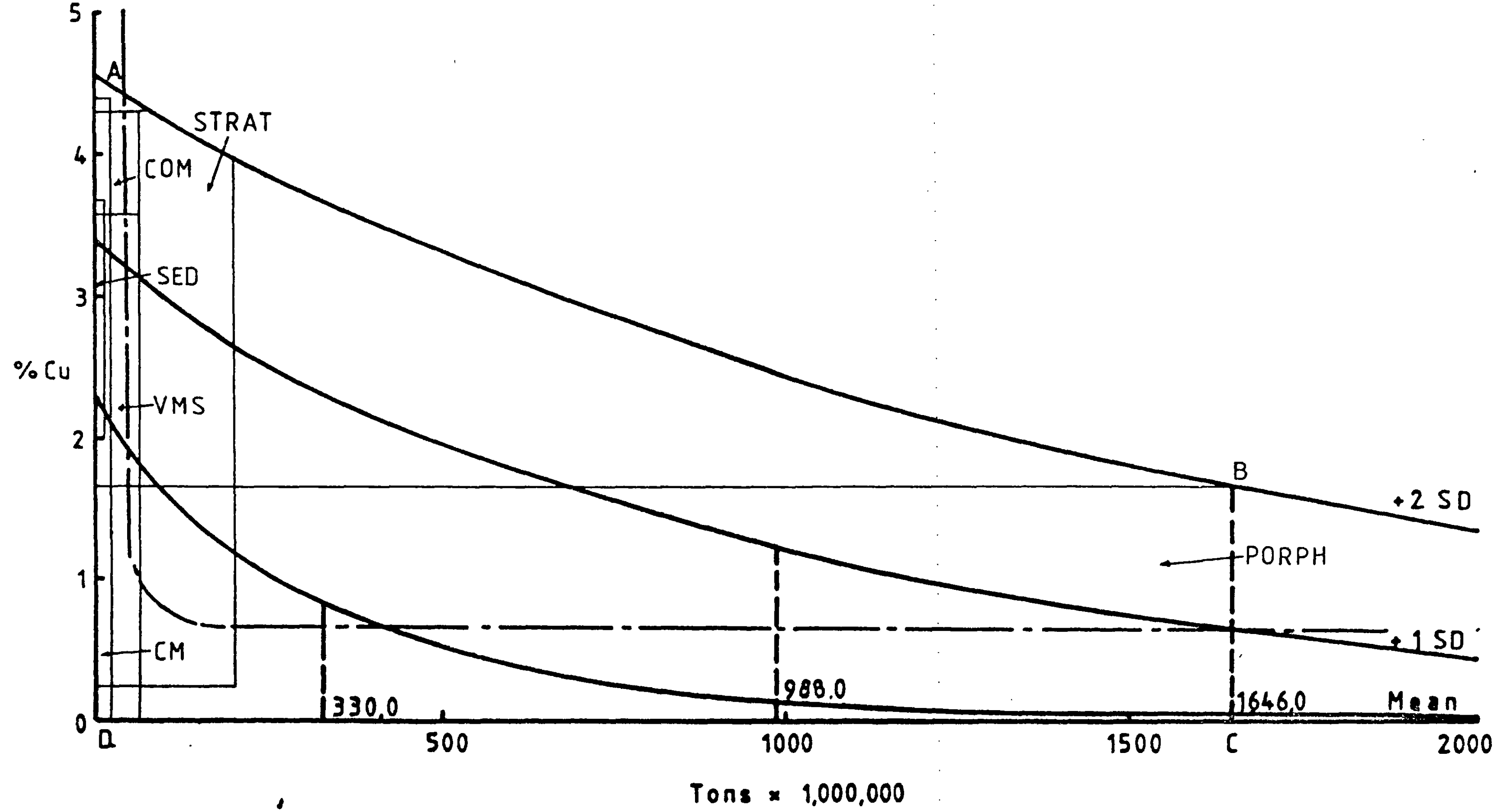


Figure 55 LEAD - DEPOSIT ALLOCATION DIAGRAM

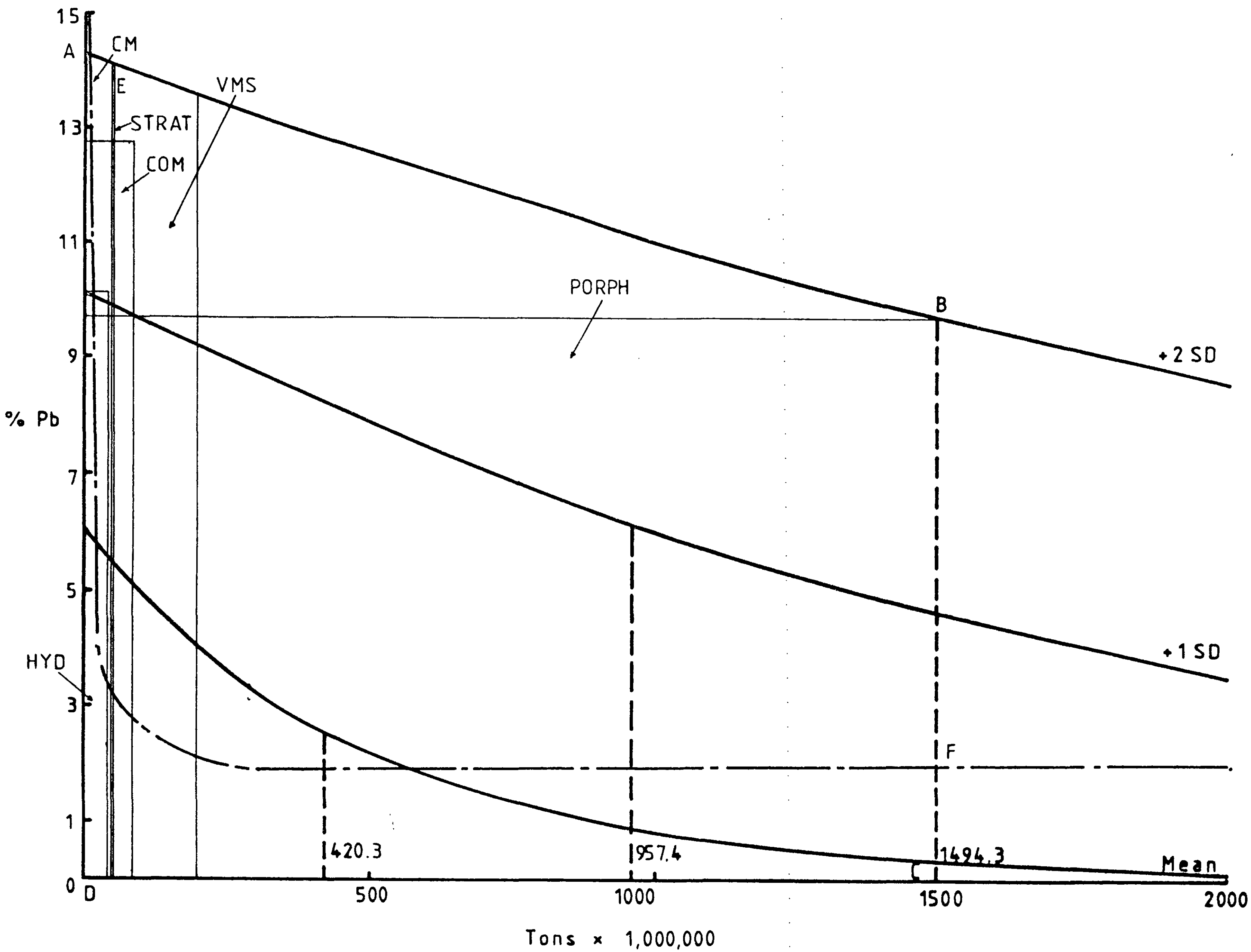


Figure 56 **ZINC - DEPOSIT ALLOCATION DIAGRAM**

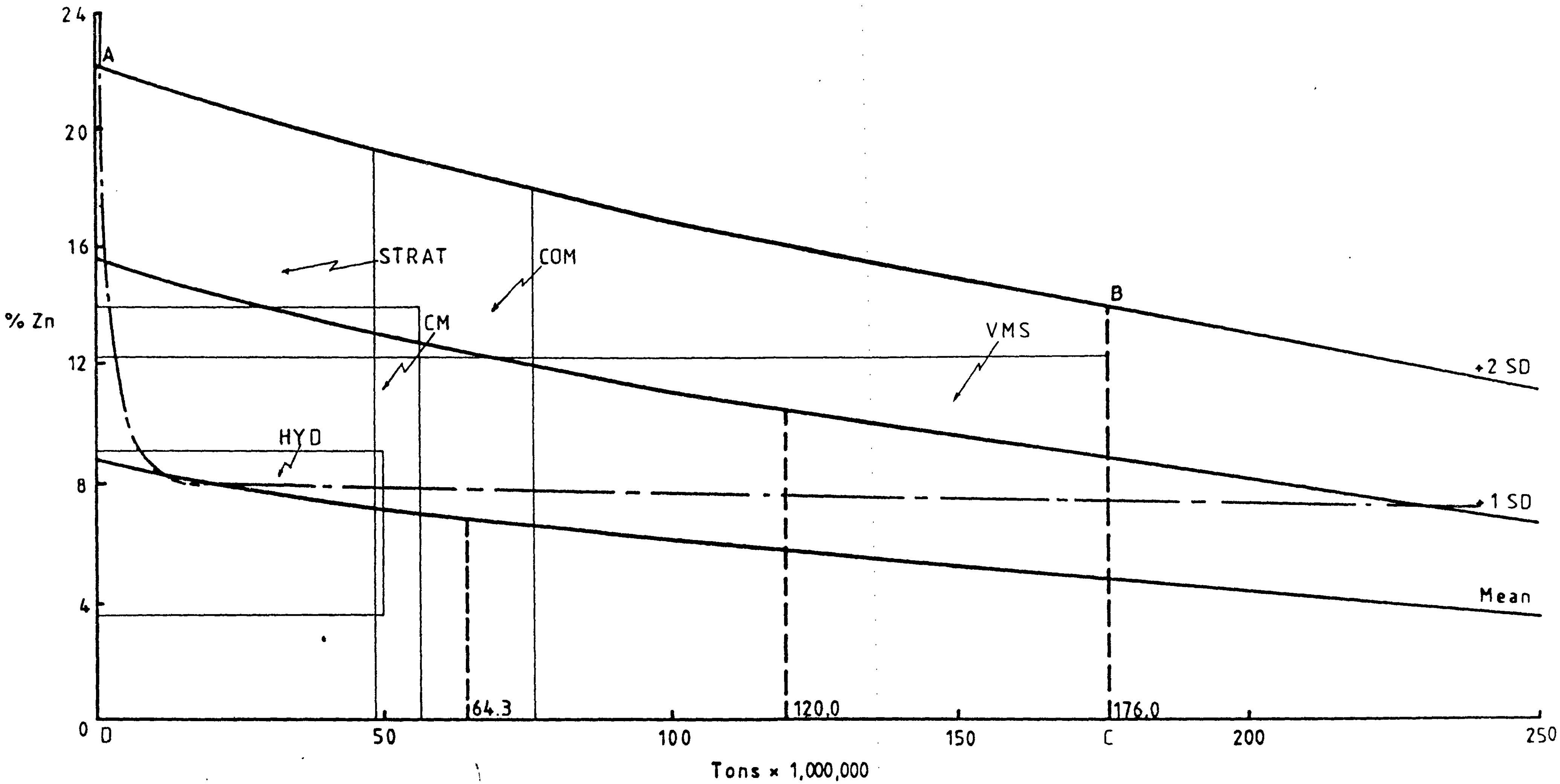


Figure 57 GOLD - DEPOSIT ALLOCATION DIAGRAM

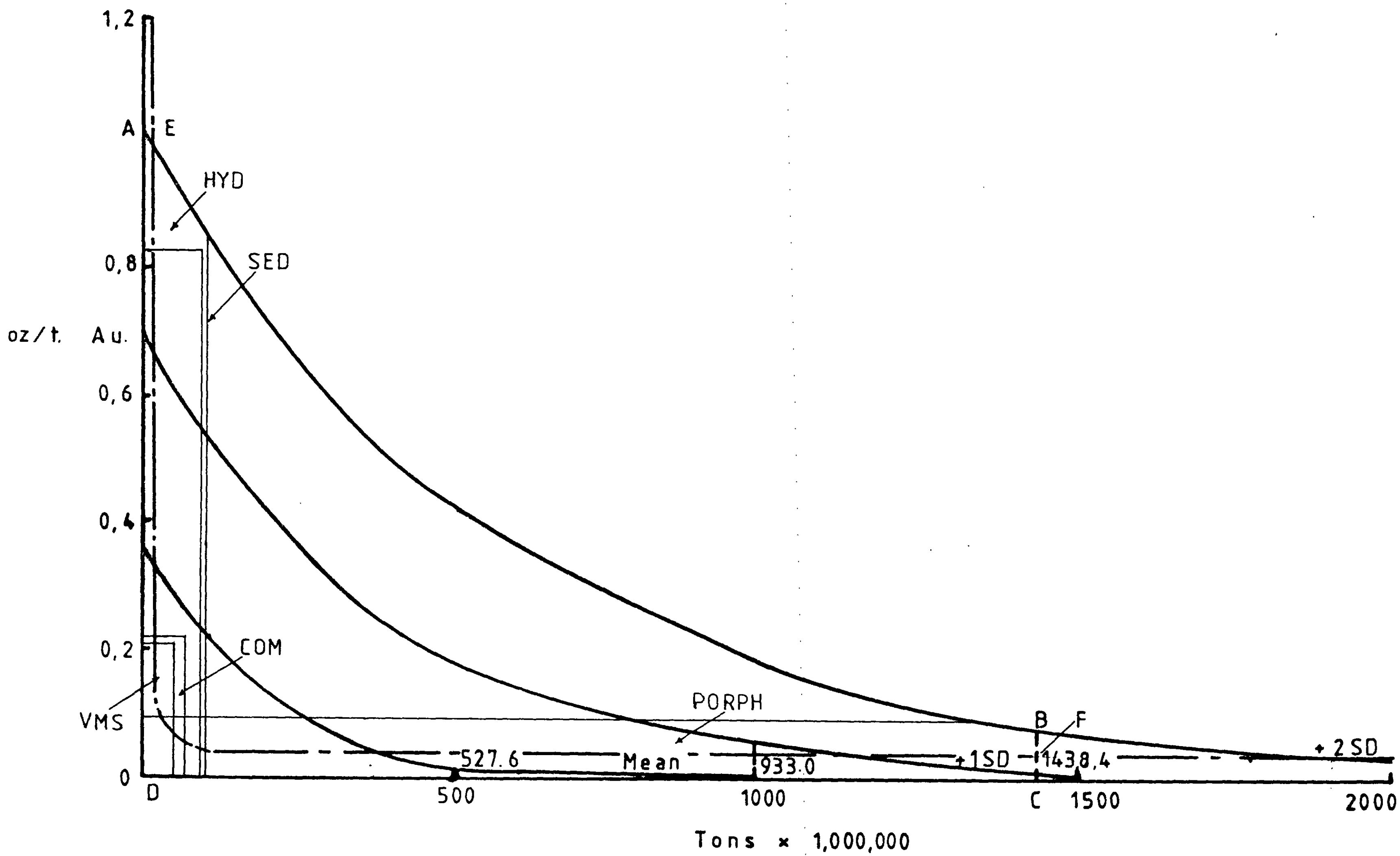
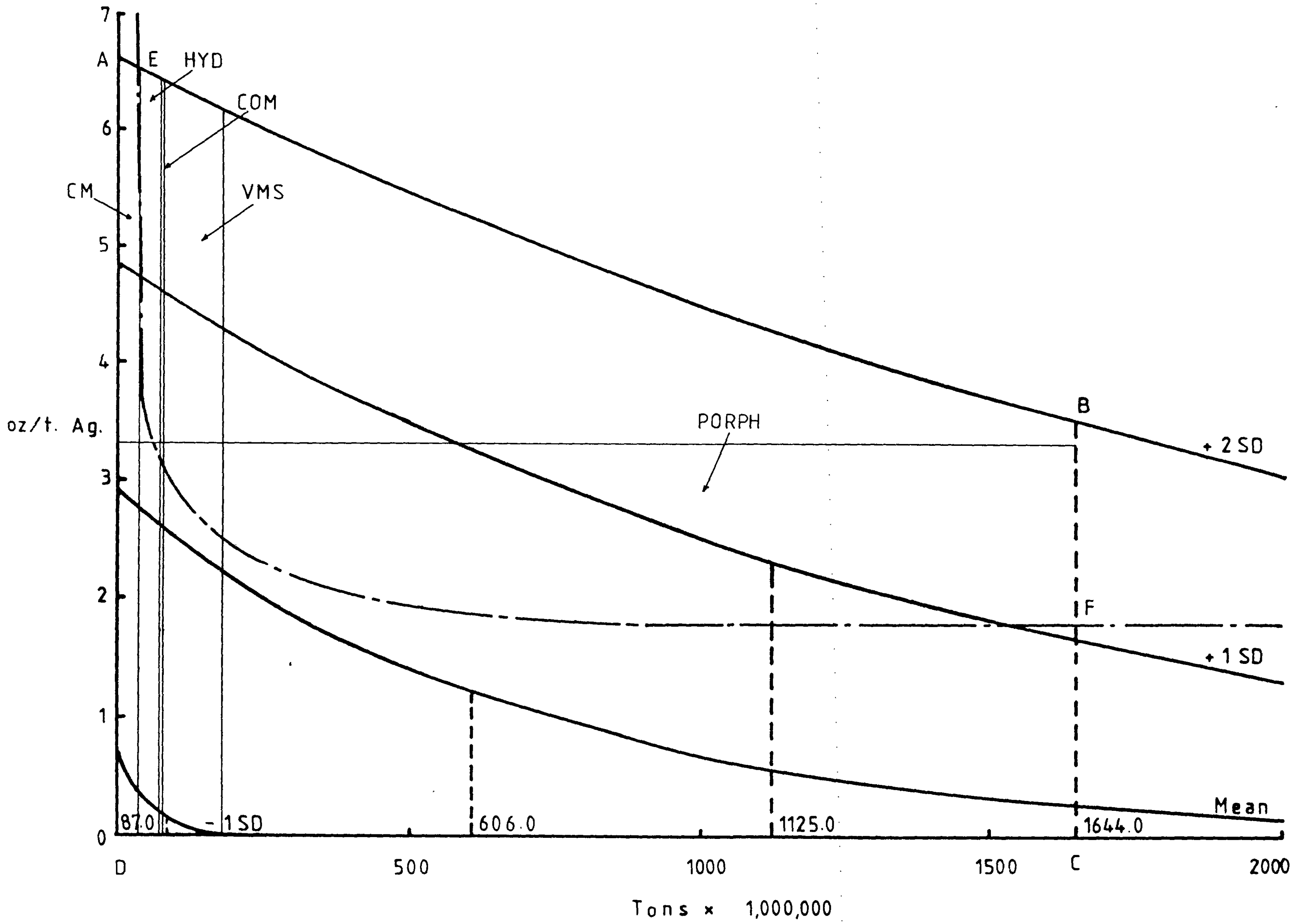


Figure 58 SILVER - DEPOSIT ALLOCATION DIAGRAM



25.0 GRADE - TONNAGE CUTOFF SPECIFICATION

The final task to be completed is the specification of guidelines to the minimum grade tonnage limits for each deposit type that justifies exploration within each commodity. The logic behind the task is as follows.

At 97.73% confidence level the upper limit of grade and tonnage is plus two standard deviations above the mean. The possible range of grade and size thus runs from zero to plus two standard deviations. The proportion of that range that will produce a successful result is defined by the excess, XS(i,j), described in Chapter 15. Therefore, the cutoffs may be derived as follows:

Lower grade limit = 0.00
Lower tonnage limit = 0.00

Upper grade limit = mean grade + (2.0 * s.d)
Upper size limit = mean size + (2.0 * s.d)

Proportion of the grade range that will meet cutoff criteria = upper grade limit * XS(i,j)

and, for tonnage = upper size limit * XS(i,j)

So, limiting cutoff = upper limit - acceptable range

Hence, the cutoff for grade and tonnage are determined. This calculation was carried out for each deposit type justifying expenditure for each commodity. The details of the calculation are given in Appendix L, and the results are summarized below:

Cutoff Grades, % or oz./ton

Deposit Type	CU	PB	ZN	AU	AG
Porphyry	0.66	2.09	–	0.0410	2.05
Sedimentary	–	–	–	0.1640	–
Contact Meta.	–	4.79	8.56	–	–
Stratiform	1.63	4.75	9.36	–	–
VMS	2.84	4.25	7.68	0.1150	3.32
Hydrothermal	–	7.76	8.89	0.3380	9.01
Complex	3.00	3.75	10.05	0.1000	4.49

Cutoff tonnages, M

Deposit Type	CU	PB	ZN	AU	AG
Porphyry	658.40	328.79	–	661.66	1019.28
Sedimentary	–	–	–	16.90	–
Contact Meta.	–	13.45	33.86	–	–
Stratiform	71.14	16.80	15.26	–	–
VMS	47.00	50.45	117.07	29.10	76.11
Hydrothermal	–	34.60	43.91	9.86	46.07
Complex	46.34	22.65	22.65	32.27	55.37

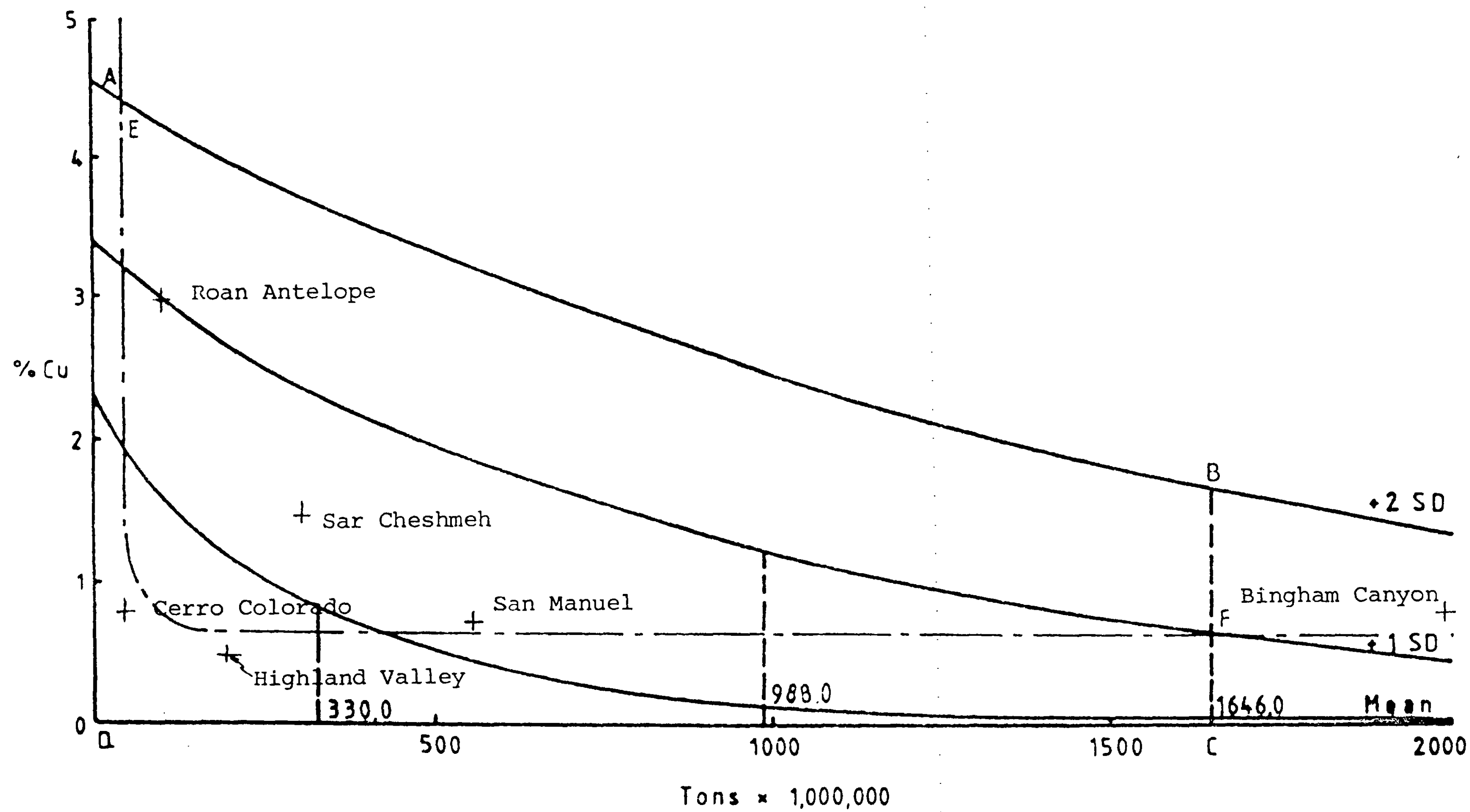
26.0 INVENTORY EVALUATION

A brief evaluation of the inventory of basic deposits was performed in order to further illustrate the consequences of the application of the theory described in this thesis. The results of the numerical example were applied to the inventory of deposits from which the Commodity Source Profiles were derived.

The data used in this evaluation comprised the basic deposit information for copper, lead, zinc, gold and silver as presented in Appendix C; the Commodity Profitability Thresholds as illustrated in Figures 49 to 53; and the cutoff grades and tonnages calculated in Chapter 25.

The potential of each deposit within each of the above mentioned commodities was assessed by testing if the grade - tonnage combination recorded in Appendix C, fell above or below the threshold on the appropriate CPT diagram. Deposits which fell above the threshold were deemed to have passed this test and could be considered potentially satisfactory investment targets. Deposits which fell below the threshold were adjudged to have failed to satisfy the corporate financial requirement. The reason for failure, either too low a grade or insufficient tonnage, is indicated in the results of this analysis,

Figure 59 COPPER - CASE STUDY EXAMPLE



LEAD - CASE STUDY EXAMPLE

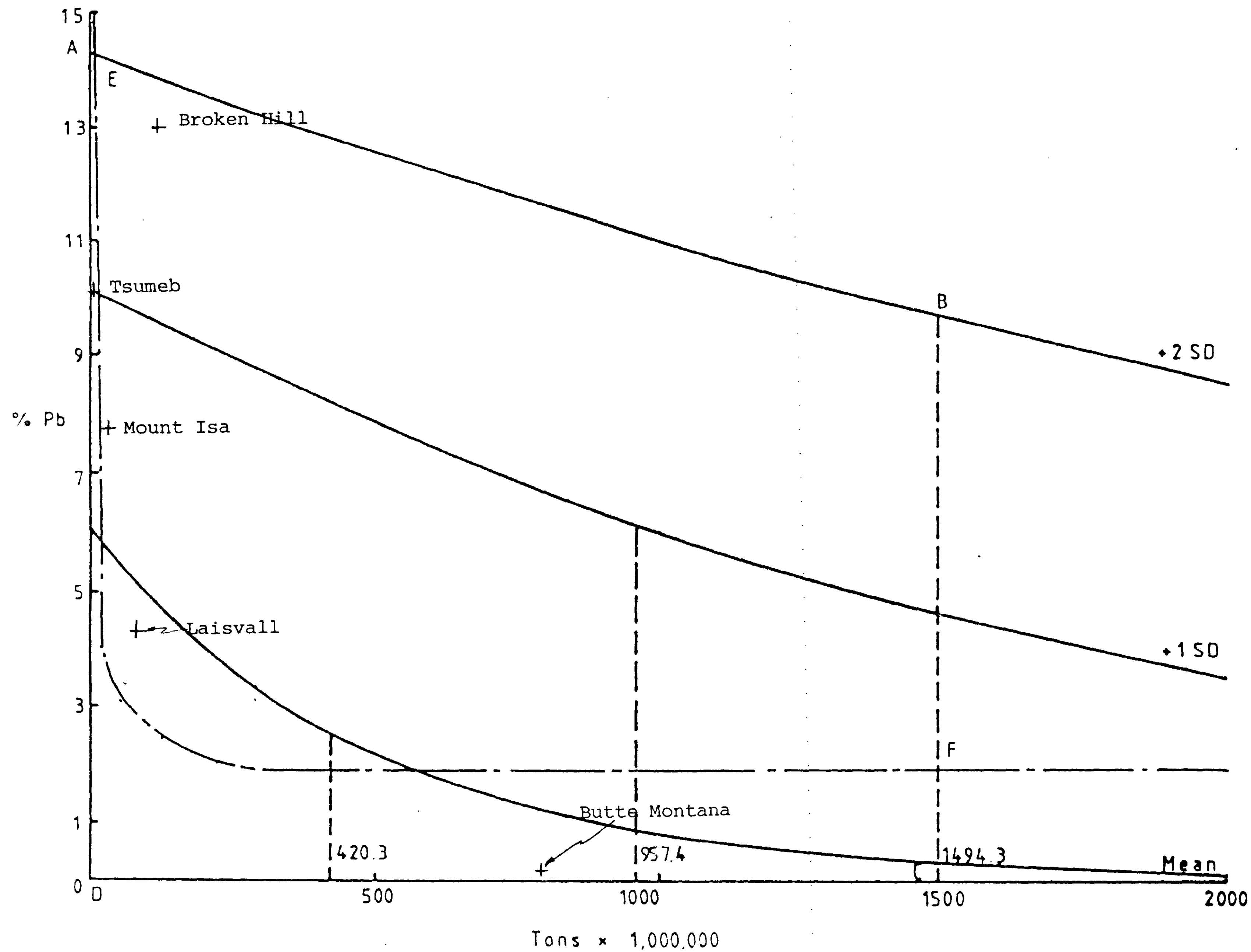


Figure 61 ZINC - CASE STUDY EXAMPLE

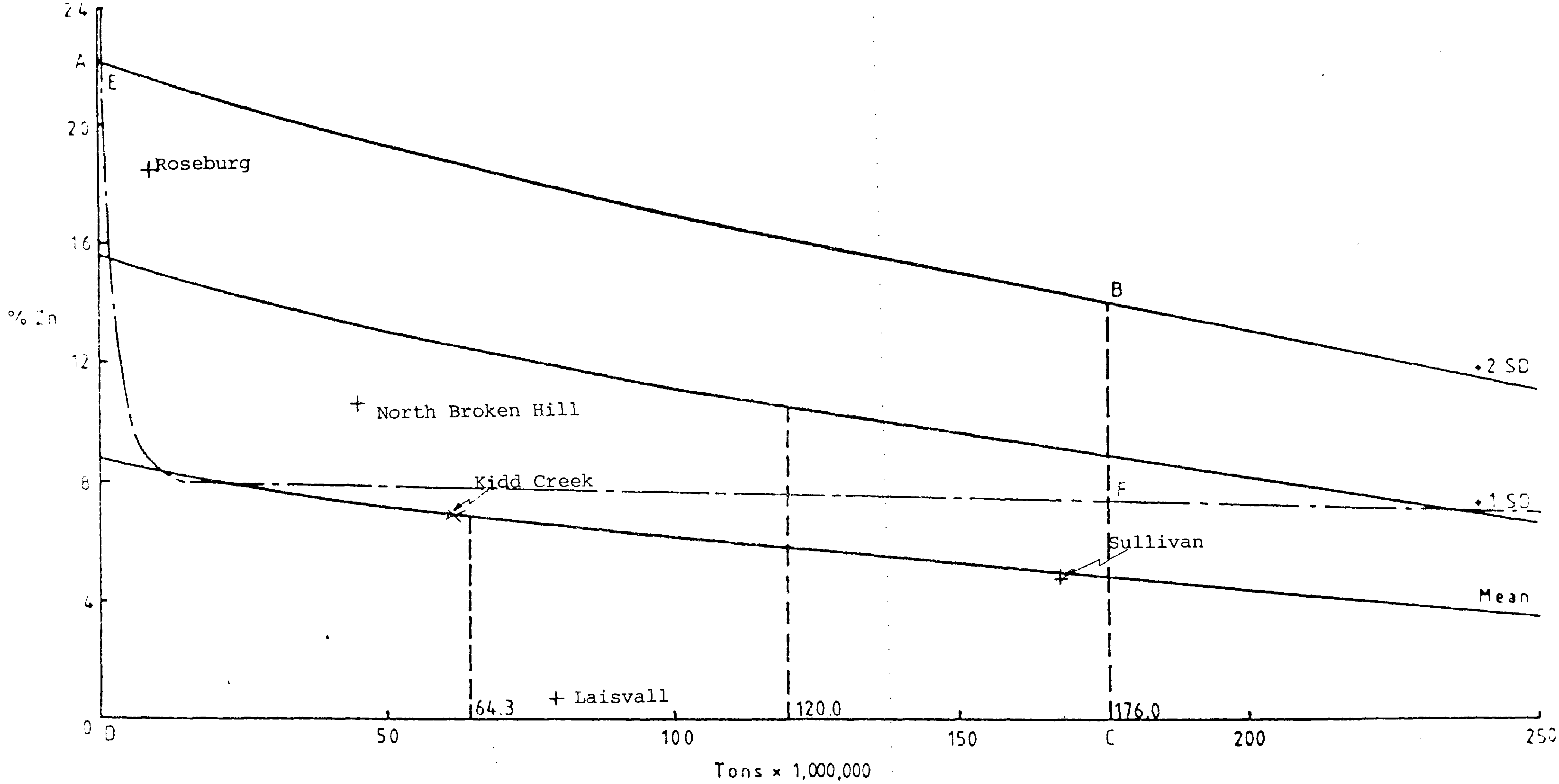


Figure 62 SILVER - CASE STUDY EXAMPLE

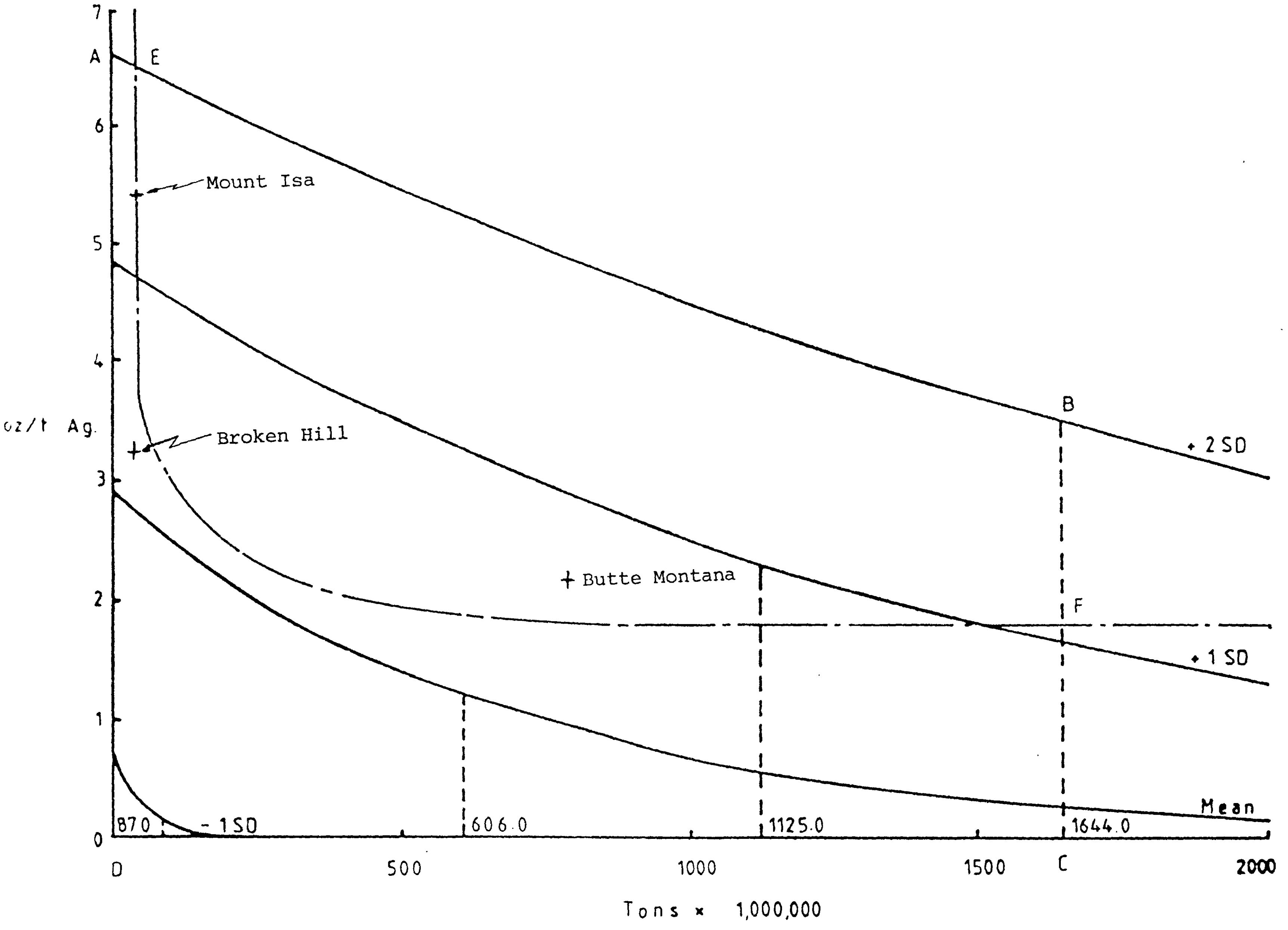
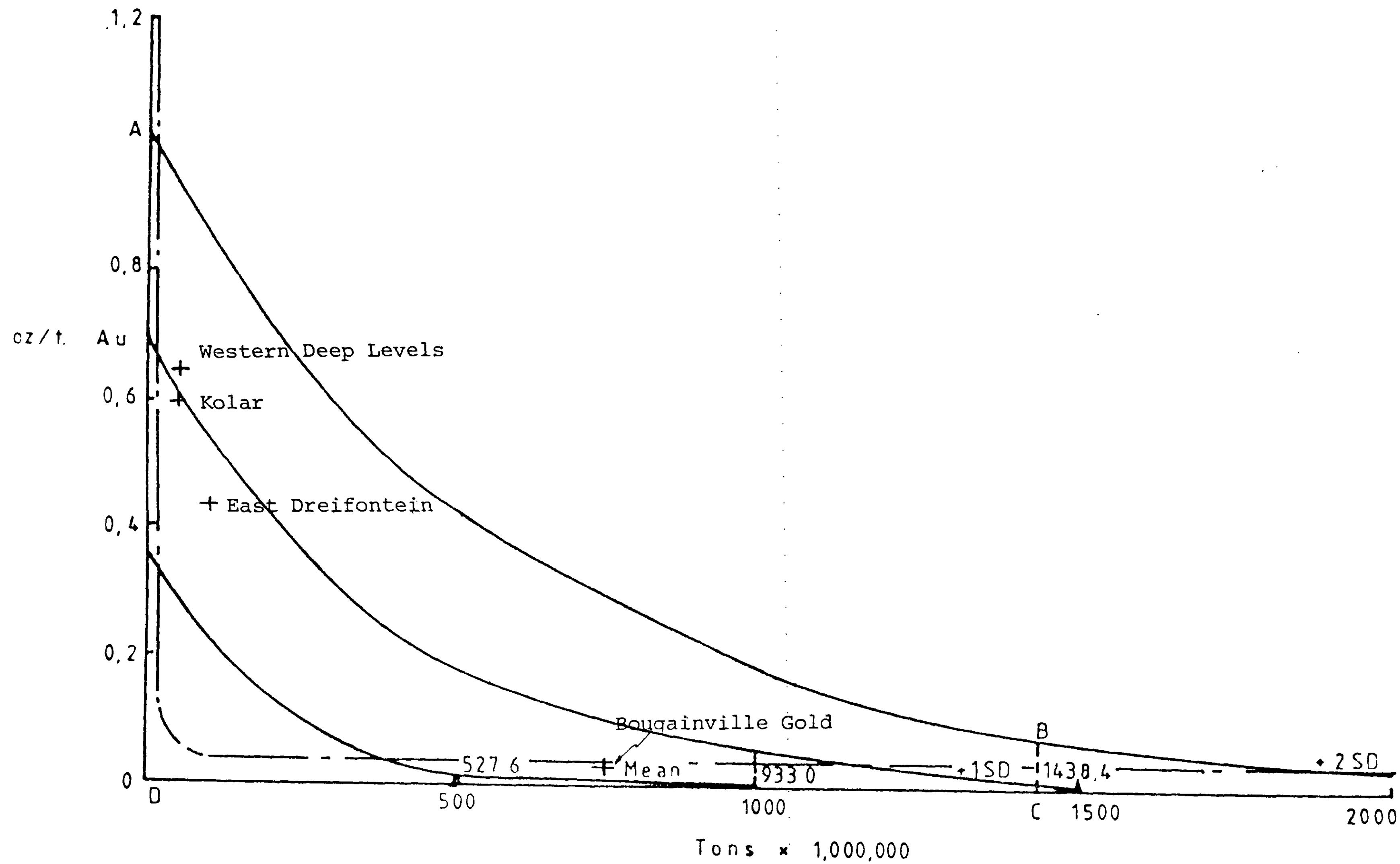


Figure 63 GOLD - CASE STUDY EXAMPLE



which are presented in Appendix M, by an arrow (<==) following the relevant characteristic.

A few illustrative examples, taken from the list in Appendix M, are presented in Figures 59 to 63. Referring to these figures it is apparent that the potential of a particular deposit is a function of the relative position of the deposit and the profitabilty threshold. Possible targets from an inventory may, therefore, be ranked and assessed in this manner, without the need for a detailed feasibility study of each deposit. Thus the use of such strategic guidelines allows for the unified evaluation of many oppurtunities within a short time. This wider examination of possible oppurtunities will of itself increase the corporation's chance of successful investment in exploration.

It should be re-emphasised that all the deposits contained in the list in Appendix M are treated as single commodity deposits. In fact some are multi-commodity deposits and should, more correctly, be treated in the grade equivalent terms described in Chapter 17, section 17.1. Such a procedure will result in certain deposits, which fail when treated on a single element basis, passing the threshold when viewed as multi-commodity deposits because of the effective addition of extra grade in to the original tonnage.

Useful information may also be gleaned from an examination of the manner in which unsuccessful deposits have failed. If a deposit is found to have a reasonable grade, but lacks tonnage, there may be sufficient geological encouragement to justify continued exploration. On the other hand, if the results of continuing exploration show only an increase in tonnage with little or no change in grade, then if the deposit is of too low a grade a decision may be made to terminate exploration. Deposits which fail because they do not meet the grade grade requirement may be re-examined to see if they contain a smaller higher grade section, and its potential may in turn be tested. A prospect lacking both grade and tonnage may be relegated to a less significant position in the scheme of things.

Going beyond the stratgeic planning use of these guidelines, insight may be gained during the actual exploration process by monitoring the progress of the potential of a prospect towards the goal defined by the threshold. The achievement of the threshold may be regarded as the point at which exploration ceases, the prospect becomes a project and development starts.

27.0 SUMMARY AND CONCLUSION

In the preceeding chapters a framework for rational decision making in the orientation of mineral exploration efforts has been developed. This logic was then used to illustrate how an exploration investment strategy may be developed for a hypothetical corporation.

In the Introduction it was argued that complexity compression was the appropriate method for investigating the problems addressed by this thesis. This method requires that a basic objective be stated. For this thesis the basic objective was defined as the deduction of a rational decision - making process which answered in general terms the following questions:-

- Can investment in exploration be justified in competition with other alternatives ?
- If so, how much of the potential investment may reasonably be consummed by exploration ?
- What is the best blend of commodities, deposit types, sizes and grades ?

The constraints on the decision - making system to be deduced were defined by Thuessen (32).

The basic axioms underlying the deduced logic were also stated in the Introduction. According to the rules of complexity aggregation these axioms must have two properties:-

- they explain complexity at the highest level, and
- explanations of lower levels of complexity can be derived from the higher levels by logical inferrence.

It is apparent, a priori, that the axioms, as stated, possess these properties, therefore, it may be concluded that they were reasonable basic assumptions.

In the Introduction five appropriate criteria were defined against which to judge the validity of the deduced logic:-

1. Conformity to intuitative experience.
2. Clarity of propositional content.
3. Internal logical consistency.
4. External logical consistency.
5. Status of a logic scheme.

In the author's opinion the system of reasoning described in this thesis satisfies all five criteria. The justification for this opinion is as follows.

The illustrative example in Part 2 of this thesis demonstrates, by the absence of contradiction in its numerical flow, that the system of reasoning possesses internal logical consistency. This satisfies the third criteria.

The logic takes externally defined financial and geologic concepts, treats them in a numericially consistent manner and provides results expressed in these same originally defined terms. This then provides a coherent interface to external systems of reasoning, and in so doing satisfies Criteria 4, the need for external logical consistency.

Regarding Criteria 5, its status as a logical scheme, this may be judged from the general results produced in Part 2. These results would indicate that the effort that should be aportioned to each of the five commodities considered is as follows:-

Commodity	RCS(i)%	Effort,%
Copper	20.79	21.53
Lead	23.55	18.89
Zinc	15.27	16.13
Gold	24.74	25.19
Silver	15.65	18.64

The effort percent is a re-expression of the budget distribution presented in Chapter 24.

Referring to the above table, the difference between RCS and effort is due to the influence of the qualitative factors. From this table the commodities may be ranked in order of attraction gold, copper, lead, silver and zinc; with gold very much to the fore. In simple terms the strategy would seem to indicate that finding a gold mine was a very good scheme. To that extent the logic is in accord with experience. Looking at the results in more detail, gold is indicated as justifying 56% more effort in exploration than zinc, with copper at 34% and lead and silver both at approximately 16.5%.

An examination of the change in potential induced by consideration of the qualitative influences indicates that gold is a very good exploration target not only from a geological point of view, but also from a qualitative view as the potential increases from one to the other. Whereas for lead the reverse is true. Copper and zinc stay about the same and silver increases markedly. The significant difference between the geologic and qualitative assessments of silver can be translated as the rarity of primary silver deposits on the one hand and the relatively good price performance of silver since the last war on the other. The decrease in the lead potential is due to the fall in the use of this commodity and the general pessimism surrounding its future. Regarding zinc, geologically it is hard to find a profitable deposit, and the qualitative optimism associated with this commodity is insufficient to raise its potential above relatively poor. Copper would seem to have a reasonable geologic potential and only an indifferent qualitative potential prevents it being top of the ranking. This indifferent qualitative potential reflects its relative abundance and consequent poor profit potential.

Summarizing, the logic described in this thesis produces a strategy for investment which indictates that gold should receive the most attention followed by copper at 86% that of gold, lead and silver at 75% and zinc at 64%. Current (1983) performances in the minerals industry do in fact agree with this conclusion, with gold mining being about the only sector, of those considered, that is showing sustained profit and exploration activity. The other commodities copper, lead, silver and zinc are not showing significant exploration activity at the moment. Clearly then the logic produces the same strategy as current industrial practice, therefore, it may be concluded that it status as a valid logic scheme is demonstrated, and so satisfies Criteria 5.

The most difficult criteria to satisfy is the second - clarity of propositional content. It is paradoxical to demand of a philosopher that he prove that his logic is clear. The question arises, clear to whom ? The propositions may be as clear as day to the philosopher but not so to the reader. Lack of understanding on the part of the reader may not be caused by cloudy propositional content on the part of the philosopher, but rather because of lack of perception on the part of the reader. Cosequently, all the author may do is to remind the reader that the basic objective has been defined in the Introduction to this thesis. As have the underlying assumptions and constraints. In subsequent chapters the arguement has been developed in a step-by-step manner until a general theory satisfying the basic objective, based upon the fundamental axioms and within the defined constraints was deduced. The use of this theory was then exemplified by a detailed numerical example. Having completed these requirements the author now claims to have fulfilled his part of the obligation to provide propositional content, and consequently to have satisfied Criteria 2.

Regarding the first Criteria, substantiation of the the author's claim is also difficult. The difficulty arises in producing a universally accepted definition of "intuitative experience" in the context of the strategic planning of mineral exploration. If an explicit definition existed, then there would be no need for this thesis. However, general experiences may be defined. Usually successful mines have certain characteristics:-

- they are large
- they are relatively high grade
- their commodity is of relatively high unit value.

There may well be other contributions to success like good management, but, all things being equal, a large, high grade deposit of a valuable commodity should provide a significant profit and, therefore, attract a large proportion of any potential investment. In fact, in general useage, a good investment is said to be a "gold mine".

Consequently, if the logic directs exploration towards large, high grade deposits of high unit value commodities then it may be concluded that the logic does actually conform to intuitative experience. Examination of the results summarized above confirms that the system of reasoning described in this thesis does satisfy Criteria 1.

Having addressed each Criteria in turn, the author rests. The reader is invited to assess the validity of the author's contention to have satisfied the requirements of philosophical deduction defined by the rules of complexity compression.

It is undoubtedly true that many of the detailed steps, in the reasoning described in this thesis are open to debate. Indeed, the promotion of such a debate was one of the initial objectives of this research. It is not claimed that the above approach has produced a final solution to the problem of strategic planning of exploration. It is to be expected that as more knowledge is gained concerning the decision making process, and the workings of the component systems, the precision of specific parts of the overall approach will be enhanced by the modification of certain assumptions and logical steps. However, it is argued that the fundamental approach could provide a reasonable and repeatable logic for use in the rational orientation of exploration efforts.

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APPENDIX A

Details of the Direct Solution for the Commodity Profitablity Threshold.

MINIMUM RESERVE ANALYSIS

A minimum reserve analysis is one which determines the grade-price-tonnage combination which will generate a predefined DCFROR.

The first step in such an analysis is to determine if any profit at all is generated. Clearly, if no operating profit is produced, then there is no possibility of producing a rate of return.

A1.1 Operating Cost

Bearing in mind the desire to consider concepts in geologic terms, it is helpful to convert operating costs in to equivalent grades. This is illustrated by the following example:

Let, YY = mining cost in $/ton,

ZZ = processing & other costs in $/ton.

PRICEVAL = net smelter return in $/lb. of commodity.

All tons are short tons, hence:

If VAL7 = operating cost grade equivalent,

VAL7 = (YY + ZZ) / (PRICEVAL * 20) ..(1)

The resulting value will be in grade percent.

A1.2 Diluted Grade

In any mining operation a certain amount of waste is retained in the material delivered to the mill for processing, i.e. dilution. This material has a grade which is different, and usually lower, than that of the ore. Therefore, the in-situ grade must be converted to a diluted grade after being subjected to the mining process. This diluted grade may be calculated thus:

Diluted grade = (ABG * (1 - PERDIL) + GD * PERDIL)

- where: ABG = in-situ grade %

PERDIL = percent mining dilution

GD = grade % of the diluting material.

The results of this equation (2) are in % grade.

A1.3 Recovered Grade

Depending upon the value of the head grade and mineralogy the recovered grade of the commodity will be less than the feed grade to the processing plant, as a function of the mineral processing recovery. Hence,

Recovered grade = Diluted grade * S1 ... (3)

- where: S1 = mineral processing recovery.

A1.4 Operating Profit - Net Grade

In financial terms the operating profit is the revenue less the operating cost. Rearranging the above three equations the net grade may be found as follows:

Let, NGRAD = net grade %, then,

NGRAD = (((ABG *(1 - PERDIL) +GD *PERDIL) *S1) -VAL7)

If the net grade is less than or equal to zero then, clearly, no amount of ore reserve will satisfy the need to make a demanded profit, and such a target is not a candidate for exploration. The advantage of use these units is that the financial implications may be translated straight in to geological terms, and the margin for profit shown in terms everyone can understand. Hence, by rearranging the last equation, breakeven grades for a variety of operating costs and conditions can be determined. If this information is then used in conjunction with a grade frequency distribution then a good picture of the potential of the orebody may be obtained.

A1.5 Cummulative Capital

In the event that a positive net grade results, the analysis continues by consideration of capital investment. This capital investment is composed of several elements:

A1.5.1 Mine Development Capital

This is the expenditure incurred for the basic operating infrastucture of the mine. For underground mines this includes such items as shafts, raises, passes, drifts, etc., needed before ore can be produced on a continuous basis. For an open-pit this would include pre-strip, roads, etc.

This initial development cost is a function of the location of the deposit, that is to say depth below surface. The deeper the shaft the higher its cost and so on. As the precision required for the analysis is first order, then a simple linear dependency between depth to the base of the deposit and mine development cost is sufficient. Hence:

Mine development
Capital, = DEPTHVAL * D

- where: DEPTHVAL = unit infrastructure cost $/ft
D = depth to base of deposit, ft.

A1.5.2 Pre-Project Expenditure

This is the amount of money spent before a decision is made to go to the development phase of a project. So it is a lump sum including the cost of such items as exploration drilling, feasibility studies, etc.

This expenditure has taken place by time zero in the project life, and is therefore, increasing in value during the pre-production period, at a rate equal to the DCFROR demanded for the project. So the amount of expenditure to be amortized over the production life is:

```
Cummulative pre-production
cost, SIGPROD                 = CAP *( 1 + DISPCT.I) **W


                 - where: W = pre-production time, yr.
                 DISPCT.I = required DCFROR
                      CAP = pre-project  expenditure
                            at time zero.
```

A1.5.3 Amortizable Base

The mine development capital and the pre-project expenditure represent an investment in discovery that must be repaid by the mine over its life, and one which is independent of production rate.

In order to bring them in to consideration in a consistent manner the discounted value of this basic expenditure must be calculated.

For the purposes of this analysis it is reasonable to assume that this investment will take place in equal, annual investments over the pre-production period. Hence, the average annual investment will be:

((DEPTHVAL * D) + (CAP * ((1 + DISPCT.I) ** W))/W

It is also possible that the company will take advantage of an investment tax credit; hence the amount of actual outlay may be reduced by an amount equal to the ITC. The cumualtive discounted value may then be calculated.

This calculation is simplified by the assumption that investment takes place in equal, annual amounts. This may be illustrated as folloes:

Let the annual investment = a(i)

- where: i = year,

and let the discount factor for year i = d(i).

Then,

Cumulative value = a(1)*d(1) + ... + a(n)*d(n)

but, a(1) = a(2) = ... = a(n)

Therefore,

Cumulative value = a * (d(1) + ... + d(n)).

Let VAL5 = (d(1) + ... d(n)).

Therefore, in this case, including the adjustment for ITC, the amortizable base, SIGDEEP, may be calculated as follows:

SIGDEEP = ((1.0 - B1)*(((DEPTHVAL * D)+(CAP * ((1.0 + DISPCT.I) ** W)))/W)) * VAL5

where:B1 = investment tax credit factor.

Consideration must now be given to variables which are a function of production rate.

A.6 Working Capital

This is assumed to be equal to three months' operating cost. So:

Working Capital, SIGWRK = 0.25*(((1.0+F1)*YY)+ZZ)

- where: F1 = mine development factor
  YY = mining cost / ton
  ZZ = processing & other costs / ton.

A word of explanation on Mine Development Factor may help at this point. In any mining method a certain effort is put in to work which does not directly produce ore, but which is indirectly necessary for its production, such as the driving of access drifts. This effort is here expressed as a percentage of the effort expended in mining ore.

The repayment of working capital at the end of the life of the project is ignored in this analysis.

A.7 Ongoing Development

As was explained above, once the mine is operating, continuous ongoing development is needed to assure the continuity of ore production. This may be calculated as follows:

Ongoing dev., SIGDEV = F1 * YY

(Symbols as above).

A.8 Net Revenue

This may be calculated on a per ton basis as a function of net grade and net smelter return. In this analysis short tons are used:

Net revenue, SIGREV = NGRAD * 20 * PRICEVAL

- where, PRICEVAL = net smelter return, $/lb.

A.9 Depreciation

In order to make this analysis as accurate as possible, after tax cashflow must be considered. Therefore, depreciation and depletion allowances must be applied. Depreciation may be determined as follows:

A.9.1 Initial Depreciable Capital

Let initial depreciable value = NDEPR

NDEPR =(1.0 + QPCT) * (K + E1)

- where: E1 = mine equipment capital factor $/TPY

K = mill equipment capital factor $/TPY

QPCT = other capital factor

This is assuming a linear relationship between the amount of mine or mill equipment capital needed and annual production rate. The other capital needed for the infrastructure, etc., of the mine is expressed as a percentage of the sum of the mine and mill capital.

Assuming that this expenditure is made in equal annual increments over the pre-production period of the project, its cumulative discounted value may be calculated as follows:

Cum. Disc. Value = (NDEPR/W) * VAL5

According to the rules on depreciation this value may be written down over the life of the equipment. These equipment lives are set in arbitrary ranges for various categories of machinery, and the write down method may be varied. Over - sophisication in the depreciation schedule is not required in this analysis, therefore an average depreciable life of eight years will be used for all equipment, and the depreciation will be calculated on a straight line basis, with a salvage value of zero assumed. Hence, the annual equipment write - down may be calculated thus:

Annual write down = (NDEPR/W) * (VAL5/8)

A.9.2 Ongoing & Replacement Depreciation

The chances are, in real operations, that the initial equipment will wear out and need replacement before the end of the project. Hence, the depreciation due to ongoing and replacement capital equipment must be calculated.

For simplicity it may be assumed that the working life of the equipment is equal to its depreciable life. Therefore, the number of sets of replacement equipment required is a function of mine production life and depreciable life. So,

Number of replacement sets needed = (M1 - W)/8

-where, M1 is the project life and W is the pre - production period. The cost of each set of equipment in a constant value analysis will be NDEPR.

Therefore, the annual ongoing and replacement cost, assuming annual increments, will be:

((M1 - W)/8) * (1.0/(M1 - W)) = NDEPR/8

The cumulative discounted value of this cost over the production life of the project will be:

(NDEPR/8) * VAL6

-where, VAL6 is the sum of the discount factors at the demanded rate over the production period.

Writing down the cumulative ongoing and replacement cost over an eight year period produces equal annual depreciation amounts = (NDEPR/64) * VAL6.

Therefore, the total depreciation available in any one year (SIGDEPR) may be defined as follows:

$$SIGDEPR = \text{Initial depr.} + \text{Ongoing depr.}$$

$$= (NDEPR/W) * (VAL5/8) + (NDEPR/64) * VAL6$$

This estimate is, of course, only an approximation, but is sufficiently accurate for this analysis.

A.10 Depletion

The cost depletion method of calculation is used, thus:

Maximum allowable depletion = SIGDEPL1

$$SIGDEPL1 = (SIGREV - SIGDEPR) * 0.5$$

The depletion permitted for commodity " X ", SIDEPL2 may be calculated as follows:

$$SIGDEPL2 = H1 * SIGREV$$

- where; H1 = depletion factor allowed.

Assuming USA rules, then the IRS states that :

(i) Depletion deductions may be taken if depreciation is less than revenue.

(ii) The depletion taken must be the lesser of maximum allowable depletion and the depletion permitted for that commodity.

Expressing these rules mathematically, and using the terms as defined above, the depletion actually taken (SIGDEPL3) may be calculated as follows:

```
IF( SIGDEPR.GE.SIGREV )     SIGDEPL3 = 0
IF( SIGDEPL2.LT.SIGDEPL1 ) SIGDEPL3 = SIGDEPL2
IF( SIGDEPL2.GE.SIGDEPL1 ) SIGDEPL3 = SIGDEPL1
```

In a situation where a depletion deduction is not allowed, setting the depletion fatcor to zero will eliminate any depletion deduction.

A.11 Deductions

Ignoring amortization as being insignificantly small, the total tax deductions applicable (SIGDED) may be defined as equal to the sum of the depreciation and depletion, hence:

```
SIGDED = SIGDEPR + SIGDEPL3
```

A.12 After Tax Profit

In order to calcuate the after tax profit in \$/ton of ore (SIGPROF) it is first necessary to determine the taxable income in \$/ton of ore (PRO1). This may be done as follows:

PRO1 = SIGREV - SIGDEV -SIGDED

If PRO1 is equal to or less than zero, then the effective tax rate (N1) is also equal to zero, in the following equation:

SIGPROF = PRO1 * (1.0 - N1) + SIGDED

Note: tax deductions are not real cash expenditures, but rather accounting conveniences, therefore, they need to be added back into the cashflow.

A.13 Base Tonnage

Now that after tax cashflow has been calculated, the base tonnage (SIGTON1) required to amortize the development and pre - project capital may be calculated as a function of that profit:

SIGTON1 = SIGDEEP/((SIGPROF/(M1-W))*VAL6)

This profit is, naturally, accumulated at the appropriate discount rate over the life of the mine.

A.14 Initial Equipment Capital Investment

In addition to the amortizable base, capital is needed for mining and processing equipment, other facilities and working capital. Investment tax credit may be taken on this expenditure. This initial capital investment is a function of production rate, and, for the sake of this analysis, it is assumed to be a linear function; thus, the initial equipment capital investment (SIGCAP) may be determined as follows:

SIGCAP = ((1.0 + QPCT)*(K + E1))-B1(C1*K + E1) +SIGWRK

- where, C1 is the proportion of mill capital that can be depreciated.

A.15 Minimum Required Mineable Tonnage

The minimum required mineable tonnage (TR) may now be calculated as a function of the base tonnage and the ratio between the cumulative net present value of the capital investment and the cumulative net present value of the profit. Hence,

TR = SIGTON1*(1.0+((SIGCAP/W)*VAL5)/((SIGPROF/(M1-W)))*VAL6))

or conceptually:

TONS = BASE * ((1 + CAPITAL)/ PROFIT)

A.16 Minimum Required In - Situ Tonnage

As a geologist looking for a deposit, knowledge of the minimum mining reserve is of limited value, an exploration target is needed. That is the definition of an actual deposit size. To achieve this target definition, the minimum mineable reserve is divided by the mining recovery (R1), to produce the minimum required in - situ reserve (ISTR), thus,

$$ISTR = TR/R1$$

The resulting tonnage is a function of grade, price, depth and the rate of return demanded.

It is not always obvious that a set of concepts such as these described above do, infact, make a reasonable, coherent theory. In order to allay such scepticism Appendix B will deal with an example which will show that the theory is actually cogent.

APPENDIX B

Proof of Minimum Reserve Analysis

Before going any further it is necessary to prove the validity of the technique described in Appendix A. The best way of doing this is to take a project and analyse it in two ways. First, using minimum reserve analysis, and second using conventional cashflow analysis, the results may then be compared. For the purposes of this study, if the results are within +/-25%, then it may be concluded that the minimum reserve analysis is suitable for use as a data reduction tool in strategic studies, such as those proposed by this thesis. It will also mean that a direct solution to the problem of generating a Commodity Source Profile will have been found.

B.1 Example Project

For the purposes of this exercise, an underground copper mine using block caving as the mining method, and conventional sulphide floatation as the concentrating system will be assumed. It will further be assumed that the operation is in the USA, and that advantage of the depletion allowance will be taken.

The project parameters may be summarized as follows:

Project life, M1	13 years
Pre - production life, W	3 years
Required return, DISPCT.I	15 %
Mill capital factor, K	$10./TPY
Mine equipment capital factor, E1	$14./TPY
Mine capital depth factor, D	$9000./foot
Depth to base orebody, DEPTHVAL	3000. feet
Investment tax credit, B1	10 %
Proportion of mill capital depreciable, C1	75 %
Pre - project expenditure, CAP	$5.0 M
Mining cost, YY	$5.00 /ton
Processing & other cost, ZZ	$2.50 /ton
Mine development factor, F1	15 %
Commodity	copper
In - situ copper grade, ABG	2.00 % Cu.
Mining dilution, PERDIL	10.00 %
Grade of dilution, GD	0.50 % Cu.
Net smelter return, PRICEVAL	$1.00 /lb. Cu.
Mining recovery, R1	100.0 %
Processing recovery, S1	90.0 %
Tax rate, N1	50.0 %
Depletion factor, H1	15.0 %
Other capital factor, QPCT	25.0 %

B.2 Approach

The first step is to calculate the cumulative discount factors for the production and pre-production periods. Using the standard definition of net present value, the discount factor for any one year is given by :

Discount factor = 1.0/(1.0 + i) ** n

- where, i = discount rate

n = year.

In this case, i is 0.15, and n is 1 to 3 for the pre - production period and 4 to 13 for the production period.

Therefore, cumulative discount factors for the pre - production period, VAL5 = 2.28. Cumulative discount factors for the production period, VAL6 = 3.3

Amortizable Base

SIGDEEP = ((1.0-B1)*(((DEPTHVAL*D)+(CAP*((1.0+ DISPCT.I)**W)))/W))*VAL5

Substituting, SIGDEEP = $23,670,000.00

Depreciation

NDEPR = (1.0 + QPCT) * (K + E1)

So, NDEPR = 30.00

And, SIGDEPR = 0.125*NDEPR*((VAL5/W)+(0.125*VAL6))

So, SIGDEPR = $ 4.40 / TPY

Working Capital

SIGWRK = 0.25*(((1.0+F1)*YY)+ZZ)

So, SIGWRK = $ 2.06 TPY

Ongoing Development

SIGDEV = F1 * YY

SO, SIGDEV = $ 0.75 / TPY

Net Grade

NGRAD = (((ABG*(1.0-PERDIL)+GD*PERDIL)*S1)-VAL7

VAL7 = (YY+ZZ)/(PRICEVAL*20)

So, VAL7 = 0.375 %

And, NGRAD = 1.290 %

Net Revenue

SIGREV = NGRAD*20*PRICEVAL

So, SIGREV = $ 25.80 /ton.

Depletion

SIGDEPL1 = (SIGREV-SIGDEPR)*0.5

So, SIGDEPL1 = $ 10.7 /ton

SIGDEPL2 = H1*SIGREV

So, SIGDEPL2 = $ 3.87 / ton

IF(SIGDEPL2.LT.SIGDEPL1) SIGDEPL3 = SIGDEPL2

IF(SIGDEPL2.GE.SIGDEPL2) SIGDEPL3 = SIGDEPL1

So, SIGDEPL3 = $ 3.87 / ton

Deduction

SIGDED = SIGDEPR + SIGDEPL3

So, SIGDED = $ 8.27 / ton

After Tax Profit

PRO1 = SIGREV - SIGDEV - SIGDED

SIGPROF = PRO1*(1.0-N1)+SIGDED

So, PRO1 = $ 16.78 / ton

And, SIGPROF = $ 16.66 / ton

Base Tonnage

SIGTON1 = SIGDEEP/((SIGPROF/(M1-W))*VAL6

So, SIGTON1 = 4,305,000.0 tons

Initial Capital Investment

SIGCAP = ((1.0+QPCT)*(K+E1))-B1*(C1*K+E1)+SIGWRK

So, SIGCAP = $ 29.91 / ton

Minimum Required Mineable Tonnage

TR = SIGTON1*(1.0+((SIGCAP/W)*VAL5)/((SIGPROF/(M1-W)))*VAL6))

From which, TR = 22,105,000.0 tons

Minimum Required In - situ Reserve

ISTR = TR / R1

So, ISTR = 22,105,000.0 tons

That is to say, in order to make a DCFROR of 15 %

at a NSR of $ 1.00 /pound of copper, a deposit of at least 22 million tons is needed if the average grade is 2.0 % Cu and the depth to the base of the deposit is 3000 feet. This can now be cross - checked using conventional cashflow analysis.

B.3 Cashflow Analysis Approach

In order to achieve a positive rate of return the investment capital must be repaid, at the desired profit, by the net cashflow over the life of the project.

Calculate the Value of the Net Cashflow

Mine production life	=	10 years
deposit size	=	22.105m tons
Hence, annual production rate	=	2,211,000 tons.

The net profit calculated above was $ 16.66 / ton of ore, so the annual profit = $ 36,835,000.00. Assuming, for the sake of consistency with the MRA model that this cashflow is occurs in each of the production years. Thus the NPV @ 15 % of the cumulative cashflow may be found as follows:

Cum. NPV15 = 36835000 * VAL6

= $ 121,556,000.00

Calculate Initial Capital Investment

Annual production rate = 2,211,000 tons.

Again to be consistent with the MRA model, assume that the same linear relationships are true for determining the capital investment, independent of the financial modelling used. Thus:

- mine equipment	=	2,211,000 * E1
	= $	30,954,000.00
- mill equipment	=	2,211,000 * K
	= $	22,110,000.00
- working capital	=	2,211,000 * SIGWRK
	= $	4,555,000.00
- mine development cost	=	DEPTHVAL * D
	= $	27,000,000.00
- pre- project expenditure	=	SIGPROD
	= $	7,604,000.00
- sub - total	= $	92,223,000
- other capital @ 25 % of mine & mill capital	= $	13,266,000
- Total initial capital	= $	105,489,000.00

Compare Expenditure and Profit

- initial expenditure	= $	105,489,000.00
- cumulative profit	= $	121,556,000.00
- correlation	=	0.87
- apparent error	=	13 %

Now according to the original definition, if the error between the two methods was less than 25 %, then MRA would be considered as suitable for the purposes of this type of strategic analysis. Clearly then, such a conclusion may be drawn.

However, consideration of the workings of the cashflow analysis is useful, as it reveals a further reduction in the apparent error.

B.4 Apparent Error

Under the conventions of calculating a DCFROR, it is assumed that initial capital is repaid as quickly as possible from the profits of the project. In the comparison made above it was assumed that the initial capital would be repaid in equal annual amounts throughout the production life of the project. In order to obtain a correct assessment of the apparent error it is necessary to compare both systems on the same basis. This may be understood by considering the following table:

Project Year	Production Year	Profit $ M	Outstanding Capital $M	Net Capital,$M
3	0	0	105.489	105.489
4	1	36.835	121.312	84.477
5	2	36.835	97.149	60.314
6	3	36.835	69.361	32.526
7	4	36.835	37.405	0.570
8	5	0.656	0.656	0.000

During the pre - production period the initial capital investment is neither charged interest nor discounted because the opportunity cost and the potential profit are assumed equal. Hence, the net effect is offsetting. However, once production starts it becomes necessary to compare the actual return with the opportunity cost, so interest at a rate equal to the expected return is charged.

In this case payback of the initial capital does not actually occur until early in the fifth production year. The assumption is also made that the profit is received at the end of each production year.

The total profit required to service the initial capital investment was \$ 147.996M. The initial capital investment was \$ 105.489M, hence the interest paid was \$ 42.507M.

The cumulative discount factor for the interest payments for the years 4 to 7 is 1.88. The final \$ 0.656M payment in year 8 is ignored.

Hence, averaging and adding:

Cumulative NPV15 of the interest = (42.507/4)*1.88

= \$ 19.98 M.

This should be deducted from the accumulated profit, hence the adjusted cumulative NPV15 of the profit = 121.556 - 19.98 = \$ 101.576 M.

Comparing this with the initial expenditure of \$105.489M produces a correlation of 96.3 %, or an apparent error of 3.7 %. Such an error is clearly insignificant in terms of the problem being solved, and so MRA may be regarded as accurate. In fact, this actual error will tend to lead to a slight over - estimate of the tonnage needed. If an error is to occur such an over - estimate is preferrable to an under - estimate.

APPENDIX C

Basic Deposit Data Sorted by Commodity and Deposit Type

COPPER

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
GRANDUC	32500000.	1.9300
INGERBELLE	43500000.	.5600
SIMILKAMEEN	4500000.	.8700
CASA GRANDE	150000000.	.9000
LAKEHEAD 1	40500000.	.4000
GREYHOUND	800000.	.7900
HIGHLAND VALLEY	200000000.	.4500
CERRO VERDE	180000000.	1.0900
CASA GRANDE	448000000.	.7000
VANCOUNVER 1	180000000.	.5200
LORNEX 1	293000000.	.4270
BOUGAINVILLE 1	760000000.	.4700
GREENAVLE 1	45000000.	.1000
SAR CHESMEH	300000000.	1.5000
COPPERMINE IS.	22000000.	.5000
MORUCOCHA	183000000.	.8000
SACUTON	17000000.	.8000
SACUTON 2	12000000.	1.5000
MICHIGUILLAY	592000000.	.7500
SAN MANUEL	565000000.	.7200
EAST JERSEY	70000000.	.6000
GASPE	60000000.	1.0000
CUAJONE	500000000.	1.0000
PIMA 1	120000000.	.5100
CHALCOBAMBA	35000000.	1.5000
TOQUEPALA	468000000.	1.0000
GAUGA	23000000.	.5000
ATLAS	189700000.	.6600
PHILEX 1	60000000.	.7000
STO. NINO	50000000.	.5000
MARTINDUQUE 2	50000000.	.6500
BLACK MOUNTAIN	20000000.	.6300
BAGUIO	24000000.	.5500
MAMUT	70000000.	.6000
VALLEY COPPER	500000000.	.5000
SCHAFT CREEK	200000000.	.4000
COPPER CREEK	200000000.	.9000
LA CARIDAD	100000000.	.8000
FERNANDO VAL.	80000000.	1.0000
COLON	50000000.	1.0000
CERRO COLO 1	40000000.	.8000
EL SALVADOR I	1000000000.	.5000
CHUQUI	4000000000.	1.0000
BINGHAM 1	2000000000.	.8000
BUTTE 1	800000000.	2.4300
PALABORA 1	400000000.	.6900

NUMBER IN THIS GROUP = 46

COPPER

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
MICILLA	2000000.	2.5000
MUSOSHI	30000000.	2.6000
CADIA 2	1000000.	.8900
HORNE	3300000.	2.4400

NUMBER IN THIS GROUP = 4

COPPER
==========

STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
BWANA MKUMBA	5760000.	3.4800
AVOCA	6000000.	1.0000
MATSITAMA	6420000.	2.2400
SKOURIOTISSA	20000000.	.5800
KALENGWA 2	250000.	3.4500
SKOURIES 1	17700000.	1.0500
MATTAGAMI 2	18000000.	.7000
ANTAMINA 1	11000000.	1.9000
JABAL SAYID	8000000.	2.5000
MUFULIRA	167067000.	3.3700
CHAMBISHI	38785000.	3.0500
BALUBA	112000000.	2.4100
LUANSHYA	85516000.	2.8600
RHOKANA	125327000.	2.7700
NCHANGA	259405000.	4.0100
BANCROFT	96882000.	3.5100
MARTINDUQUE 1	4800000.	2.0000
ROAN	93500000.	3.0000
ROAN ANTELOPE	96300000.	2.9500
NACIEMENTO	11000000.	.6500

NUMBER IN THIS GROUP =   20

COPPER
==========

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
ORANGE 1	2540000.	1.0550
ABERLOW 1	3000000.	1.2000
SABENA	4000000.	.7000
VAL D'OR	500000.	3.2300
TIPPERARY 1	6000000.	1.2000
SNOW LAKE	1000000.	3.0000
GOUDREAU 1	500000.	1.5600
BLACK COPPER	1170000.	.6700
FLEXAR 1	270000.	4.2300
INGUARAN 1	4400000.	2.0000
AMOS 1	2500000.	1.1000
BATIALO 2	500000.	4.0000
GECO 1	27000000.	2.1000
ROSITA	3689000.	1.2500
HUANZALA 3	2200000.	1.0000
MATCHLESS	2400000.	1.7000
BOTREST	6420000.	2.2400
TIMNA	11000000.	1.6000
AL AMAR 1	5500000.	.7000
SCOTIA 2	1250000.	.2500
MT. ISA 6	1500000.	3.8000
MT. ISA 7	45000000.	3.2000
MT. LYELL	41900000.	1.4000
WARREGO 1	3500000.	2.6000
LEPANTO 3	500000.	4.0000
ABERLOW 1	3900000.	1.2000
COPPERMINE RIVER	3000000.	3.4800
SAN ANTONIO 1	5000000.	1.4000

NUMBER IN THIS GROUP =   28

COPPER

OXIDE

NAME	TONS.S	GRADE % OR OZ/TON
EXOTICA	153000000.	1.6100

NUMBER IN THIS GROUP = 1

COPPER

VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
ERTSBERG 1	33000000.	2.5000
PIKWE 1	27670000.	1.1600
SELIBE 1	13500000.	1.5700
KIDD CREEK 2	62500000.	1.3300
TINTAYA	7000000.	3.0000
MT. MORGAN 1	9530000.	1.0800
LEPANTO 1	8900000.	2.9700
R.T. PATINIO 1	40000000.	.8000
MADANKUDAN 1	3000000.	2.7500
KOSAKA 1	10000000.	2.0000

NUMBER IN THIS GROUP = 10

COPPER

COMPLEX SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
MT. CURSON 1	3200000.	1.0400
BATHURST 3	60800000.	.2800
BATHURST 5	13000000.	1.1400
BATHURST 6	18000000.	.3700
ANDERSON LAKE 1	17600000.	3.0000
MADRIGAL 1	1000000.	3.0000
TSUMEB 2	7000000.	3.6600
ROSEBURG 3	8650000.	.8900
HORNE 1	58000000.	2.4000

NUMBER IN THIS GROUP = 9

GOLD
==========

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
BOUGAINVILLE 2	760000000.	.0200
PHILEX 2	60000000.	.0280
CERRO COLO 2	18000000.	.0800
EL SALVADOR II	1000000000.	.0050
BUTTE 6	800000000.	.0080

NUMBER IN THIS GROUP = 5

COPPER
==========

HYDROTHERMAL

NAME	TONS.S	GRADE % OR OZ/TON
JUNO 1	200000.	.5000
PEKO 2	900000.	3.6000
IVANHOE 1	160000.	4.2000
DARIBO	400000.	2.5000

NUMBER IN THIS GROUP = 4

GOLD

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
H.B.FONTEIN	28100000.	.4000
H.B.FONTEIN 2	20600000.	.0400
WIT NIGEL	4210000.	.2700
BRAKEN	28000000.	.4700
E.G. MAIN	2000000.	.2500
E.G. KIMBERLEY	5000000.	.2300
GROOTVLEI MAIN	14000000.	.2100
GROOTVLEI KIM.	18000000.	.2200
KINROSS	23000000.	.3600
LESLIE	37000000.	.3300
MARIVALE MAIN	16000000.	.2600
MARIVALE KIM.	17000000.	.2600
ST. HELENA	95000000.	.5200
WINKELHAAK	50000000.	.3000
CORTEZ	3400000.	.2900
EAGLE	1600000.	.4100
DONALDA	3150000.	.3500
ELSBURG	54000000.	.3500
VIRGINIA SA 1	37000000.	.2980
MERRIESPRUIT 1	16000000.	.2800
E. DAGGERFONTEIN	24800000.	.1700
VAAL REEFS	66100000.	.4800
DOME	2030000.	.2790
CAMPBELL RED LAK	1300000.	.6900
LUZ	3280000.	.0950
DOORNFONTEIN	29060000.	.4300
E. DRIEFONTEIN	100000000.	.4400
KLOOF	11590000.	.5500
LIBANON	25900000.	.4000
LUIPAARDSVLEI	6360000.	.2700
SPAARWATER	710000.	.3600
SUB NIGEL	2660000.	.4300
VENTERSPOST	21880000.	.4400
VLAKFONTEIN	9390000.	.4600
W. DRIEFONTEIN	64880000.	.8110
EAST DAGGA	24270000.	.3000
F.S. GEDULD	49540000.	1.2700
P. BRAND	75550000.	.6600
P. STEYN	68900000.	.3800
S.A. LANDS	13850000.	.3900
WELKOM	53650000.	.4000
W. DEEPS	49150000.	.6500
W. HOLDINGS	70180000.	.7000
W. REEFS	48250000.	.4300
BLYVOOR	58740000.	.7000
DURBAN DEEP	38600000.	.2000
E. RAND PROP.	52500000.	.2700
HARMONY	64890000.	.3800
WESTERN AREAS	53980000.	.3500
GROOTVLEI	32000000.	.2200
BUFFELSFONTEIN	68380000.	.4700
S. ROODEPOORT	8020000.	.3100
STILFONTEIN	22520000.	.4600
W. RAND CONS.	34920000.	.2200
HARTEBEESTFONTEI	52760000.	.4100
LORAINE	80440000.	.4100
RAND LEASES	60000000.	.4100
ZANDPAN	28080000.	.4000
ASHANTI	37000000.	1.0400
KALGOORLIE	6100000.	.1900
GREAT BOULDER	1530000.	.2400
N. KALGOORLIE	2100000.	.2500
KOLAR	45300000.	.5900

NUMBER IN THIS GROUP =   63

GOLD

STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
SKOURIES 2	17700000.	.0340

NUMBER IN THIS GROUP = 1

GOLD

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
GOUDREAU 2	500000.	.1350
FLEXAR 3	270000.	.0300
BATIALO 1	500000.	3.5000
NORSEMAN	530000.	.5000
WARREGO 2	3500000.	.0600
LEPANTO 4	500000.	3.5000

NUMBER IN THIS GROUP = 6

GOLD

VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
ERTSBERG 4	33000000.	.0200
MT. MORGAN 2	9530000.	.0900
LEPANTO 2	8900000.	.1730
R.T. PATINIO 2	40000000.	.0700

NUMBER IN THIS GROUP = 4

GOLD

COMPLEX SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
LEADVILLE 4	2401000.	.0840
ANDERSON LAKE 4	17600000.	.0380
ROSEBURG 5	8650000.	.1100
HORNE 2	58000000.	.1800

NUMBER IN THIS GROUP = 4

SILVER
==========

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
CERRO COLO 3	18000000.	1.3500
EL SALVADOR III	1000000000.	.0500
BUTTE 5	800000000.	2.1500

NUMBER IN THIS GROUP = 3

GOLD
==========

HYDROTHERMAL

NAME	TONS.S	GRADE % OR OZ/TON
EL SALVADOR 1	118000.	.1500
BULLFINCH	16000.	4.0000
FALCON	760000.	.3200
FERGUSSON 1	60000.	.0440
EL DORADO	1000000.	.4800
EL SAL 1	118000.	.1500
JUNO 3	200000.	3.0000
PERU 1	900000.	.1000
IVANHOE 2	160000.	.0700
EMPEROR	970000.	.4500
HOLLINGER 1	60000000.	.3200
HOMESTAKE 1	135000000.	.3200

NUMBER IN THIS GROUP = 12

SILVER
==========

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
ATLIN 1	150000.	20.0000
ABERLOW 2	3000000.	1.6600
TIPPERARY 2	6000000.	1.6600
FLEXAR 4	270000.	.1200
INGUARAN 2	4400000.	.3000
FARRELL 2	60000.	14.1000
MT. ISA 2	600000.	2.0000
MT. ISA 5	34000000.	5.4000
ABERLOW 2	3900000.	1.8700
SAN EULAILA 3	35000000.	7.4100

NUMBER IN THIS GROUP = 10

SILVER
==========

COMPLEX SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
MT. CORSON 2	3200000.	.3500
LEADVILLE 3	2401000.	2.6400
BATHURST 4	60800000.	2.4000
BATHURST 9	18000000.	1.8400
ANDERSON LAKE 5	17600000.	.6100
MADRIGAL 4	1000000.	6.0000
TSUMEB 1	7000000.	2.1300
ROSEBURG 4	8650000.	5.1000
LAISVALL 3	80000000.	.2900

NUMBER IN THIS GROUP = 9

SILVER
==========

VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
ERTSBERG 3	33000000.	.3000
KIDD CREEK 3	62500000.	4.8500
ANVIL 3	63000000.	1.0000
N. BROKEN HILL 2	45000000.	7.3900
R.T. PATINIO 3	40000000.	1.7000
SULLIVAN 3	170000000.	1.7700
BROKEN HILL 3	120000000.	1.9300

NUMBER IN THIS GROUP = 7

SILVER
==========

STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
MOGUL 3	10200000.	.9000
SILVERMINES 3	14000000.	.8700

NUMBER IN THIS GROUP = 2

SILVER
==========

HYDROTHERMAL

NAME	TONS.S	GRADE % OR OZ/TON
EL SALVADOR 2	118000.	10.0000
FERGUSSON 2	60000.	6.9000
FRANCES LAKE 1	400000.	4.2000
EL SAL 2	118600.	10.0000
HOLLINGER 2	60000000.	.0700
BUNKER HILL 1	40000000.	3.2200

NUMBER IN THIS GROUP = 6

LEAD
==========
OXIDE

NAME	TONS.S	GRADE % OR OZ/TON
ANGOURAN 1	15000000.	7.0000

NUMBER IN THIS GROUP = 1

LEAD
==========
STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
ZBA 1	44100000.	12.3000
MOGUL 2	10200000.	2.8000
SILVERMINES 1	14000000.	2.8000

NUMBER IN THIS GROUP = 3

LEAD
==========
CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
ATLIN 2	150000.	5.0000
HUANZALA 2	2200000.	7.0000
FARRELL 3	60000.	12.8000
MT. ISA 1	600000.	5.5000
MT. ISA 3	34000000.	7.4000
SAN ANTONIO 2	5000000.	.9000
SAN EULAILA 1	35000000.	12.0000

NUMBER IN THIS GROUP = 7

LEAD
==========
PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
PINE POINT 1	40500000.	5.0000
BUTTE 4	800000000.	.1000

NUMBER IN THIS GROUP = 2

LEAD
==========
HYDROTHERMAL

NAME	TONS.S	GRADE % OR OZ/TON
FERGUSSON 3	60000.	6.0000
FRANCES LAKE 2	400000.	8.0000
ICHMOOL	1300000.	4.0000
MOATE 2	110000.	1.0000
BUNKER HILL 2	40000000.	4.0000

NUMBER IN THIS GROUP = 5

LEAD
==========
COMPLEX SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
LEADVILLE 1	2401000.	5.1300
BATHURST 2	60800000.	3.5000
BATHURST 7	18000000.	2.3560
ANDERSON LAKE 3	17600000.	.2000
MADRIGAL 2	1000000.	6.0800
TSUMEB 3	7000000.	10.5000
BELANATANA 2	730000.	2.9000
BELTANA 4	97000.	12.0000
ROSEBURG 2	8650000.	5.6000
LAISVALL 1	80000000.	4.3000

NUMBER IN THIS GROUP = 10

LEAD
==========
TRI - STATE

NAME	TONS.S	GRADE % OR OZ/TON
PITCHER 2	200000000.	.8000

NUMBER IN THIS GROUP = 1

LEAD
==========
VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
ANVIL 1	63000000.	4.0000
N. BROKEN HILL 1	4500000.	12.9700
MADANKUDAN 2	3000000.	1.2000
SULLIVAN 1	170000000.	4.0000
BROKEN HILL 1	120000000.	13.0000
KOSAKA 2	10000000.	1.7000

NUMBER IN THIS GROUP = 6

ZINC

OXIDE

NAME	TONS.S	GRADE % OR OZ/TON
ANGOURAN 2	15000000.	28.0000

NUMBER IN THIS GROUP = 1

ZINC

STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
MATTAGAMI 1	18000000.	10.0000
ANTAMINA 2	11000000.	1.5000
ZBA 2	44100000.	26.3000
MOGUL 1	10200000.	8.2000
SILVERMINES 2	14000000.	7.4000

NUMBER IN THIS GROUP = 5

ZINC

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
TENNESSE	50000000.	5.0000
FLEXAR 2	270000.	.4000
GECO 2	27000000.	5.1000
HUANZALA 1	2200000.	13.0000
AL AMAR 2	5500000.	5.0000
FARRELL 1	60000.	7.3000
MT. ISA 4	34000000.	5.6000
SAN ANTONIO 3	5000000.	1.6000
SAN EULAILA 2	35000000.	11.0000

NUMBER IN THIS GROUP = 9

ZINC

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
BUTTE 2	600000000.	.7400

NUMBER IN THIS GROUP = 1

ZINC

HYDROTHERMAL

NAME	TONS.S	GRADE % OR OZ/TON
FERGUSSON 4	60000.	6.7000
FRANCES LAKE 3	400000.	9.0000
MOATE 1	110000.	6.4000
BUNKER HILL 3	40000000.	5.0000

NUMBER IN THIS GROUP = 4

ZINC

COMPLEX SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
LEADVILLE 2	2401000.	9.9500
BATHURST 1	60800000.	8.8800
BATHURST 6	18000000.	5.6300
ANDERSON LAKE 2	17600000.	3.8000
MADRIGAL 3	1000000.	4.0000
TSUMEB 4	7000000.	3.1000
BELTANA 1	730000.	37.0000
BELTANA 3	730000.	24.9000
ROSEBURG 1	8650000.	18.6000
LAISVALL 2	80000000.	.6000

NUMBER IN THIS GROUP = 10

ZINC

TRI - STATE

NAME	TONS.S	GRADE % OR OZ/TON
PITCHER 1	200000000.	3.2000

NUMBER IN THIS GROUP = 1

ZINC

VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
PINE POINT 2	40500000.	5.0000
KIDD CREEK 1	62500000.	7.0800
ANVIL 2	63000000.	5.0000
N. BROKEN HILL 3	45000000.	10.7000
MADANKUDAN 3	3000000.	3.5000
SULLIVAN 2	170000000.	5.0000
BROKEN HILL 2	120000000.	11.0000
KOSAKA 3	10000000.	5.0000

NUMBER IN THIS GROUP = 8

IRON

STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
HELEN	280000000.	33.0000

NUMBER IN THIS GROUP = 1

IRON

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
ORANGE 2	5100000.	44.6000

NUMBER IN THIS GROUP = 1

IRON

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
L. WASWANIPI	40000000.	30.0000
CASSALA	500000000.	44.0000
CADIA	6040000.	43.3000
MT. WHALEBACK	1000000000.	64.0000
HAMPSHIRE	5000000.	64.0000
CHESBAR	40000000.	30.0000
NEW MYLAMAQUE 1	50000000.	27.0000
SNETTISHAM	1000000000.	20.0000
EL CERO	350000000.	64.0000
MACUIRA RIDGE	50000000.	57.8000
AGALTECA	10000000.	60.0000
WADI WASSAT	200000000.	90.0000
WADI SAWAWIN	300000000.	48.0000
BAILADILLA	3000000000.	65.0000
DALIARI	50000000.	61.5000
MONCORVO	800000000.	40.0000
CURI	200000000.	48.0000
CAUE	450000000.	67.0000
CAUE 2	650000000.	50.0000
CONCEIACAO 2	800000000.	50.0000
PALABORA 2	400000000.	27.0000

NUMBER IN THIS GROUP = 21

URANIUM

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
H.B.FONTEIN	28100000.	.0340
REXSPAR	1700000.	.0930
KITTS	250000.	.5000
FERNANDEZ	1350000.	.1200
VIRGINIA SA 2	37000000.	.0300
MERRIESPRUIT 2	16000000.	.0090
POCOS DE CALDA	1000000.	.7500
RANSTED	3500000.	.0400

NUMBER IN THIS GROUP = 8

NICKEL

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
GT. BOULDER	500000.	4.0000
GEENVALE 1	45000000.	1.5500

NUMBER IN THIS GROUP = 2

NICKEL

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
LAKEHEAD 2	40500000.	.2000
GREENVALE 2	45000000.	1.5000

NUMBER IN THIS GROUP = 2

MANGANESE

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
TAMBAU	10000000.	52.4000

NUMBER IN THIS GROUP = 1

MANGANESE

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
BUTTE 3	800000000.	.5700

NUMBER IN THIS GROUP = 1

IRON

VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
ERTSBERG 2	33000000.	40.6000

NUMBER IN THIS GROUP = 1

NICKEL

VOLCANOGENIC MASSIVE SULPHIDE

NAME	TONS.S	GRADE % OR OZ/TON
PIKWE 2	27670000.	1.4900
SELIBE 2	13500000.	.6600
KAMBALDA	14290000.	3.4000
KAMBALDA	14300000.	3.4000
KAMBALDA 1	14300000.	3.6500
KAMBALDA 2	1000000.	2.0000

NUMBER IN THIS GROUP = 6

NICKEL

OXIDE

NAME	TONS.S	GRADE % OR OZ/TON
GT. BOULDER OX.	250000.	1.1300
SCOTIA 3	250000.	1.1300

NUMBER IN THIS GROUP = 2

NICKEL

LATERITE

NAME	TONS.S	GRADE % OR OZ/TON
GLOBAL	50000000.	1.7000
STRATHCONA	91700000.	1.9700
BUHAO	64000000.	1.5500
GREENVILLE 1	45000000.	1.5000
DOMINICA	62000000.	1.5500
QUEENSLAND 1	45000000.	1.5000
ARAGUA	44000000.	1.6000
CAPE TIBURON	80000000.	2.0000
MINDANAO	200000000.	1.3000
PALAWAN	200000000.	1.3000

NUMBER IN THIS GROUP = 10

NICKEL

STRATIFORM

NAME	TONS.S	GRADE % OR OZ/TON
NEPEAN	500000.	4.0000
WIDGIEMOOLTHA	1000000.	1.5000

NUMBER IN THIS GROUP = [illegible]

NICKEL

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
GT. BOULDER 2	1250000.	3.0700
GT. BOULDER SULF	500000.	.6400
NEPEAN 2	500000.	4.0000
REDROSS	1000000.	4.4000
C2	500000.	4.0000
SCOTIA 1	1250000.	3.0700
SCOTIA 4	500000.	.6400

NUMBER IN THIS GROUP = 7

MOLYBDENUM

PORPHYRY

NAME	TONS.S	GRADE % OR OZ/TON
VANCOUVER 2	180000000.	.1350
LORNEX 2	293000000.	.0140
HENDERSON	303000000.	.2000
ENDAKO	239000000.	.0899
PIMA 2	120000000.	.0100
EL SALVADOR IV	1000000000.	.0200
BINGHAM 2	2000000000.	.0300
CLIMAX	550000000.	.1200

NUMBER IN THIS GROUP = 8

MERCURY

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
TYAUGHTON CR.	1400000.	.0550

NUMBER IN THIS GROUP = 1

TIN

HYDROTHERMAL

NAME	TONS.S	GRADE % OR OZ/TON
WHEAL JANE	5000000.	1.2500
WHITE CRYSTAL	660000.	.4200
WILD CHERRY	930000.	.3800
PAHANG	340000.	2.3700

NUMBER IN THIS GROUP = 4

TIN

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
THURSDAY IS.	10500000.	.1000

NUMBER IN THIS GROUP = 1

CADMIUM

OXIDE

NAME	TONS.S	GRADE % OR OZ/TON
ANGOURAN 3	15000000.	2.5000

NUMBER IN THIS GROUP = 1

TITANIUM

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
NEW MYLAMAQUE 2	50000000.	6.0000

NUMBER IN THIS GROUP = 1

COLUMBIUM

SEDIMENTARY

NAME	TONS.S	GRADE % OR OZ/TON
OKA	2500000.	.4000

NUMBER IN THIS GROUP = 1

COBALT

LATERITE

NAME	TONS.S	GRADE % OR OZ/TON
GREENVILLE 2	45000000.	.1000
QUEENSLAND 2	45000000.	.1900
GEENVALE 2	45000000.	.1100

NUMBER IN THIS GROUP = 3

MOLYBDENUM

CONTACT METAMORPHIC

NAME	TONS.S	GRADE % OR OZ/TON
PINE CREEK 2	10000000.	.2400

NUMBER IN THIS GROUP = 1

```
TUNGSTEN
==========
          CONTACT METAMORPHIC
          -------------------------------
                    NAME                                    TONS.S      GRADE % OR OZ/TON
                    ----                                    ------      -----------------
                    TAMWORTH                               500000.           1.4000
                    PINE CREEK 1                         10000000.            .4000

       NUMBER IN THIS GROUP =      2

       NUMBER OF LINES IN BASE FILE =  421

**** ANALYSIS COMPLETE ****
```

```
BISMUTH
==========
          HYDROTHERMAL
          -------------------------------
                    NAME                                    TONS.S      GRADE % OR OZ/TON
                    ----                                    ------      -----------------
                    JUNO 2                                 200000.           1.0000

       NUMBER IN THIS GROUP =      1
```

APPENDIX D

Statistical Analysis of the Basic Deposit Data

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE HYDROTHERMAL

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
200000.	.5000	-.6324	-1.3523	5.3010	-.3010	-.8339	-1.4623
900000.	3.6000	1.4267	.5532	5.9542	.5563	1.2981	.5593
160000.	4.2000	-.7501	.9220	5.2041	.6232	-.9206	.7172
400000.	2.5000	-.0441	-.1229	5.6021	.3979	.2564	.1859

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE COMPLEX SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
3200000.	1.0400	-.7774	-.5623	6.5051	.0170	-.9582	-.2144
60800000.	.2800	1.7661	-1.2613	7.7839	-.5528	1.2909	-1.6184
13000000.	1.1400	-.3447	-.4834	7.1139	.0569	.1126	-.1162
18000000.	.3700	-.1239	-1.0904	7.2553	-.4318	.3612	-1.3202
17600000.	3.0000	-.1416	.9826	7.2455	.4771	.3440	.9191
1000000.	3.0000	-.8746	.9826	6.0000	.4771	-1.6466	.9191
7000000.	3.6600	-.6096	1.5029	6.8451	.5635	-.3603	1.1318
8650000.	.8900	-.5368	-.6865	6.9370	-.0506	-.1986	-.3811
58000000.	2.4000	1.6425	.5097	7.7634	.3802	1.2549	.6803

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
33000000.	2.5000	.6052	.6975	7.5185	.3979	.8511	.7631
27670000.	1.1600	.3244	-.9030	7.4420	.0645	.6622	-.8572
13500000.	1.5700	-.4219	-.4133	7.1303	.1959	-.1077	-.2186
62500000.	1.3300	2.1589	-.6999	7.7959	.1239	1.5363	-.5687
7000000.	3.0000	-.7642	1.2947	6.8451	.4771	-.8123	1.1478
9530000.	1.0800	-.6310	-.9985	6.9791	.0334	-.4813	-1.0080
8900000.	2.9700	-.6642	1.2589	6.9494	.4728	-.5547	1.2266
40000000.	.8000	.9739	-1.3330	7.6021	-.0969	1.0575	-1.6413
3000000.	2.7500	-.9749	.9951	6.4771	.4393	-1.7213	.9642
10000000.	2.0000	-.6062	.1003	7.0000	.3010	-.4297	.2922

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE STRATIFORM

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
5760000.	3.4800	-.7716	1.0291	6.7604	.5416	-.8379	.8540
6000000.	1.0000	-.7681	-1.2784	6.7782	0.0000	-.8136	-1.1599
6420000.	2.2400	-.7621	-.1247	6.8075	.3502	-.7734	.1426
20000000.	.5800	-.5659	-1.6692	7.3010	-.2366	-.0985	-2.0396
250000.	3.4500	-.8512	1.0012	5.3979	.5378	-2.7013	.8401
17700000.	1.0500	-.5992	-1.2319	7.2480	.0212	-.1711	-1.0811
18000000.	.7000	-.5948	-1.5576	7.2553	-.1549	-.1611	-1.7359
11000000.	1.9000	-.6959	-.4410	7.0414	.2788	-.4536	-.1233
8000000.	2.5000	-.7392	.1172	6.9031	.3979	-.6427	.3199
167067000.	3.3700	1.5581	.9267	8.2229	.5276	1.1623	.8022
38785000.	3.0500	-.2946	.6290	7.5887	.4843	.2949	.6411
112000000.	2.4100	.7628	.0345	8.0492	.3820	.9248	.2607
85516000.	2.8600	.3803	.4522	7.9320	.4564	.7645	.5372
125327000.	2.7700	.9552	.3685	8.0980	.4425	.9915	.4855
259405000.	4.0100	2.8917	1.5222	8.4140	.6031	1.4236	1.0830
96882000.	3.5100	.5444	1.0570	7.9862	.5453	.6386	.8679
4800000.	2.0000	-.7855	-.2480	6.6812	.3010	-.9451	-.0405
93500000.	3.0000	.4956	.5825	7.9708	.4771	.8175	.6144
96300000.	2.9500	.5360	.5359	7.9836	.4698	.8351	.5872
11000000.	.6500	-.6959	-1.6041	7.0414	-.1871	-.4536	-1.8556

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
2540000.	1.0550	-.3683	-.7994	6.4048	.0233	-.0913	-.6610
3000000.	1.2000	-.3264	-.6761	6.4771	.0792	.1376	-.4698
4000000.	.7000	-.2418	-1.1013	6.6021	-.1549	.2605	-1.2700
500000.	3.2300	-.5450	1.0500	5.6990	.5092	-1.3506	1.0003
6000000.	1.2000	-.0685	-.6761	6.7782	.0792	.5747	-.4698
1000000.	3.0000	-.5017	.8544	6.0000	.4771	-.8135	.6916
500000.	1.5600	-.5450	-.3700	5.6990	.1931	-1.3506	-.0803
1170000.	.6700	-.4870	-1.1268	6.0682	-.1739	-.6919	-1.3351
270000.	4.2300	-.5650	1.9003	5.4314	.6263	-1.8281	1.4007
4400000.	2.0000	-.2071	.0041	6.6435	.3010	.3344	.2886
2500000.	1.1000	-.3717	-.7612	6.3979	.0414	-.1036	-.5990
500000.	4.0000	-.5450	1.7047	5.6990	.6021	-1.3506	1.3177
27000000.	2.1000	1.7509	.0891	7.4314	.3222	1.7401	.3610
3689000.	1.2500	-.2687	-.6336	6.5669	.0969	.1978	-.4092
2200000.	1.0000	-.3977	-.8462	6.3424	0.0000	-.2027	-.7405
2400000.	1.7000	-.3804	-.2510	6.3802	.2304	-.1753	.0473
6420000.	2.2400	-.0321	.2082	6.8075	.3502	.6271	.4569
11000000.	1.6000	.3647	-.3360	7.0414	.2041	1.0443	-.0427
5500000.	.7000	-.1118	-1.1013	6.7404	-.1549	.5073	-1.2700
1250000.	.2500	-.4800	-1.4829	6.0969	-.6021	-.6407	-2.7987
1500000.	3.8000	-.4584	1.5346	6.1761	.5798	-.4994	1.2415
45000000.	3.2000	3.3105	1.0245	7.6532	.5051	2.1359	.9864
41900000.	1.4000	3.0429	-.5061	7.6222	.1461	2.0816	-.2409
3500000.	2.6000	-.2851	.5143	6.5441	.4150	.1571	.6781
500000.	4.0000	-.5450	1.7047	5.6990	.6021	-1.3506	1.3177
3900000.	1.2000	-.2554	-.6761	6.5911	.0792	.2409	-.4698
3000000.	3.4800	-.3264	1.2625	6.4771	.5416	.1376	1.1109
5000000.	1.4000	-.1551	-.5061	6.6990	.1461	.4334	-.2409

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE SEDIMENTARY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
2000000.	2.5000	-.5060	.4820	6.3010	.3979	-.4278	.4898
30000000.	2.6000	1.4966	.6348	7.4771	.4150	1.4144	.5653
1000000.	.8900	-.5775	-1.4951	6.0000	-.0506	-.8934	-1.4981
3300000.	2.4400	-.4130	.4383	6.5185	.3874	-.0872	.4430

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE PORPHYRY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
32500000.	1.9300	-.4521	2.6909	7.5119	.2856	-.7524	1.9536
43500000.	.5600	-.4354	-.5966	7.6385	-.2518	-.5713	-.4382
4500000.	.8700	-.4947	.1473	6.6532	-.0605	-1.9807	.3748
150000000.	.9300	-.2735	.2193	8.1761	-.0458	.1977	.4420
40500000.	.4000	-.4400	-.9906	7.6075	-.3979	-.6157	-1.1649
800000.	.7900	-.5003	-.0447	5.9031	-.1024	-3.0537	.1837
200000000.	.4500	-.1975	-.8606	8.3010	-.3468	.3765	-.9315
180000000.	1.0900	-.2279	.6752	8.2553	.0374	.3110	.8215
448000000.	.7000	.1794	-.2607	8.6513	-.1549	.8775	-.0560
180000000.	.5200	-.2279	-.6926	8.2553	-.2840	.3110	-.6450
293000000.	.4270	-.0562	-.9158	8.4669	-.3696	.6137	-1.0355
760000000.	.4700	.6536	-.8126	8.8808	-.3279	1.2058	-.8453
45000000.	.1000	-.4331	-1.7005	7.6532	-1.0000	-.5512	-3.9118
300000000.	1.5000	-.0455	1.6591	8.4771	.1761	.6284	1.4541
22000000.	.5000	-.4681	-.7406	7.3424	-.3010	-.9948	-.7227
183000000.	.8000	-.2234	-.0207	8.2625	-.0969	.3213	.2086
17000000.	.8000	-.4757	-.0207	7.2304	-.0969	-1.1550	.2086
12000000.	1.5000	-.4833	1.6591	7.0792	.1761	-1.3713	1.4541
592000000.	.7500	.3983	-.1407	8.7723	-.1249	1.0506	.0807
565000000.	.7200	.3572	-.2127	8.7520	-.1427	1.0216	-.0002
70000000.	.6000	-.3951	-.5006	7.8451	-.2218	-.2757	-.3615
60000000.	1.0000	-.4113	.4592	7.7782	0.0000	-.3715	.6507
500000000.	1.0000	.2584	.4592	8.6990	0.0000	.9457	.6507
120000000.	.5100	-.3191	-.7166	8.0792	-.2924	.0591	-.6835
35000000.	1.5000	-.4483	1.6591	7.5441	.1761	-.7063	1.4541
468000000.	1.0000	.2098	.4592	8.6702	0.0000	.9046	.6507
23000000.	.5000	-.4666	-.7406	7.3617	-.3010	-.9672	-.7227
189700000.	.6600	-.2132	-.3567	8.2781	-.1805	.3436	-.1726
60000000.	.7000	-.4113	-.2607	7.7782	-.1549	-.3715	-.0560
50000000.	.5000	-.4255	-.7406	7.6990	-.3010	-.4848	-.7227
50000000.	.6500	-.4255	-.3817	7.6990	-.1871	-.4848	-.2029
20000000.	.6300	-.4711	-.4287	7.3010	-.2007	-1.0540	-.2648
24000000.	.5500	-.4651	-.5206	7.3802	-.2596	-.9407	-.5339
70000000.	.8000	-.3951	-.0207	7.8451	-.0969	-.2757	.2086
500000000.	.5000	.2584	-.7406	8.6990	-.3010	.9457	-.7227
200000000.	.4000	-.1975	-.9806	8.3010	-.3979	.3765	-1.1649
200000000.	.9000	-.1975	.2193	8.3010	-.0458	.3765	.4420
100000000.	.8000	-.3495	-.0207	8.0000	-.0969	-.0541	.2086
80000000.	1.0000	-.3799	.4592	7.9031	0.0000	-.1928	.6507
50000000.	1.0000	-.4255	.4592	7.6990	0.0000	-.4848	.6507
40000000.	.8000	-.4407	-.0207	7.6021	-.0969	-.6234	.2086
1000000000.	.5000	1.0184	-.7406	9.0000	-.3010	1.3753	-.7227
4000000000.	1.0000	5.5753	.4592	9.6021	0.0000	2.2375	.6507
2000000000.	.8000	2.5384	-.0207	9.3010	-.0969	1.6069	.2086
800000000.	2.4300	.7144	3.8908	8.9031	.3856	1.2377	2.4101
400000000.	.6900	.1064	-.2847	8.6021	-.1612	.8071	-.0345

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STATISTICAL INFORMATION FOR COPPER
DEPOSIT TYPE COMPLEX SULPHIDE

CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0     20.0
                      ----     ----     ----     ----     ----
THEORETICAL CHI =     6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     5.667
FOR GRADE CALCULATED CHI =     1.222
FOR LOG - TONS CALCULATED CHI =      .333
FOR LOG - GRADE CALCULATED CHI =     3.889
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          11218245.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                  4.
EQUIVALENT LOG - GRADE MEAN =    1.2708
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    2.5463
BASIC MEAN TONNAGE =          20805556.
BASIC TONNAGE STANDARD DEVIATION =          22645701.
BASIC MEAN GRADE =    1.7533
BASIC GRADE STANDARD DEVIATION =    1.2687

                ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR COPPER
DEPOSIT TYPE HYDROTHERMAL

CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0     20.0
                      ----     ----     ----     ----     ----
THEORETICAL CHI =     6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     2.000
FOR GRADE CALCULATED CHI =     0.000
FOR LOG - TONS CALCULATED CHI =     0.000
FOR LOG - GRADE CALCULATED CHI =     2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          327615.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                  2.
EQUIVALENT LOG - GRADE MEAN =    2.0850
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    2.6551
BASIC MEAN TONNAGE =          415000.
BASIC TONNAGE STANDARD DEVIATION =          339951.
BASIC MEAN GRADE =    2.7000
BASIC GRADE STANDARD DEVIATION =    1.6269

                ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE STRATIFORM

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.400

FOR GRADE CALCULATED CHI = 1.500

FOR LOG - TONS CALCULATED CHI = 5.200

FOR LOG - GRADE CALCULATED CHI = 3.600

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 23607405.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 5.

EQUIVALENT LOG - GRADE MEAN = 2.05[illegible]7

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 1.8575

BASIC MEAN TONNAGE = 59185600.

BASIC TONNAGE STANDARD DEVIATION = 69240466.

BASIC MEAN GRADE = 2.3740

BASIC GRADE STANDARD DEVIATION = 1.0748

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR COPPER

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 1.200

FOR GRADE CALCULATED CHI = 3.600

FOR LOG - TONS CALCULATED CHI = 2.000

FOR LOG - GRADE CALCULATED CHI = 2.000

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 14926222.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 3.

EQUIVALENT LOG - GRADE MEAN = 1.7414

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 1.6[illegible]62

BASIC MEAN TONNAGE = 21510000.

BASIC TONNAGE STANDARD DEVIATION = 18986424.

BASIC MEAN GRADE = 1.9160

BASIC GRADE STANDARD DEVIATION = .8372

***** ANALYSIS COMPLETE *****

```
DEPOSIT TYPE SEDIMENTARY

CONFIDENCE LEVEL % =  1.0       2.5       5.0      10.0      20.0
                     ----      ----      ----      ----      ----
THEORETICAL CHI =    6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =     6.000
FOR GRADE CALCULATED CHI =     6.000
FOR LOG - TONS CALCULATED CHI =     2.000
FOR LOG - GRADE CALCULATED CHI =     6.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          3751166.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =              4.
EQUIVALENT LOG - GRADE MEAN =   1.9383
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   1.6812
BASIC MEAN TONNAGE =          9075060.
BASIC TONNAGE STANDARD DEVIATION =          13981744.
BASIC MEAN GRADE =   2.1075
BASIC GRADE STANDARD DEVIATION =    .8143

                ***** ANALYSIS COMPLETE *****
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DEPOSIT TYPE CONTACT METAMORPHIC

CONFIDENCE LEVEL % =  1.0       2.5       5.0      10.0      20.0
                     ----      ----      ----      ----      ----
THEORETICAL CHI =    6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =   59.714
FOR GRADE CALCULATED CHI =    2.857
FOR LOG - TONS CALCULATED CHI =    3.714
FOR LOG - GRADE CALCULATED CHI =    3.714
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          2857787.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =              4.
EQUIVALENT LOG - GRADE MEAN =   1.6467
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   1.9612
BASIC MEAN TONNAGE =          6790679.
BASIC TONNAGE STANDARD DEVIATION =          11542014.
BASIC MEAN GRADE =   1.9952
BASIC GRADE STANDARD DEVIATION =   1.1761

                ***** ANALYSIS COMPLETE *****
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                    STATISTICAL INFORMATION FOR COPPER
     DEPOSIT TYPE PORPHYRY

CONFIDENCE LEVEL % =  1.0       2.5       5.0      10.0      20.0
                      ----      ----      ----      ----      ----

THEORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =   61.478
FOR GRADE CALCULATED CHI =    7.343
FOR LOG - TONS CALCULATED CHI =      .435
FOR LOG - GRADE CALCULATED CHI =    5.304
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =        109107400.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =              5.
EQUIVALENT LOG - GRADE MEAN =    .7201
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   1.6564
BASIC MEAN TONNAGE =        329967391.
BASIC TONNAGE STANDARD DEVIATION =        657916116.
BASIC MEAN GRADE =    .8086
BASIC GRADE STANDARD DEVIATION =    .4167

              ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE HYDROTHERMAL

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
60000.	6.0000	-.4701	.5369	4.7782	.7782	-.9385	.5799
400000.	8.0000	-.4509	1.3038	5.6021	.9031	-.2116	.9402
1300000.	4.0000	-.4[illegible]	-.2[illegible]1	6.1139	.6021	.2562	.0721
110000.	1.0000	-.4672	-1.38[illegible]5	5.0414	0.0000	-.7031	-1.6642
40000000.	4.0000	1.7881	-.2301	7.6021	.6021	1.587[illegible]	.0721

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE COMPLEX SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
2401000.	5.1300	-.6173	-.0351	6.3804	.7101	-.3714	.2907
60800000.	3.5000	1.4754	-.4865	7.7839	.5441	1.1632	-.0432
18000000.	2.3560	-.1583	-.8134	7.2553	.3722	.5852	-.3389
17600000.	.2000	-.0727	-1.4006	7.2455	-.6990	.5745	-2.5430
1000000.	6.0800	-.6675	.2281	6.0000	.7839	-.7873	.4391
7000000.	10.5000	-.4525	1.4323	6.8451	1.0212	.1367	.9152
730000.	2.9000	-.6772	-.6527	5.8633	.4624	-.9367	-.2075
97000.	12.0000	-.6999	1.6678	4.9868	1.0792	-1.8951	1.0329
8650000.	5.6000	-.3934	.0951	6.9370	.7482	.2372	.3672
80000000.	4.3000	2.1634	-.2650	7.9031	.6335	1.2935	.1365

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
63000000.	4.0000	.0179	-.3956	7.7993	.6021	.5629	-.0495
4500000.	12.9700	-.8175	1.2588	6.6532	1.1129	-.9434	1.1339
3000000.	1.2000	-.8389	-.9120	6.4771	.0792	-1.1816	-1.2608
170000000.	4.0000	1.5458	-.3956	8.2304	.6021	1.1304	-.0495
120000000.	13.0000	.8318	1.2643	8.0792	1.1139	.9309	1.1362
10000000.	1.7000	-.7390	-.8198	7.0000	.2304	-.4921	-.9104

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE STRATIFORM

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
44100000.	12.3000	1.1486	1.1547	7.6444	1.0899	1.1303	1.1547
10200000.	2.8000	-.6766	-.5774	7.0086	.4472	-.7705	-.5774
14000000.	2.8000	-.4720	-.5774	7.1461	.4472	-.3595	-.5774

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
150000.	5.0000	-.6721	-.5408	5.1761	.6990	-1.0025	-.1485
2200000.	7.0000	-.5451	-.3565	6.3424	.8451	.0721	.2299
60000.	12.8000	-.6777	1.3520	4.7782	1.1072	-1.3692	.9086
600000.	5.5000	-.6442	-.4195	5.7782	.7404	-.4478	-.0413
34000000.	7.4000	1.4245	.0416	7.5315	.8692	1.1676	.2924
5000000.	.9000	-.3717	-1.5357	6.6990	-.0458	.4016	-2.0769
35000000.	12.0000	1.4864	1.1578	7.5441	1.0792	1.1792	.8356

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE PORPHYRY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
40500000.	5.0000	-.7071	.7071	7.6075	.6990	-.7071	.7071
800000000.	.1000	.7071	-.7071	8.9031	-1.0000	.7071	-.7071

```
CONFIDENCE LEVEL % =  1.0       2.5       5.0       10.0      20.0
                      ----      ----      ----      ----      ----

THEORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =     7.600
FOR GRADE CALCULATED CHI =     1.200
FOR LOG - TONS CALCULATED CHI =     2.000
FOR LOG - GRADE CALCULATED CHI =     2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =           5248533.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                8.
EQUIVALENT LOG - GRADE MEAN =    3.6777
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    3.1424
BASIC MEAN TONNAGE =          19627800.
BASIC TONNAGE STANDARD DEVIATION =          27906639.
BASIC MEAN GRADE =    5.2566
BASIC GRADE STANDARD DEVIATION =    3.6104

                    ***** ANALYSIS COMPLETE *****
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CONFIDENCE LEVEL % =  1.0       2.5       5.0       10.0      20.0
                      ----      ----      ----      ----      ----

THEORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =     8.600
FOR GRADE CALCULATED CHI =      .600
FOR LOG - TONS CALCULATED CHI =      .600
FOR LOG - GRADE CALCULATED CHI =     3.800
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =           672236.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                13.
EQUIVALENT LOG - GRADE MEAN =    3.7764
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    2.2221
BASIC MEAN TONNAGE =          8374000.
BASIC TONNAGE STANDARD DEVIATION =          17686483.
BASIC MEAN GRADE =    4.6000
BASIC GRADE STANDARD DEVIATION =    2.6977

                    ***** ANALYSIS COMPLETE *****
```

DEPOSIT TYPE STRATIFORM

CONFIDENCE LEVEL % = 1.0   2.5   5.0   10.0   20.0

THEORETICAL CHI =   6.63   5.02   3.84   2.71   1.64

FOR TONS CALCULATED CHI = 1.000
FOR GRADE CALCULATED CHI = 3.667
FOR LOG - TONS CALCULATED CHI = 1.000
FOR LOG - GRADE CALCULATED CHI = 3.667
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 18466635.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 2.
EQUIVALENT LOG - GRADE MEAN = 4.5957
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.3501
BASIC MEAN TONNAGE = 22766667.
BASIC TONNAGE STANDARD DEVIATION = 1857265.
BASIC MEAN GRADE = 5.9667
BASIC GRADE STANDARD DEVIATION = 5.4848

***** ANALYSIS COMPLETE *****

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

CONFIDENCE LEVEL % = 1.0   2.5   5.0   10.0   20.0

THEORETICAL CHI =   6.63   5.02   3.84   2.71   1.64

FOR TONS CALCULATED CHI = 3.333
FOR GRADE CALCULATED CHI = 2.000
FOR LOG - TONS CALCULATED CHI = .667
FOR LOG - GRADE CALCULATED CHI = 2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 23616602.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 6.
EQUIVALENT LOG - GRADE MEAN = 4.2018
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.7[illegible]20
BASIC MEAN TONNAGE = 61750000.
BASIC TONNAGE STANDARD DEVIATION = 70028387.
BASIC MEAN GRADE = 6.1450
BASIC GRADE STANDARD DEVIATION = 5.4219

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE PORPHYRY

```
CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0     20.0
                      ----     ----     ----     ----     ----
THEORETICAL CHI =     6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     2.000
FOR GRADE CALCULATED CHI =     2.000
FOR LOG - TONS CALCULATED CHI =     2.000
FOR LOG - GRADE CALCULATED CHI =     2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =        180000000.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =          8.
EQUIVALENT LOG - GRADE MEAN =     .7071
EQUIVALENT LOG - GRADE STANDARD DEVIATION =  15.8984
BASIC MEAN TONNAGE =        420250000.
BASIC TONNAGE STANDARD DEVIATION =        537047600.
BASIC MEAN GRADE =   2.5500
BASIC GRADE STANDARD DEVIATION =   3.4648
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***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR LEAD

DEPOSIT TYPE CONTACT METAMORPHIC

```
CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0     20.0
                      ----     ----     ----     ----     ----
THEORETICAL CHI =     6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     2.714
FOR GRADE CALCULATED CHI =     1.571
FOR LOG - TONS CALCULATED CHI =     .429
FOR LOG - GRADE CALCULATED CHI =     .429
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =        1837344.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =          12.
EQUIVALENT LOG - GRADE MEAN =   5.7060
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   2.4333
BASIC MEAN TONNAGE =        11001429.
BASIC TONNAGE STANDARD DEVIATION =        16145206.
BASIC MEAN GRADE =   7.2286
BASIC GRADE STANDARD DEVIATION =   4.1210
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***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE HYDROTHERMAL

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
60000.	6.7000	-.5065	-.1452	4.7782	.8261	-.7652	.0450
400000.	9.0000	-.4894	1.3419	5.6021	.9542	-.1204	1.2690
110000.	6.4000	-.5040	-.2262	5.0414	.8062	-.5532	-.1450
40000000.	5.0000	1.5000	-1.0705	7.6021	.6990	1.4448	-1.1690

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE COMPLEX SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
2401000.	9.9500	-.6226	-.1452	6.3804	.9978	-.5631	.3064
60800000.	8.8900	1.4757	-.2367	7.7839	.9484	1.2866	.2116
18000000.	5.6300	-.0607	-.5149	7.2553	.7505	.5899	-.1683
17600000.	3.8000	-.0751	-.6715	7.2455	.5798	.5770	-.4961
1000000.	4.0000	-.6709	-.6544	6.0000	.6021	-1.0645	-.4533
7000000.	3.1000	-.4556	-.7314	6.8451	.4914	.0493	-.6658
730000.	37.0000	-.6806	2.1700	5.8633	1.5682	-1.2445	1.4014
730000.	24.9000	-.6806	1.1344	5.8633	1.3962	-1.2446	1.0712
8650000.	18.6000	-.3963	.5952	6.9370	1.2695	.1734	.8280
80000000.	.6000	2.1649	-.9454	7.9031	-.2218	1.4436	-2.0350

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
40500000.	5.0000	-.4244	-.5417	7.6075	.6990	.0471	-.4746
62500000.	7.0800	-.0313	.1923	7.7959	.8500	.3739	.3828
63000000.	5.0000	-.0223	-.5417	7.7993	.6990	.3799	-.4746
45000000.	10.7000	-.3440	1.4697	7.6532	1.0294	.1264	1.4007
3000000.	3.5000	-1.0944	-1.0710	6.4771	.5441	-1.9139	-1.3538
170000000.	5.0000	1.8896	-.5417	8.2304	.6990	1.1279	-.4746
120000000.	11.0000	.9961	1.5756	8.0792	1.0414	.8654	1.4689
10000000.	5.0000	-.9693	-.5417	7.0000	.6990	-1.0068	-.4746

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE STRATIFORM

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
18000000.	10.0000	-.1035	-.3732	7.2553	1.0000	.1417	.2772
11000000.	1.5000	-.5996	-.9876	7.0414	.1761	-.6927	-1.5621
44100000.	26.3000	1.7463	1.68[illegible]4	7.6444	1.4200	1.6571	1.2147
10200000.	8.2000	-.6563	-.2668	7.0086	.9138	-.8205	.0848
14000000.	7.4000	-.3870	-.3529	7.1461	.8692	-.2846	-.3147

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
50000000.	5.0000	1.7099	-.2487	7.6990	.6990	.9879	.1465
270000.	.4000	-.9272	-1.3925	5.4314	-.3979	-1.2293	-2.2155
27000000.	5.1000	.4934	-.2238	7.4314	.7076	.7263	.1650
2200000.	13.0000	-.8182	1.7466	6.3424	1.1139	-.3385	1.0410
5500000.	5.0000	-.6436	-.2487	6.7404	.6990	.7976	.1465
60000.	7.3000	-.9314	.3233	4.7782	.8633	-1.6683	.5034
34000000.	5.6000	.8637	-.3995	7.5315	.7482	.8242	.2524
5000000.	1.6000	-.6711	-1.0941	6.6990	.2041	.0102	-.9191
35000000.	11.0000	.9165	1.2433	7.5441	1.0414	.8365	.6838

STATISTICAL INFORMATION FOR URANIUM

DEPOSIT TYPE SEDIMENTARY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
28100000.	.0340	1.1925	-.5930	7.4487	-1.4685	1.1432	-.5676
1700000.	.0930	-.6552	-.3784	6.2304	-1.0315	-.4336	.1086
250000.	.5000	-.7561	1.1024	5.3979	-.3010	-1.5112	1.2389
1350000.	.1200	-.6795	-.2801	6.1303	-.9208	-.5532	.2798
37000000.	.0300	1.8020	-.6076	7.5682	-1.5229	1.2979	-.6517
16000000.	.0090	.3402	-.6840	7.2041	-2.0458	.8266	-1.4608
1000000.	.7500	-.7039	2.0120	6.0000	-.1249	-.7319	1.5113
3500000.	.0400	-.5299	-.5712	6.5441	-1.3979	-.0277	-.4584

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE HYDROTHERMAL

FIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
ORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

TONS CALCULATED CHI = 6.000

GRADE CALCULATED CHI = 2.000

LOG - TONS CALCULATED CHI = 2.000

LOG - GRADE CALCULATED CHI = 0.000

ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

IVALENT LOG - TONNAGE MEAN = 570054.

IVALENT LOG - TONNAGE STANDARD DEVIATION = 13.

IVALENT LOG - GRADE MEAN = 6.6278

IVALENT LOG - GRADE STANDARD DEVIATION = 1.2726

IC MEAN TONNAGE = 10142500.

IC TONNAGE STANDARD DEVIATION = 19915564.

IC MEAN GRADE = 6.7750

IC GRADE STANDARD DEVIATION = 1.6581

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR URANIUM

DEPOSIT TYPE SEDIMENTARY

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 1.000

FOR GRADE CALCULATED CHI = 7.000

FOR LOG - TONS CALCULATED CHI = 3.000

FOR LOG - GRADE CALCULATED CHI = 1.000

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 3676662.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 6.

EQUIVALENT LOG - GRADE MEAN = .0791

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 4.4287

BASIC MEAN TONNAGE = 11112500.

BASIC TONNAGE STANDARD DEVIATION = 14366293.

BASIC MEAN GRADE = .1970

BASIC GRADE STANDARD DEVIATION = .2749

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

NFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
EORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

R TONS CALCULATED CHI = 4.000
R GRADE CALCULATED CHI = 3.000
R LOG - TONS CALCULATED CHI = 4.000
R LOG - GRADE CALCULATED CHI = 3.000
ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
UIVALENT LOG - TONNAGE MEAN = 38047834.
UIVALENT LOG - TONNAGE STANDARD DEVIATION = 4.
UIVALENT LOG - GRADE MEAN = 6.0617
UIVALENT LOG - GRADE STANDARD DEVIATION = 1.5[illegible]
SIC MEAN TONNAGE = 64250000.
SIC TONNAGE STANDARD DEVIATION = 55965551.
SIC MEAN GRADE = 6.5350
SIC GRADE STANDARD DEVIATION = 2.8339

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE COMPLEX SULPHIDE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 5.000
FOR GRADE CALCULATED CHI = 2.000
FOR LOG - TONS CALCULATED CHI = 2.000
FOR LOG - GRADE CALCULATED CHI = 2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 6422337.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 6.
EQUIVALENT LOG - GRADE MEAN = 6.8897
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 3.3183
BASIC MEAN TONNAGE = 19691100.
BASIC TONNAGE STANDARD DEVIATION = 27858091.
BASIC MEAN GRADE = 11.6460
BASIC GRADE STANDARD DEVIATION = 11.6840

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE CONTACT METAMORPHIC

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 3.000
FOR GRADE CALCULATED CHI = 2.111
FOR LOG - TONS CALCULATED CHI = 2.111
FOR LOG - GRADE CALCULATED CHI = 5.667
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 4881714.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 11.
EQUIVALENT LOG - GRADE MEAN = 4.2751
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.9135
BASIC MEAN TONNAGE = 17670000.
BASIC TONNAGE STANDARD DEVIATION = 18908022.
BASIC MEAN GRADE = 6.0000
BASIC GRADE STANDARD DEVIATION = 4.0215

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR ZINC

DEPOSIT TYPE STRATIFORM

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 6.600
FOR GRADE CALCULATED CHI = 3.800
FOR LOG - TONS CALCULATED CHI = .600
FOR LOG - GRADE CALCULATED CHI = .600
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 16564045.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 2.
EQUIVALENT LOG - GRADE MEAN = 7.5131
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.8151
BASIC MEAN TONNAGE = 19460000.
BASIC TONNAGE STANDARD DEVIATION = 14110209.
BASIC MEAN GRADE = 10.6800
BASIC GRADE STANDARD DEVIATION = 9.2955

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE HYDROTHERMAL

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
118000.	.1500	-.4019	-.4895	5.0719	-.8239	-.6127	-.5155
16000.	4.0000	-.4044	2.4847	4.2041	.6021	-1.3633	1.8968
760000.	.3200	-.3863	-.3582	5.8808	-.4949	.0871	.0412
60000.	.0440	-.4033	-.5714	4.7782	-1.3565	-.8668	-1.4166
1000000.	.4800	-.3804	-.2346	6.0000	-.3188	.1912	.3391
118000.	.1500	-.4019	-.4895	5.0719	-.8239	-.6127	-.5155
200000.	3.0000	-.3999	1.7121	5.3010	.4771	-.4144	1.6854
900000.	.1000	-.3829	-.5281	5.9542	-1.0000	.1506	-.8134
160000.	.0700	-.4009	-.5513	5.2041	-1.1549	-.4983	-1.0754
970000.	.4500	-.3811	-.2578	5.9868	-.3468	.1788	.2916
60000000.	.3200	1.0575	-.3582	7.7782	-.4949	1.7284	.0412
135000000.	.3200	2.8855	-.3582	8.1303	-.4949	2.0331	.0412

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE COMPLEX SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
2401000.	.0840	-.7700	-.3212	6.3804	-1.0757	-1.2119	-.0917
17600000.	.0380	-.1624	-1.0954	7.2455	-1.4202	.2833	-1.3116
8650000.	.1100	-.5202	.1180	6.9370	-.9586	-.2499	.3230
58000000.	.1800	1.4526	1.2976	7.7634	-.7447	1.1784	1.0803

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
33000000.	.0200	.6334	-1.0713	7.5185	-1.6990	.7406	-1.3598
9530000.	.0900	-.8323	.0275	6.9791	-1.0458	-.8184	.3049
8900000.	.1730	-.8717	1.3303	6.9494	-.7620	-.9042	1.0282
40000000.	.0700	1.0706	-.2865	7.6021	-1.1549	.9820	.0267

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
500000.	.1350	-.3749	-.6693	5.6990	-.8697	-.2637	-.4501
270000.	.0300	-.5597	-.7302	5.4314	-1.5229	-.9660	-1.1879
500000.	3.5000	-.3749	1.2848	5.6990	.5441	-.2637	1.1468
530000.	.5000	-.35[illegible]8	-.4573	5.7243	-.3010	-.1973	.1922
3500000.	.0600	2.0352	-.7128	6.5441	-1.2218	1.9543	-.8479
500000.	3.5000	-.3749	1.2848	5.6990	.5441	-.2637	1.1468

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE SEDIMENTARY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
28100000.	.4000	-.1672	.0075	7.4487	-.3979	.3124	.2395
20600000.	.0400	-.4557	-1.7469	7.3139	-1.3979	.1698	-1.1687
4210000.	.2700	-1.0862	-.6260	6.6243	-.5686	-1.1714	-.5129
28000000.	.4700	-.1711	.3486	7.4472	-.3279	.3097	.5483
2000000.	.2500	-1.1712	-.7235	6.3010	-.6021	-1.7532	-.6613
5000000.	.2300	-1.0559	-.8210	6.6990	-.6383	-1.0359	-.8199
14000000.	.2100	-.7096	-.9184	7.1461	-.6778	-.2321	-.9941
18000000.	.2200	-.5558	-.8697	7.2553	-.6576	-.0357	-.9050
23000000.	.3600	-.3634	-.1874	7.3617	-.4437	.1559	.0378
37000000.	.3300	.1751	-.3336	7.5682	-.4815	.5275	-.1288
16000000.	.2600	-.6327	-.6748	7.2041	-.5850	-.1278	-.5852
17000000.	.2600	-.5942	-.6748	7.2304	-.5850	-.0834	-.5852
95000000.	.5200	2.4062	.5923	7.9777	-.2840	1.2546	.7418
50000000.	.3000	.6752	-.4798	7.6990	-.5229	.7629	-.3112
3400000.	.2900	-1.1174	-.5286	6.5315	-.5376	-1.3384	-.3761
1600000.	.4100	-1.1866	.0562	6.2041	-.3872	-1.9276	.2868
3150000.	.3500	-1.1270	-.2362	6.4983	-.4559	-1.3931	-.0161
54000000.	.3500	.8291	-.2362	7.7324	-.4559	.8230	-.0161
37000000.	.2980	.1751	-.4896	7.5682	-.5258	.5275	-.3240
16000000.	.2800	-.6327	-.5773	7.2041	-.5528	-.1278	-.4433
24800000.	.1700	-.2942	-1.1134	7.3945	-.7696	.2148	-1.3986
66100000.	.4800	1.2945	.3974	7.8202	-.3188	.9811	.5686
2030000.	.2790	-1.1701	-.5822	6.3075	-.5544	-1.7415	-.4532
1300000.	.6900	-1.1982	1.4208	6.1139	-.1612	-2.0899	1.2834
3280000.	.0950	-1.1220	-1.4758	6.5159	-1.0223	-1.3665	-2.5127
29060000.	.4300	-.1303	.1537	7.4633	-.3665	.3387	.3780
100000000.	.4400	2.5985	.2024	8.0000	-.3565	1.3047	.4220
11590000.	.5500	-.5023	.7385	7.0641	-.2596	-.3738	.8492
25900000.	.4000	-.2519	.0075	7.4133	-.3979	.2487	.2395
6360000.	.2700	-1.0035	-.6260	6.8035	-.5686	-.8489	-.5129
710000.	.3600	-1.2209	-.1874	5.8513	-.4437	-2.5627	.0378
2660000.	.4300	-1.1458	.1537	6.4249	-.3665	-1.5332	.3780
21880000.	.4400	-.4065	.2024	7.3400	-.3565	.1159	.4220
9390000.	.4600	-.8870	.2999	6.9727	-.3372	-.5443	.5071
64880000.	.8110	1.2476	2.0104	7.8121	-.0910	.9665	1.5927
24270000.	.3000	-.3146	-.4798	7.3851	-.5229	.1979	-.3112
49540000.	1.2700	.6575	4.2473	7.6950	.1038	.7556	2.4514
75550000.	.6600	1.6580	1.2746	7.8782	-.1805	1.0855	1.1983
68900000.	.3800	1.4022	-.0900	7.8382	-.4202	1.0135	.1413
13850000.	.3900	-.7154	-.0412	7.1414	-.4089	-.2426	.1911
53650000.	.4000	.8156	.0075	7.7296	-.3979	.8179	.2395
49150000.	.6500	[illegible]	1.2258	7.6915	-.1871	.7495	1.1593
70180000.	.7000	1.4515	1.4695	7.8462	-.1549	1.0279	1.3109
48250000.	.4300	.6079	.1537	7.6835	-.3665	.7350	.3780
58740000.	.7000	1.0114	1.4695	7.7689	-.1549	.8888	1.3109
38600000.	.2000	.2367	-.9672	7.5866	-.6990	.5606	-1.0875
52500000.	.2700	.7714	-.6260	7.7202	-.5686	.8114	-.5129
64890000.	.3800	1.2480	-.0900	7.8122	-.4202	.9665	.1413
53980000.	.3500	.8283	-.2362	7.7322	-.4559	.8227	-.0161
32000000.	.2200	-.0172	-.8697	7.5051	-.6576	.4140	-.9050
68380000.	.4700	1.3822	.3486	7.8349	-.3279	1.0076	.5483
8020000.	.3100	-.9397	-.4311	6.9042	-.5086	-.6676	-.2465
22520000.	.4600	-.3819	.2999	7.3526	-.3372	.1394	.5071
34920000.	.2200	.0951	-.8697	7.5431	-.6576	.4823	-.9050
52760000.	.4100	.7814	.0562	7.7223	-.3872	.8049	.2868
80440000.	.4100	1.8461	.0562	7.9055	-.3872	1.1345	.2868
60000000.	.4100	1.0599	.0562	7.7782	-.3872	.9054	.2868
23090000.	.4000	-.1680	.0075	7.4484	-.3979	.3119	.2395
37000000.	1.0400	.1751	3.1264	7.5682	.0170	.5275	2.0688
6100000.	.1900	-1.0135	-1.1159	6.7853	-.7212	-.8315	-1.1857
1530000.	.2400	-1.1893	-.7722	6.1847	-.6198	-1.9625	-.7384
2100000.	.2500	-1.1674	-.7235	6.3222	-.6021	[illegible]	-.6613
45300000.	.5900	.4944	.9334	7.6561	-.2291	[illegible]	.9316

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE PORPHYRY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
760000000.	.0200	.5103	-.2697	8.8808	-1.6990	.6499	.1066
60000000.	.0280	-1.0267	-.0066	7.7782	-1.5528	-.7353	.4161
18000000.	.0800	-1.1189	1.7039	7.2553	-1.0969	-1.3921	1.3916
1000000000.	.0050	1.0372	-.7631	9.0000	-2.3010	.7935	-1.1683
800000000.	.0080	.5981	-.6645	8.9031	-2.0969	.6779	-.7360

```
DEPOSIT TYPE COMPLEX SULPHIDE

CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0     20.0
                      ----     ----     ----     ----     ----

THEORETICAL CHI =     6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     2.000
FOR GRADE CALCULATED CHI =     0.000
FOR LOG - TONS CALCULATED CHI =     0.000
FOR LOG - GRADE CALCULATED CHI =     0.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          12066664.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                  4.
EQUIVALENT LOG - GRADE MEAN =     .0892
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    1.9161
BASIC MEAN TONNAGE =          21662750.
BASIC TONNAGE STANDARD DEVIATION =          25014985.
BASIC MEAN GRADE =     .1030
BASIC GRADE STANDARD DEVIATION =     .0593

                    ***** ANALYSIS COMPLETE *****
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DEPOSIT TYPE HYDROTHERMAL

CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0     20.0
                      ----     ----     ----     ----     ----

THEORETICAL CHI =     6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     22.667
FOR GRADE CALCULATED CHI =     22.667
FOR LOG - TONS CALCULATED CHI =     1.333
FOR LOG - GRADE CALCULATED CHI =     2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          602733.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                  14.
EQUIVALENT LOG - GRADE MEAN =     .3026
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    3.9605
BASIC MEAN TONNAGE =          16608500.
BASIC TONNAGE STANDARD DEVIATION =          41030275.
BASIC MEAN GRADE =     .7837
BASIC GRADE STANDARD DEVIATION =     1.2945

                    ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE CONTACT METAMORPHIC

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 11.333

FOR GRADE CALCULATED CHI = 2.000

FOR LOG - TONS CALCULATED CHI = 6.000

FOR LOG - GRADE CALCULATED CHI = .667

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 630139.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 2.

EQUIVALENT LOG - GRADE MEAN = .3379

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 7.6789

BASIC MEAN TONNAGE = 966667.

BASIC TONNAGE STANDARD DEVIATION = 124476[illegible].

BASIC MEAN GRADE = 1.2875

BASIC GRADE STANDARD DEVIATION = 1.7220

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR GOLD

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.000

FOR GRADE CALCULATED CHI = 0.000

FOR LOG - TONS CALCULATED CHI = 4.000

FOR LOG - GRADE CALCULATED CHI = 2.000

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 18292127.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 2.

EQUIVALENT LOG - GRADE MEAN = .0683

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.4692

BASIC MEAN TONNAGE = 22857500.

BASIC TONNAGE STANDARD DEVIATION = 16012179.

BASIC MEAN GRADE = .0883

BASIC GRADE STANDARD DEVIATION = .0637

***** ANALYSIS COMPLETE *****

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STATISTICAL INFORMATION FOR GOLD
    DEPOSIT TYPE PORPHYRY

CONFIDENCE LEVEL % =  1.0       2.5       5.0      10.0      20.0
                     ----      ----      ----      ----      ----

THEORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =     2.200
FOR GRADE CALCULATED CHI =     3.800
FOR LOG - TONS CALCULATED CHI =     2.200
FOR LOG - GRADE CALCULATED CHI =     2.200
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =         230922083.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                 6.
EQUIVALENT LOG - GRADE MEAN =     .0178
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   2.9664
BASIC MEAN TONNAGE =         527600000.
BASIC TONNAGE STANDARD DEVIATION =         455443520.
BASIC MEAN GRADE =     .0282
BASIC GRADE STANDARD DEVIATION =     .0304

                    ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR GOLD
    DEPOSIT TYPE SEDIMENTARY

CONFIDENCE LEVEL % =  1.0       2.5       5.0      10.0      20.0
                     ----      ----      ----      ----      ----

THEORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =     4.365
FOR GRADE CALCULATED CHI =     3.476
FOR LOG - TONS CALCULATED CHI =     5.508
FOR LOG - GRADE CALCULATED CHI =     6.524
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =         18641230.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                 4.
EQUIVALENT LOG - GRADE MEAN =     .3530
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   1.6860
BASIC MEAN TONNAGE =         32447619.
BASIC TONNAGE STANDARD DEVIATION =         25996316.
BASIC MEAN GRADE =     .3985
BASIC GRADE STANDARD DEVIATION =     .2052

                    ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE HYDROTHERMAL

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
118000.	10.0000	-.6289	1.0772	5.0719	1.0000	-.6761	.6472
60000.	6.9000	-.6311	.2948	4.7782	.8388	-.6938	.4507
400000.	4.2000	-.6183	-.3865	5.6021	.6232	-.2831	.1879
118000.	10.0000	-.6289	1.0772	5.0719	1.0000	-.6761	.6472
60000000.	.0700	1.6311	-1.4288	7.7782	-1.1549	1.3298	-1.9833
40000000.	3.2200	.8763	-.6358	7.6021	.5079	1.1993	.6472

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE COMPLEX SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
3200000.	.3500	-.6628	-1.0031	6.5051	-.4559	-.7559	-1.3387
2401000.	2.6400	-.6908	.1322	6.3804	.4216	-.9528	.4918
60800000.	2.4000	1.3601	.0132	7.7839	.3802	1.2629	.4055
18000000.	1.8400	-.1430	-.2644	7.2553	.2648	.4284	.1647
17600000.	.6100	-.1571	-.8742	7.2455	-.2147	.4229	-.8354
1000000.	6.0000	-.7400	1.7950	6.0000	.7782	-1.5533	1.2355
7000000.	2.1300	-.5293	-.1206	6.8451	.3284	-.2192	.2973
8650000.	5.1000	-.4714	1.3518	6.9370	.7076	-.0741	1.0883
80000000.	.2900	2.0343	-1.0329	7.9031	-.5376	1.4511	-1.5090

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
33000000.	.3000	-.8578	-.9594	7.5185	-.5229	-1.1334	-1.7187
62500000.	4.8500	-.2722	.8551	7.7959	.6857	-.0533	.9466
63000000.	1.0000	-.2623	-.6812	7.7993	0.0000	-.0459	-.5656
45000000.	7.3900	-.6196	1.8681	7.6532	.8686	-.6122	1.3499
40000000.	1.7000	-.7188	-.4011	7.6021	.2304	-.8078	-.0574
170000000.	1.7700	1.8616	-.3732	8.2304	.2480	1.6189	-.0188
120000000.	1.9300	.8691	-.3094	8.0792	.2856	1.0348	.0641

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
150000.	20.0000	-.6253	2.1856	5.1761	1.3010	-1.1543	1.3462
3000000.	1.6600	-.4178	-.5697	6.4771	.2201	.2149	-.2037
6000000.	1.6600	-.1994	-.5697	6.7782	.2201	.5317	-.2037
270000.	.1200	-.6166	-.8011	5.4314	-.9208	-.8857	-1.8397
4400000.	.3000	-.3159	-.7740	6.6435	-.5229	.3933	-1.2691
60000.	14.1000	-.6319	1.2932	4.7782	1.1492	-1.5732	1.1285
600000.	2.0000	-.5926	-.5186	5.7782	.3010	-.5207	-.0877
34000000.	5.4000	1.8394	-.0078	7.5315	.7324	1.3246	.5308
3900000.	1.8700	-.3523	-.5381	6.5911	.2718	.3348	-.1295
35000000.	7.4100	1.9122	.2942	7.5441	.8698	1.3378	.7279

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE PORPHYRY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
18000000.	1.3500	-1.1331	.1573	7.2553	.1303	-1.1533	.4601
1000000000.	.0500	.7592	-1.0693	9.0000	-1.3010	.6261	-1.1472
800000000.	2.1500	.3778	.9121	8.9031	.3324	.5272	.6871

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE STRATIFORM

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
10200000.	.9000	-.7071	.7071	7.0086	-.0458	-.7071	.7071
14000000.	.8700	.7071	-.7071	7.1461	-.0605	.7071	-.7071

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE COMPLEX SULPHIDE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 5.667

FOR GRADE CALCULATED CHI = .333

FOR LOG - TONS CALCULATED CHI = .333

FOR LOG - GRADE CALCULATED CHI = 3.889

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 9636897.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 4.

EQUIVALENT LOG - GRADE MEAN = 1.5340

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 3.0158

BASIC MEAN TONNAGE = 22072333.

BASIC TONNAGE STANDARD DEVIATION = 28474849.

BASIC MEAN GRADE = 2.3733

BASIC GRADE STANDARD DEVIATION = 2.0170

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE HYDROTHERMAL

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 7.333

FOR GRADE CALCULATED CHI = .667

FOR LOG - TONS CALCULATED CHI = 3.333

FOR LOG - GRADE CALCULATED CHI = 11.333

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 963898.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 22.

EQUIVALENT LOG - GRADE MEAN = 2.9456

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 6.6189

BASIC MEAN TONNAGE = 16782667.

BASIC TONNAGE STANDARD DEVIATION = 26496204.

BASIC MEAN GRADE = 5.7317

BASIC GRADE STANDARD DEVIATION = 3.9626

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE STRATIFORM

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.000

FOR GRADE CALCULATED CHI = 2.000

FOR LOG - TONS CALCULATED CHI = 2.000

FOR LOG - GRADE CALCULATED CHI = 2.000

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 11949895.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 1.

EQUIVALENT LOG - GRADE MEAN = .8849

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 1.0243

BASIC MEAN TONNAGE = 12100000.

BASIC TONNAGE STANDARD DEVIATION = 2687005.

BASIC MEAN GRADE = .8850

BASIC GRADE STANDARD DEVIATION = .0212

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.714

FOR GRADE CALCULATED CHI = 2.714

FOR LOG - TONS CALCULATED CHI = 2.714

FOR LOG - GRADE CALCULATED CHI = 1.571

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 64749122.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 2.

EQUIVALENT LOG - GRADE MEAN = 1.8051

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.8410

BASIC MEAN TONNAGE = 76214286.

BASIC TONNAGE STANDARD DEVIATION = 50380103.

BASIC MEAN GRADE = 2.7057

BASIC GRADE STANDARD DEVIATION = 2.5075

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE PORPHYRY

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 1.000
FOR GRADE CALCULATED CHI = 1.000
FOR LOG - TONS CALCULATED CHI = 3.667
FOR LOG - GRADE CALCULATED CHI = 1.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 243288080.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 10.
EQUIVALENT LOG - GRADE MEAN = .5255
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 7.7713
BASIC MEAN TONNAGE = 606000000.
BASIC TONNAGE STANDARD DEVIATION = 518948938.
BASIC MEAN GRADE = 1.1833
BASIC GRADE STANDARD DEVIATION = 1.0599

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR SILVER

DEPOSIT TYPE CONTACT METAMORPHIC

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 17.200
FOR GRADE CALCULATED CHI = 3.600
FOR LOG - TONS CALCULATED CHI = 2.000
FOR LOG - GRADE CALCULATED CHI = 2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 1874604.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 9.
EQUIVALENT LOG - GRADE MEAN = 2.3024
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 4.9820
BASIC MEAN TONNAGE = 8738000.
BASIC TONNAGE STANDARD DEVIATION = 13733586.
BASIC MEAN GRADE = 5.4520
BASIC GRADE STANDARD DEVIATION = 6.6562

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR IRON

DEPOSIT TYPE SEDIMENTARY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
40000000.	30.0000	-.6450	-1.1620	7.6021	1.4771	-.7786	-1.1739
5000000000.	44.0000	.0426	-.3498	6.6990	1.6435	.6213	-.1693
6040000.	43.3000	-.6958	-.3904	6.7810	1.6365	-1.8265	-.2114
1000000000.	64.0000	.7901	.8106	9.0000	1.8062	1.0055	.8135
5000000.	64.0000	-.6973	.8106	6.6990	1.8062	-1.9312	.8135
40000000.	30.0000	-.6450	-1.1620	7.6021	1.4771	-.7786	-1.1739
50000000.	27.0000	-.6300	-1.3360	7.6990	1.4314	-.6549	-1.4503
1000000000.	20.0000	.7901	-1.7421	9.0000	1.3010	1.0055	-2.2375
350000000.	64.0000	-.1816	.8106	8.5441	1.8062	.4236	.8135
50000000.	57.8000	-.6300	.4559	7.6990	1.7619	-.6549	.5463
10000000.	60.0000	-.6898	.5785	7.0000	1.7782	-1.5470	.6443
200000000.	90.0000	-.4058	2.3190	8.3010	1.9542	.1135	1.7078
300000000.	48.0000	-.2563	-.1177	8.4771	1.6812	.3382	.0589
3000000000.	65.0000	1.7798	.8686	9.4771	1.8129	1.6145	.8542
50000000.	61.5000	-.6300	.6655	7.6990	1.7889	-.6549	.7090
800000000.	40.0000	.4911	-.5818	8.9031	1.6021	.8818	-.4193
200000000.	48.0000	-.4058	-.1177	8.3010	1.6812	.1135	.0589
450000000.	67.0000	-.0321	.9846	8.6532	1.8261	.5629	.9337
650000000.	50.0000	.2669	-.0017	8.8129	1.6990	.7568	.1660
800000000.	50.0000	.4911	-.0017	8.9031	1.6990	.8818	.1660
400000000.	27.0000	-.1068	-1.3360	8.6021	1.4314	.4977	-1.4503

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
27670000.	1.4900	1.5987	-.7666	7.4420	.1732	.8596	-.5057
13500000.	.6600	-.0802	-1.4421	7.1303	-.1805	.2446	-1.7251
14290000.	3.4000	.0134	.7855	7.1550	.5315	.2934	.7298
14300000.	3.4000	.0146	.7855	7.1553	.5315	.2940	.7298
14300000.	3.6500	.0146	.9887	7.1553	.5623	.2940	.8360
1000000.	2.0000	-1.5612	-.3521	6.0000	.3010	-1.9855	-.0648

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE OXIDE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
251000.	1.1400	.7071	.7071	5.3997	.0569	.7071	.7071
250000.	1.1300	-.7071	-.7071	5.3979	.0531	-.7071	-.7071

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE LATERITE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
50000000.	1.7000	-.6257	.4319	7.6990	.2304	-.6881	.4946
91700000.	1.9700	.0579	1.5641	7.9624	.2945	.3624	1.5038
64000000.	1.5500	-.3962	-.1971	7.8062	.1903	-.2605	-.1379
45000000.	1.5000	-.7076	-.4068	7.6532	.1761	-.8705	-.3624
62000000.	1.5500	-.4290	-.1971	7.7924	.1903	-.3155	-.1379
45000000.	1.5000	-.7076	-.4068	7.6532	.1761	-.8705	-.3624
44000000.	1.6000	-.7240	.0126	7.6435	.2041	-.9095	.0795
80000000.	2.0000	-.1339	1.6899	7.9031	.3010	.1260	1.6073
200000000.	1.3000	1.8331	-1.2454	8.3010	.1139	1.7131	-1.3422
200000000.	1.3000	1.8331	-1.2454	8.3010	.1139	1.7131	-1.3422

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE STRATIFORM

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
500000.	4.0000	-.7071	.7071	5.6990	.6021	-.7071	.7071
1000000.	1.5000	.7071	-.7071	6.0000	.1761	.7071	-.7071

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
1250000.	3.0700	1.2687	.1513	6.0969	.4871	1.2178	.3721
500000.	.6400	-.7807	-1.3902	5.6990	-.1938	-.7911	-1.4455
500000.	4.0000	-.7807	.7413	5.6990	.6021	-.7911	.6788
1000000.	4.4000	.5855	.9950	6.0000	.6435	.7286	.7893
500000.	4.0000	-.7807	.7413	5.6990	.6021	-.7911	.6788
1250000.	3.0700	1.2687	.1513	6.0969	.4871	1.2178	.3721
500000.	.6400	-.7807	-1.3902	5.6990	-.1938	-.7911	-1.4455

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE SEDIMENTARY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
500000.	4.0000	-.7071	.7071	5.6990	.6021	-.7071	.7071
45000000.	1.5500	.7071	-.7071	7.6532	.1903	.7071	-.7071

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE PORPHYRY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
40500000.	.2000	-.7071	-.7071	7.6075	-.6990	-.7071	-.7071
45000000.	1.5000	.7071	.7071	7.6532	.1761	.7071	.7071

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            STATISTICAL  INFORMATION FOR IRON
 DEPOSIT TYPE SEDIMENTARY

FIDENCE LEVEL % =  1.0       2.5       5.0       1.0      10.0
                   ----      ----      ----      ----     ----
ORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

 TONS CALCULATED CHI =     6.524
 GRADE CALCULATED CHI =     1.667
 LOG - TONS CALCULATED CHI =     1.667
 LOG - GRADE CALCULATED CHI =     1.667
ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
IVALENT LOG - TONNAGE MEAN =        162978619.
IVALENT LOG - TONNAGE STANDARD DEVIATION =                  6.
IVALENT LOG - GRADE MEAN =  -6.9335
IVALENT LOG - GRADE STANDARD DEVIATION =   1.4641
IC MEAN TONNAGE =        471478095.
IC TONNAGE STANDARD DEVIATION =        668963302.
IC MEAN GRADE =  50.0286
IC GRADE STANDARD DEVIATION =  17.2365

                   ***** ANALYSIS COMPLETE *****
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            STATISTICAL  INFORMATION FOR NICKEL
 DEPOSIT TYPE PORPHYRY

CONFIDENCE LEVEL % =  1.0       2.5       5.0       10.0      20.0
                      ----      ----      ----      ----      ----
THEORETICAL CHI =     6.63      5.02      3.84      2.71      1.64

FOR TONS CALCULATED CHI =     2.000
FOR GRADE CALCULATED CHI =     2.000
FOR LOG - TONS CALCULATED CHI =     2.000
FOR LOG - GRADE CALCULATED CHI =     2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =        42690748.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =                  1.
EQUIVALENT LOG - GRADE MEAN =     .5477
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   4.1568
BASIC MEAN TONNAGE =        42750000.
BASIC TONNAGE STANDARD DEVIATION =        3181981.
BASIC MEAN GRADE =     .8500
BASIC GRADE STANDARD DEVIATION =     .9192

                   ***** ANALYSIS COMPLETE *****
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           STATISTICAL INFORMATION FOR NICKEL
 DEPOSIT TYPE SEDIMENTARY

FIDENCE LEVEL % =  1.0      2.5      5.0      10.0      20.0
                   ----     ----     ----     ----      ----
ORETICAL CHI =     6.63     5.02     3.84     2.71      1.64

 TONS CALCULATED CHI =     2.000
 GRADE CALCULATED CHI =     2.000
 LOG - TONS CALCULATED CHI =     2.000
 LOG - GRADE CALCULATED CHI =     2.000
ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
IVALENT LOG - TONNAGE MEAN =          4743416.
IVALENT LOG - TONNAGE STANDARD DEVIATION =              24.
IVALENT LOG - GRADE MEAN =    2.4900
IVALENT LOG - GRADE STANDARD DEVIATION =   1.955
IC MEAN TONNAGE =         22750000.
IC TONNAGE STANDARD DEVIATION =          31466252.
IC MEAN GRADE =    2.7750
IC GRADE STANDARD DEVIATION =    1.7324

                   ***** ANALYSIS COMPLETE *****
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           STATISTICAL INFORMATION FOR NICKEL
 DEPOSIT TYPE CONTACT METAMORPHIC

CONFIDENCE LEVEL % =  1.0      2.5      5.0      10.0      20.0
                      ----     ----     ----     ----      ----
THEORETICAL CHI =     6.63     5.02     3.84     2.71      1.64

FOR TONS CALCULATED CHI =     5.000
FOR GRADE CALCULATED CHI =     2.714
FOR LOG - TONS CALCULATED CHI =     7.286
FOR LOG - GRADE CALCULATED CHI =     2.714
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =          717251.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =              2.
EQUIVALENT LOG - GRADE MEAN =    2.2271
EQUIVALENT LOG - GRADE STANDARD DEVIATION =   2.3695
BASIC MEAN TONNAGE =          785714.
BASIC TONNAGE STANDARD DEVIATION =          365963.
BASIC MEAN GRADE =    2.8314
BASIC GRADE STANDARD DEVIATION =    1.5764

                   ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE STRATIFORM

IDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
RETICAL CHI =	6.63	5.02	3.84	2.71	1.64

TONS CALCULATED CHI = 2.000

GRADE CALCULATED CHI = 2.000

LOG - TONS CALCULATED CHI = 2.000

LOG - GRADE CALCULATED CHI = 2.000

CTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

VALENT LOG - TONNAGE MEAN = 707107.

VALENT LOG - TONNAGE STANDARD DEVIATION = 2.

VALENT LOG - GRADE MEAN = 2.4495

VALENT LOG - GRADE STANDARD DEVIATION = 2.0008

C MEAN TONNAGE = 750000.

C TONNAGE STANDARD DEVIATION = 353552.

C MEAN GRADE = 2.7500

C GRADE STANDARD DEVIATION = 1.7678

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE LATERITE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.000

FOR GRADE CALCULATED CHI = 1.200

FOR LOG - TONS CALCULATED CHI = 1.200

FOR LOG - GRADE CALCULATED CHI = 1.200

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 74386456.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 2.

EQUIVALENT LOG - GRADE MEAN = 1.5815

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 1.1577

BASIC MEAN TONNAGE = 88170000.

BASIC TONNAGE STANDARD DEVIATION = 61005829.

BASIC MEAN GRADE = 1.5970

BASIC GRADE STANDARD DEVIATION = .2385

***** ANALYSIS COMPLETE *****

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            STATISTICAL INFORMATION FOR NICKEL
 DEPOSIT TYPE OXIDE

IDENCE LEVEL % =  1.0      2.5      5.0     10.0     20.0
                  ----     ----     ----    ----     ----
RETICAL CHI =    6.63     5.02     3.84     2.71     1.64

TONS CALCULATED CHI =     2.000
GRADE CALCULATED CHI =     2.000
LOG - TONS CALCULATED CHI =     2.000
LOG - GRADE CALCULATED CHI =     2.000
CTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
VALENT LOG - TONNAGE MEAN =            250500.
VALENT LOG - TONNAGE STANDARD DEVIATION =                  1.
VALENT LOG - GRADE MEAN =    1.1350
VALENT LOG - GRADE STANDARD DEVIATION =    1.0062
C MEAN TONNAGE =           250500.
C TONNAGE STANDARD DEVIATION =                  707.
C MEAN GRADE =    1.1350
C GRADE STANDARD DEVIATION =     .0071

                  ***** ANALYSIS COMPLETE *****
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            STATISTICAL INFORMATION FOR NICKEL
   DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

CONFIDENCE LEVEL % =  1.0      2.5      5.0     10.0     20.0
                      ----     ----     ----    ----     ----
THEORETICAL CHI =    6.63     5.02     3.84     2.71     1.64

FOR TONS CALCULATED CHI =     2.000
FOR GRADE CALCULATED CHI =     3.333
FOR LOG - TONS CALCULATED CHI =     6.000
FOR LOG - GRADE CALCULATED CHI =     3.333
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN =        10147083.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION =              3.
EQUIVALENT LOG - GRADE MEAN =    2.0885
EQUIVALENT LOG - GRADE STANDARD DEVIATION =    1.9499
BASIC MEAN TONNAGE =        14176667.
BASIC TONNAGE STANDARD DEVIATION =         8440329.
BASIC MEAN GRADE =    2.4333
BASIC GRADE STANDARD DEVIATION =    1.2306

                  ***** ANALYSIS COMPLETE *****
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STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE OXIDE

IDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
RETICAL CHI =	6.63	5.02	3.84	2.71	1.64

TONS CALCULATED CHI = 2.000

GRADE CALCULATED CHI = 2.000

LOG - TONS CALCULATED CHI = 2.000

LOG - GRADE CALCULATED CHI = 2.000

CTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

VALENT LOG - TONNAGE MEAN = 250500.

VALENT LOG - TONNAGE STANDARD DEVIATION = 1.

VALENT LOG - GRADE MEAN = 1.1350

VALENT LOG - GRADE STANDARD DEVIATION = 1.0062

C MEAN TONNAGE = 250500.

C TONNAGE STANDARD DEVIATION = 707.

C MEAN GRADE = 1.1350

C GRADE STANDARD DEVIATION = .0071

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR NICKEL

DEPOSIT TYPE VOLCANOGENIC MASSIVE SULPHIDE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.000

FOR GRADE CALCULATED CHI = 3.333

FOR LOG - TONS CALCULATED CHI = 6.000

FOR LOG - GRADE CALCULATED CHI = 3.333

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 10147083.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 3.

EQUIVALENT LOG - GRADE MEAN = 2.0385

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 1.9499

BASIC MEAN TONNAGE = 14176667.

BASIC TONNAGE STANDARD DEVIATION = 8440329.

BASIC MEAN GRADE = 2.4333

BASIC GRADE STANDARD DEVIATION = 1.2306

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR TIN

DEPOSIT TYPE HYDROTHERMAL

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
5000000.	1.2500	1.4909	.1553	6.6990	.0969	1.3366	.4644
660000.	.4200	-.4894	-.7335	5.8195	-.3768	-.3723	-.7697
930000.	.3800	-.3662	-.7764	5.9685	-.4202	-.0727	-.6830
340000.	2.3700	-.6354	1.3546	5.5315	.3747	-.9517	1.1863

STATISTICAL INFORMATION FOR MOLYBDENUM

DEPOSIT TYPE PORPHYRY

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
180000000.	.1350	-.6368	.8219	8.2553	-.8697	-.8237	.9123
293000000.	.0140	-.4594	-.9036	8.4669	-1.8539	-.2996	-1.0535
303000000.	.2000	-.4437	1.7488	8.4814	-.6990	-.2635	1.2533
239000000.	.0899	-.5441	.1788	8.3784	-1.0462	-.5187	.5596
120000000.	.0100	-.7310	-.9606	8.0792	-2.0000	-1.2599	-1.3454
1000000000.	.0200	.6505	-.8130	9.0000	-1.6990	1.0210	-.7441
2000000000.	.0300	2.2204	-.6754	9.3010	-1.5229	1.7666	-.3924
550000000.	.1200	-.0559	.6080	8.7404	-.9208	.3778	.8101

DEPOSIT TYPE PORPHYRY

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 6.000
FOR GRADE CALCULATED CHI = 4.000
FOR LOG - TONS CALCULATED CHI = 1.000
FOR LOG - GRADE CALCULATED CHI = 2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 387096570.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 3.
EQUIVALENT LOG - GRADE MEAN = .0472
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 3.1671
BASIC MEAN TONNAGE = 585625000.
BASIC TONNAGE STANDARD DEVIATION = 637004808.
BASIC MEAN GRADE = .0774
BASIC GRADE STANDARD DEVIATION = .0701

***** ANALYSIS COMPLETE *****

DEPOSIT TYPE HYDROTHERMAL

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 6.000
FOR GRADE CALCULATED CHI = 2.000
FOR LOG - TONS CALCULATED CHI = 2.000
FOR LOG - GRADE CALCULATED CHI = 2.000
IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL
EQUIVALENT LOG - TONNAGE MEAN = 1010692.
EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 3.
EQUIVALENT LOG - GRADE MEAN = .8292
EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.4199
BASIC MEAN TONNAGE = 1732500.
BASIC TONNAGE STANDARD DEVIATION = 2191641.
BASIC MEAN GRADE = 1.1050
BASIC GRADE STANDARD DEVIATION = .9338

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR COBALT

DEPOSIT TYPE LATERITE

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
45200000.	.1200	1.0000	1.0000	7.6551	-.9208	.9996	.9845
45100000.	.1000	0.0000	-1.0000	7.6542	-1.0000	.0007	-1.0148
45000000.	.1100	-1.0000	0.0000	7.6532	-.9586	-1.0004	.0303

DEPOSIT TYPE CONTACT METAMORPHIC

TONS	GRADE	S.N. TONS	S.N. GRADE	LOG TONS	LOG GRADE	SNL TONS	SNL GRADE
500000.	1.4000	-.7071	.7071	5.6990	.1461	-.7071	.7071
10000000.	.4000	.7071	-.7071	7.0000	-.3979	.7071	-.7071

STATISTICAL INFORMATION FOR COBALT

DEPOSIT TYPE LATERITE

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 1.000

FOR GRADE CALCULATED CHI = 1.000

FOR LOG - TONS CALCULATED CHI = 1.000

FOR LOG - GRADE CALCULATED CHI = 1.000

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 45099426.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 1.

EQUIVALENT LOG - GRADE MEAN = .1097

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 1.0355

BASIC MEAN TONNAGE = 451[illegible].

BASIC TONNAGE STANDARD DEVIATION = 10000[illegible].

BASIC MEAN GRADE = .1100

BASIC GRADE STANDARD DEVIATION = .0100

***** ANALYSIS COMPLETE *****

STATISTICAL INFORMATION FOR TUNGSTEN

DEPOSIT TYPE CONTACT METAMORPHIC

CONFIDENCE LEVEL % =	1.0	2.5	5.0	10.0	20.0
THEORETICAL CHI =	6.63	5.02	3.84	2.71	1.64

FOR TONS CALCULATED CHI = 2.000

FOR GRADE CALCULATED CHI = 2.000

FOR LOG - TONS CALCULATED CHI = 2.000

FOR LOG - GRADE CALCULATED CHI = 2.000

IF ACTUAL CHI IS LESS THAN THEORETICAL CHI THEN DISTRIBUTION IS NORMAL

EQUIVALENT LOG - TONNAGE MEAN = 2036068.

EQUIVALENT LOG - TONNAGE STANDARD DEVIATION = 3.

EQUIVALENT LOG - GRADE MEAN = .7483

EQUIVALENT LOG - GRADE STANDARD DEVIATION = 2.4250

BASIC MEAN TONNAGE = 5250000.

BASIC TONNAGE STANDARD DEVIATION = 6717514.

BASIC MEAN GRADE = .9000

BASIC GRADE STANDARD DEVIATION = .7071

***** ANALYSIS COMPLETE *****

APPENDIX E

Details of the Exponential Models of the Commodity Source Profiles of Copper, Lead, Zinc, Gold, Silver and Nickel.

COMMODITY SOURCE PROFILE DATA

COPPER

Type	Mean Grade	Mean Tons	SD Grade	SD Tons
PORPH	0.81	330.00	0.42	658.00
SED	2.11	9.00	0.81	14.00
CM	2.00	6.80	1.18	11.50
STRAT	2.37	59.20	1.08	69.20
VMS	1.92	21.50	0.84	19.00
COM	1.75	20.80	1.27	22.70
HYD	2.70	0.42	1.63	0.34

Note: Certain abbreviations have been used in the above table and are used elsewhere in this thesis, the definition of the terms is as follows:

PORPH = porphyry
SED = sedimentary
STRAT = stratiform
CM = contact metamorphic
VMS = volcanogenic massive sulphide
COM = complex
HYD = hydrothermal
LAT = laterite
OX = oxide

GOLD

Type	Mean Grade	Mean Tons	SD Grade	SD Tons
PORPH	0.028	527.60	0.030	455.40
SED	0.400	32.50	0.210	26.00
CM	1.290	1.00	1.720	1.25
VMS	0.088	22.90	0.064	16.00
COM	0.103	21.70	0.059	25.00
HYD	0.784	16.60	1.300	41.00

SILVER

Type	Mean Grade	Mean Tons	SD Grade	SD Tons
PORPH	1.18	606.00	1.06	519.00
CM	5.45	8.70	6.66	13.70
STRAT	0.89	12.10	0.02	2.70
VMS	2.71	76.20	2.51	50.40
COM	2.37	22.10	2.02	28.50
HYD	5.73	16.80	3.96	26.50

LEAD

Type	Mean Grade	Mean Tons	SD Grade	SD Tons
PORPH	2.55	420.30	3.47	537.10
CM	7.23	11.00	4.12	16.20
STRAT	5.97	22.80	5.49	18.60
VMS	6.15	61.80	5.42	70.00
COM	5.26	19.70	3.61	27.90
HYD	4.60	8.40	2.61	17.70

ZINC

Type	Mean Grade	Mean Tons	SD Grade	SD Tons
CM	6.00	17.70	4.02	18.90
STRAT	10.68	19.50	9.30	14.10
VMS	6.54	64.30	2.83	56.00
COM	11.65	19.70	11.68	27.90
HYD	6.78	10.10	1.66	19.90

NICKEL

Type	Mean Grade	Mean Tons	SD Grade	SD Tons
PORPH	0.85	42.80	0.92	3.20
SED	2.78	22.78	1.73	31.50
CM	2.83	0.79	1.58	0.37
STRAT	2.75	0.75	1.77	0.35
LAT	1.60	88.20	0.24	61.00
OX	1.14	0.25	0.01	0.01
VMS	2.43	14.20	1.23	8.40

Exponential Model Parameters

The general form of the curve fitted was:

$$y = e^{mx} . b$$

"m" parameter values

Commodity	Standard Deviations				
	-2.0	-1.0	Mean	+1.0	+2.0
Copper	-	0.1215	-0.0030	-0.0010	-0.00061
Lead	-	-	-0.0020	-0.0005	-0.00026
Zinc	-	-	-0.0037	-0.0034	-0.00276
Gold	-	-5.1724	-0.0050	-0.0026	-0.00172
Silver	-	-0.0208	-0.0015	-0.0007	-0.00039
Nickel	-	0.0074	-0.0056	-0.0028	-0.00201

"b" parameter values

Stand.	Commodity					
Dev.	Copper	Lead	Zinc	Gold	Silver	Nickel
-2.0	-	-	-	-	-	-
-1.0	0.80414	-	-	-	0.64339	1.11952
Mean	2.24080	6.048	8.837	0.362	2.93646	2.14714
+1.0	3.37861	10.17	15.56	0.699	4.88977	3.08233
+2.0	4.52504	14.28	22.22	1.021	6.64194	3.98610

Correlation Coefficients for Models Fits

Stand.	Commodity					
Dev.	Copper	Lead	Zinc	Gold	Silver	Nickel
-2.0	-	-	-	-	-	-
-1.0	0.24	-	-	-1.00	-0.83	0.70
Mean	-0.92	-0.89	-0.26	-0.70	-0.47	-0.37
+1.0	-0.93	-0.70	-0.27	-0.63	-0.30	-0.26
+2.0	-0.92	-0.56	-0.27	-0.60	-0.23	-0.23

APPENDIX F

Predicted Values for Inflation and Commodity Prices

BASIC DEFLATED COMMODITY PRICE DATA

YEAR	INFLATION	COPPER $/LB.	LEAD $/LB.	ZINC $/LB.	GOLD $/OZ.	SILVER $/OZ.
1945.	1.000	.1200	.0700	.0300	34.71	.52
1946.	1.006	.1391	.0795	.1093	34.50	.80
1947.	1.351	.1554	.1110	.0814	25.69	.53
1948.	1.350	.1703	.1333	.1333	25.70	.56
1949.	1.136	.1760	.1320	.1584	27.90	.63
1950.	1.348	.1632	.0964	.1335	25.76	.55
1951.	1.576	.1586	.1142	.1269	22.03	.56
1952.	1.390	.1799	.1223	.1439	24.89	.61
1953.	1.328	.2183	.1154	.0979	26.23	.64
1954.	1.361	.2204	.1028	.0882	25.74	.62
1955.	1.305	.2912	.1150	.1073	26.85	.68
1956.	1.350	.3111	.1185	.1037	25.92	.67
1957.	1.327	.2260	.1130	.1055	26.33	.69
1958.	1.276	.2037	.0940	.0940	27.51	.70
1959.	1.241	.2499	.0967	.1128	28.28	.73
1960.	1.193	.2682	.1006	.1173	29.56	.76
1961.	1.191	.2519	.0924	.1092	29.53	.78
1962.	1.187	.2611	.0842	.1095	29.58	.91
1963.	1.197	.2589	.0919	.1086	29.30	1.07
1964.	1.193	.2683	.1174	.1257	29.42	1.[illegible]8
1965.	1.237	.2910	.1293	.1213	28.40	1.[illegible]4
1966.	1.270	.2913	.1181	.1181	27.68	1.[illegible]2
1967.	1.184	.3210	.1183	.1267	29.73	1.31
1968.	1.171	.3757	.1110	.1195	32.96	1.84
1969.	1.274	.4003	.1177	.1256	32.26	1.68
1970.	1.323	.4461	.1210	.1210	27.14	1.34
1971.	1.207	.4309	.1160	.1492	33.61	1.28
1972.	1.352	.3772	.111[illegible]	.1405	42.98	1.25
1973.	2.172	.2717	.0737	.1381	44.76	1.18
1974.	2.737	.2521	.0840	.1425	58.12	1.72
1975.	2.469	.2794	.0891	.162[illegible]	65.24	1.79
1976.	2.693	.2525	.0854	.1485	46.34	1.62
1977.	2.846	.2319	.1089	.1300	51.92	1.62
1978.	2.999	.2067	.1134	.1167	64.44	1.60
1979.	3.296	.2821	.1608	.1123	92.53	3.36
1980.	3.572	.2408	.1792	.1176	171.44	4.24
1981.	4.001	.2025	.0825	.1053	114.94	2.[illegible]1
1982.	4.401	.1659	.0591	.0932	82.84	2.17
AVERAGE	1.750	.2529	.1052	.1196	42.18	1.22
ST.DEV.	.903	.0788	.0200	.0195	29.64	.79

PREDICTED VALUES FOR INFLATION

RANGE OF PREDICTION IN YEARS = 14

YEAR	DEFLATED VALUE	1983 BASED VALUE
1983	4.2586	1.0000
1984	4.1010	.9630
1985	3.9294	.9227
1986	3.7461	.8797
1987	3.5850	.8418
1988	3.4641	.8134
1989	3.3468	.7859
1990	3.2209	.7563
1991	3.1024	.7285
1992	2.9901	.7021
1993	2.8906	.6788
1994	2.8106	.6600
1995	2.7575	.6475
1996	2.7393	.6432

AVERAGE VALUE 1983 $ = .7874
STANDARD DEVIATION = .1198

PREDICTED VALUES FOR COPPER
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RANGE OF PREDICTION IN YEARS = 10

YEAR	DEFLATED VALUE	1983 BASED VALUE
1983	.1718	.7316
1984	.1778	.7572
1985	.1848	.7869
1986	.1925	.8199
1987	.1985	.8455
1988	.2026	.8630
1989	.2095	.8924
1990	.2188	.9318
1991	.2245	.9562
1992	.2265	.9647

AVERAGE VALUE 1983 $ = .8549
STANDARD DEVIATION = .0819

AVERAGE ERROR OF ESTIMATION = -.0013
STANDARD DEVIATION OF ERROR = .0005
PRECISION OF THE ESTIMATE +/- % = -.1555

AVERAGE DISCOUNTED PRICE @ 0 % DISCOUNT RATE = .8549

AVERAGE DISCOUNTED PRICE @ 5 % DISCOUNT RATE = .3105

AVERAGE DISCOUNTED PRICE @ 10 % DISCOUNT RATE = .1670

AVERAGE DISCOUNTED PRICE @ 15 % DISCOUNT RATE = .1108

AVERAGE DISCOUNTED PRICE @ 20 % DISCOUNT RATE = .0822

AVERAGE DISCOUNTED PRICE @ 25 % DISCOUNT RATE = .0651

PREDICTED VALUES FOR LEAD

RANGE OF PREDICTION IN YEARS = 5

YEAR	DEFLATED VALUE	1983 BASED VALUE
1983	.0621	.2647
1984	.0721	.3071
1985	.0843	.3591
1986	.0907	.3863
1987	.0930	.3959

AVERAGE VALUE 1983 $ = .3426
STANDARD DEVIATION = .0556

AVERAGE ERROR OF ESTIMATION = -.00[illegible]
STANDARD DEVIATION OF ERROR = .0000
PRECISION OF THE ESTIMATE +/- % = -.0034

AVERAGE DISCOUNTED PRICE @ 0 % DISCOUNT RATE = .3426

AVERAGE DISCOUNTED PRICE @ 5 % DISCOUNT RATE = .1248

AVERAGE DISCOUNTED PRICE @ 10 % DISCOUNT RATE = .0674

AVERAGE DISCOUNTED PRICE @ 15 % DISCOUNT RATE = .0450

AVERAGE DISCOUNTED PRICE @ 20 % DISCOUNT RATE = .0335

AVERAGE DISCOUNTED PRICE @ 25 % DISCOUNT RATE = .0266

PREDICTED VALUES FOR ZINC

RANGE OF PREDICTION IN YEARS = 6

YEAR	DEFLATED VALUE	1983 BASED VALUE
1983	.0973	.4144
1984	.1012	.4311
1985	.1042	.4438
1986	.1076	.4584
1987	.1108	.4719
1988	.1119	.4767

AVERAGE VALUE 1983 $ = .4494
STANDARD DEVIATION = .0242

AVERAGE ERROR OF ESTIMATION = -.0000
STANDARD DEVIATION OF ERROR = .0000
PRECISION OF THE ESTIMATE +/- % = -.0033

AVERAGE DISCOUNTED PRICE @ 0 % DISCOUNT RATE = .4494

AVERAGE DISCOUNTED PRICE @ 5 % DISCOUNT RATE = .1639

AVERAGE DISCOUNTED PRICE @ 10 % DISCOUNT RATE = .0888

AVERAGE DISCOUNTED PRICE @ 15 % DISCOUNT RATE = .0595

AVERAGE DISCOUNTED PRICE @ 20 % DISCOUNT RATE = .0445

AVERAGE DISCOUNTED PRICE @ 25 % DISCOUNT RATE = .0355

PREDICTED VALUES FOR GOLD

RANGE OF PREDICTION IN YEARS = 13

YEAR	DEFLATED VALUE	1983 BASED VALUE
1983	79.2058	337.3065
1984	75.4760	321.4227
1985	72.0829	306.9728
1986	68.8688	293.2853
1987	66.0153	281.13[illegible]
1988	63.7870	271.6439
1989	60.9709	259.6511
1990	57.4481	244.6487
1991	54.4420	231.8469
1992	52.9081	225.315[illegible]
1993	55.6318	236.9139
1994	59.7149	254.3025
1995	60.7992	258.92[illegible]

AVERAGE VALUE 1983 $ = 271.0279
STANDARD DEVIATION = 35.1931

AVERAGE ERROR OF ESTIMATION = -113.[illegible]422
STANDARD DEVIATION OF ERROR = 46.8695
PRECISION OF THE ESTIMATE +/- % = -41.7087

AVERAGE DISCOUNTED PRICE @ 0 % DISCOUNT RATE = 271.0279

AVERAGE DISCOUNTED PRICE @ 5 % DISCOUNT RATE = 99.9899

AVERAGE DISCOUNTED PRICE @ 10 % DISCOUNT RATE = 55.2949

AVERAGE DISCOUNTED PRICE @ 15 % DISCOUNT RATE = 37.8962

AVERAGE DISCOUNTED PRICE @ 20 % DISCOUNT RATE = 28.9827

AVERAGE DISCOUNTED PRICE @ 25 % DISCOUNT RATE = 23.5728

PREDICTED VALUES FOR SILVER

RANGE OF PREDICTION IN YEARS = 22

YEAR	DEFLATED VALUE	1983 BASED VALUE
1983	2.1200	9.0282
1984	2.0698	8.8144
1985	2.0240	8.6195
1986	1.9801	8.4325
1987	1.9336	8.2344
1988	1.8836	8.0214
1989	1.8376	7.8257
1990	1.8087	7.7027
1991	1.7875	7.6123
1992	1.7541	7.47[illegible]1
1993	1.7103	7.2835
1994	1.6629	7.0816
1995	1.6116	6.8631
1996	1.5694	6.6835
1997	1.5403	6.5597
1998	1.5085	6.4243
1999	1.4723	6.2699
2000	1.4396	6.1309
2001	1.4470	6.1621
2002	1.5130	6.4433
2003	1.5530	6.6138
2004	1.5484	6.5941

AVERAGE VALUE 1983 $ = 7.3123
STANDARD DEVIATION = .9160

AVERAGE ERROR OF ESTIMATION = -.1004
STANDARD DEVIATION OF ERROR = .0433
PRECISION OF THE ESTIMATE +/- % = -1.3737

AVERAGE DISCOUNTED PRICE @ 0 % DISCOUNT RATE = 7.3123

AVERAGE DISCOUNTED PRICE @ 5 % DISCOUNT RATE = 2.7403

AVERAGE DISCOUNTED PRICE @ 10 % DISCOUNT RATE = 1.5397

AVERAGE DISCOUNTED PRICE @ 15 % DISCOUNT RATE = 1.0649

AVERAGE DISCOUNTED PRICE @ 20 % DISCOUNT RATE = .8171

AVERAGE DISCOUNTED PRICE @ 25 % DISCOUNT RATE = .6645

APPENDIX G

Details of the Capital and Operating Costs, Operating Parameters and Financial Factors used for each Mining Method as Input to the Minimum Reserve Analysis

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: CUPIT

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.59
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	55275.09
MILL CAPITAL FACTOR $/TPY	8.53
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.94
PROCESSING & OTHER COSTS $/TON	9.04

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	14901232.
AVERAGE OPERATING STRIP RATIO W:O	.05
PROCESSING RECOVERY	.95
TAILINGS GRADE	.04

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.05
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: CUZERO

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.60
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	0.00
MILL CAPITAL FACTOR $/TPY	8.53
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.85
PROCESSING & OTHER COSTS $/TON	9.04

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	0.
AVERAGE OPERATING STRIP RATIO W:O	0.00
PROCESSING RECOVERY	.95
TAILINGS GRADE	.04

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	0.00
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: CUSELF

OPEN STOPING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	12.17
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	12989.71
MILL CAPITAL FACTOR $/TPY	13.70
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	24.01
PROCESSING & OTHER COSTS $/TON	10.55

OPERATING PARAMETERS

MINING DILUTION	15.00
MINING RECOVERY	.60
PROCESSING RECOVERY	.87
TAILINGS GRADE	.31

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.30
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: CUAS

ARTIFICIALLY SUPPORTED MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	39.69
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	12989.71
MILL CAPITAL FACTOR $/TPY	23.31
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	30.66
PROCESSING & OTHER COSTS $/TON	12.92

OPERATING PARAMETERS

MINING DILUTION	5.00
MINING RECOVERY	.95
PROCESSING RECOVERY	.88
TAILINGS GRADE	.26

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.20
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

? 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: CUAC

ARTIFICIAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	13.91
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13006.97
MILL CAPITAL FACTOR $/TPY	18.10
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	9.28
PROCESSING & OTHER COSTS $/TON	11.68

OPERATING PARAMETERS

MINING DILUTION	20.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.88
TAILINGS GRADE	.23

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.25
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: CUNC

NATURAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	9.53
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13145.03
MILL CAPITAL FACTOR $/TPY	8.54
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	7.45
PROCESSING & OTHER COSTS $/TON	9.04

OPERATING PARAMETERS

MINING DILUTION	25.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.95
TAILINGS GRADE	.04

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.15
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: PBNC

NATURAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	9.44
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13145.03
MILL CAPITAL FACTOR $/TPY	8.06
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	7.45
PROCESSING & OTHER COSTS $/TON	8.89

OPERATING PARAMETERS

MINING DILUTION	25.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.81
TAILINGS GRADE	.49

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.15
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: PBPIT

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.59
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	52713.89
MILL CAPITAL FACTOR $/TPY	8.05
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.84
PROCESSING & OTHER COSTS $/TON	8.89

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	14901232.
AVERAGE OPERATING STRIP RATIO W:O	.04
PROCESSING RECOVERY	.81
TAILINGS GRADE	.49

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.04
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

```
                              INPUT VALUES NEEDED FOR
                              MINIMUM RESERVE ANALYSIS
                              SCENARIO: PBAS
                              ------------------------

        ARTIFICIALLY SUPPORTED MINING

                              CAPITAL COST FACTORS
                              --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                          44.23

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.                   12989.71

MILL CAPITAL FACTOR $/TPY                                    24.46

EXPLORATION & FEASIBILITY EXPENDITURE $                   5000000.

OTHER CAPITAL FACTOR                                           .25

                              OPERATING COSTS
                              ---------------

MODIFIED MINING COST $/TON                                   30.66

PROCESSING & OTHER COSTS $/TON                               13.19

                              OPERATING PARAMETERS
                              --------------------

MINING DILUTION                                               5.00

MINING RECOVERY                                                .95

PROCESSING RECOVERY                                            .77

TAILINGS GRADE                                                1.06

                              FINANCIAL FACTORS
                              -----------------

MINE DEVELOPMENT FACTOR                                        .20

MILL DEPRECIATION FACTOR                                       .75

INVESTMENT TAX CREDIT FACTOR                                   .10

EFFECTIVE TAX RATE                                             .50


    ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
```

```
                              INPUT VALUES NEEDED FOR
                              MINIMUM RESERVE ANALYSIS
                              SCENARIO: PBAC
                              ------------------------

        ARTIFICIAL CAVING

                              CAPITAL COST FACTORS
                              --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                          11.73

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.                   13006.97

MILL CAPITAL FACTOR $/TPY                                    13.45

EXPLORATION & FEASIBILITY EXPENDITURE $                   5000000.

OTHER CAPITAL FACTOR                                           .25

                              OPERATING COSTS
                              ---------------

MODIFIED MINING COST $/TON                                    9.28

PROCESSING & OTHER COSTS $/TON                               10.48

                              OPERATING PARAMETERS
                              --------------------

MINING DILUTION                                              20.00

MINING RECOVERY                                                .85

PROCESSING RECOVERY                                            .75

TAILINGS GRADE                                                1.52

                              FINANCIAL FACTORS
                              -----------------

MINE DEVELOPMENT FACTOR                                        .25

MILL DEPRECIATION FACTOR                                       .75

INVESTMENT TAX CREDIT FACTOR                                   .10

EFFECTIVE TAX RATE                                             .50


    ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
```

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: PBZERO

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.59
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	0.00
MILL CAPITAL FACTOR $/TPY	8.05
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.77
PROCESSING & OTHER COSTS $/TON	8.89

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	0.
AVERAGE OPERATING STRIP RATIO W:O	0.00
PROCESSING RECOVERY	.81
TAILINGS GRADE	.49

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	0.00
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: PBSELF

OPEN STOPING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	14.01
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	12989.71
MILL CAPITAL FACTOR $/TPY	17.78
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	24.01
PROCESSING & OTHER COSTS $/TON	11.60

OPERATING PARAMETERS

MINING DILUTION	15.00
MINING RECOVERY	.60
PROCESSING RECOVERY	.75
TAILINGS GRADE	1.47

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.30
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: ZNPIT

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.57
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	45995.50
MILL CAPITAL FACTOR $/TPY	6.84
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.57
PROCESSING & OTHER COSTS $/TON	8.47

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	14901232.
AVERAGE OPERATING STRIP RATIO W:O	.02
PROCESSING RECOVERY	.76
TAILINGS GRADE	.17

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.02
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: ZNNC

NATURAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	9.24
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13145.03
MILL CAPITAL FACTOR $/TPY	6.84
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	7.45
PROCESSING & OTHER COSTS $/TON	8.47

OPERATING PARAMETERS

MINING DILUTION	25.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.76
TAILINGS GRADE	.17

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.15
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: ZNAC

ARTIFICIAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	11.67
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13006.97
MILL CAPITAL FACTOR $/TPY	13.29
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	9.28
PROCESSING & OTHER COSTS $/TON	10.43

OPERATING PARAMETERS

MINING DILUTION	20.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.67
TAILINGS GRADE	2.16

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.25
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: ZNAS

ARTIFICIALLY SUPPORTED MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	39.32
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	12989.71
MILL CAPITAL FACTOR $/TPY	23.22
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	30.66
PROCESSING & OTHER COSTS $/TON	12.90

OPERATING PARAMETERS

MINING DILUTION	5.00
MINING RECOVERY	.95
PROCESSING RECOVERY	.67
TAILINGS GRADE	2.25

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.20
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS

```
          INPUT VALUES NEEDED FOR
          MINIMUM RESERVE ANALYSIS
          SCENARIO: ZNZERO
          ------------------------

OPEN PIT MINING

          CAPITAL COST FACTORS
          --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                  1.57

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.              0.00

MILL CAPITAL FACTOR $/TPY                            6.84

EXPLORATION & FEASIBILITY EXPENDITURE $          5000000.

OTHER CAPITAL FACTOR                                  .25

          OPERATING COSTS
          ---------------

MODIFIED MINING COST $/TON                           1.54

PROCESSING & OTHER COSTS $/TON                       8.47

          OPERATING PARAMETERS
          --------------------

MINING DILUTION                                       .15

MINING RECOVERY                                      1.00

PRE-STRIP REQUIREMENT TONS                             0.

AVERAGE OPERATING STRIP RATIO W:O                    0.00

PROCESSING RECOVERY                                   .76

TAILINGS GRADE                                        .17

          FINANCIAL FACTORS
          -----------------

MINE DEVELOPMENT FACTOR                              0.00

MILL DEPRECIATION FACTOR                              .75

INVESTMENT TAX CREDIT FACTOR                          .10

EFFECTIVE TAX RATE                                    .50


   ***** ANALYSIS COMPLETE *****
```

```
          INPUT VALUES NEEDED FOR
          MINIMUM RESERVE ANALYSIS
          SCENARIO: ZNSELF
          ------------------------

OPEN STOPING

          CAPITAL COST FACTORS
          --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                 14.57

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.          12989.71

MILL CAPITAL FACTOR $/TPY                           18.79

EXPLORATION & FEASIBILITY EXPENDITURE $          5000000.

OTHER CAPITAL FACTOR                                  .25

          OPERATING COSTS
          ---------------

MODIFIED MINING COST $/TON                          24.01

PROCESSING & OTHER COSTS $/TON                      11.85

          OPERATING PARAMETERS
          --------------------

MINING DILUTION                                     15.00

MINING RECOVERY                                       .60

PROCESSING RECOVERY                                   .65

TAILINGS GRADE                                       3.74

          FINANCIAL FACTORS
          -----------------

MINE DEVELOPMENT FACTOR                               .30

MILL DEPRECIATION FACTOR                              .75

INVESTMENT TAX CREDIT FACTOR                          .10

EFFECTIVE TAX RATE                                    .50


   ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
```

```
INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AUSELF
------------------------

OPEN STOPING

CAPITAL COST FACTORS
--------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                          13.28
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.                   12989.71
MILL CAPITAL FACTOR $/TPY                                    16.33
EXPLORATION & FEASIBILITY EXPENDITURE $                   5000000.
OTHER CAPITAL FACTOR                                           .25

OPERATING COSTS
---------------

MODIFIED MINING COST $/TON                                   24.01
PROCESSING & OTHER COSTS $/TON                               11.23

OPERATING PARAMETERS
--------------------

MINING DILUTION                                              15.00
MINING RECOVERY                                                .60
PROCESSING RECOVERY                                           1.00
TAILINGS GRADE                                                0.00

FINANCIAL FACTORS
-----------------

MINE DEVELOPMENT FACTOR                                        .30
MILL DEPRECIATION FACTOR                                       .75
INVESTMENT TAX CREDIT FACTOR                                   .10
EFFECTIVE TAX RATE                                             .50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
 3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3
```

```
INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AUAS
------------------------

ARTIFICIALLY SUPPORTED MINING

CAPITAL COST FACTORS
--------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                         199.59
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.                   12989.71
MILL CAPITAL FACTOR $/TPY                                    47.63
EXPLORATION & FEASIBILITY EXPENDITURE $                   5000000.
OTHER CAPITAL FACTOR                                           .25

OPERATING COSTS
---------------

MODIFIED MINING COST $/TON                                   30.66
PROCESSING & OTHER COSTS $/TON                               18.07

OPERATING PARAMETERS
--------------------

MINING DILUTION                                               5.00
MINING RECOVERY                                                .95
PROCESSING RECOVERY                                            .97
TAILINGS GRADE                                                 .04

FINANCIAL FACTORS
-----------------

MINE DEVELOPMENT FACTOR                                        .20
MILL DEPRECIATION FACTOR                                       .75
INVESTMENT TAX CREDIT FACTOR                                   .10
EFFECTIVE TAX RATE                                             .50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
 3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3
```

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AUNC

NATURAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	9.36
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13145.03
MILL CAPITAL FACTOR $/TPY	7.62
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	7.45
PROCESSING & OTHER COSTS $/TON	8.74

OPERATING PARAMETERS

MINING DILUTION	25.00
MINING RECOVERY	.85
PROCESSING RECOVERY	1.00
TAILINGS GRADE	0.00

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.15
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AUAC

ARTIFICIAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	14.71
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13006.97
MILL CAPITAL FACTOR $/TPY	19.42
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATIND COSTS

MODIFIED MINING COST $/TON	9.28
PROCESSING & OTHER COSTS $/TON	12.00

OPERATING PARAMETERS

MINING DILUTION	20.00
MINING RECOVERY	.85
PROCESSING RECOVERY	1.00
TAILINGS GRADE	0.00

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.25
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AUZERO

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.58
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	0.00
MILL CAPITAL FACTOR $/TPY	7.62
EXPLORATION &.FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.69
PROCESSING & OTHER COSTS $/TON	8.74

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	0.
AVERAGE OPERATING STRIP RATIO W:O	0.00
PROCESSING RECOVERY	1.00
TAILINGS GRADE	0.00

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	0.00
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AUPIT

OPEN PIT MINING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	1.58
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	56179.51
MILL CAPITAL FACTOR $/TPY	7.62
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	1.78
PROCESSING & OTHER COSTS $/TON	8.74

OPERATING PARAMETERS

MINING DILUTION	.15
MINING RECOVERY	1.00
PRE-STRIP REQUIREMENT TONS	16618458.
AVERAGE OPERATING STRIP RATIO W:O	.05
PROCESSING RECOVERY	1.00
TAILINGS GRADE	0.00

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.05
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS

```
                    INPUT VALUES NEEDED FOR
                    MINIMUM RESERVE ANALYSIS
                    SCENARIO: AGAS
                    ------------------------

       ARTIFICIALLY SUPPORTED MINING

                    CAPITAL COST FACTORS
                    --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                    39.69

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.             12989.71

MILL CAPITAL FACTOR $/TPY                              23.31

EXPLORATION & FEASIBILITY EXPENDITURE $             5000000.

OTHER CAPITAL FACTOR                                     .25

                    OPERATING COSTS
                    ---------------

MODIFIED MINING COST $/TON                             30.66

PROCESSING & OTHER COSTS $/TON                         12.92

                    OPERATING PARAMETERS
                    --------------------

MINING DILUTION                                         5.00

MINING RECOVERY                                          .95

PROCESSING RECOVERY                                      .70

TAILINGS GRADE                                          1.61

                    FINANCIAL FACTORS
                    -----------------

MINE DEVELOPMENT FACTOR                                  .20

MILL DEPRECIATION FACTOR                                 .75

INVESTMENT TAX CREDIT FACTOR                             .10

EFFECTIVE TAX RATE                                       .50


    ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
 3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3
```

```
                    INPUT VALUES NEEDED FOR
                    MINIMUM RESERVE ANALYSIS
                    SCENARIO: AGSELF
                    ------------------------

       OPEN STOPING

                    CAPITAL COST FACTORS
                    --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                    16.34

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.             12989.71

MILL CAPITAL FACTOR $/TPY                              21.56

EXPLORATION & FEASIBILITY EXPENDITURE $             5000000.

OTHER CAPITAL FACTOR                                     .25

                    OPERATING COSTS
                    ---------------

MODIFIED MINING COST $/TON                             24.01

PROCESSING & OTHER COSTS $/TON                         12.51

                    OPERATING PARAMETERS
                    --------------------

MINING DILUTION                                        15.00

MINING RECOVERY                                          .60

PROCESSING RECOVERY                                      .79

TAILINGS GRADE                                           .19

                    FINANCIAL FACTORS
                    -----------------

MINE DEVELOPMENT FACTOR                                  .30

MILL DEPRECIATION FACTOR                                 .75

INVESTMENT TAX CREDIT FACTOR                             .10

EFFECTIVE TAX RATE                                       .50


    ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
 3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3
```

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AGNC

NATURAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	9.32
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13145.03
MILL CAPITAL FACTOR $/TPY	7.36
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	7.45
PROCESSING & OTHER COSTS $/TON	8.65

OPERATING PARAMETERS

MINING DILUTION	25.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.77
TAILINGS GRADE	.27

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.15
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AGAC

ARTIFICIAL CAVING

CAPITAL COST FACTORS

MINE EQUIPMENT CAPITAL FACTOR $/TPY	11.47
MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.	13006.97
MILL CAPITAL FACTOR $/TPY	12.73
EXPLORATION & FEASIBILITY EXPENDITURE $	5000000.
OTHER CAPITAL FACTOR	.25

OPERATING COSTS

MODIFIED MINING COST $/TON	9.28
PROCESSING & OTHER COSTS $/TON	10.28

OPERATING PARAMETERS

MINING DILUTION	20.00
MINING RECOVERY	.85
PROCESSING RECOVERY	.73
TAILINGS GRADE	.72

FINANCIAL FACTORS

MINE DEVELOPMENT FACTOR	.25
MILL DEPRECIATION FACTOR	.75
INVESTMENT TAX CREDIT FACTOR	.10
EFFECTIVE TAX RATE	.50

***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

1. SELECT A MINING METHOD
2. CALCULATE CAPITAL & OPERATING COSTS
3. EXIT FROM THE SYSTEM

ENTER A NUMBER FROM 1 TO 3

```
INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AGZERO
------------------------

      OPEN PIT MINING

                  CAPITAL COST FACTORS
                  --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                 1.58

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.             0.00

MILL CAPITAL FACTOR $/TPY                           7.35

EXPLORATION & FEASIBILITY EXPENDITURE $         5000000.

OTHER CAPITAL FACTOR                                 .25

                  OPERATING COSTS
                  ---------------

MODIFIED MINING COST $/TON                          1.64

PROCESSING & OTHER COSTS $/TON                      8.65

                  OPERATING PARAMETERS
                  --------------------

MINING DILUTION                                      .15

MINING RECOVERY                                     1.00

PRE-STRIP REQUIREMENT TONS                            0.

AVERAGE OPERATING STRIP RATIO W:O                   0.00

PROCESSING RECOVERY                                  .77

TAILINGS GRADE                                       .27

                  FINANCIAL FACTORS
                  -----------------

MINE DEVELOPMENT FACTOR                             0.00

MILL DEPRECIATION FACTOR                             .75

INVESTMENT TAX CREDIT FACTOR                         .10

EFFECTIVE TAX RATE                                   .50


    ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
```

```
INPUT VALUES NEEDED FOR
MINIMUM RESERVE ANALYSIS
SCENARIO: AGPIT
------------------------

      OPEN PIT MINING

                  CAPITAL COST FACTORS
                  --------------------

MINE EQUIPMENT CAPITAL FACTOR $/TPY                 1.58

MINE CAPITAL DEPTH FACTOR $/FT. OR $/M.         48894.44

MILL CAPITAL FACTOR $/TPY                           7.35

EXPLORATION & FEASIBILITY EXPENDITURE $         5000000.

OTHER CAPITAL FACTOR                                 .25

                  OPERATING COSTS
                  ---------------

MODIFIED MINING COST $/TON                          1.68

PROCESSING & OTHER COSTS $/TON                      8.65

                  OPERATING PARAMETERS
                  --------------------

MINING DILUTION                                      .15

MINING RECOVERY                                     1.00

PRE-STRIP REQUIREMENT TONS                     14901232.

AVERAGE OPERATING STRIP RATIO W:O                    .03

PROCESSING RECOVERY                                  .77

TAILINGS GRADE                                       .27

                  FINANCIAL FACTORS
                  -----------------

MINE DEVELOPMENT FACTOR                              .03

MILL DEPRECIATION FACTOR                             .75

INVESTMENT TAX CREDIT FACTOR                         .10

EFFECTIVE TAX RATE                                   .50


    ***** ANALYSIS COMPLETE *****

WHAT DO YOU WANT TO DO

 1. SELECT A MINING METHOD
 2. CALCULATE CAPITAL & OPERATING COSTS
```

APPENDIX H

Results of Minimum Reserve Analysis of Copper, Lead, Zinc, Gold and Silver

MINIMUM IN - SITU RESERVE
SCENARIO: CU.ZERO

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 500.

PRICE:		1.0187	.9368	.8549	.7730	.6911
GRADE:	1.6500	2854056.4	3496411.3	4416604.7	5809292.6	8079205.8
GRADE:	1.2300	6109622.5	7825021.8	10465276.2	15956868.1	34859353.9
GRADE:	.8100	46675729.0	143248305.0	2981404463.9	0.0	0.0
GRADE:	.3900	0.0	0.0	0.0	0.0	0.0
GRADE:	0.0000	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: CUPIT

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 500.

PRICE:		1.0187	.9368	.8549	.7730	.6911
GRADE:	1.6500	16913681.8	20790470.5	26376983.3	34898085.4	48936454.3
GRADE:	1.2300	36745261.5	47355905.2	63869672.9	99492374.5	247083860.6
GRADE:	.8100	343299822.7	1310880127.1	*************	0.0	0.0
GRADE:	.3900	0.0	0.0	0.0	0.0	0.0
GRADE:	0.0000	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: CUNC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		1.0187	.9368	.8549	.7730	.6911
GRADE:	1.6500	181316089.4	360124172.7	1088334309.3	20169310664.4	0.0
GRADE:	1.2300	*************	0.0	0.0	0.0	0.0
GRADE:	.8100	0.0	0.0	0.0	0.0	0.0
GRADE:	.3900	0.0	0.0	0.0	0.0	0.0
GRADE:	0.0000	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: CUAC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		1.0187	.9368	.8549	.7730	.6911
GRADE:	3.6000	16235953.7	19865192.6	25038915.6	32815611.5	45364367.8
GRADE:	2.7600	31893615.7	40464234.1	53409571.9	86802650.3	209008133.3
GRADE:	1.9200	166443078.4	398168968.7	2122226742.0	0.0	0.0
GRADE:	1.0800	0.0	0.0	0.0	0.0	0.0
GRADE:	.2400	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: CUSELF

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		1.0187	.9368	.8549	.7730	.6911
GRADE:	4.5300	33227456.4	45135219.2	65816909.2	114719245.1	516734344.9
GRADE:	3.4500	110791865.9	298766698.5	2957590946.4	0.0	0.0
GRADE:	2.3700	0.0	0.0	0.0	0.0	0.0
GRADE:	1.2900	0.0	0.0	0.0	0.0	0.0
GRADE:	.2100	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: CUAS

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		1.0187	.9368	.8549	.7730	.6911
GRADE:	3.7300	202690351.8	1509924382.0	0.0	0.0	0.0
GRADE:	2.9200	0.0	0.0	0.0	0.0	0.0
GRADE:	2.1100	0.0	0.0	0.0	0.0	0.0
GRADE:	1.3000	0.0	0.0	0.0	0.0	0.0
GRADE:	.4900	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: PBNC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		.4600	.4000	.3400	.2800	.2200
GRADE:	9.4700	16428489.0	22325554.4	32831345.3	54818976.9	127842656.7
GRADE:	6.0200	48602789.4	72635385.0	142379452.8	786661586.7	0.0
GRADE:	2.5500	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: PBSELF

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		.4600	.4000	.3400	.2800	.2200
GRADE:	16.9500	9698368.4	14269110.0	23203719.8	47143828.5	469023073.6
GRADE:	11.4600	31447548.1	54658082.1	229434801.6	0.0	0.0
GRADE:	5.9700	0.0	0.0	0.0	0.0	0.0
GRADE:	.4800	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: PBAS

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		.4600	.4000	.3400	.2800	.2200
GRADE:	9.8200	45672440.3	198040847.4	0.0	0.0	0.0
GRADE:	7.2100	0.0	0.0	0.0	0.0	0.0
GRADE:	4.6000	0.0	0.0	0.0	0.0	0.0
GRADE:	1.9900	0.0	0.0	0.0	0.0	0.0

```
MINIMUM IN - SITU RESERVE
SCENARIO: PBZERO
-------------------------


RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % =   5.00
========================================================

DEPTH:                500.
                     --------

PRICE:                       .4600           .4000           .3400           .2800           .2200
---------------------------------------------------------------------------------------------------

GRADE:   9.4700          567322.9        720137.3        965566.3       1407233.6       2360527.5
GRADE:   6.0200         1290050.2       1717733.6       2458314.2       3945134.7       7792172.3
GRADE:   2.5500        12275796.7      25832847.1     168345073.0             0.0             0.0
```

```
MINIMUM IN - SITU RESERVE
SCENARIO: PBPIT
-------------------------


RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % =   5.00
========================================================

DEPTH:                500.
                     --------

PRICE:                       .4600           .4000           .3400           .2800           .2200
---------------------------------------------------------------------------------------------------

GRADE:   9.4700         3165830.7       4022036.5       5399423.0       7884225.9      13269411.4
GRADE:   6.0200         7224253.6       9635194.5      13823294.4      22273841.4      44342259.5
GRADE:   2.5500        70606873.0     156063130.9    1183249320.3             0.0             0.0
```

```
MINIMUM IN - SITU RESERVE
SCENARIO: PBAC
-------------------------


RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % =   5.00
========================================================

DEPTH:               5000.
                     --------

PRICE:                       .4600           .4000           .3400           .2800           .2200
---------------------------------------------------------------------------------------------------

GRADE:  16.9900         4987758.1       6476305.6       8952210.6      13635116.8      24579178.2
GRADE:  11.5700        10638625.7      14524099.7      21534277.0      36529593.2      97475124.3
GRADE:   6.1500        55634430.8     124069025.7     701423552.3             0.0             0.0
GRADE:    .7300               0.0             0.0             0.0             0.0             0.0
```

MINIMUM IN - SITU RESERVE
SCENARIO: ZNAC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		.4900	.4700	.4500	.4300	.4100
GRADE:	12.2000	10511438.8	11504151.8	12666277.0	14040433.3	15683808.0
GRADE:	9.3700	19289098.0	21416016.2	23958082.2	27034589.7	30811636.1
GRADE:	6.5400	53140652.8	62300616.3	77891650.5	104704062.5	149457948.3
GRADE:	3.7100	0.0	0.0	0.0	0.0	0.0
GRADE:	.8800	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: ZNSELF

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		.4900	.4700	.4500	.4300	.4100
GRADE:	29.2800	3603389.7	3899853.2	4242911.5	4643578.0	5116433.6
GRADE:	19.9300	8006639.0	8840479.2	9834656.2	11035254.4	12506585.8
GRADE:	10.7000	57007223.3	73848449.8	108977735.4	179669767.6	359271465.6
GRADE:	1.3800	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: ZNAS

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		.4900	.4700	.4500	.4300	.4100
GRADE:	10.1000	60908391.8	91631323.0	155066708.1	323219973.5	1082471637.4
GRADE:	8.4400	113274798.2	14952637315.8	0.0	0.0	0.0
GRADE:	6.7800	0.0	0.0	0.0	0.0	0.0
GRADE:	5.1200	0.0	0.0	0.0	0.0	0.0
GRADE:	3.4600	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: AUSELF

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		341.4100	306.2200	271.0300	235.8400	200.6500
GRADE:	.8200	1504276.5	1736964.4	2047736.9	2480698.4	3118502.5
GRADE:	.6100	2238459.5	2604188.6	3096988.5	3790413.6	4823522.8
GRADE:	.4000	4093466.4	4318073.8	5806387.1	7215887.0	9347613.8
GRADE:	.1900	13281528.6	15961775.5	19685835.5	25105480.4	33487443.2

MINIMUM IN - SITU RESERVE
SCENARIO: AUAS

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		341.4100	306.2200	271.0300	235.8400	200.6500
GRADE:	4.7300	115049.3	131886.8	154101.9	184587.6	228637.6
GRADE:	3.0100	205659.6	237783.5	280536.6	339740.9	426080.5
GRADE:	1.2900	670513.2	786044.2	940766.4	1155896.6	1469723.0

MINIMUM IN - SITU RESERVE
SCENARIO: AUZERO

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 500.

PRICE:		341.4100	306.2200	271.0300	235.8400	200.6500
GRADE:	.2080	442048.1	512473.3	606364.1	736573.2	926631.6
GRADE:	.1480	706522.0	823886.2	980851.4	1199058.8	1517905.5
GRADE:	.0880	1492220.2	1750694.8	2095916.4	2573968.9	3267085.7
GRADE:	.0280	7903824.7	9180434.3	11060950.8	13920325.6	18150592.5

MINIMUM IN - SITU RESERVE
SCENARIO: AUPIT

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 500.

PRICE:		341.4100	306.2200	271.0300	235.8400	200.6500
GRADE:	.2080	2595296.6	3009702.6	3562476.9	4329573.0	5450198.4
GRADE:	.1480	4152484.5	4844257.6	5770078.9	7058216.6	8942512.9
GRADE:	.0880	8790635.5	10319633.4	12363853.2	15198195.6	19314258.0
GRADE:	.0280	47006004.2	54669276.7	65997952.3	83261853.6	108910745.4

MINIMUM IN - SITU RESERVE
SCENARIO: AUNC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		341.4100	306.2200	271.0300	235.8400	200.6500
GRADE:	.2080	9887936.7	11611577.8	13925714.3	17153211.3	21879546.3
GRADE:	.1480	16407085.0	19323526.5	23228309.2	28648411.5	36524366.1
GRADE:	.0880	35892372.1	42229795.7	50611106.9	62051563.2	78295489.2
GRADE:	.0280	276677205.3	369579978.7	521266581.0	795149862.3	1371408248.1

MINIMUM IN - SITU RESERVE
SCENARIO: AUAC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		341.4100	306.2200	271.0300	235.8400	200.6500
GRADE:	3.3900	204337.4	230297.4	263715.7	308294.0	370624.5
GRADE:	2.0900	350843.7	397588.9	458316.6	540210.3	656232.1
GRADE:	.7900	1140800.2	1314216.1	1544360.6	1862239.8	2324869.2

MINIMUM IN - SITU RESERVE
SCENARIO: AGZERO

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 500.

PRICE:		9.1500	8.2300	7.3100	6.3900	5.4700
GRADE:	3.3000	2215145.9	2591202.6	3087847.1	3765390.8	4728081.2
GRADE:	2.2400	3926068.8	4585095.6	5446468.3	6605526.3	8222148.5
GRADE:	1.1300	9786900.9	11525923.2	14026216.9	17505968.1	22565062.1
GRADE:	.1200	640141487.8	15294622563.1	74613471693.8	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: AGPIT

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 500.

PRICE:		9.1500	8.2300	7.3100	6.3900	5.4700
GRADE:	3.3000	11622054.3	13600549.7	16215214.3	19785165.0	24862768.9
GRADE:	2.2400	20632232.3	24108242.5	28655462.2	34780638.0	43335100.5
GRADE:	1.1800	51626411.0	60859041.6	74147112.5	92676955.4	119684415.6
GRADE:	.1200	60503051160.8	*************	*************	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: AGNC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		9.1500	8.2300	7.3100	6.3900	5.4700
GRADE:	3.3000	53853036.0	62911054.0	74690855.9	91252710.3	119618165.5
GRADE:	2.2400	95877695.0	115307272.8	143023532.2	197299589.7	292246380.3
GRADE:	1.1800	410532733.4	555166338.0	796034129.2	1243500674.9	2228534869.0
GRADE:	.1200	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: AGAC

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		9.1500	8.2300	7.3100	6.3900	5.4700
GRADE:	7.7300	12350709.0	14521161.4	17415103.9	21410615.4	27175462.9
GRADE:	5.2200	22543850.0	26521980.4	31789616.4	38992201.5	49243177.1
GRADE:	2.7100	61413780.6	76085492.8	102755210.1	147705468.6	232972749.0
GRADE:	.2000	0.0	0.0	0.0	0.0	0.0

MINIMUM IN - SITU RESERVE
SCENARIO: AGAS

RATE OF RETURN ASSUMED FOR THIS ANALYSIS WAS % = 5.00

DEPTH: 5000.

PRICE:		9.1500	8.2300	7.3100	6.3900	5.4700
GRADE:	18.7700	3000587.9	3530056.7	4240992.9	5232141.2	6681891.4
GRADE:	12.1100	5925321.2	7002931.5	8447303.1	10453433.9	13368256.1
GRADE:	5.4500	20869212.2	24979364.3	34101835.2	49819315.9	80660049.6

APPENDIX I

Details of the Calculation of Operating Cutoff Grades

BASIC LOGIC

In Appendix A above, two terms NGRAD, the net operating grade, and VAL7, the operating cost expressed as a grade were defined as follows:

NGRAD = (ABG*(1.0-PERDIL)+GD*PERDIL)*S1-VAL7

VAL7 = (YY+ZZ) / (PRICEVAL * CON)

- where, CON is either 1 for gold & silver, or 20 for non - precious metals.

In the limit, when no profit is being made, and operating costs are only just being covered, NGRAD is equal to zero. So,

((VAL7/S1) - (GD*PERDIL)) / (1.0-PERDIL) = ABG

For the purposes of this thesis the grade of the diluting material is assumed to be zero, so the above expression simplifies to:

ABG = (VAL7 / S1) / (1.0 - PERDIL)

-where, ABG is the limiting or cutoff grade. Hence, using the appropriate input values from Appendix G, and substituting in the above equation produced the following results:

CUTOFF GRADE RESULTS

Mining	Commodity				
Method	Copper	Lead	Zinc	Gold	Silver
ZERO	0.67	1.94	1.46	0.039	1.83
PIT	0.79	2.29	1.73	0.046	2.16
NAT.C.	1.35	3.96	3.10	0.080	3.81
ART.C.	1.73	4.84	4.09	0.100	4.58
S.S.	2.72	8.22	7.21	0.150	7.44
ART.S.	3.03	8.82	7.60	0.200	8.97

The above mining method abreviations have the following meanings:

ZERO = zero strip pit
PIT = open pit mining
NAT.C. = natural caving
ART.C. = artificial caving
S.S. = self - supporting
ART.S. = artificially supported

APPENDIX J

Details of the Relative Socio - Political Index Calculation.

The data used below is taken from the 1979 USBM Mineral Commodity Summaries.

COPPER

Region	Reserves	Relative Reserves	Regional SPI	SPI Increment
East	36000	0.07	0.69	0.048
Europe	97000	0.18	1.13	0.203
N. Am.	142000	0.25	1.37	0.343
Austr.	9000	0.02	1.37	0.027
Africa	69000	0.13	0.62	0.081
USSR	54000	0.10	0.90	0.040
S. Am.	142000	0.25	0.80	0.200
Total SPI for copper =				0.942

LEAD

Region	Reserves	Relative Reserves	Regional SPI	SPI Increment
East	0	0	0.69	0
Europe	30000	0.24	1.13	0.27
N. Am.	38000	0.30	1.37	0.41
Austr.	17000	0.14	1.37	0.19
Africa	0	0	0.62	0
USSR	27000	0.21	0.90	0.19
S. Am.	14000	0.11	0.80	0.09
Total SPI for lead =				1.15

ZINC

Region	Reserves	Relative Reserves	Regional SPI	SPI Increment
East	0	0	0.69	0
Europe	54000	0.36	1.13	0.41
N. Am.	50000	0.34	1.37	0.47
Austr.	19000	0.13	1.37	0.18
Africa	0	0	0.62	0
USSR	17000	0.11	0.90	0.10
S. Am.	10000	0.07	0.80	0.06
Total SPI for zinc =				1.22

GOLD

Region	Reserves	Relative Reserves	Regional SPI	SPI Increment
East	0	0	0.69	0
Europe	0	0	1.13	0
N. Am.	155	0.13	1.37	0.18
Austr.	200	0.17	1.37	0.29
Africa	580	0.49	0.62	0.30
USSR	260	0.21	0.90	0.19
S. Am.	0	0	0.80	0
Total SPI for gold =				0.96

SILVER

Region	Reserves	Relative Reserves	Regional SPI	SPI Increment
East	0	0	0.69	0
Europe	420	0.07	1.13	0.08
N. Am.	2220	0.36	1.37	0.49
Austr.	0	0	1.37	0
Africa	0	0	0.62	0
USSR	2000	0.33	0.90	0.30
S. Am.	1460	0.24	0.80	0.19
Total SPI for silver =				1.06

Summarizing:

Commodity	SPI	RSPI
Copper	0.94	0.18
Lead	1.15	0.22
Zinc	1.22	0.23
Gold	0.96	0.18
Silver	1.06	0.19

APPENDIX K

Details of the Deposit Allocation Calculation

K.1 Calculation of DA(i) Parameter Values

COPPER

Deposit	PORPH	SED	CM	STRAT	VMS	COM	HYD
Av.grade	0.810	2.11	2.00	2.37	1.90	1.75	2.70
Av.tons	330.0	9.00	6.80	59.20	21.50	20.80	0.42
+2S.D.G	1.700	3.73	4.36	4.53	3.58	4.29	5.96
+2SD.Ton	1646.	37.0	29.80	197.60	59.50	66.20	0.11
Base Ar.	33.21	1.84	2.00	16.65	6.38	5.68	0.65
XS(i,j)	0.60	0.00	0.00	0.64	0.21	0.30	0.00
Target	71.05	71.05	71.05	71.05	71.05	71.05	71.0
DA(i)	0.59	0.00	0.00	0.32	0.05	0.04	0.00

Note:

The meaning of the abbreviations used in the above and subsequent tables is as follows:

Av. grade	=	mean grade
Av. tons	=	mean tonnage
+2S.D.G	=	+ 2 standard deviations of grade
+2SD.Ton	=	+ 2 standard deviations of tonnage
Base Ar.	=	basic area
XS(i,j)	=	as previously defined
Target	=	target area
DA(i)	=	as previously defined

LEAD

Deposit	PORPH	CM	STRAT	VMS	COM	HYD
Av.grade	2.55	7.23	5.97	6.15	5.26	4.60
Av.tons	420.30	11.00	22.80	61.80	19.70	8.40
+2S.D.G	9.49	15.47	16.95	16.99	12.48	9.82
+2SD.Ton	1494.50	43.40	60.00	201.80	75.50	43.80
Base Ar.	147.23	6.12	8.40	27.89	9.44	4.30
XS(i,j)	0.78	0.69	0.72	0.75	0.70	0.21
Target	155.21	155.21	155.21	155.21	155.21	155.21
DA(i)	0.74	0.03	0.04	0.13	0.05	0.01

ZINC

Deposit	CM	STRAT	VMS	COM	HYD
Av.grade	6.00	10.68	6.54	11.65	6.78
Av.tons	17.70	19.50	64.30	19.70	10.10
+2S.D.G	14.04	29.28	12.20	35.01	10.10
+2SD.Ton	55.50	47.70	176.30	75.50	49.90
Base Ar.	38.96	49.77	99.79	75.45	16.57
XS(i,j)	0.39	0.68	0.37	0.70	0.12
Target	84.82	84.82	84.82	84.82	84.82
DA(i)	0.11	0.24	0.26	0.38	0.01

GOLD

Deposit	PORPH	SED	CM	VMS	COM	HYD
Av.grade	0.028	0.40	1.29	0.088	0.103	0.748
Av.tons	527.6	32.50	1.00	22.900	21.700	16.60
+2S.D.G	0.088	0.82	4.73	0.216	0.221	3.384
+2SD.Ton	1438.8	84.50	3.50	54.900	71.700	98.60
Base Ar.	12.23	6.93	1.66	1.190	1.590	9.48
XS(i,j)	0.54	0.80	0.00	0.470	0.550	0.90
Target	47.27	47.27	47.27	47.270	47.270	47.27
DA(i)	0.30	0.25	0.00	0.030	0.040	0.38

SILVER

Deposit	PORPH	CM	STRAT	VMS	COM	HYD
Av.grade	1.18	5.45	0.89	2.71	2.37	5.73
Av.tons	606.0	8.70	12.10	76.20	22.10	16.80
+2S.D.G	3.30	18.77	0.93	7.73	6.41	13.65
+2SD.Ton	1644.0	36.10	17.50	177.00	79.10	69.80
Base Ar.	108.5	4.77	0.02	17.64	10.14	8.93
XS(i,j)	0.38	0.00	0.00	0.57	0.30	0.34
Target	90.86	90.86	90.86	90.86	90.86	90.86
DA(i)	0.72	0.00	0.00	0.18	0.05	0.05

K.2 Calculation of Budget Allocation

COPPER

Deposit Type	MIC(i),$M	DA(i)	Budget/Type,$M
Porphry	5.98	0.59	3.53
Sedimentary	5.98	0.00	0.00
Contact Meta.	5.98	0.00	0.00
Stratiform	5.98	0.32	1.91
VMS	5.98	0.05	0.30
Complex	5.98	0.04	0.24
Hydrothermal	5.98	0.00	0.00
Total			5.98

LEAD

Deposit Type	MIC(i),$M	DA(i)	Budget/Type,$M
Porphry	5.34	0.74	3.95
Contact Meta.	5.34	0.03	0.16
Stratiform	5.34	0.04	0.21
VMS	5.34	0.13	0.69
Complex	5.34	0.05	0.27
Hydrothermal	5.34	0.01	0.06
Total			5.34

ZINC

Deposit Type	MIC(i),$M	DA(i)	Budget/Type,$M
Contact Mata.	4.56	0.11	0.50
Stratiform	4.56	0.24	1.09
VMS	4.56	0.26	1.19
Complex	4.56	0.38	1.73
Hydrothermal	4.56	0.01	0.05
Total			4.56

GOLD

Deposit Type	MIC(i),$M	DA(i)	Budget/Type,$M
Porphyry	7.12	0.30	2.14
Sedimentary	7.12	0.25	1.78
Contact Meta.	7.12	0.00	0.00
VMS	7.12	0.03	0.21
Complex	7.12	0.04	0.29
Hydrothermal	7.12	0.38	2.70
Total			7.12

SILVER

Deposit Type	MIC(i),$M	DA(i)	Budget/Type,$M
Porphry	5.27	0.72	3.80
Contact Meta.	5.27	0.00	0.00
Stratiform	5.27	0.00	0.00
VMS	5.27	0.18	0.95
Complex	5.27	0.05	0.26
Hydrothermal	5.27	0.05	0.26
Total			5.27

APPENDIX L

Details of the Cutoff Grade - Tonnage Calculations

COPPER

Deposit	PORPH	STRAT	VMS	COM
XS(i,j)	0.60	0.64	0.21	0.30
SD grade	0.42	1.08	0.84	1.27
SD tons	658.00	69.20	19.00	22.70
Av.grade	0.81	2.37	1.92	1.75
Av. tons	330.00	59.20	21.50	20.80
CUTOFF				
Grade	0.66	1.63	2.84	3.00
Tonnage	658.40	71.14	47.00	46.34

LEAD

Deposit	PORPH	CM	STRAT	VMS	COM	HYD
XS(i,j)	0.78	0.69	0.72	0.75	0.70	0.21
SD grade	3.47	4.12	5.49	5.42	3.61	2.61
SD tons	537.10	16.20	18.60	70.00	27.90	17.70
Av.grade	2.55	7.23	5.97	6.15	5.26	4.60
Av. tons	420.30	11.00	22.80	61.80	19.70	8.40
CUTOFF						
Grade	2.09	4.79	4.75	4.25	3.75	7.76
Tonnage	328.79	13.45	16.80	50.45	22.65	34.60

ZINC

Deposit	CM	STRAT	VMS	COM	HYD
XS(i,j)	0.39	0.68	0.37	0.70	0.12
SD grade	4.02	9.30	2.83	11.68	1.66
SD tons	18.90	14.10	56.00	27.90	19.90
Av.grade	6.00	10.68	6.54	11.65	6.78
Av. tons	17.70	19.50	64.30	19.70	10.10
CUTOFF					
Grade	8.56	9.36	7.68	10.05	8.89
Tonnage	33.86	15.26	111.07	22.65	43.91

GOLD

Deposit	PORPH	SED	VMS	COM	HYD
XS(i,j)	0.54	0.80	0.47	0.55	0.90
SD grade	0.030	0.21	0.064	0.059	1.30
SD tons	455.400	26.00	16.00	25.00	41.00
Av.grade	0.028	0.40	0.088	0.103	0.784
Av. tons	527.600	32.50	22.90	21.70	16.80
CUTOFF					
Grade	0.041	0.164	0.115	0.100	0.338
Tonnage	661.660	16.900	29.10	32.27	9.860

SILVER

Deposit	PORPH	VMS	COM	HYD
XS(i,j)	0.38	0.57	0.30	0.34
SD grade	1.06	2.51	2.02	3.96
SD tons	519.00	50.40	28.50	26.50
Av.grade	1.18	2.71	2.37	5.73
Av. tons	606.00	76.20	22.10	16.80
CUTOFF				
Grade	2.05	3.22	4.49	9.01
Tonnage	1019.28	76.11	55.37	46.07

APPENDIX M

Case Study Results for Copper, Lead, Zinc, Gold and Silver.

COPPER

Porphyry

Name	Tons	Grade	Pass/Fail
Granduc	32,500,000.<==	1.93	Fail
Ingerbelle	43,500,000.<==	0.56<==	Fail
Similkameen	4,500,000.<==	0.87	Fail
** Casa Grande	150,000,000.	0.90	Pass **
Lakehead 1	40,500,000.<==	0.40<==	Fail
Greyhound	800,000.<==	0.79	Fail
Highland Valley	200,000,000.<==	0.45<==	Fail
** Cerro Verde	180,000,000.	1.09	Pass **
** Casa Grande 2	448,000,000.	0.70	Pass **
Vancouver 1	180,000,000.	0.52<==	Fail
Lornex 1	293,000,000.	0.43<==	Fail
Bougainville 1	760,000,000.	0.47<==	Fail
Greenvale 1	45,000,000.<==	0.10<==	Fail
** Sar Cheshmeh	300,000,000.	1.50	Pass **
Coppermine Is.	22,000,000.<==	0.50<==	Fail
** Morococha	183,000,000.	0.80	Pass **
Sacoton	17,000,000.<==	0.80	Fail
Sacoton 2	12,000,000.<==	1.50	Fail
** Michiquillay	592,000,000.	0.75	Pass **
** San Manuel	565,000,000.	0.72	Pass **
East Jersey	70,000,000.<==	0.60<==	Fail
Gaspe	60,000,000.<==	1.00	Fail
** Cuajone	500,000,000.	1.00	Pass **
Pima 1	120,000,000.<==	0.51<==	Fail
Chalcobamba	35,000,000.<==	1.50	Fail
** Toquepala	468,000,000.	1.00	Pass **
Gaoua	23,000,000.<==	0.50<==	Fail
Atlas	189,700,000.<==	0.66	Fail
Philex 1	60,000,000.<==	0.70	Fail
Sto. Nino	50,000,000.<==	0.50<==	Fail
Marinduque 2	50,000,000.<==	0.65<==	Fail
Black Mountain	20,000,000.<==	0.63<==	Fail
Bagoto	24,000,000.<==	0.55<==	Fail
Mamut	70,000,000.<==	0.80	Fail
Valley Copper	500,000,000.	0.50<==	Fail
Schaft Creek	200,000,000.<==	0.40<==	Fail
** Copper Creek	200,000,000.	0.90	Pass **
La Caridad	100,000,000.<==	0.80	Fail
** Fernando Val.	80,000,000.	1.00	Pass **
Colon	50,000,000.<==	1.00	Fail
Cerro Colorado 1	40,000,000.<==	0.80	Fail
El Salvador 1	1000,000,000.	0.50<==	Fail
** Chuiquicamata	4000,000,000.	1.00	Pass **
** Bingham 1	2000,000,000.	0.80	Pass **
** Butte 1	800,000,000.	2.43	Pass **
** Palabora 1	400,000,000.	0.69	Pass **

Sedimentary

Micilla	2,000,000.<==	2.50	Fail
Musoshi	30,000,000.<==	2.60	Fail
Cadia 2	1,000,000.<==	0.89	Fail
Horne	3,300,000.<==	2.44	Fail

Stratiform

Bwana Mkumba	5,760,000.<==	3.48	Fail
Avoca	6,000,000.<==	1.00<==	Fail
Matsitama	6,420,000.<==	2.24	Fail
Skouriotissa	20,000,000.<==	0.58<==	Fail
Kalengwa	250,000.<==	3.45	Fail
Skouries 1	17,700,000.<==	1.05<==	Fail
Mattagami 2	18,000,000.<==	0.70<==	Fail
Antamina 1	11,000,000.<==	1.90	Fail
Jabal Sayid	8,000,000.<==	2.50	Fail
** Mufilira	167,067,000.	3.37	Pass **
Chambishi	38,785,000.<==	3.05	Fail
** Baluba	112,000,000.	2.41	Pass **
** Luanshya	85,516,000.	2.86	Pass **
** Rhokana	125,327,000.	2.77	Pass **
** Nchanga	259,405,000.	4.01	Pass **
** Bancroft	96,882,000.	3.51	Pass **
Martinduque 1	4,800,000.<==	2.00	Fail
** Roan	93,500,000.	3.00	Pass **
** Roan Antelope	96,300,000.	2.95	Pass **
Naciemento	11,000,000.<==	0.65<==	Fail

Contact Matamorphic

Orange 1	2,540,000.<==	1.05<==	Fail
Aberlow	3,000,000.<==	1.20<==	Fail
Sabena	4,000,000.<==	0.70<==	Fail
Val d'Or	500,000.<==	3.23	Fail
Tipperary 1	6,000,000.<==	1.20<==	Fail
Snow Lake	1,000,000.<==	3.00	Fail
Goudreau	500,000.<==	1.56<==	Fail
Black Copper	1,170,000.<==	0.67<==	Fail
Flexar 1	270,000.<==	4.23	Fail
Inguaran 1	4,400,000.<==	2.00	Fail
Amos 1	2,500,000.<==	1.10<==	Fail
Batialo 2	500,000.<==	4.00	Fail
Eeco 1	27,000,000.<==	2.10	Fail
Rosita	3,689,000.<==	1.25<==	Fail
Huanzala 3	2,200,000.<==	1.00<==	Fail
Matchless	2,400,000.<==	1.70	Fail
Butrest	6,420,000.<==	2.24	Fail
Timna	11,000,000.<==	1.60<==	Fail

Al Amar 1	5,500,000.<==	0.70<==	Fail
Scotia 2	1,250,000.<==	0.25<==	Fail
Mt. Isa 6	1,500,000.<==	3.80	Fail
Mt. Isa 7	45,000,000.<==	3.20	Fail
Mt. Lyell	41,900,000.<==	1.40<==	Fail
Warrego 1	3,500,000.<==	2.60	Fail
Lepanto 3	500,000.<==	4.00	Fail
Aberlow 1	3,900,000.<==	1.20<==	Fail
Coppermine River	3,000,000.<==	3.48	Fail
San Antonio 1	5,000,000.<==	1.40<==	Fail

Oxide

** Exotica	153,000,000.	1.61	Pass **

Volcanogenic Massive Sulphide

Ertsberg	33,000,000.<==	2.50	Fail
Pikwe 1	27,670,000.<==	1.16<==	Fail
Selibe 1	13,500,000.<==	1.57<==	Fail
** Kidd Creek 2	62,500,000.	1.33	Pass **
Tintaya	7,000,000.<==	3.00	Fail
Mt. Morgan 1	9,530,000.<==	1.08<==	Fail
Lepanto 1	8,900,000.<==	2.97	Fail
F. T. Patinio 1	40,000,000.<==	0.80<==	Fail
Madankudan	3,000,000.<==	2.75	Fail
Kusaka 1	10,000,000.<==	2.00<==	Fail

Complex Sulphide

Mt. Curson 1	3,200,000.<==	1.04<==	Fail
Bathurst 3	60,800,000.	0.28<==	Fail
Bathurst 5	13,000,000.<==	1.14<==	Fail
Bathurst 6	18,000,000.<==	0.37<==	Fail
Anderson Lake 1	17,600,000.<==	3.00	Fail
Madrigal 1	1,000,000.<==	3.00	Fail
Tsumeb 2	7,000,000.<==	3.66	Fail
Roseburg 3	8,650,000.<==	0.89<==	Fail
** Horne 1	58,000,000.	2.40	Pass **

Hydrothermal

Juno 1	200,000.<==	0.50<==	Fail
Peko 2	900,000.<==	3.60	Fail
Ivanhoe 1	160,000.<==	4.20	Fail
Daribo	400,000.<==	2.50	Fail

GOLD
====

Porphyry

Bougainville 2	760,000,000.	0.020<==	Fail
Philex 2	60,000,000.	0.020<==	Fail
Cerro Colorado 2	18,000,000.<==	0.080	Fail
El Salvador 2	1000,000,000.	0.005<==	Fail
Butte 6	800,000,000.	0.008<==	Fail

Sedimentary

** H. B. Fontein	28,100,000.	0.400	Pass **
H. B. Fontein 2	20,600,000.	0.020<==	Fail
Wit Nigel	4,210,000.<==	0.270	Fail
** Braken	28,000,000.	0.470	Pass **
E. G. Main	2,000,000.<==	0.250	Fail
E. G. Kimberley	5,000,000.<==	0.230	Fail
Groutvlei Main	14,000,000.<==	0.210	Fail
** Groutvlei Kim.	18,000,000.	0.220	Pass **
** Kinross	23,000,000.	0.360	Pass **
** Leslie	37,000,000.	0.330	Pass **
Marivale Main	16,000,000.<==	0.260	Fail
** Marivale Kim.	17,000,000.	0.260	Pass **
** St. Helena	95,000,000.	0.520	Pass **
** Winkelhaak	50,000,000.	0.300	Pass **
Cortez	3,400,000.<==	0.290	Fail
Eagle	1,600,000.<==	0.410	Fail
Donalda	3,150,000.<==	0.350	Fail
** Elsburg	54,000,000.	0.350	Pass **
** Virginia SA 1	37,000,000.	0.298	Pass **
Merriespruit 1	16,000,000.<==	0.280	Fail
** E. Daggerfontein	17,000,000.	0.170	Pass **
** Vaal Reefs	66,100,000.	0.480	Pass **
Dome	2,030,000.<==	0.279	Fail
Campbell Red Lake	1,300,000.<==	0.690	Fail
Luz	3,280,000.<==	0.095	Fail
** Doornfontein	29,060,000.	0.430	Pass **
** E. Driefontein	100,000,000.	0.440	Pass **
Kloof	11,590,000.<==	0.550	Fail
** Libanon	25,900,000.	0.400	Pass **
Luipaardsvlei	6,360,000.<==	0.270	Fail
Spaarwater	710,000.<==	0.360	Fail
Sub Nigel	2,660,000.<==	0.430	Fail
** Venterspost	21,880,000.	0.440	Pass **
Vlakfontein	9,390,000.<==	0.460	Fail
** W. Dreifontein	64,880,000.	0.811	Pass **
** East Dagga	24,270,000.	0.300	Pass **
** F. S. Geduld	49,540,000.	1.270	Pass **
** P. Brand	75,550,000.	0.660	Pass **
** P. Steyn	68,900,000.	0.380	Pass **
S. A. Lands	13,850,000.<==	0.390	Fail

** Welkom	53,650,000.	0.400	Pass **
** W. Deeps	49,150,000.	0.650	Pass **
** W. Holdings	70,180,000.	0.700	Pass **
** W. Reefs	48,250,000.	0.430	Pass **
** Blyvoor	58,740,000.	0.700	Pass **
** Durban Deep	38,600,000.	0.200	Pass **
** E. Rand Prop.	52,500,000.	0.270	Pass **
** Harmony	64,890,000.	0.380	Pass **
** Western Areas	53,980,000.	0.350	Pass **
** Grootvlei	32,000,000.	0.220	Pass **
** Buffelsfontein	68,380,000.	0.470	Pass **
S. Roodepoort	8,020,000.<==	0.310	Fail
** Stilfontein	22,520,000.	0.460	Pass **
** W. Rand Cons.	34,920,000.	0.220	Pass **
** Hartebeestfontein	52,760,000.	0.410	Pass **
** Loraine	80,440,000.	0.410	Pass **
** Rand Leases	60,000,000.	0.410	Pass **
** Zandpan	28,080,000.	0.400	Pass **
** Ashanti	37,000,000.	1.040	Pass **
Kalgoorlie	6,100,000.<==	0.190	Fail
Great Boulder	1,530,000.<==	0.240	Fail
N. Kalgoorlie	2,100,000.<==	0.250	Fail
** Kolar	45,300,000.	0.590	Pass **

Stratiform

Skouries 2	17,700,000.	0.034<==	Fail

Contact Metamorphic

Goudreau	500,000.<==	0.135	Fail
Flexar 3	270,000.<==	0.030<==	Fail
Batialo 1	500,000.<==	3.500	Fail
Norseman	530,000.<==	0.500	Fail
Warrego 2	3,500,000.<==	0.060	Fail
Lepanto 2	500,000.<==	3.500	Fail

Volcanogenic Massive Sulphide

Ertsberg 4	33,000,000.	0.020<==	Fail
Mt. Morgan 2	9,530,000.<==	0.090<==	Fail
Lepanto 2	8,900,000.<==	0.173	Fail
R. T. Patinio	40,000,000.	0.070<==	Fail

Complex Sulphide

Leadville 4	2,401,000.<==	0.084<==	Fail
Anderson Lake	17,600,000.<==	0.038<==	Fail
Roseburg 5	8,650,000.<==	0.110	Fail
** Horne 2	58,000,000.	0.180	Pass **

Hydrothermal

El Salvador 1	118,000.<==	0.150<==	Fail
Bullfinch	16,000.<==	4.000	Fail
Falcon	760,000.<==	0.320<==	Fail
Fergusson 1	60,000.<==	0.044<==	Fail
El Dorado	1,000,000.<==	0.480	Fail
El Sal 1	118,000.<==	0.150<==	Fail
Juno 3	200,000.<==	3.000	Fail
Peko 1	900,000.<==	0.100<==	Fail
Ivanhoe 2	160,000.<==	0.070<==	Fail
Emperor	970,000.<==	0.450	Fail
** Hollinger 1	60,000,000.	0.320	Pass **
** Homestake 1	135,000,000.	0.320	Pass **

SILVER

Contact Metamorphic

Atlin 1	150,000.<==	20.00	Fail
Aberlow 2	3,000,000.<==	1.66<==	Fail
Tipperary 2	6,000,000.<==	1.66<==	Fail
Flexar 4	270,000.<==	0.12<==	Fail
Inguaran 2	4,400,000.<==	0.30<==	Fail
Farrell 2	60,000.<==	14.10	Fail
Mt. Isa 2	600,000.<==	2.00<==	Fail
Mt. Isa 5	34,000,000.<==	5.40	Fail
Aberlow 3	3,900,000.<==	1.87<==	Fail
San Eulaila 3	35,000,000.<==	7.41	Fail

Complex Sulphide

Mt. Curson 2	3,200,000.<==	0.35<==	Fail
Leadville 3	2,401,000.<==	2.64<==	Fail
** Bathurst 4	60,080,000.	2.40	Pass **
Bathurst 9	18,000,000.<==	1.84<==	Fail
Anderson Lake 5	17,600,000.<==	0.61<==	Fail
Madrigal 4	1,000,000.<==	6.00	Fail
Tsumeb 1	7,000,000.<==	2.13<==	Fail

Roseburg 4	8,650,000.<==	5.10	Fail
Laisvall 3	80,000,000.	0.29<==	Fail

Volcanogenic Massive Sulphide

Ertsberg 3	33,000,000.<==	0.30<==	Fail
** Kidd Creek 3	62,500,000.	4.85	Pass **
Anvil 3	63,000,000.	1.00<==	Fail
N. Broken Hill 2	45,000,000.<==	7.39	Fail
R. T. Patinio 3	40,000,000.<==	1.70<==	Fail
Sullivan 3	170,000,000.	1.77<==	Fail
Broken Hill 3	120,000,000.	1.93<==	Fail

Stratiform

Mogul 3	10,200,000.<==	0.90<==	Fail
Silvermines 3	14,000,000.<==	0.87<==	Fail

Hydrothermal

El Salvador 2	118,000.<==	10.00	Fail
Fergusson 2	60,000.<==	6.90<==	Fail
Frances Lake 1	400,000.<==	4.20<==	Fail
Hollinger 2	60,000,000.	0.07<==	Fail
Bunker Hill 1	40,000,000.<==	3.22<==	Fail

Porphyry

Cerro Colorado 3	18,000,000.<==	1.35<==	Fail
El Salvador 3	1000,000,000.<==	0.05<==	Fail
** Butte 5	800,000,000.	2.15	Pass **

LEAD
====

Porphyry

** Pine Point 1	40,500,000.	5.00	Pass **
Butte 4	800,000,000.	0.10<==	Fail

Oxide

Angouran	15,000,000.<==	7.00	Fail

Stratiform

** Zba 1	44,100,000.	12.00	Pass **
Mogul 2	10,200,000.<==	2.80<==	Fail
Silvermines 1	14,000,000.<==	2.80<==	Fail

Contact Metamorphic

Atlin 2	150,000.<==	5.00	Fail
Huanzala 2	2,200,000.<==	7.00	Fail
Farrell 3	60,000.<==	12.80	Fail
Mt. Isa 1	600,000.<==	5.50	Fail
** Mt. Isa 3	34,000,000.	7.40	Pass **
San Antonio 2	5,000,000.<==	0.90<==	Fail
** San Eulaila 1	35,000,000.	12.00	Pass **

Hydrothermal

Fergusson 3	60,000.<==	6.00<==	Fail
Frances Lake 2	400,000.<==	8.00	Fail
Ichmoul	1,300,000.<==	4.00<==	Fail
Moate 2	110,000.<==	1.00<==	Fail
** Bunker Hill 2	40,000,000.	4.00	Pass **

Complex Sulphide

Leadville 1	2,401,000.<==	5.13	Fail
** Bathurst 2	60,800,000.	3.50	Pass **
Bathurst 7	18,000,000.<==	2.35<==	Fail
Anderson Lake 3	17,600,000.<==	0.20<==	Fail
Madrigal 2	1,000,000.<==	6.08	Fail
Tsumeb 3	7,000,000.<==	10.50	Fail
Belanatana 2	730,000.<==	2.90<==	Fail
Beltana 4	97,000.<==	12.00	Fail
Roseburg 2	8,650,000.<==	5.60	Fail
** Laisvall 1	80,000,000.	4.30	Pass **

Tri - State

Pitcher 2	200,000,000.	0.80<==	Fail

Volcanogenic Massive Sulphide

** Anvil 1	63,000,000.	4.00	Pass **
N. Broken Hill 1	4,500,000.<==	12.97	Fail
Madankadan 2	3,000,000.<==	1.20<==	Fail
** Sullivan 1	170,000,000.	4.00	Pass **
** Broken Hill 1	120,000,000.	13.00	Pass **
Kosaka 2	10,000,000.<==	1.70<==	Fail

ZINC

Oxide

** Angouran 2	15,000,000.	28.00	Pass **

Stratiform

** Mattagami 1	18,000,000.	10.00	Pass **
Antamina 2	11,000,000.<==	1.50<==	Fail
** Zba 2	44,100,000.	26.30	Pass **
Mogul 1	10,200,000.<==	8.20<==	Fail
Silvermines 2	14,000,000.<==	7.40<==	Fail

Contact Metamorphic

Tennesse	50,000,000.	5.00<==	Fail
Flexar 2	270,000.<==	0.40<==	Fail
Geco 2	27,000,000.<==	5.10<==	Fail
Huanzala 1	2,200,000.<==	13.00	Fail
El Amar 2	5,500,000.<==	5.00<==	Fail
Farrell 1	60,000.<==	7.30	Fail
Mt. Isa 1	34,000,000.<==	5.60<==	Fail
San Antonio	5,000,000.<==	1.60<==	Fail
San Eulaila 2	35,000,000.<==	11.00	Fail

Porphyry

Butte 2	800,000,000.	0.74<==	Fail

Hydrothermal

	Fergusson 4	60,000.<==	6.70<==	Fail
	Frances Lake 3	400,000.<==	9.00	Fail
	Moate 1	110,000.<==	6.40<==	Fail
	Bunker Hill 3	40,000,000.	5.00<==	Fail

Complex Sulphide

	Leadville 2	2,401,000.<==	9.95<==	Fail
**	Bathurst 1	60,800,000.	8.88	Pass **
	Bathurst 6	18,000,000.	5.63<==	Fail
	Anderson Lake 2	17,600,000.	3.80<==	Fail
	Madrigal 3	1,000,000.<==	4.00<==	Fail
	Tsumeb 4	7,000,000.<==	3.10<==	Fail
**	Beltana 1	730,000.	37.00	Pass **
	Beltana 3	730,000.<==	24.90	Fail
**	Roseburg 1	8,650,000.	18.60	Pass **
	Laisvall 2	80,000,000.	0.60<==	Fail

Tri - State

	Pitcher 1	200,000,000.	3.20<==	Fail

Volcanogenic Massive Sulphide

	Pine Point 2	40,500,000.<==	5.00<==	Fail
**	Kidd Creek 1	62,500,000.	7.08	Pass **
	Anvil 2	63,000,000.	5.00<==	Fail
**	N. Broken Hill 3	45,000,000.	10.70	Pass **
	Madankadan 3	3,000,000.<==	3.50<==	Fail
	Sullivan 2	170,000,000.	5.00<==	Fail
**	Broken Hill 2	120,000,000.	11.00	Pass **
	Kosaka 3	1,000,000.<==	5.00<==	Fail