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MINERAL EXPLORATION

RISK ANALYSIS ASSESSMENT OF THE INFLUENCE OF
GEOLOGICAL FACTORS
ON EXPLORATION AND MINING INVESTMENT ALTERNATIVES:
DEVELOPMENT OF A MICROCOMPUTER SIMULATION MODEL.

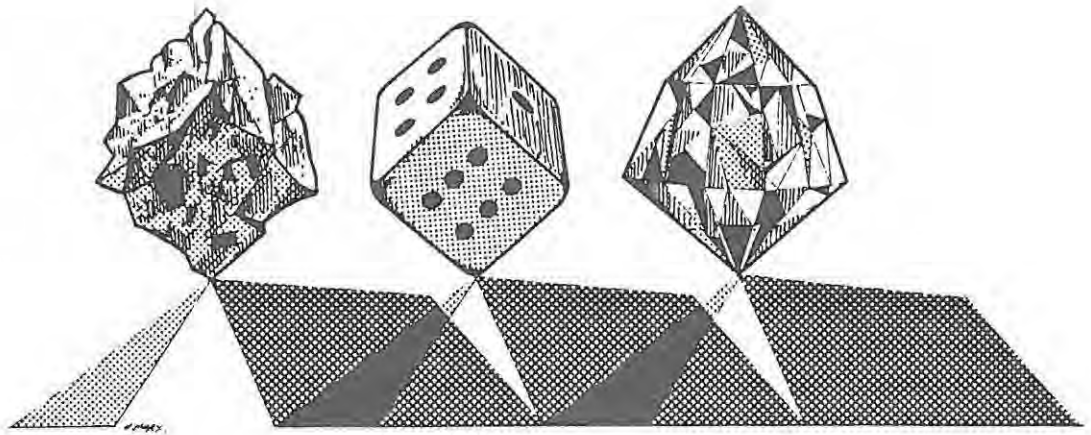
by

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HIGH RISK ? HIGH REWARD

ABSTRACT

A microcomputer risk analysis model is developed and used in an exploration - delineation simulation to analyse in particular the effects of geological factors on exploration - mining investment alternatives. Analysis of results indicate that geological parameters can have profound effects on such investment alternatives and that the role of the geologist in determining and evaluating the significance of the various geological factors is critical.

Simulation examples highlight some of the key geological parameters and show how changes in these parameters influence both the expected mean results and the standard deviations of such means.

The risk analysis model provides an ideal means of conveying the importance of the different geological factors on exploration - delineation - mining investment alternatives and may be used as a geological education aid.

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GENERAL INTRODUCTION.

The main aims of this dissertation are threefold:

to foster the awareness of both the explorationist and the development planning / mining geologist with respect to the economic effects and risk implications of various geological parameters associated with mineral prospects;

to illustrate the relative ease with which a viable, robust risk analysis system can be instituted, using modern high-capacity micro-computers and readily available sophisticated software packages;

and to attempt to convey the belief that it is geologists first and foremost who should make exploration and delineation decisions, but that to do so rationally and intelligently requires that they understand and are able to balance and partly quantify the relationship between geology and economics.

After a brief look at the mineral exploration environment, attention is focussed on mineral prospect evaluation. Recognition of the complex interplay between geology and economics develops naturally into a perceived need for quantification or at least the probabilistic specification of the interacting variables, and methods to study their affects. Coherent geological-economic models are required from the outset.

The basic principles of risk analysis are outlined and then applied to mineral prospect models at different stages of data collection. The method used allows for quantitative analysis and the effects of variable geological parameters on overall economic performance are specifically demonstrated.

SECTION A : MINERAL EXPLORATION.

OBJECTIVES AND INCENTIVES.

The prime objective of any mineral exploration programme is to locate economically viable mineral deposits. Economic viability of a mineral deposit is a complex function of geology, geography and time. Different aspects of these factors are both independent and dependent on each other and are discussed in full in the later section on prospect evaluation. Methods of assessing economic viability are also discussed in full in the section on risk analysis.

The prime incentive for any mineral exploration programme is a favourable probability that a sufficient number of economically viable deposits will be located and exploited such that the combined discovery-exploitation costs are less than the accrued revenue. More specifically, the difference between revenue and total costs should meet or exceed some predetermined level, a level that is normally a function of the cost of capital.

Quantification of the discovery risk involved is normally only possible if empirically derived historically based probability factors are determinable. Extensive treatment of this aspect of exploration is given for example for the Shield and Cordilleran regions of Canada (Mackenzie, 1981). Treatment of the subject is beyond the scope of this dissertation and the interested reader is referred to the work of Mackenzie (op. cit.) as example.

Suffice to say that the incentives are extremely sensitive to various combinations of the above mentioned factors of geology, geography and time. Embodied in the concept of time in particular are such factors as price variations, exchange fluctuations, changing technology and changing government and environmental policy. Certain geographical factors are also very time dependent given that infrastructure for example may change significantly with time.

Also of major significance and consequence is the extent to which exploration funding is tax deductible (government policy). This factor in particular may severely hamper if not curtail exploration by companies or groups not able to benefit from exploration tax write-offs against previously generated profits.

Although not necessarily the case, it seems logical to assume that any company, group or individual involved in mineral exploration, perceives it to be an acceptable risk (worth taking ahead of other investment opportunities?) and expects some chance of success, tempered by the knowledge of the high risk - high return nature of mineral exploration in general.

PHILOSOPHY.

The best way to structure and equip a successful exploration team is subject to considerable debate. Different philosophies abound but may often be linked to the degree of bureaucracy that is in turn so often a function of the size of the money providing organization. How much autonomy, what size budget, what levels of manpower, what level of technical support and expertise .. etc. are all decision variables. Rather than be drawn into the debate at this point, the following list of characteristics of exploration organizations that are **known** to be creative, vital and most of all successful is offered (Mackenzie, 1981).

- * High quality staff and orientation toward people. These organizations have outstanding key people who challenge subordinates and are sensitive to their needs for achieving job satisfaction.
- * Sound basis of operations. The organizations work within corporate guidelines toward higher level goals and objectives.
- * Creative and productive atmosphere. The organizations encourage independent, creative, and innovative thinking in an environment free from stifling controls and administrative distractions. Adequate facilities and state-of-the-art geotechnical support are requisite.
- * High standards. The organizations achieve high standards of performance, integrity, and ethics.
- * Entrepreneurial form of organization. The entrepreneurial style is best suited as it fosters imagination, innovation and flexibility in high-risk and high reward environments.

* Morale and team spirit. The organizations have high morale, enthusiasm, and a "can-do" attitude. The team enjoys camaraderie and loyalty, and has the ability to engage in new opportunities with confidence and commitment.

There appears to be a certain level of agreement that a tight-knit group of some six to ten well qualified professional geologists supported by a medium sized annual budget (large enough for a degree of freedom, but not so large as to attract peripheral administrators - and raised eyebrows) constitutes the ingredients of a successful exploration team. In many situations, a lot hinges on how and by whom the ingredients are blended.

AREA AND COMMODITY SELECTION.

By area selection is meant the "geographical" area, and this is normally strongly influenced both by political risk and perceived "prospectivity" for the commodities in question. Also important are logistical and familiarity factors. Exploration within the confines of a known political framework from a proximal organizational homebase is often far more comfortable (and possibly successful) than far flung ventures in areas of political uncertainty.

Of major importance in area selection is the possibility or otherwise of tax concessions usually for profit generating operations within the same geographical boundaries as the exploration programme. This often effectively restricts exploration incentive and thus effort to areas either within or directly adjacent to currently operating mine properties. The major current phase of gold exploration in the Witwatersrand basin is a case in point, with most of the new shaft sinking and development (if not all of the exploration) taking place either on or adjacent to existing operational mine lease areas.

The question of commodity selection is a difficult one, especially so if grassroots exploration is anticipated. Even with the most well-founded exploration programmes, grassroots ventures would not normally be expected to result in producing mines in less than anything from eight to fifteen years (and more often than not - NEVER) after programme inception. Given this long time delay between inception and possible success and production, commodity market and price forecasting should ideally be targeted a decade

or more into the future - this seldom appears to be the case in practice.

Considering the problems of long-term price and exchange rate forecasting, such forecasting as is carried out is perhaps best done so on the combined basis of historical constant value (indexed) prices and the anticipated future supply-demand balance. Supply-demand forecasting for many commodities is bedevilled though by the present era of rampaging technology and the ever-present "threat?" of substitution.

Many commodity selection decisions are doubtless strongly influenced by the level of existing exploitation, treatment and marketing experience and skills concerning particular deposit types that any particular company or group may possess. This is perhaps best highlighted by noting the frequency of burnt fingers in situations where decisions to mine are taken where the necessary deposit-specific experience and knowledge is lacking.

RESOURCE APPRAISAL.

The essence of resource appraisal or rather, what constitutes a resource might well be summarized as follows: The **crustal abundance** of the various elements of interest (commodities) is heterogeneous. In particular, a very small percentage of the crust contains concentrations of these commodities that range from one to several orders of magnitude above average crustal levels. It is these areas of concentration above background that constitute the **resource base**. The **mineral endowment** of an area represents that subset of the resource base which has been selected on common sense grounds and which may or may not satisfy economic and other criteria and become part of the resources of the area.

In that at least one of the aims of resource appraisal of any particular area is to ascertain the "indicated exploration potential" of that area, the very process of resource appraisal is (or should be) intimately associated with area and commodity selection, as well as with target generation.

Methods employed in resource appraisal vary and include those that are primarily geological, economic, statistical or mathematical in nature. Excellent treatment of the subject is given by Harris (1984) and in summary form by the same author (Harris, 1981). A personal viewpoint is that the most realistic, accurate and moreover useful resource appraisal studies are

likely to be multidisciplinary in nature, and if anything should lean heavily towards modern geological-geotectonic-geogenesis thinking, and less towards mathematics and statistics. In this vein it is satisfying to see how totally reliant the relevant computer driven artificial intelligence expert systems (e.g. PROSPECTOR) are on sound geological knowledge - without which the systems would not exist.

The most useful outcome (from an explorationists point of view) of the resource appraisal of an area is not only the resource potential of the various subregions of the area, but the predicted type and size of potential ore deposits - in other words, the development of a geological model(s) of the region and of the expected ore deposit types. These working hypotheses then form the basis for both target selection and, to a large extent, preliminary thoughts on the most suitable exploration methodology.

TARGET GENERATION.

If a resource appraisal of an area has been conducted, and a particular commodity (or commodities) has been selected (as for previously discussed reasons), then the process of target generation should follow naturally. Target generation without prior resource appraisal would normally require many of the techniques and procedures implicit in resource appraisal itself.

Working on the assumption that mineral exploration is not purely a matter of luck (in which case there would presumably be no need for area, commodity or target selection, you would simply start drilling wherever you happened to be), a sound starting point is a thorough working knowledge of the tectonic setting and nature of ore deposits. It must immediately be stressed however, that this knowledge by itself is of little direct use without a knowledge of the geology and tectonics of the area in question. It is no good simply knowing that porphyry copper deposits are often located in subduction related arc settings if you have no idea whether or not the selected area includes such arc or fossil-arc settings!

Having then generated a geological model of the area (preferably during the resource appraisal phase) and having proposed working hypotheses of the likely deposit-types and settings of the selected commodity(ies), the next step is to select the most favourable sub-areas within the region for actual physical exploration. This phase may already overlap with the next step,

that of the methodology adopted, in that some data from reconnaissance methods may already be available (e.g. aeromagnetics).

Even at this early stage of exploration, the geologist is already equipped with a knowledge of the general geographical conditions (he knows where he is going to concentrate his efforts), and the broad outlines of the type and likely dimensions of any deposits (from his geological models). Used correctly, this information, together with what should be readily and freely available additional information (either from within the parent company or from companies that may be mining deposits similar to those envisaged in the model), can form the basis of an initial albeit poorly constrained economic feasibility study. Traditionally the realm of the vastly experienced consultant or director, such studies are often conducted on sundry small scraps of paper. Although such methods of evaluation may be fairly useful in the broad sense and for the norm, they cannot hope to incorporate more than the most basic parameters (tonnage, grade, mining costs, prices), and more seriously perhaps, may fail to consider information implicit to the current geological model, but outside the sphere of experience of the estimator.

One of the stated aims of this dissertation is to illustrate the applicability of risk analysis models to exploration and delineation decision making at all of the normal GO - NO GO decision points. The important point to make at this juncture is that:

THE MORE CORRECT AND MORE PRECISELY CONSTRAINED THE INITIAL AND LATER GEOLOGICAL MODELS, THE MORE REALISTIC, MEANINGFUL AND USEFUL WILL BE ANY "PRE- OR POST-DISCOVERY" ECONOMIC FEASIBILITY STUDY.

An analysis of the effects and influences of geological parameters on deposit economics will clarify this statement. It will also become clear which of the parameters have the greatest effect and thus most need be constrained, therefore indicating which aspects of the model (or indeed actual deposit) be further researched or clarified.

METHODOLOGY.

The particular exploration methodology employed is very much a function of the terrane and the nature of the target (geological model). It may also be a function of budget and time constraints, although, if the expected targets and likely probability of success warrant exploration, budgets should not be prohibitively restrictive (if the terrane and the model fundamentally require a particular technique - use it!). Many of the methods and techniques that may be employed and that are specifically useful in certain situations have been discussed in a separate exercise (Mallinson, 1985), and fall outside of the scope of this dissertation.

It is however probably fair to say that for many different forms of exploration in widely diverse terranes, a broadly similar approach would be adopted, that is, one of successive scaling down of aerial size and of increased detailed coverage. Once again, as with resource appraisal, any exploration effort should make use of all of the available data and a multidisciplinary approach is advocated. A principally geochemically orientated study may for example rely heavily on geological maps which in turn may be vastly improved by the addition and use of geophysical data.

Whatever the system or sequence adopted, at some stage diamond drilling is normally required, and siting of drill holes is often best carried out by detailed structural mapping if surface exposures allow, or by a combination of detailed soil/rock geochemical patterns and geophysical delineation when applicable (IP, EMP, magnetics, self potential etc.).

For the purpose of this dissertation and most of the further discussion, an anomaly or mineral occurrence becomes a mineral prospect once a diamond drill hole (DDH) has intersected an "interesting" mineral horizon of a mineable width.

SECTION B : MINERAL PROSPECT EVALUATION.

INTRODUCTION.

The prime reason for evaluating any mineral prospect is to ascertain the economic potential of the prospect. The proviso is added that there may well be some incentive to further assess prospects that at an early stage are found to be economically unattractive. This would be the case if it was felt that further assessment expenditure could improve or modify geological models that should be at the centre of the whole exploration endeavour.

Even before a "prospect" as previously defined is located, the need for some form of economic evaluation is apparent. The whole exploration programme is presumably founded on the hope of locating "economically attractive" deposits. Much of what follows in this section will be aimed at quantifying just what constitutes an "economically attractive" deposit, and in emphasizing the role of geological factors in making it so or not so.

A useful way of introducing the topic is to look briefly at the WHY?, HOW?, WHERE?, and WHEN? of mineral prospect evaluation.

WHY?

This question is already partly answered in the above and previous sections. More specifically though, any person charged with expenditure decision making must have as sound as possible justification. Throughout both the exploration and the evaluation phases of programmes, numerous GO - NO GO decision points are reached. Add to this the likelihood that the decision maker (who should most certainly at an early stage be the exploration geologist in charge) most probably has numerous prospects in the pipeline that require decisions. Ideally, the prospects need to be ranked in order of priority and to do this intelligently and confidently, and not purely because government minimum prospect expenditure laws demand it, requires that economic assessments be conducted. Notice that it does not necessarily follow that the most promising looking prospect be accorded the first priority at any point in time, as it may be equally valid and expedient to prove the non-viability of some prospects in order that options may be

relinquished and efforts then channelled on the more promising prospects.

HOW?

There are doubtless many differing opinions and philosophies on just how prospects should be evaluated. These may range from general lack of real interest, through hunches or sheer guesses to some form of coherent, sound and reproducible method. Ideally, the method used should allow for **comparative** decision making between options, preferably with an indication and quantification of the expected returns and the associated risks involved. The technique of on-site micro-computer aided risk analysis outlined later meets most of these requirements.

WHERE?

All too often as the various stages of further investigation and delineation begin to indicate the makings of a potentially attractive proposition, the role of decision maker passes up through the hierarchy and is rapidly taken out of the hands of the person best equipped, if not to single-handedly take further decisions, then at least to be a key member if not the key member of a decision team. This stems partly from the norm where there is often an inability on the part of the exploration geologist to handle the rapidly accumulating data on all aspects of the prospect, let alone handle the raw geological data. Breaks in continuity brought about by premature withdrawal of the explorationist from a "going" prospect can (and do) have serious consequences, as so many of the more detailed (and subtle) geological parameters can have such a marked effect on the attractiveness or apparent attractiveness of a prospect. The handing over of only a geological reserve estimation does not do justice to the contribution that the on-site geologist can and should make to any feasibility study.

For these reasons it is strongly advocated that the on-site geologist be equipped with the tools of recording the pertinent geological parameters and for making his own albeit preliminary risk and reward assessments. This would certainly require that in addition to the normal field equipment, a microscope and on-site computing facilities would be requisite. In addition, the incumbent should be accorded the necessary authority within the company structure to enable early and frequent interdisciplinary working

groups to be convened at his initiation.

WHEN?

As to when prospect evaluation should be carried out, the answer is CONTINUALLY. Preferably, as each element of new data is acquired, this should be incorporated into the evaluation. This is probably only possible via the data handling capacities of computers. Once again it must be stressed that head office mainframe computers are not the answer as 'continually' may even mean on a daily basis.

The techniques of prospect evaluation by means of risk analysis that are discussed and employed in this section are widely applicable to almost any form or type of mineral deposit or prospect. However, in order to meet the principal aim of this dissertation, namely that of using a risk analysis technique to study geological influences, the particular computer program that is developed is specifically tailored for copper-silver base metal deposit types. In much of the discussion that follows, general principles are first outlined and then illustrated with reference to this particular simulation model.

INVESTMENT ANALYSIS.

SETTING THE SCENE.

"Find a deposit of at least 10 million tonnes at a grade of at least 2% copper" - a not uncommon example of the type of charge issued to the exploration geologist. It is not difficult to illustrate that requests of this nature without further qualifications are of minimal value, even should just such a deposit be located the following day. To focus on but a few of the geological reasons why this should be so, we compare two deposits, both of which satisfy the requirements of grade and tonnage and both of which are equally favourably situated with respect to all geographical and political factors.

The first deposit has a strike length of 800 m and a depth extent of 400 m. It is 10 m wide, regular, continuous, vertical and has well defined

geological boundaries. The density is constant at about 3 g/cm³ and the ore mineral is bornite. The grain size of both ore and gangue minerals is coarse. A well developed zone of secondary enrichment exists and the depth of oxidation is minimal. Ground conditions are excellent ... etc..

The **second** deposit has an identical strike length of 800 m. However, the depth extent is 1200 m. It is about 3 m wide on average but discontinuous and highly irregular in shape. It has variable dips that range from near vertical to near horizontal and is further disrupted by numerous medium scale faults. The density is similar to that of the first deposit but more variable. The ore mineral is fine-grained chalcopyrite in addition to which economically insignificant but metallurgically problematical amounts of sphalerite and galena occur as small inclusions. Arsenic and bismuth levels are high. The boundary envelope is not geologically well defined and the situation is complicated by poor ground conditions, talcose shear zones and ... etc..

This overplayed "economic viability assassination" of the second deposit is intentional as it leaves no doubt as to the vast differences in the deposits that have the some tonnage-grade characteristics. However, ALL deposits differ geologically and herein lurks the greater danger :

THE GEOLOGICAL ATTRIBUTES OF DEPOSITS NORMALLY INCLUDE MANY HIGHLY SIGNIFICANT CHARACTERISTICS OF A SUBTLE (AND NOT SO SUBTLE!) NATURE THAT MAY EASILY BE OVERLOOKED SIMPLY BECAUSE OF THE NON-RECOGNITION AND NON-APPRECIATION OF THEIR SIGNIFICANCE.

To whom then should be given the charge of the final evaluations and feasibility studies? Solely the mining engineer?, the metallurgist?, the economist-accountant!?. Surely a team effort is called for that must be spearheaded by the geologist, at least until such time as he is **COMPLETELY SATISFIED** that the ultimate decision makers are **FULLY AWARE AND HAVE TAKEN COGNISANCE OF ALL OF THE SIGNIFICANT GEOLOGICAL FACTORS**. Only then should he hand over and leave the subtleties and niceties of the various financing alternatives, the blasting patterns, the size of store holdings etc. to others for more detailed consideration.

REVENUES, COSTS AND CASHFLOWS.

The economic outcome of a mineral investment alternative is described by the actual cash revenue and costs which are anticipated if the alternative is undertaken. Alternatives embody both revenues and costs during one or more fixed time periods. Estimates of revenues and costs are normally combined for each period of time.

Cash Flow is the difference between Revenues and Costs for a specific time period. For evaluation purposes, an annual period is suitable and usually used.

Thus: $\text{Annual Cash Flow} = \text{Annual Revenue} - \text{Annual Costs}.$

If the annual revenues exceed the annual costs, the net revenue is referred to as a **positive cash flow** whereas if the reverse is true, a **negative cash flow** results.

The cash flow estimates form the basis of any **feasibility, sensitivity and risk assessments** and determine the quality of the results.

Revenue accrues mainly from mineral or metal product sales as well as from miscellaneous equipment or plant salvage and sales. For our purposes, only revenue from product sales need be considered. This is simply the product of the metals recovered and the prevailing metal prices. The amount of metals recovered is in turn a direct function of the available in situ metal grades, the tonnage processed and the mining, extraction and refining efficiencies. The metal prices are a complex function of supply-demand, politics and exchange rates. It is important when forecasting metal prices to do so in local currency terms, thus automatically incorporating exchange rate forecasting.

Costs may be distinguished as **Capital Expenditure, Operating Costs or Taxation Payments**. Capital expenditure is very much a function of the scale of the venture and both Capital Expenditure and Operating Costs are complex functions of the geographical and geological parameters. Taxation is as it were a direct function of geography - where it is located dictates as to who rules the country and makes the decisions on taxation levels.

Details of methods and procedures used for estimating revenues and costs are

included in SECTION C.

CONSTANT, CURRENT AND REAL TERM CASH FLOWS.

In order to measure the economic performance of an investment alternative from the cash flow estimates, the cash flows must be expressed in **CONSTANT** money terms. This means that neither prices nor costs are escalated over the cash flow period, but are held constant at year 0 value. This method of ignoring both cost escalation and commodity price inflation may lead to slightly distorted values of calculated returns, especially if the rate of price inflation differs from the rate of cost escalation. For this reason, it is more correct to calculate the cash flows in **CURRENT** money terms, allowing for inflation and escalation, and then to deflate the resulting cash flows to reflect the various cash flows in **REAL** money terms, in other words, in the same terms as at the year of inception.

For our purposes, the small differences that can occur by ignoring inflation are not significant and all of the analyses carried out in Section C are done so in constant money terms, with 1980 set as the base year.

MEASURES OF ECONOMIC PERFORMANCE.

There are a number of different criteria for defining economic returns on mining investments. None of the methods is suited to all situations and it is preferable to test alternatives on the basis of multiple criteria.

All of the methods that are employed are based on some or other form of analysis of the Cash Flows. A ten year period of analysis (10 cash flow periods) is normally adequate to calculate the return on investment. This is because most of the more useful techniques are based on the **time value** of money and employ a compounded discount to successive cash flows, so that by the time the cash flow periods exceed 10 years their effect on the analysis is normally very small. Standardization on the number of cash flow periods also makes for easier and more meaningful comparison between alternatives. It is common practice to record all of the preproduction cash outflow (no revenues at this stage) in constant money terms at inception or year 0.

Production cash flows are also estimated in constant money terms, or as previously discussed, in current money terms that must first be corrected to real money (deflated) terms before any of the measuring criteria are applied.

As example, an investment alternative may indicate the following cash flow situation.

INCEPTION - YEAR 0 (pre-prod. CAPITAL)	- R30 000 000
YEAR 1	5 000 000
YEAR 2	5 000 000
:	:
YEAR 10	5 000 000

R30 million is invested, and R50 million is accrued in equal installments over a period of 10 years. The main criteria for defining returns on the investment as represented by the cash flows are described with reference to this example and are as follows:

Payback Period indicates the time required to recoup the initial investment capital. Thereafter, average returns per annum on the investment are calculated. In the above example the payback period would be 6 years and thereafter the annual returns on investment would be constant at 16,67%. Sole use of this criterion suffers from the setback that it does not take into account the time value of money.

Net Present Value (NPV) and the associated measures of Present Value Ratio and Potential Capital Gain. The Present Value of each of the cash flows, that is, their would be value at year 0 (inception), may be computed by compoundly discounting successive cash flows by some specified discount rate. The NPV is then the sum of the present values of all of the cash flows so discounted. Should the NPV be positive, then the Internal Rate of Return (IRR) of the investment alternative exceeds the chosen discount rate. Should the NPV be negative, the reverse is true. In our chosen example, the following NPV's have been calculated:

NPV @ 5% = + R8 198 738
NPV @ 10% = + R 657 123
NPV @ 15% = - R4 266 223

This indicates that the investment alternative would meet and exceed required discounting rates of 10% or less. The IRR is indicated as slightly larger than 10%.

Discounted Cash Flow (DCF) or Internal Rate of Return (IRR) is that rate of discount that causes the negative and positive cash flows to be exactly equal and thus to cancel. Thus, the NPV at a discounting rate equal to the IRR would be zero. In the example, the IRR is calculated to be 10.56%. It should be noted that the return indicated (the 10.56%) is obtained **only** on that part of the capital investment not yet repaid from year to year.

Wealth Growth Rate (WGR) and External Rate of Return (ERR) both require that the cost of capital to the investor and the reinvestment rate available to the investor be known, before either can be calculated. Returns are computed assuming that cash surplus generated from the investment is reinvested at the investor's average reinvestment rate.

It is not intended to further discuss or expand on the various criteria and their merits and demerits as detailed discussion here is not necessary for an appreciation of the effects of geological parameters on investment alternatives. The factors are however important when **ACTUAL** decisions need be taken and the reader is referred to Mackenzie (1982) and Krige (1979,1983) as selected references offering more details.

A very important point to note however is that so far none of the measures mentioned give any indication as to the size of the investment. An initial investment of R30 with ensuing cash flows of R5 for 10 years would yield **EXACTLY** the same results as those for the figures used, that were in millions of rands. For this reason it is necessary to use some quantity to measure the expected size of the investment.

The following criteria have been chosen to form the basis of many of the

comparisons of alternatives as presented in SECTION D. IRR is used as the one parameter in conjunction with the TOTAL POTENTIAL REVENUE (TPR) of the investment alternative calculated over the same time period as that used for the IRR. For mining investments, the Total Potential Revenue as here defined is as follows:

$$\text{TPR} = \text{Expected tonnage processed} \times \text{expected recovered metal(s) grade} \\ \times \text{expected metal(s) prices.}$$

IRR is plotted against TPR as shown in figure B-1 and different alternatives may be plotted and compared on the same set of axes.

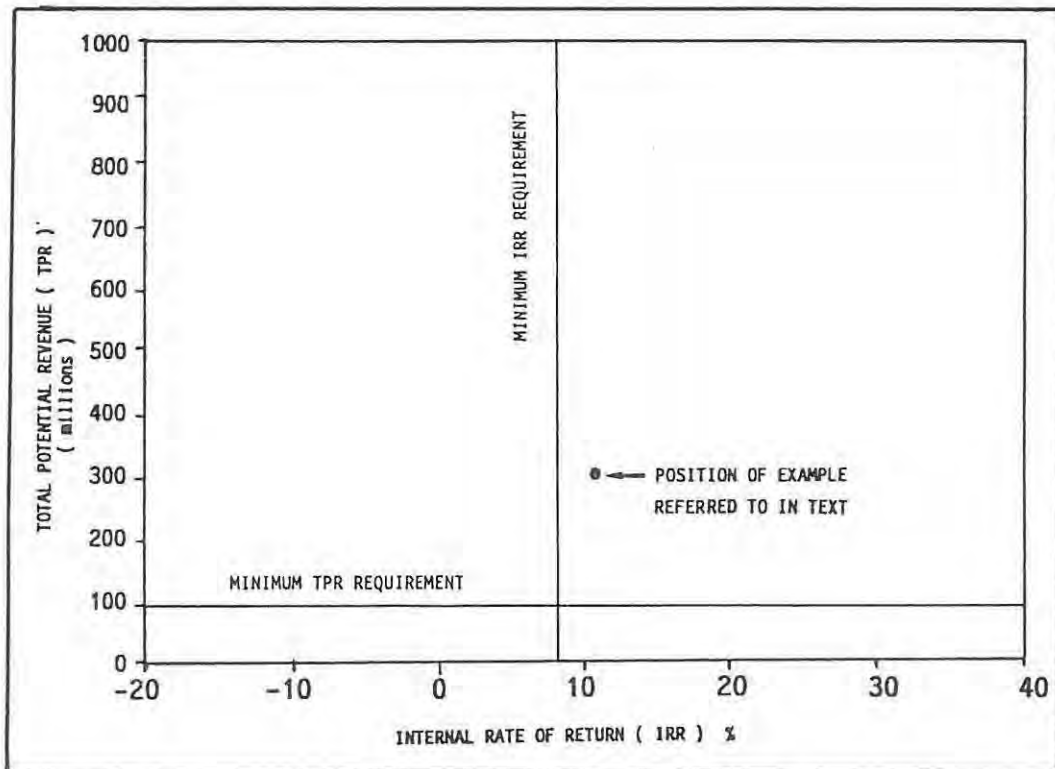


FIGURE B-1. Internal Rate of Return vs Total Potential Revenue. The plot provides a useful framework for comparing investment alternatives.

MINIMUM REQUIREMENTS.

We have seen that a minimum requirement specified in terms of tonnage and grade does not provide a sound basis for assessment. A far more meaningful and useful system includes a minimum threshold IRR on the one hand and a minimum TPR on the other. Example minimum requirements specified in these terms are plotted in figure B-1 and represent a minimum IRR requirement of 8% and a minimum TPR of R100 million. The position of the investment alternative example used earlier has also been plotted on the graph of figure B-1 assuming that it represents a mining investment alternative with a TPR value of R300 million.

The question that immediately springs to mind is HOW SURE ARE WE OF OUR ESTIMATED CASH FLOWS? Phrasing this differently, we may well ask: WHAT IF our price and cost estimates for example are either too high or too low? How CERTAIN are we of achieving the IRR and TPR that we have indicated and what is the RISK associated with our investment alternative?

The what if? type of questions can be answered by a **Sensitivity Analysis** that determines the sensitivity of our cash flows and thus IRR to the various parameters that control costs and revenues. However, sensitivity analysis alone cannot provide us with a quantification of the risks involved, it can only provide us with a ranking of the variables which most influence our analysis. Risk analysis on the other hand, provides us with a quantified measure of the risks involved and as we shall see, automatically furnishes us with a sensitivity analysis.

RISK ANALYSIS.

INTRODUCTION .

Strictly speaking, the term Risk Analysis is something of a misnomer in that a risk analysis model provides the means of evaluating investment alternatives AND quantifies the associated risks involved. It does not only provide a measure of the risks. This means that a risk analysis of an investment alternative provides a measure of the quality of the investment and the probability of achieving any specified requirement.

Before looking at the reasons for opting for a risk analysis model, we look briefly at some of the alternative methods that may be employed to handle risk.

METHODS OF HANDLING RISK.

The following methods have been and continue to be used to handle risk:

(Note that the risks we refer to here exclude POLITICAL RISKS such as nationalization, instability etc.. In such cases, criteria such as increased expected returns in conjunction with shorter payback periods (recoup your investment before the next military coup!) may be subjectively applied).

Conservatism in estimates of tonnages, grades, recoveries, prices etc. - a single set of conservative factors used in conjunction with a conservative measure of return. The method suffers from being extremely subjective with no overall measure of the degree of conservatism or of the remaining risk of achieving even this estimate. Neither does it provide any indication of the upside potential of the investment. It is also most unsatisfactory for the selection from different alternatives.

Single analysis on estimates at likely levels and applying a conservative criteria for returns - same problem as above.

Sensitivity analysis involving a series of cash flows with, in each case, all factors at likely levels except one which is varied over its likely range. This method will rank the risk factors in order of importance but will not quantify the overall risk.

Risk analysis which can provide an overall assessment of risk for the project whichever criteria are applied, and which as we have said previously supplies all the sensitivity analyses as a byproduct.

It is apparent then, that a risk analysis technique is the best suited for handling the risk element associated with investment alternatives. In

particular, in recent times significant increases in the risk element associated with exploration and exploitation of new mining ventures have taken place. These are due *inter alia* to escalation of uncertainties associated with government policies and intervention across a wide spectrum of significant factors. Then too, investment decisions must be taken in a climate of unstable international currency exchange rates and differential inflation rates.

Krige (1979) has noted that against the background of uncertainties, some of which cannot be quantified even subjectively, it is essential that critical (ALL?) decisions on capital investments in new mining ventures should be taken only after at least a full assessment has been made of the overall risk associated with those uncertainties that can be quantified on some reasonable basis. He further notes that apart from subjective judgement, it is unlikely that there will ever be any alternative to risk analysis for this purpose.

OBJECTIVES AND APPLICATIONS.

The main objective of risk analysis is to provide the decision maker with as realistic as is practical a simulation of what could be expected to happen if the project is proceeded with. This will assist in the decision making process on:

- * the viability of the project and the chances of success as opposed to the risk of failure,
- * the 'best' mining plan, i.e. scale of production, sequence of mining, product form etc., and
- * the 'best' scheme for the capitalization of the venture.

To reiterate, our aim is to use a risk analysis simulation to assess the role of geological parameters on factors such as 'best' mining plan, viability of project etc..

BASIC PRINCIPLES.

The basic principles of risk analysis are in fact rather simple. The method and procedure is diagrammized in figure B-2. At the heart of any risk analysis is the computation of estimated CASH FLOWS. Various INPUT DATA is required to estimate the cash flows. The cash flows together with analysis of these flows in terms of investment return criteria form a portion of the OUTPUT DATA.

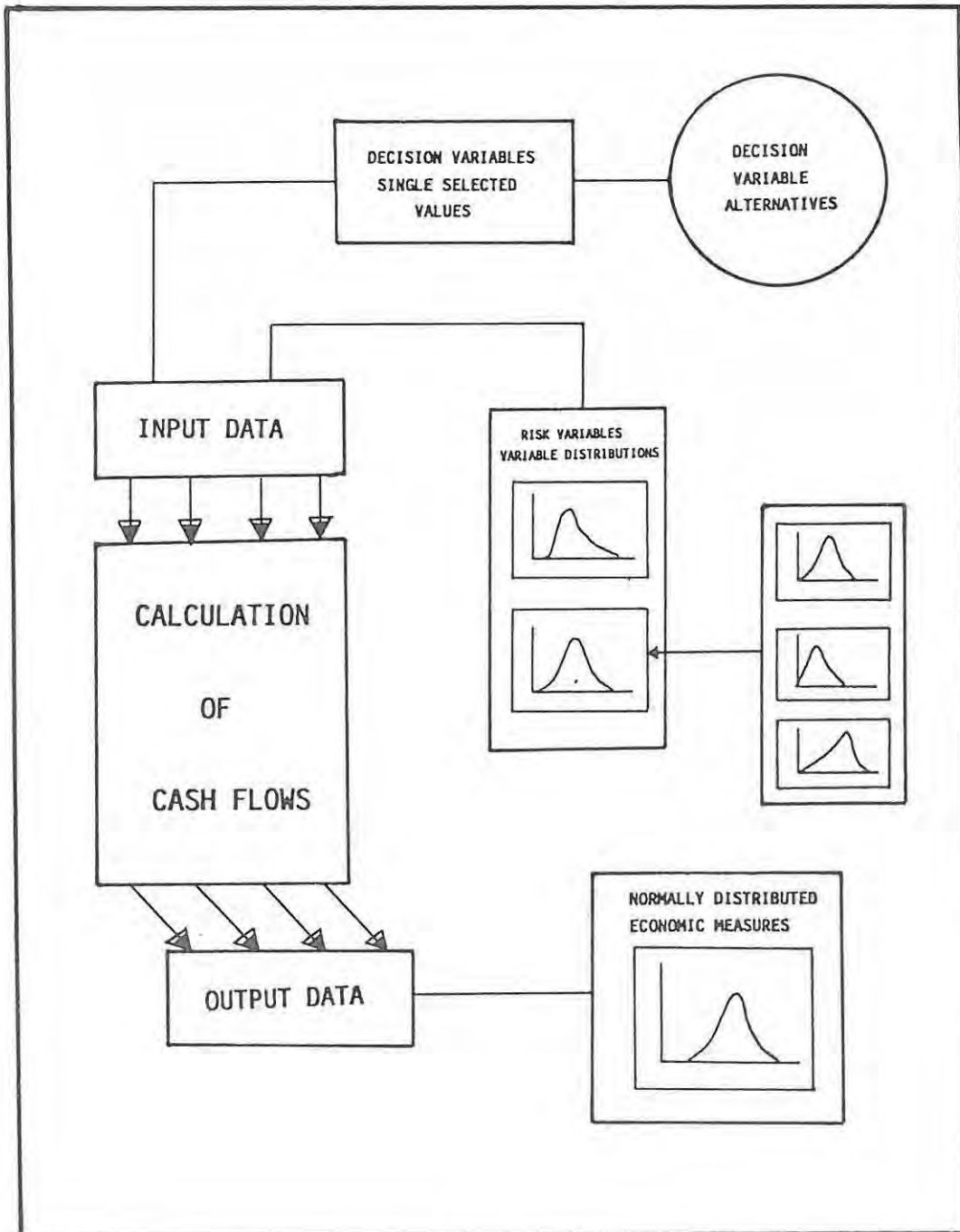


FIGURE B-2. Flowchart illustrating the basic principles of risk analysis.

Input data for cash flow calculation includes both **DECISION VARIABLES** and **RISK VARIABLES**. An example of a decision variable would be the scale of production. At some stage a decision from a number of production alternatives is taken and once the scale of production is finalized it no longer constitutes a variable. Risk variables on the other hand, would include inherent uncertainty factors where uncertainty cannot be removed by decision but may be gradually reduced only with time or in some cases with additional expenditure. Examples of risk variables would include grades, costs, revenues etc.. Some of the risk variables are objectively quantifiable (e.g. grade - requires additional expenditure to be better quantified) whereas others must be subjectively (though hopefully intelligently) estimated (e.g. prices).

Many of the key risk variable input data are themselves functions of a number of uncertainty factors, as is illustrated in figure B-2.

From the cash flow estimates it is then possible to assess the project using the various economic measuring criteria previously discussed (IRR, PAYBACK, NPV, WGR, TPR etc.).

The risk analysis method best employed uses a **Monte Carlo** random selection procedure to choose a particular set of **INPUT DATA** from the various single point input data (in which case the same value is always selected) or from the variable input data. In the Monte Carlo technique, the variable input data are most conveniently expressed in the form of population frequency distributions. These are often normally or near normally distributed for many of the risk variables but may have other distributions including split-normal, log normal or other forms.

Having selected a particular set of input data, a single set of cash flows is then estimated. The procedure is then repeated and a **NEW** set of input data are **RANDOMLY SELECTED** and a new complete set of cash flows is calculated, from which a **new** set of economic measures may be calculated.

The whole process may be repeated indefinitely but normally 50 to 100 iterations are quite sufficient. A feature of a risk analysis is that the results that are produced (e.g. IRR) are themselves usually **NORMALLY DISTRIBUTED** or nearly so (see SECTION D) irrespective of the population frequency distributions of the input data - unless a single particular

non-normally distributed input parameter totally **DOMINATES** the risk assessment. This is **seldom** the case in mineral prospect or mine evaluations.

Methods of analysing risk analysis output data are best described and discussed in SECTION D although we have already seen (in figure B-1) an example of the type of analysis that may be expected. On a plot such as that shown in figure B-1, we are now in a position to illustrate the risks involved by plotting not only the expected "single" likely value, but all combinations of values that constitute possible outcomes given randomly selected possible inputs - see Section D.

FACTORS INFLUENCING ECONOMIC ATTRACTIVENESS.

The main factors that influence the economic attractiveness of a mining investment alternative may be conveniently grouped into Geological and Geographical factors, and Time Dependent factors. Notice that there is considerable overlap and interaction of these groupings and that many of the important variables are influenced by all three of these factors, as represented in figure B-3.

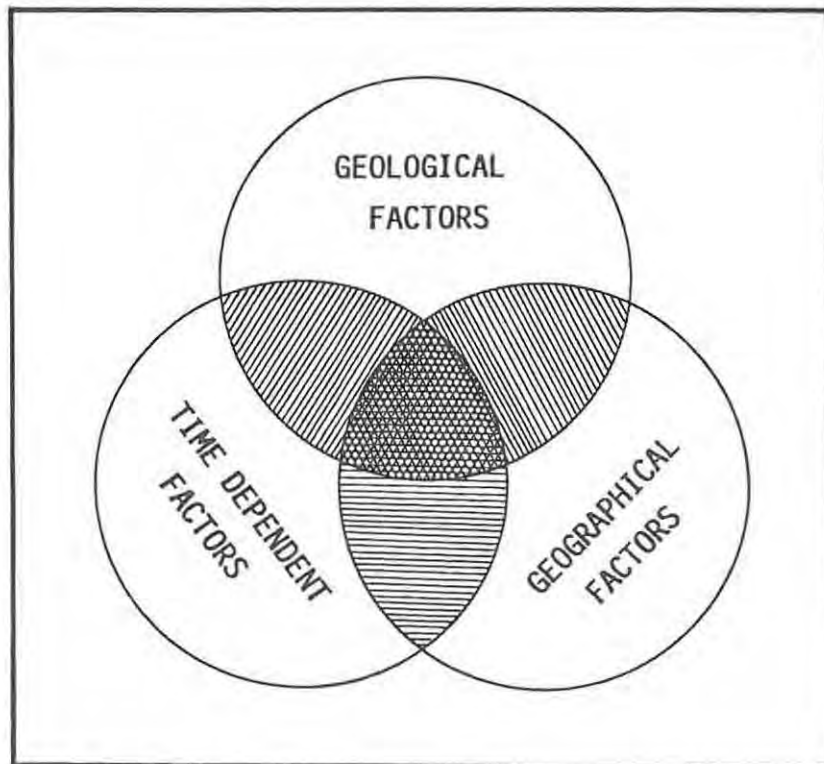


FIGURE B-3. The main factors that influence mining investments.

Ultimately, all of the factors in some way affect either the REVENUES accruing to the project or the COSTS incurred by the project or BOTH. We have seen that the cash flows of a project are generated by subtracting the costs from the revenues over specified time periods, and that these cash flows may then be subjected to economic criteria measures.

FACTORS INFLUENCING REVENUES.

There are perhaps fewer factors that directly influence revenues than those that affect costs and we deal with them first.

Revenue accrues from recovered products x product prices.

Recovered Products.

Recovered products are a function of *in situ* metal grades, and of the mining, extraction, concentration and refining efficiencies. These in turn are all functions of geology, geography and time - to varying degrees.

To illustrate: The *in situ* grade is primarily a function of geology, but the *in situ* grade distribution, which can greatly affect the timing and magnitude of the cash flows may for example be a partial function of climate which is geographically controlled. An example would be a zone of secondary enrichment. The mining, extraction and refining efficiencies are largely functions of geology, but once again the other factors are important and may even dominate. One thinks of time dependent advances in technology, or once again, changes in climate. Metallurgical extraction efficiencies may be directly affected by temperatures and average annual rainfall, causing for example, the deterioration of sulphides in any stockpiled situation.

Generally speaking, advances in technology have enabled lower quality material to be exploited. There has thus in effect been a balance established between commodity depletion and increased mining-extraction efficiencies that have enabled the mining industry to survive and grow.

Product Prices.

Product prices are often the risk variables that most influence investment decisions. As such, attempts must be made to forecast prices. One of the major problems is that prices tend to be highly variable in the short to medium term, although this variation often follows a medium to long term cyclical trend superimposed on an even longer term static or partial growth trend (in constant money terms). The typical form of price variations applicable to many mineral/metal commodities is depicted in figure B-4.

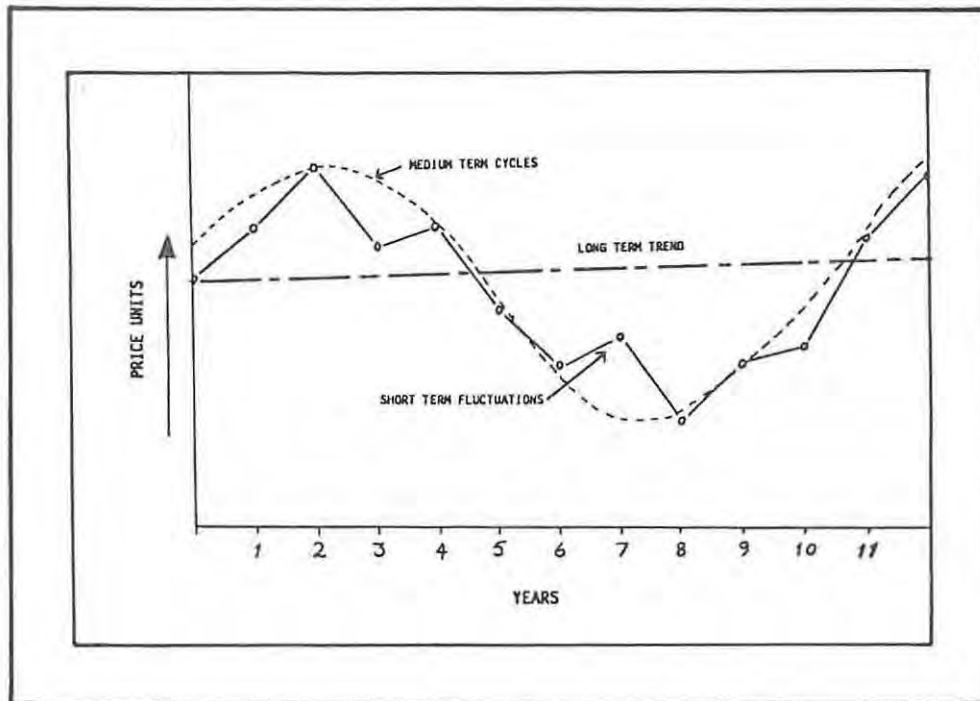


FIGURE B-4. Short, medium and long term price trends typical of many commodity metals.

Ironically, a knowledge of this cyclic nature of metal prices may be advantageous if the approximate position within a cycle can be estimated (which it often can or rather ought to be) and if the mining - milling production schedule is sufficiently FLEXIBLE to ACT on this knowledge. Implications of such flexibility are demonstrated and discussed in Section D.

Note that it is important (if not essential) that commodity price forecasting be carried out in local currencies, in other words, in the same currency as is used to pay for costs.

One method of price forecasting, based on long term constant price trends and differential inflation rates between the country in whose currency the price is fixed and the country where the venture is undertaken is discussed by Krige (1978). Krige (op. cit.) uses the method to develop a means of forecasting current future prices based on long term constant price estimates and the estimation of local inflation rates. He develops formulae for copper and gold price forecasting for the South African situation - in Rand terms. (See section C for discussion and use of these formulae.)

Two factors are central to this method of price forecasting. They are:

- * that the long term constant price trend fitted to historical data is meaningful and can be extrapolated to the future, and
- * that the local current future price will follow this long term price trend, but be a function of the differential inflation rates of the price fixing and commodity producing countries, and that the necessary local price adjustments will be controlled by the EXCHANGE RATE between the two countries.

In a well-balanced economy (particularly the export part thereof), this method would be expected to work well in the medium to long term, as long as the exchange rates are set by market forces and not by government decree. However, in an economy where the export sector of that economy is dominated by one commodity (e.g. South Africa - gold; Chile, Zambia - copper), the adjustments in the exchange rates will be dictated by the prevailing world price of that single commodity.

Nowhere is this better illustrated than in South Africa over the last 10 years or so in particular. The crux of the matter is as follows:

THE FORECASTING OF THE PRICE OF A COMMODITY THAT DOMINATES A COUNTRY'S EXPORTS AND FOREIGN EXCHANGE EARNINGS IN TERMS OF PRODUCER COUNTRY CURRENCY IS RELATIVELY EASY - THE FUTURE PRICE OF THE COMMODITY IN THE PRODUCER COUNTRY IS THAT PRICE IN LOCAL CURRENCY THAT IS NECESSARY TO MAINTAIN REQUIRED LEVELS OF PRODUCTION OF THE COMMODITY IN QUESTION.

This 'necessary price' may be conveniently (and surreptitiously?) set by the exchange rate, or may 'effectively' be set by some or other form of subsidy

to that industry. That this is so is illustrated in figure B-5, showing the dollar and rand gold prices as well as the rand production costs of gold over the last 15 years or so.

Why this digression? The reason is that although the risk of forecasting the price of the dominant export commodity is effectively reduced, that of forecasting prices of subsidiary commodities is greatly increased. For example, to forecast the rand price of copper for the next 10 years one need in addition to forecast the gold price!

A large differential inflation rate between the producer country and the commodity price fixing country (as currently prevails with South Africa as the producer country) coupled with an increasing gold price in current dollar terms and a less rapidly increasing copper price in pound terms, is for example doubly bad news for the copper industry in South Africa in particular.

FACTORS INFLUENCING COSTS.

For this discussion, we are most interested in the geological factors that influence costs. It is however, as we shall see, impossible to assess these influences independently of geographical and time related factors. In order then to most clearly highlight the geological factors, it becomes necessary to FIX or CURTAIL the range of variance of the geographical and time related contributions to the total assessment. This approach is adopted in Section D when actual results of running the risk analysis simulation with changing geological input parameters are analysed.

The different geological factors influence the costs incurred in different phases and areas of the development, mining, extracting and refining process. The cost areas that can be affected are illustrated in Pie Chart form in figure B-6.

The extent to which the geological factors affect the different cost areas may be portrayed by the size of the pie slices. This provides a method of integrating and illustrating the geological-geographical-time dependent interdependence. Specifically:

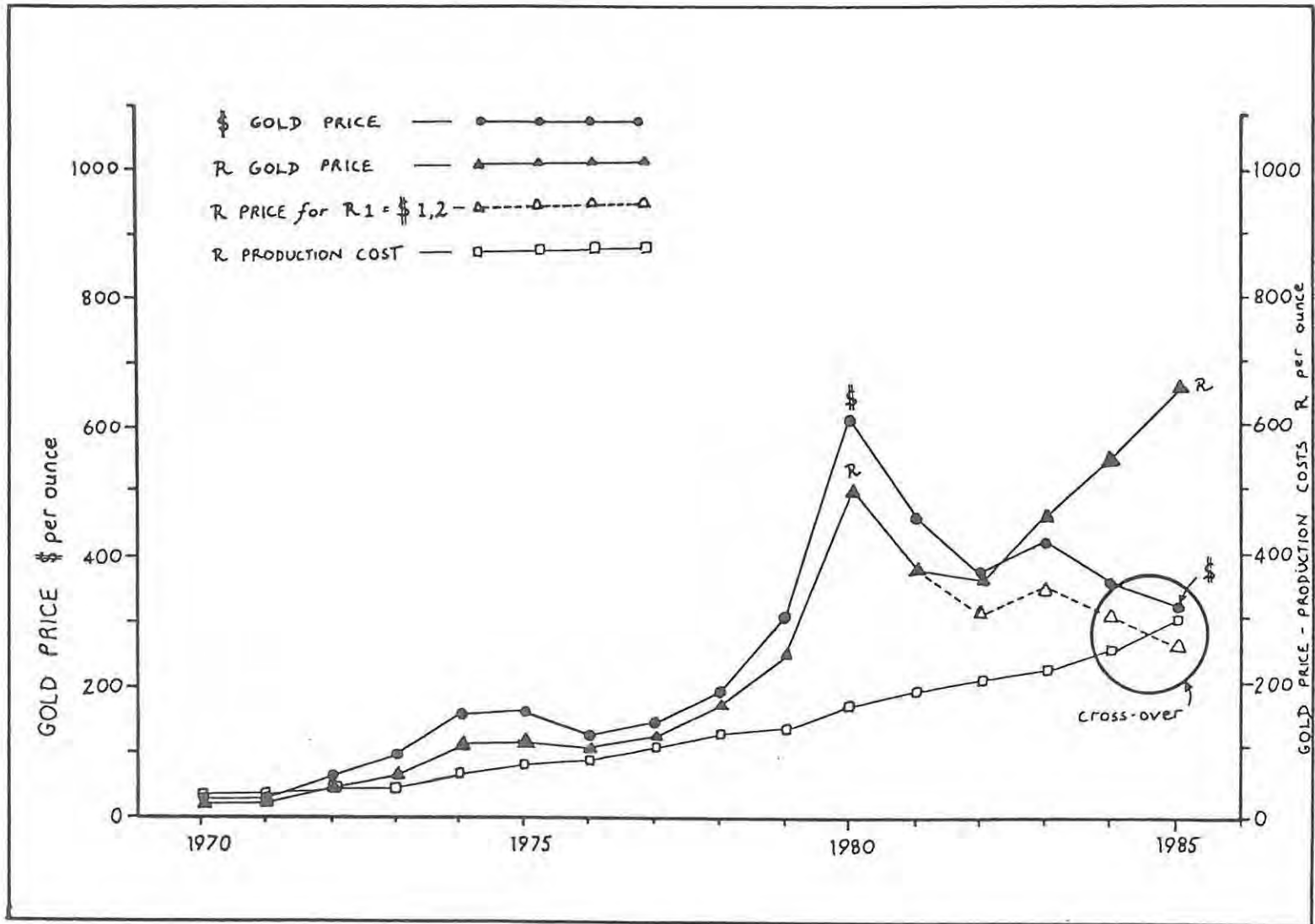


FIGURE B-5. Dollar and rand prices and production costs (rand) per ounce of gold over the last 15 years. The dollar/rand exchange is averaged as R1 = \$1,2 for the period 1970 - 1981. The dashed line joining open triangles shows the rand price had this rate of exchange been maintained. Notice that production cost figures are based on a grade of 6,6 g/t and should in fact be slightly lower in the 1970's when grades were generally somewhat higher.

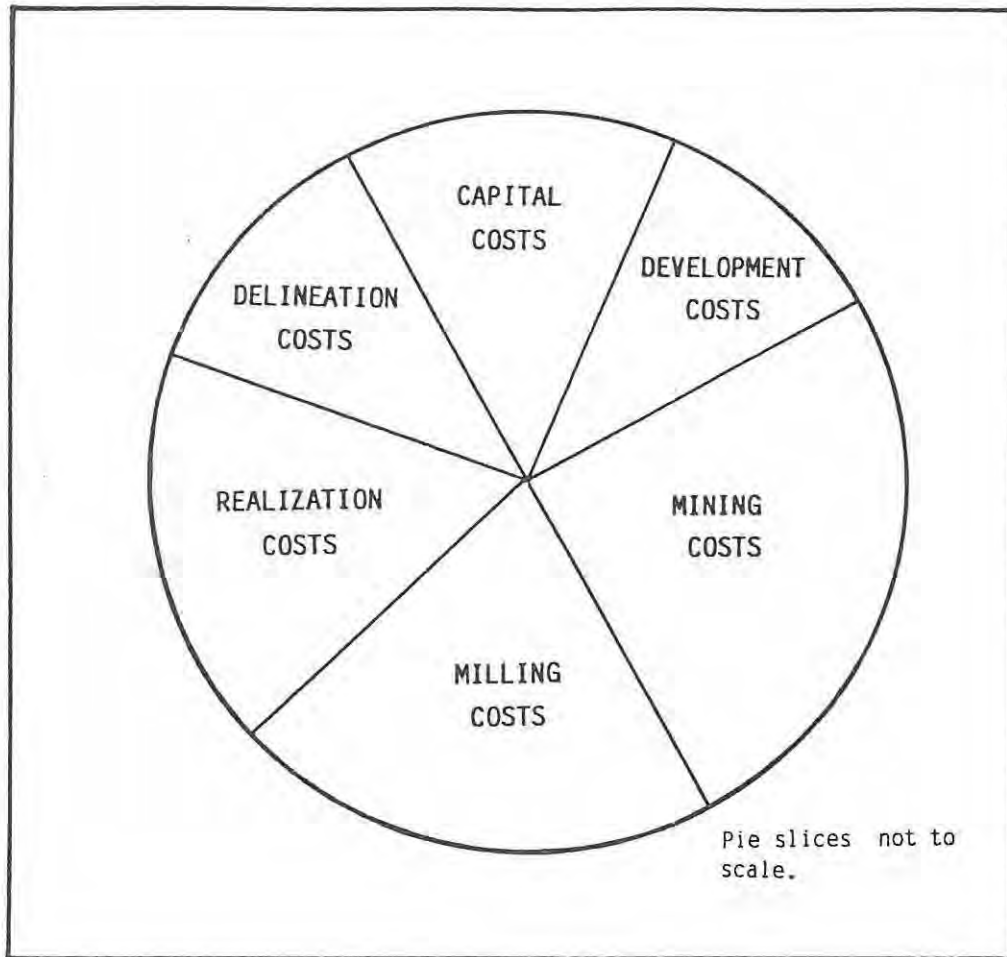


FIGURE B-6. Cost areas affected by geological factors.

- * given identical geographical and time dependent factors, geologically different deposits will be reflected by different shares in the pie, and
- * given geologically identical deposits, different geographical and time dependent situations will be reflected by different shares in the pie.

To illustrate the first case:

Two otherwise identical ore deposits produce at different grades of concentrates, say 20% Cu as against 40% Cu. Given identical geographical and time related factors, the realization costs related to concentrate handling and smelting would be double in the former case as compared to the latter.

To illustrate the second case:

Two identical ore deposits produce concentrates of identical grade. The one deposit is located at twice the distance from the smelter as the other. In this case, the realization costs slice of the pie will be greater for the latter than for the former and the latter project will be more sensitive than the former to any changes in concentrate grades or transport tariffs.

Hundreds of examples similar to these two may be theoretically constructed. Hence, in a theoretical approach, the various permutations are endless. However, in real practical examples, the concentrate grade IS found to be say approximately 30%, and once ascertained, is unlikely to change very much (unless it were poorly ascertained!). Only the certainty with which it has been ascertained can generally be improved upon.

Table B-1 lists some of the more important geological factors that may be expected to influence the various costs incurred during delineation, mining, extracting and refining products. An attempt is made to illustrate the ORDER or MAGNITUDE of these effects. $I_1 \dots I_3$ represent increasing magnitude of increased (I) costs and $D_1 \dots D_3$ represent increasing magnitude of decreased (D) costs. The table is not ranked in any specific order as we have already seen that the ranking of the influence of geological parameters is not only orebody dependent, but also a function of geography and time. Some of the effects of changing these geological factors both in fixed and in variable geographical-time frameworks are examined in Section D.

To end this section, an important comment or conclusion: It may well be the case (and is surprisingly often so) that any one or group of geological factors may so dominate the costs as to occupy almost all of the costs pie. It is thus imperative and should be one of the first priorities of ANY evaluation to be alert to this possibility and to identify such a situation - AS EARLY AS POSSIBLE. An obvious example that springs to mind is that of a bonanza of a deposit in all respects other the lack of a technically-economically feasible method of producing single metal concentrates due to complex and fine-grained ore mineral exsolutions and intergrowths. Other such examples exist!

TABLE B-1. The effect of different geological factors on the different cost areas.

GEOLOGICAL FACTORS	COST AREAS								
	DELINEATION COSTS	PRE-PRODUCTION COSTS	MINE CAPITAL COSTS	MINE DEVELOPMENT COSTS	MINE OPERATING COSTS	MILL CAPITAL COSTS	MILL OPERATING COSTS	OFF MINE REALIZATION COSTS	
increasing SIZE	I ₃	I ₃	I ₃	D ₂	D ₃	I ₃	D ₃		
increasing STRUCTURAL COMPLEXITY	I ₃	V ₁	V ₂	V ₃	V ₂				Either primary or secondary, could decrease costs if favourable eg. fold duplication.
more complex SHAPE	I ₃	I ₁	I ₁	I ₃	I ₃		I ₂	I ₂	These factors could affect the milling costs if more waste dilution occurred, especially if the waste contained constituents deleterious to the milling - floatation process.
less CONTINUITY	I ₃	I ₁	I ₁	I ₃	I ₃		I ₂	I ₂	
less REGULARITY	I ₃	I ₁	I ₁	I ₃	I ₃		I ₂	I ₂	
narrower WIDTH	I ₃	I ₁	I ₂	I ₃	I ₃				
changing ATTITUDE	V ₃	V ₃	V ₃	V ₃	V ₃				Major effect on all of these, either increasing or decreasing costs depending on what the attitude is!
less well defined BOUNDARY	I ₁			I ₁	I ₁				Could increase development and operating costs as more sampling / drilling would be necessary to define the orebody.
poorer GROUND CONDITION	I ₁	I ₁	I ₂	I ₃	I ₃		I ₂	I ₂	
less favourable TOPOGRAPHY	I ₁	I ₁	I ₁		I ₁	I ₂	I ₂		Affects roadways, access, tailings disposal etc..
less favourable GROUND WATER CONDITIONS		I ₁	I ₂	I ₃	I ₁		I ₂		More complex flotation circuits would be required if more material was oxidized.
Increasing VARIABILITY OF ORE DISTRIBUTION	I ₁		I ₁	I ₂	I ₁	I ₁	I ₁	V ₃	May well increase milling costs but could be beneficial overall in allowing for sequencing and specific timing of cash flows.
HARDER material			I ₁	I ₁	I ₁	I ₃	I ₃		
more ABRASIVE material			I ₁	I ₁	I ₂	I ₁	I ₃		
less favourable CONDITION OF ORE MINERALS						I ₁	I ₃	I ₁	
less favourable NATURE OF GANGUE						I ₁	I ₃	I ₃	
Increasing TENOR of ore						I ₁	I ₂	I ₃	
Increasing METAL CONTENT of ore minerals						D ₁	D ₂	D ₃	
increasing ORE IMPURITIES						I ₁	I ₃	I ₃	
increasing ORE TEXTURAL COMPLEXITY						I ₂	I ₃	I ₃	
smaller GRAIN SIZE						I ₃	I ₃	I ₁	
POLYMETALLIC						I ₃	I ₃	I ₃	

I = Increased costs, 1 ... 3 = increased magnitude of effect.
 D = Decreased costs, 1 ... 3 = increased magnitude of effect.
 V = Variable costs (I or D), 1 ... 3 = increased magnitude of effect.

SECTION C - SETTING UP THE RISK ANALYSIS ON A MICROCOMPUTER.

NOTE.

At this juncture, the sceptic would be excused for thinking:

"How can all of these variable parameters be meaningfully handled by a head office team of investment analysis 'experts', let alone the on-site or regional office based exploration geologist?"

I intend to illustrate, in reply to this very valid question, that a useful and meaningful system of analysis can and should be implemented, and that it is the on-site geologist who should implement such a system. Although it cannot be claimed that use of such an on-site system will provide all of the answers to allow for all of the 'correct' decisions to be taken, it is claimed that the use of such a system MUST be an improvement on taking forced or purely hunch-inspired decisions. In addition it will provide the geologist with an added degree of confidence in his decision taking, as well as increasing the level of professionalism and professional satisfaction.

INTRODUCTION.

This section introduces the workings of computer spreadsheets. The manner in which the risk analysis program has been developed on such a spreadsheet is described. The methods and assumptions used to calculate or select the various variable and fixed data inputs are detailed. From this input data, the pre-production capital costs are calculated and the production cash flows are estimated over 10 years in annual periods. The nature of the output data is described and discussed and finally the technique used for multiple iterations or runs of the risk analysis simulation is described. Output data handling and interpretation forms the basis of the discussion in SECTION D.

WHAT IS A SPREADSHEET?

A spreadsheet is structured as a grid, similar to a large sheet of graph paper. The user can enter a number, formula or text (figure C-1) into any

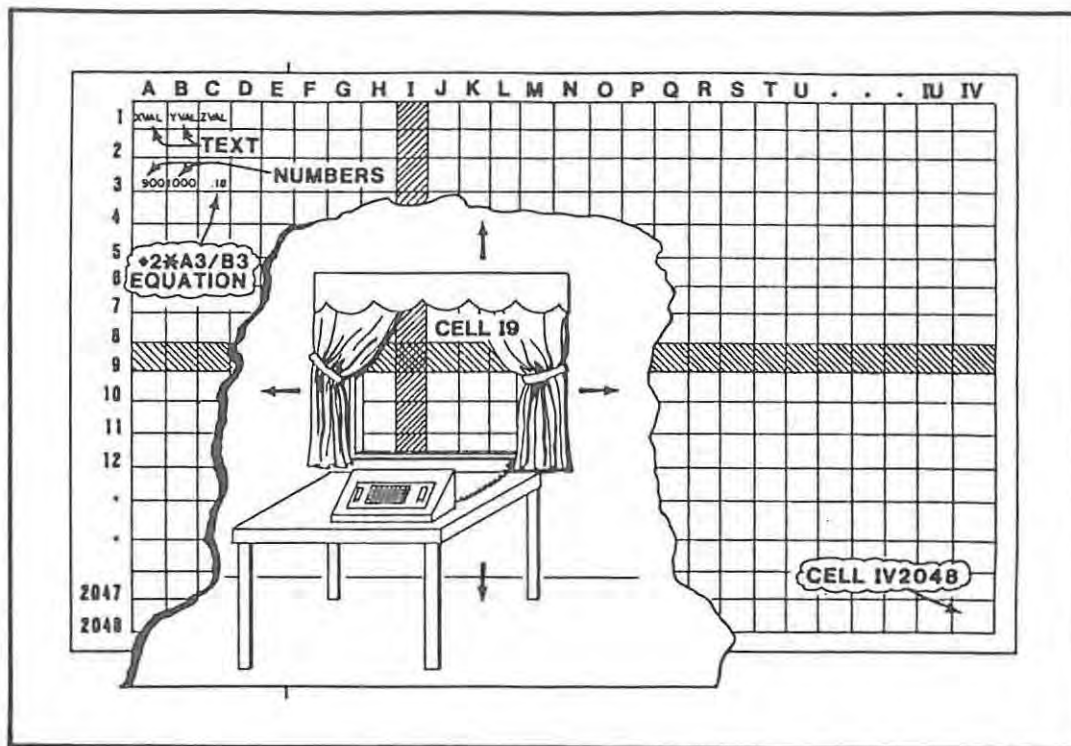


FIGURE C-1. Cartoon sketch illustrating the nature of a spreadsheet. (after Smith, 1985).

particular grid location or "cell". A highlighted CURSOR indicates the position of the active cell. The cursor is positioned with arrow keys. All cells have a location identified by columns (A, B, C, D ... IT, IU, IV) and rows (1, 2, 3, 4 ... 2046, 2047, 2048). The screen shows only part of the spreadsheet at any time, usually about 8 columns by 20 rows, depending on the selected column width. It is similar to viewing a part of a large sheet of graph paper through a window, and moving the window over the area of interest.

There are several different spreadsheet software packages available, including Lotus 1-2-3 (the one used), Supercalc, Multiplan, Framework, Abacus and many others. Most of them are very similar in operation but may vary in grid size, speed of calculation, flexibility and level of integration with other utilities such as data base and graphing utilities. All programming was carried out using Lotus 1-2-3* on an IBM PC.

* Lotus 1-2-3 is the trademark of the Lotus Development Corporation, Cambridge Massachusetts.

SPREADSHEET OPERATION.

Several factors make computations with the spreadsheet easy and flexible. Lotus 1-2-3 for example is menu-operated with commands that are usually self-descriptive. A help facility provides a listing and explanation of all of the various available commands.

Commands include those that facilitate movement around the spreadsheet (GOTO .. cell position) or for example selective viewing of the spreadsheet by the WINDOW command, which allows the user to place either a vertical or a horizontal 'split' in the screen window at the current position of the cursor. This enables different portions of the spreadsheet to be viewed simultaneously.

The COPY command allows the user to COPY cells or cell ranges that may contain numbers, text or formulae with a few keystrokes. An important feature of spreadsheets is that formulae are automatically copied **relatively**. A formula in cell A3 for example that reads $A1 + A2$ if copied into cell B3, would read $B1 + B2$. Relative copying of formulae can be suppressed by **absolute** copying, specified by a '\$' symbol. In this case, copying the absolute value of cell A3, \$A3, into B3, would yield a value in B3 equal to $A1 + A2$ and not $B1 + B2$.

Using other descriptive commands, 1-2-3 can MOVE blocks of data, and INSERT or DELETE rows or columns. Using the DATABASE utilities, one can for example SORT information with two levels of priority, FIND certain pieces of information or provide frequency DISTRIBUTIONS of data ranges, given specified class intervals.

Macros are commands allowing for more advanced programming of specific keys to do more complex or repetitive tasks. Macros may be used to set up sub-routines, menus, and conditional logic statements. Additionally, macros can be used to program a key to move the cursor, edit, or execute commands.

In addition to simple equations using multipliers or additives etc., functions are available for more complex computations. 1-2-3 has a library of trigonometric, statistical, financial, logic and data functions. Many of these functions are similar to those found in higher level programming languages. Some of the particularly useful functions that are used

extensively in the risk analysis program include:

@ RAND - random number generator, generates numbers between 0 and 1

@ SUM - sums values in a specified RANGE

@ AVG - averages values in a specified RANGE

@ IRR - calculates the internal rate of return for a given range of cash flows

and many more.

Notice that functions are prefaced by the @ key.

SO WHAT?

The real power of spreadsheet programming rests in the fact that the various cell entries can be 'linked' to other cells by simple or complex formulae. Changing the value of a cell that is linked to other cells by such formulae will automatically cause the values in the linked cells to change. In the previous trivial example, cell A3 had the formula $A1 + A2$ entered. If we entered the values 5 and 10 into cells A1 and A2 respectively, cell A3 would register the value 15. Now, if we changed the value in cell A1, say to 15, cell A3 would automatically register a value of 20. More impressively, in the risk analysis 'template' that has been constructed, changing the whole set of input variables (of grade, prices etc.) results in a complete recalculation of 10 years of cash flows, IRR's, etc. in about 3 seconds.

Once the desired 'matrix' or 'template' has been constructed on the spreadsheet, it may be SAVED for future use and RETRIEVED from a disc when required. A further major advantage of a spreadsheet system is that a retrieved template may be rapidly modified and reformatted to suit any particular user requirement. The flexibility and ease of reformatting (for example constructing summary tables, graphs, split windows etc.) simply cannot be rivalled by the more conventional programming languages, as any experienced user accustomed to the formatting headaches associated with say FORTRAN would be the first to admit. Retaylored templates may then be SAVED under separate file names, or output as hard copy using the PRINT command

and all of the associated OPTIONS.

It is not intended (nor is it practically possible) to cover and convey all of the workings of the Lotus 1-2-3 system. Suffice to say that personal experience indicates that a beginner user with NO previous computing experience of ANY form, will be able to use a spreadsheet system immediately, use it constructively in about one day, intelligently in about three days and in an advanced fashion in about one week. It may however, be necessary to use the system for some period of time before the full potential is appreciated. New subtleties, uses and applications will continue to be discovered, as is the case with any powerful programming language or system.

The basic risk analysis simulation used in this dissertation was 'set up' in about one week, although some previous experience of spreadsheets was held (not of Lotus 1-2-3 specifically). Obviously, a lot more time was spent on the PC in producing and analysing the RESULTS of the risk analyses.

THE RISK ANALYSIS TEMPLATE.

LAYOUT.

The layout of the risk analysis template (from now on referred to as RISK\$) is shown in figure C-2. RISK\$ may be conveniently divided into four sections for discussion purposes. These are:

- * AN INPUT section, where requested data must be input and where some of the costs and cost structures are computed,
- * A CASH FLOW calculation grid, where the period cash flows are computed from the input data and various formulae that connect and link this data,
- * AN OUTPUT section, where results of various cash flow analyses or any other factors of interest may be displayed and
- * A WORKING SECTION, where various random numbers or Keyboard Macros are listed.

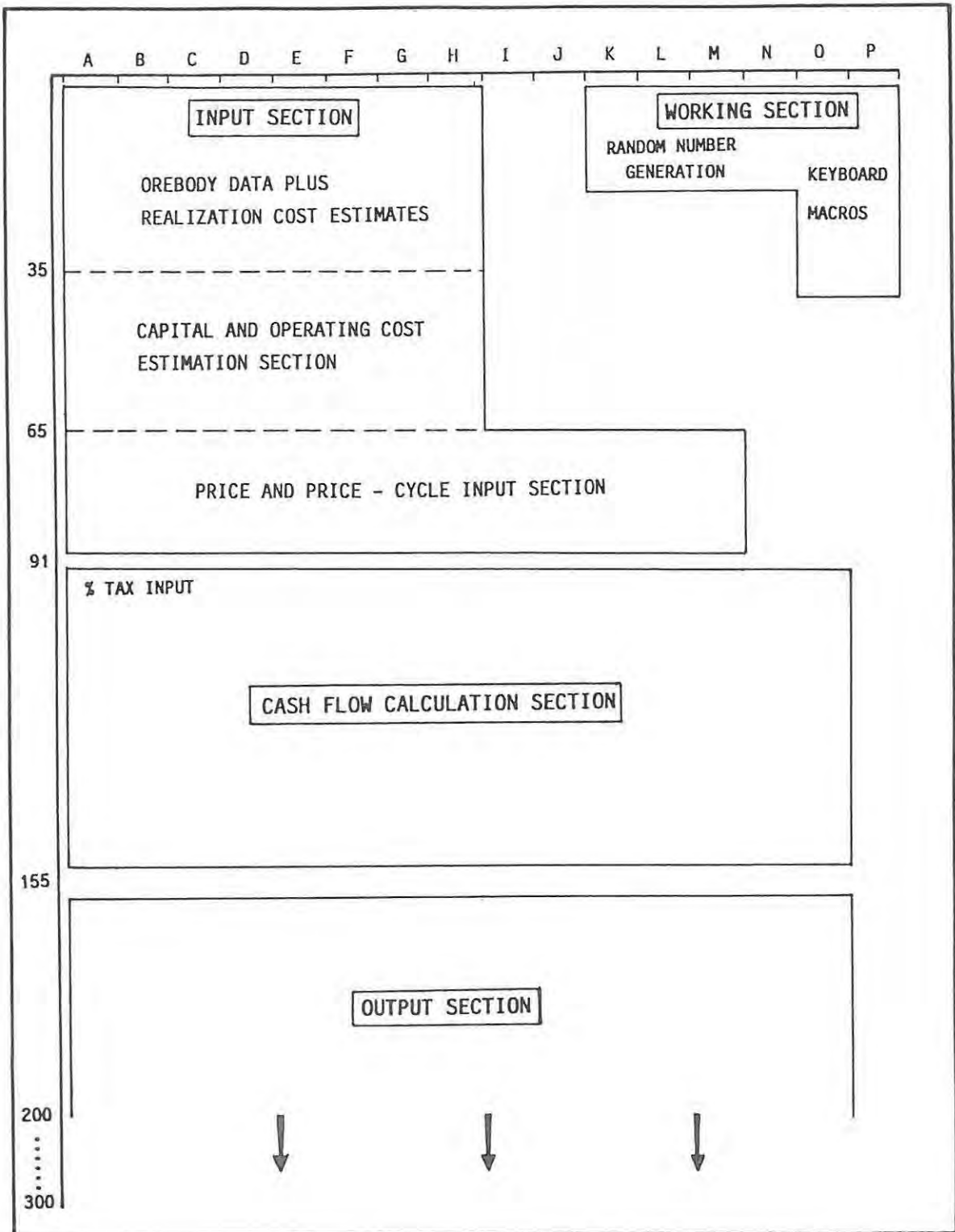


FIGURE C-2. RISK\$ spreadsheet layout.

INPUT DATA.

Various input data is required before RISK\$ can be activated. When RISK\$ is initially retrieved to become the working file, the data that was last entered and saved will occupy the input data cells. As RISK\$ has been set up as a simulation risk analysis, very little actual cost input data is actually required. For example, from an input of the reserve tonnages, the system automatically selects an annual capacity tonnage (T) - see later. Likewise, selection of a particular mining method, together with deposit size and shape parameters such as width (W) and depth of shaft (S), allows for the automatic calculation of the expected mining capital and production costs.

The required input data falls into two categories:

- * single value data and
- * variable value data.

Single value data simply require a single value input. These single values will of course vary with conditions, location and time. They are however usually fixed for given conditions of known location at a fixed point in time. An example would be railage costs per kilometer per tonne of concentrate.

Variable value data on the other hand require expected mean values and standard deviations from these means. In RISK\$, ALL of the variable input data are assumed to be normally distributed, so that a mean and a single standard deviation from this mean is all that is required as input. RISK\$ could easily be modified to accept either split-normal or lognormal input data but this facility was not deemed necessary for the main purpose of this dissertation: to use RISK\$ to analyse geological factor influence. It should be noted however, that the metal grades of many ore deposits are lognormally distributed. Although this is so, the variance of the MEAN of such lognormally distributed populations may for many practical purposes be accepted as being near-normally distributed, although it would actually be 'slightly' lognormal.

The cells into which the required orebody specific data (mostly risk variables) and the various costs related to realization charges must be

entered are shown in figure C-3, which is a direct PRINT from RISK\$ and appears as it would on the computer console.

	A	B	C	D	E	F	G	H
1	RISK ANALYSIS OF BASE METAL (COPPER PLUS SILVER) PROSPECTS							
2	=====							
3	Estimates of the following parameters must be entered.							
4	Where necessary, mean values and standard deviation should be entered.							
5	In some cases these will have been calculated from available data, in							
6	other cases "best" estimates must be made.							
7	*****							
8	***INPUT DATA***							
9	*****							
					MEAN	STANDARD		EXAMPLE
					VALUE	DEVIATION		VALUE
						(of the mean)		
11	-----							
12	DELINEATION COSTS	(DC)			1000000	-		1000000
13	MINEABLE RESERVES	(R)			10000000	100000		10158457
14	MINEABLE WIDTH	(W)			10	0.5		10.79
15	COPPER GRADE	(CuGr)			2	0.1		1.98
16	% RECOVERY	(CuREC)			92	1		91.58
17	SILVER GRADE	(AgGr)			10	0.5		9.62
18	% RECOVERY	(AgREC)			90	1		89.70
19	CONCENTRATE GRADE	(ConGr)			35	2		34.91
20	SHAFT DEPTH	(S)			400	10		409.32
21	MINING METHOD	(MM)			1	-		
22	1 = open stoping,							
23	2 = cut and fill							
24	FAVOURABILITY FACTOR (FF)							
25	1= good, 1= bad							
26	-----							
27	REALIZATION COSTS:							
28	(per appropriate unit)							
29	ORE TRANSPORT	(OreT)	RANDS		0	-		
30	CONC. TRANSPORT	(ConT)	RANDS		30	-		
31	CONC. SMELTING	(ConS)	RANDS		75	-		
32	COPPER ALLOWANCE	(CuAll)	%		0.3	-		
33	SILVER ALLOWANCE	(AgAll)	%		35	-		
34	PENALTY UNIT	(PU)	%		2	-		
35	REFINING ETC.	(REF)	RANDS		300	-		
36	-----							

FIGURE C-3. Detail of part of the input section of RISK\$, INPUT data cells are outlined.

RISK\$ uses this data to do two things:

- * randomly select risk variable values from each of the specified distributions and use them for the calculation of costs and cash flows, and
- * calculate the annual capacity tonnage and the capital and operating costs for the chosen set of risk variable values.

Example values chosen from the specified risk variable populations are shown in figure C-3. Figure C-4 is of the next portion of the spreadsheet and shows the annual capacity tonnage and the set of capital and operating costs computed from using the set of example values displayed in figure C-3.

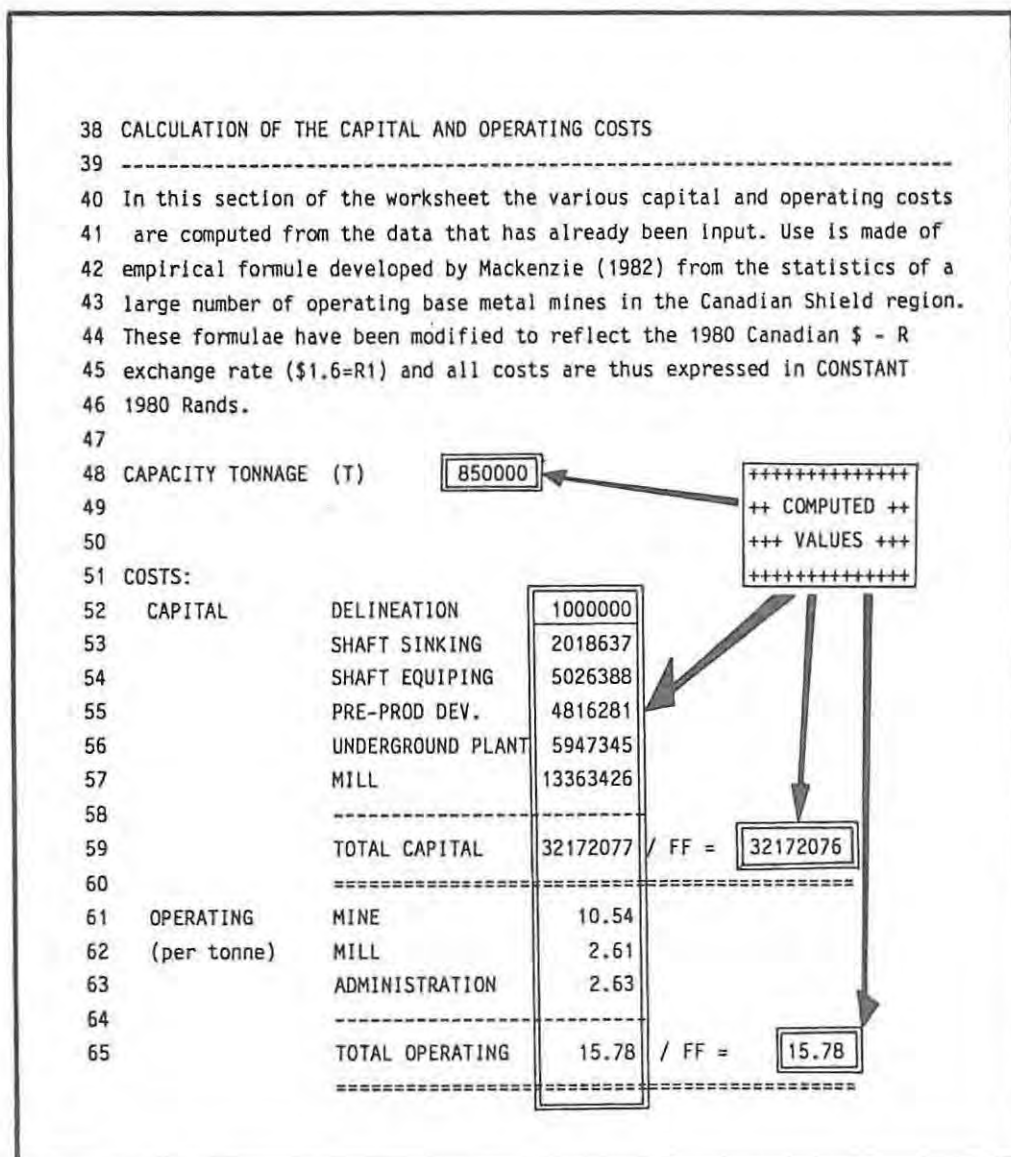


FIGURE C-4. Detail of the cost computing part of the INPUT SECTION of RISK\$.

Random Selection of a Number from a Normal Distribution Population.

The N(0,1) distribution is the standard reduced form of a normal distribution, where 0 is the mean (μ) of the distribution and 1 is the standard deviation (σ). Approximately 68% of a randomly selected set of numbers from such a N(0,1) distribution would lie between -1 and +1.

An element may be randomly selected from the N(0,1) distribution using the following formula:

$$z = \frac{\sum_{i=1}^n u_i - \frac{n}{2}}{\sqrt{\frac{n}{12}}} \dots\dots\dots(1)$$

Where: z is the randomly selected element from the N(0,1) population,
 U_i is a random number between 0 and 1, and
 n is the number of times the summation is carried out.

For practical purposes, n can be set at 12 in which case equation (1) reduces to :

$$z = \sum_{i=1}^{12} u_i - 6 \dots\dots\dots(2)$$

Figure C-5 shows the spread of 100 numbers that have been randomly selected from a N(0,1) distribution.

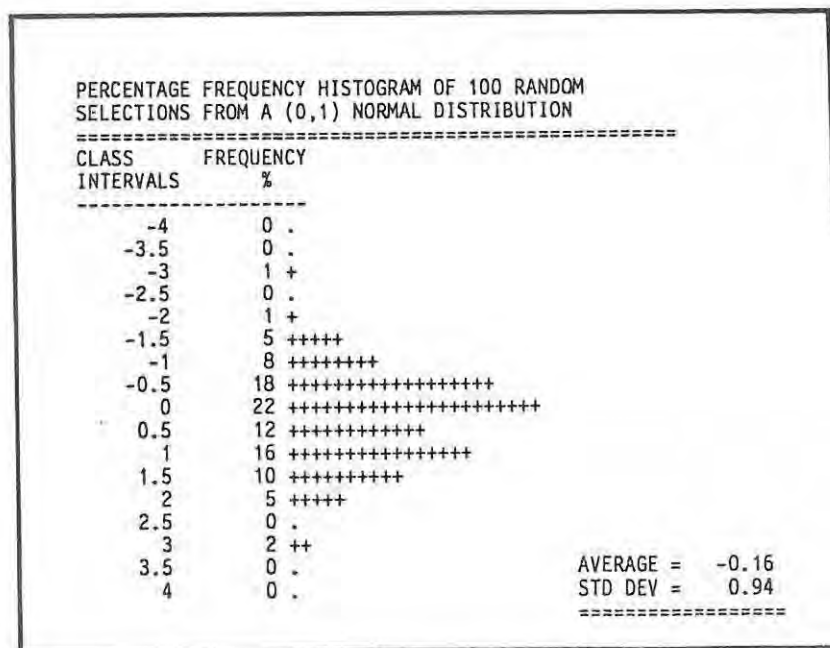


FIGURE C-5. Frequency percentage histogram of 100 random selections from a N(0,1) distribution.

Numbers selected from a $N(0,1)$ distribution may be converted to represent numbers from any specified normal distribution by multiplying the result of equation (2) by the specified standard deviation (σ) and adding the mean (μ). Thus, the equation for selecting random numbers from say a grade distribution with a mean of 2% and a standard deviation of the mean of 0.1% would be

$$z = 0.1 * \left(\sum_{i=1}^{12} U_i - 6 \right) + 2 \quad \dots\dots\dots(3)$$

and the result of 100 such z selections is shown in figure C-6.

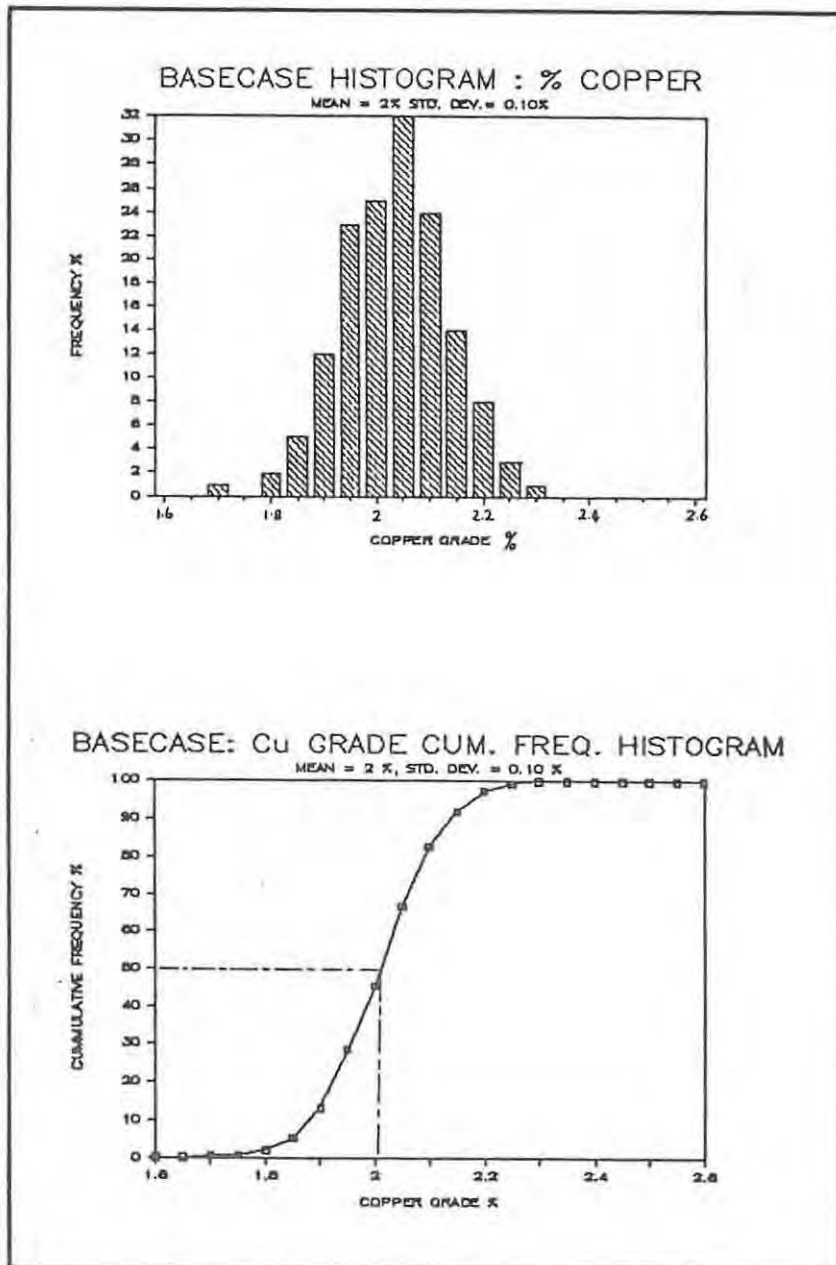


FIGURE C-6. Results of 100 copper grade selections from a normal distribution with a specified mean of 2% and standard deviation of 0.1%.

Equations such as equation (3) can be constructed using the @RAND function available on Lotus and entered into cells on the spreadsheet. This allows for random selections from specific populations to be made automatically each time RISK\$ is executed. The example values shown in figure C-3 of grades, tonnages, recoveries etc. are all selected in this fashion. A re-execution produces a completely random new set of values.

Mining Method (MM) and Favourability Factor (FF).

RISK\$ has been set up to accept two possible mining methods, open stoping and cut-and-fill. These have been chosen as there is a significant difference in the production costs associated with each method, and this is useful in highlighting the importance of parameters such as orebody dimensions, shape and orientation. RISK\$ could be modified to accept other methods such as shrinkage or open cast mining, but too many options would only serve to confuse the main issues.

The method used to calculate the capital and operating costs is based on empirical formulae developed from operating mine statistics and provides useful average cost estimates. The FAVOURABILITY FACTOR (FF) has been incorporated in RISK\$ to make allowance for the many geological factors that are not easily quantified and are nowhere properly reflected in the calculations. It allows for a negative adjustment to the likely average costs if the geological factors are worse than average, and vice-versa. Factors used to assess 'better' or 'worse' than average should NOT include grades, widths, depths etc. as these are already accounted for, but should include factors such as ground conditions, continuity, regularity, faulting, jointing etc., although some of these may have already dictated the mining method to be employed.

Calculation of Annual Capacity Tonnage, Capital and Working Costs.

The annual capacity production tonnage (T) and the capital and operating costs are calculated from the input data for each selected set of variables. Empirical equations derived by Mackenzie (1982) are used. His equations are shown in figures C-7 and C-8. The equations produce results in CONSTANT 1980 Canadian dollars. The equations as used in RISK\$ have been modified to provide answers in CONSTANT 1980 Rands, and the modified form of the

SOME GENERALIZED COSTING RELATIONSHIPS FOR UNDERGROUND MINING.

(Monetary values expressed in constant 1980 Canadian dollars).

Definitions:

T = mine or mill capacity, tons of ore per year.
R = recoverable ore reserves, tons.
UCC = unit capital cost, \$ per annual ton of capacity.
UOC = unit operating cost, \$ per ton mined or milled.
ACC = annual capital costs, \$ per year.
CC = capital cost, \$.
S = shaft depth, metres.
W = average ore width, metres.
E = stope width, metres.
N = number of persons employed.

Capacity-reserve relationship:

Within the limits $50000 < T < 6000000$

$$T = 4.22(R)^{0.756}$$

rounding T,

to the nearest 10000 for $50000 < T < 250000$
to the nearest 25000 for $250000 < T < 500000$
to the nearest 50000 for $500000 < T < 1000000$
to the nearest 100000 for $1000000 < T < 6000000$

Shaft sinking:

Timber shaft for $50000 < T < 500000$

$$CC = (0.003389(T) + 3513)(S)$$

Concrete shaft for $500000 < T < 1500000$

$$CC = (1525(T)^{0.1204})(S)$$

Multiple shafts required for $T > 1500000$

Shaft equipping and surface installations:

$$CC = 7244(T)^{0.5136}$$

Pre-production underground mine development:

$$CC = 60(T)/(W)^{0.8}$$

Pre-production period:

2 years for $50000 < T < 100000$
3 years for $100000 < T < 500000$
4 years for $500000 < T < 1500000$
5 years for $T > 1500000$

Underground mine plant equipment:

$$UCC = 3000(T)^{-0.4095}$$

Underground mine sustaining capital cost:

$$ACC = 137.47(T)^{0.6791}$$

Underground mine operating cost:

Blasthole open stoping (sublevel stoping)

$$UOC = 2853(T)^{-0.3837}$$

Shrinkage stoping

$$UOC = 1774(T)^{-0.3000}$$

Cut-and-fill stoping

$$UOC = 1736(T)^{-0.3122}$$

Manpower for underground mine and mill:

Blasthole open stoping (sublevel stoping)

$$N = 0.73(T/E)^{0.5} + 0.074(T)^{0.5}$$

Shrinkage stoping

$$N = 0.053(T)^{0.7}/(E)^{0.5} + 0.074(T)^{0.5}$$

Cut-and-fill stoping

$$N = 0.064(T)^{0.7}/(E)^{0.5} + 0.074(T)^{0.5}$$

FIGURE C-7. Generalized costing relationships for underground mining. Monetary values are expressed in constant 1980 Canadian dollars. (from Mackenzie, 1982)

GENERALIZED COSTING RELATIONSHIPS FOR AN OPEN PIT.

CONSTRAINTS: In 1980 Canadian dollars.

M= mine capacity (tons of ore+waste per year)

200 000 < M < 60 000 000

T= mine and/or mill capacity (tons of ore per year)

50 000 < T < 30 000 000

Capacity(T)-Reserve(R) relationship: $T=5.63(R)^{0.756}$

Pre-production stripping cost for an open pit:

Unconsolidated overburden: \$2.30 per cu.metre

Waste rock: \$3.00 per cu.metre

Open pit operating cost (UOC=unit operating cost,\$ per ton mined or milled):

$UOC=378(M)^{-0.3691}$

Open pit equipment and maintenance facilities (UCC=unit capital cost,\$ per

annual ton of capacity): $UCC=304(M)^{-0.3255}$

Open pit sustaining capital cost (ACC=annual capital cost,\$ per year):

$ACC=0.1623(M)+81300$

Mill capital cost (UCC=unit capital cost,\$ per annual ton of capacity):

Copper mill: $UCC=18051(T)^{-0.4794}$

Copper-molybdenum mill: $UCC=31520(T)^{-0.5114}$

Copper-zinc or lead-zinc mill: $UCC=109021(T)^{-0.5903}$

Mill operating cost (UDC=unit operating cost,\$ per ton mined or milled):

Copper mill: $UDC=345.5(T)^{-0.3251}$

Copper-molybdenum mill: $UDC=487(T)^{-0.3400}$

Copper-zinc or lead-zinc mill: $UDC=1530(T)^{-0.4122}$

Mill sustaining capital cost (ACC=annual capital cost,\$ per year):

$ACC=0.01(UCC \text{ mill})(T)$

Working capital:

3 months operating costs (mine+mill+overhead)

Administrative and overhead operating costs:

20% of UOC (mine+mill)

Pre-production period: 2 years

Manpower for open pit and mill (N= number of persons employed):

$N=0.02(M)^{0.5718}+0.074(T)^{0.5000}$

Accommodation (per person employed when necessary):

Houses= \$70 000

Permanent bunkhouse facilities= \$15 000

Amenities (per person employed when necessary)

Townsite= \$20 000

Camp= \$5 000

Road construction: \$80 000 per km

Power facilities:

Peak load (kilowatts,KW): $KW=7.9(T)^{0.4977}$

Isolated mines (assume diesel generating plant and low voltage): capital cost= $6100(KW)^{0.8}$

Where utility power is available (assume transmission line, utility substation, and low voltage distribution):

For $T < 500 000$, use 27.6 or 41.6 KW line:

capital cost= $1100(KW)^{0.8}+35000(D)$

For $T > 500 000$, use 110 KW line:

capital cost= $1100(KW)^{0.8}+75000(D)$

where D= transmission distance, km

Mine recovery: MRF= 100%

Dilution: DF= 0%

Mill recovery factor:

Copper mill: Cu-95%, Au-85%, Ag-85%

Copper-molybdenum mill: Cu-90%, Mo-80%, Au-80%, Ag-80%

Copper-zinc or lead-zinc mill: Cu-90%, Pb-90%, Zn-80%

Au-60%, Ag-75%, Cd-75%

Concentrate grade:

Cu-25%

Mo-90%MoS₂

Zn-55%

Pb-60%

Concentrate transportation cost:

Road: 3.0 cents per tonxkm

Rail: <500 km, 5.0 cents per tonxkm

500-1500 km, 3.0 cents per tonxkm

>1500 km, 2.5 cents per tonxkm

Transfer: \$4 per ton concentrate at each transfer point

FIGURE C-8. Generalized costing relationships for an open pit. Monetary values are expressed in constant 1980 Canadian dollars. (from Mackenzie, 1982)

equations used are given in figure C-9. The actual form and format of the formulae used to calculate the various capital and operating costs are shown in figure C-10 where the cell co-ordinates and contents are listed.

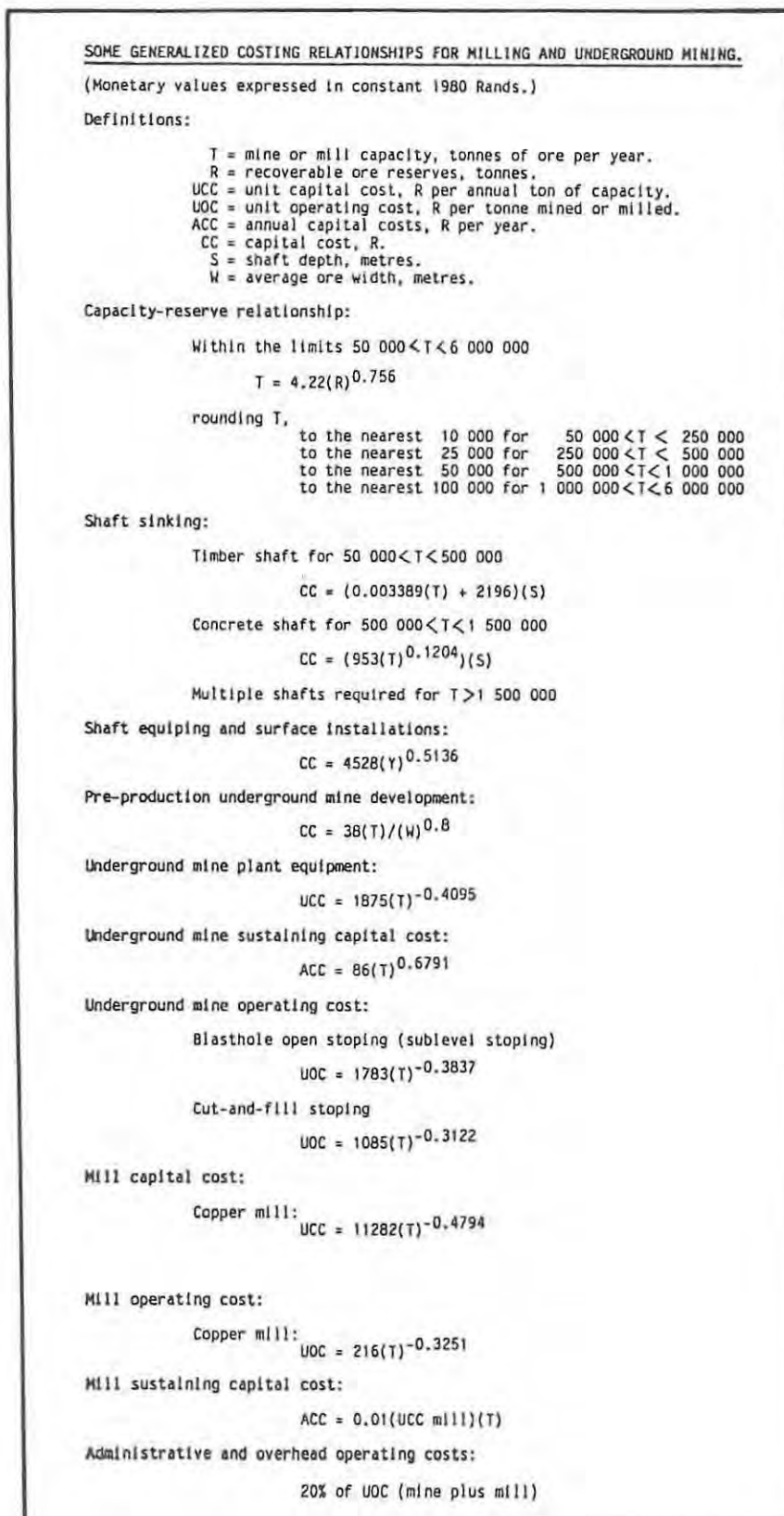


FIGURE C-9. Costing relationships for milling and underground mining. Modified after Mackenzie (1982) to show monetary values in constant 1980 rands.

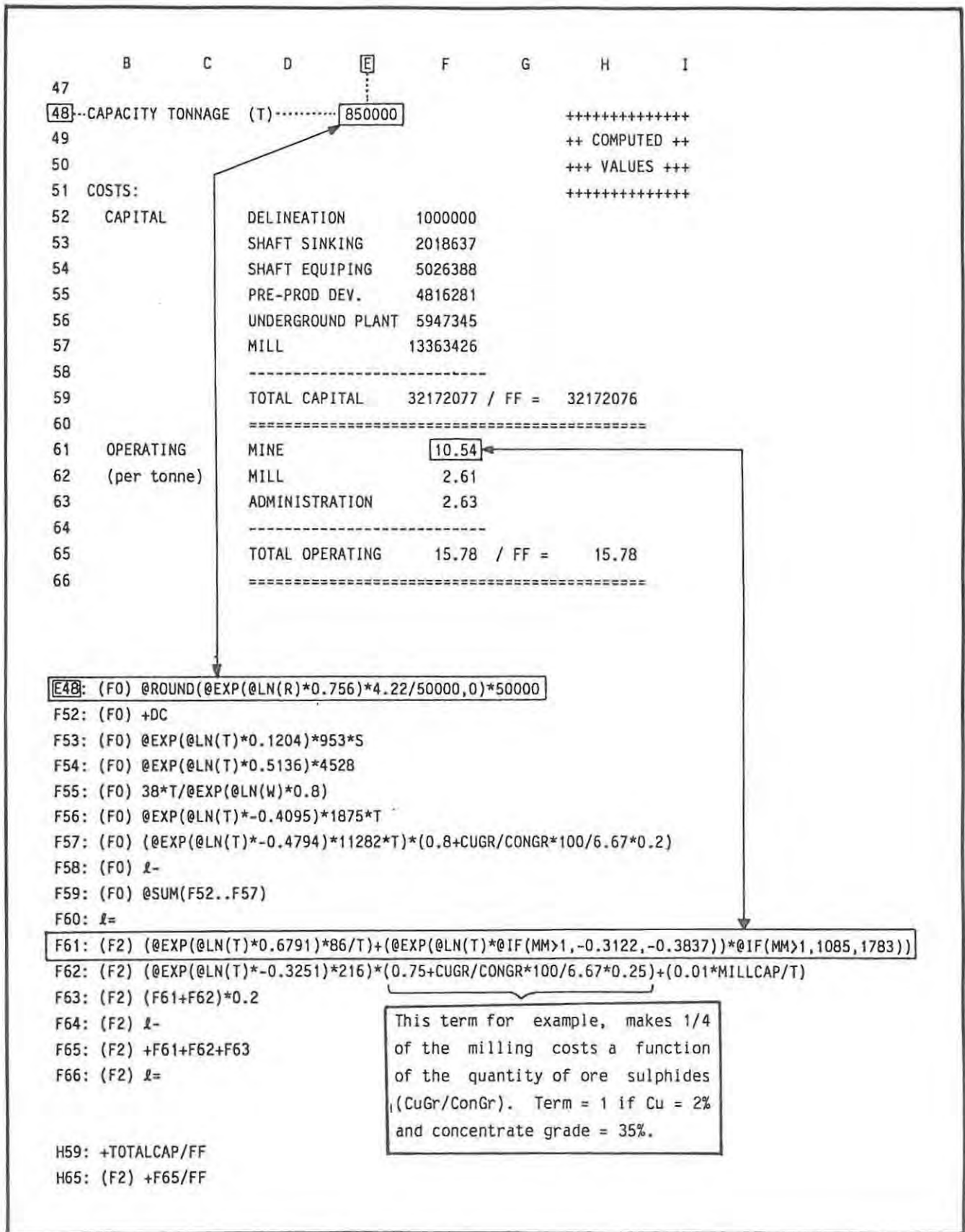


FIGURE C-10. Cell formulae used to calculate RISK\$ costing relationships. Formulae for calculating capacity tonnage and mine operating costs are boxed as illustrative examples. Formulae as per figure C-9.

Note in particular that the formula for mill operating costs has not only been modified from \$ to R, but has been set to increase with increasing headgrades and decreasing concentrate grades and vice-versa. For values of 2% copper and 30% concentrate grade, the formula produces an answer identical to the simple equation as shown in figure C-9. More than 2% Cu and less than 30% concentrate grade will result in higher milling costs.

Under most normal circumstances, the estimation of capital and operating costs would be handled somewhat differently than it is by RISK\$. With the acquisition of real data from a mineral prospect delineation programme, the various costs would be estimated, possibly using formulae such as Mackenzie's as guidelines, but modifying to suit. Remember that Mackenzie's formulae provide reasonable answers for average situations of similar deposit types, so that RISK\$ provides the most likely 'expected' average costs and the associated errors for the given set of input data.

Note however, that the way in which RISK\$ is structured is particularly useful for assessing the 'likely' investment potential and risk of anticipated deposits (geological models) not yet located, but predicted. In such cases, cost estimations based on average known costs of similar deposit types (which is what Mackenzie's formulae provide) appears a reasonable method to use. Additional inherent uncertainty in such cases can easily be simulated by RISK\$ by setting the Favourability Factor (FF) as a risk variable with a specified mean and standard deviation- see SECTION D.

Input of Estimated Prices and Price Cycles.

The expected mean values in CONSTANT 1980 Rands of the copper and silver prices are entered together with the anticipated standard deviation from the means in the cells indicated in figure C-11. In addition, the facility exists to simulate any expected or anticipated price cycle pattern (see figure C-11). This allows construction of price cycle patterns of the type illustrated in figure B-4. Prices are selected from their specified normal distributions in the same manner as for the previous risk variables. In the case of prices however, new selections are made for EACH of the cash flow periods, whereas for the previous parameters such as copper grade, any selected value is used for ALL of the cash flow periods in any one execution run. The reason for this should be apparent - the selected mean grade estimate is the mean of the total tonnage of the orebody and is orebody

dependent, the prices fluctuate continually and are NOT orebody dependent.

	A	B	C	D	E	F	G	H	I	J	K	L	M
68													
69	ESTIMATION OF COMMODITY PRICES FOR THE RELEVANT TEN YEAR CYCLE												
70	-----												
71	ESTIMATED METAL PRICE MEANS AND DEVIATIONS												
72													
73		MEAN	STANDARD										
74		VALUE	DEVIATION (of the mean)										
75	COPPER	2000	100	Copper price in Rand per tonne.									
76	SILVER	200	20	Silver price in Rand per kilogram									
77													
78													
79													
80	***** ***INPUT DATA*** *****												
81	LONG TERM PRICE CYCLE PREDICTION												
82													
83	COMMODITY	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	AVERAGE	
84	-----												
85	COPPER	1	1	1	1	1	1	1	1	1	1	1.00	
86	SILVER	1	1	1	1	1	1	1	1	1	1	1.00	
87													
88	EXAMPLE PRICES												
89	-----												
90	COPPER	2037.43	2006.21	1835.56	1969.30	2147.64	1985.15	1955.17	2062.23	1883.60	2061.17	1994.35	
91	SILVER	185.01	225.99	217.17	172.80	211.88	218.28	184.70	216.62	212.68	177.58	202.27	

FIGURE C-11. Detail of the price and price-cycle part of the RISK\$ INPUT SECTION. Input data cells are outlined.

The longterm trends in the Rand price of copper as determined over a 23 year period (1955 - 1977) are shown in figure C-12 (Krige, 1978). Using this

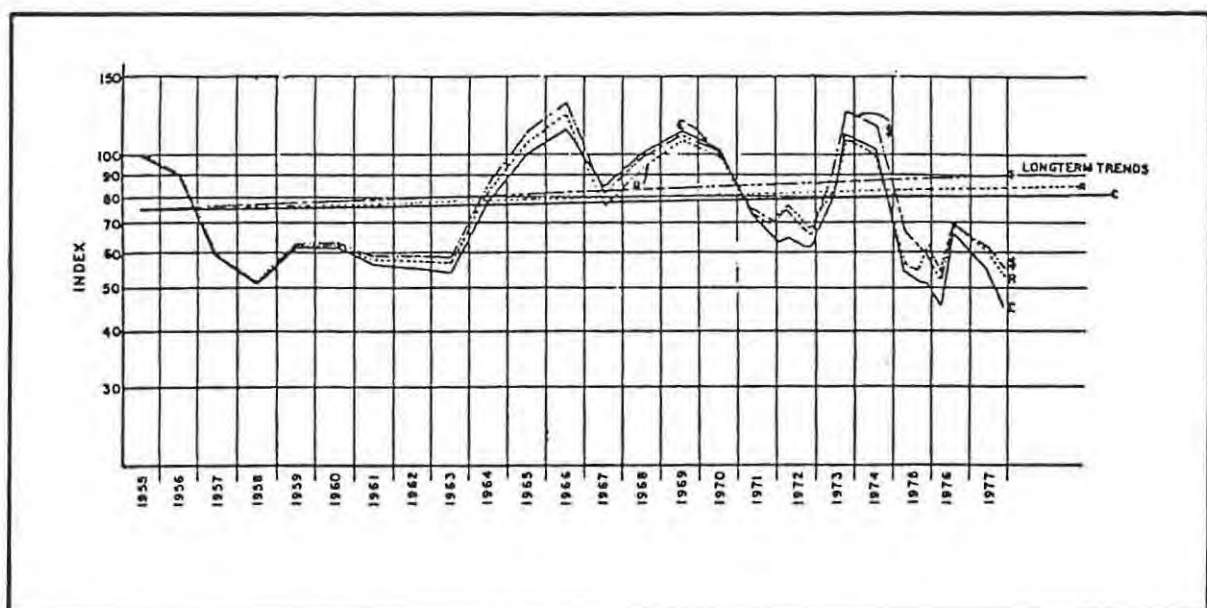


FIGURE C-12. The L.M.E. copper price per tonne (1 000 kg) in real R, \$ and £ terms indexed to 1955. Long term predicted price trends indicated. (after Krige, 1978)

long term price trend, the predicted copper price for 1980 (in current rand terms) is approximately R2 200. This is by the following formula (Krige, op. cit.):

$$\text{Price (1980)} = R1\ 658 (1.0028(1 + z/100)^y$$

where R1 658 is the estimated 1977 current price, z is the anticipated annual local inflation rate (10% was used) and y is the number of years (in this case 3).

The equation yields a value of R5 235 as the expected 1986 copper price if the South African inflation rate is taken as an annual 15% from 1980 to 1986. This is rather higher than the actual average 1986 price of about R3 500 for at least three reasons. The first is that the South African exchange rate is dominated by the gold price (as previously discussed). The second is that from 1977 to the present, the copper price as indexed to the 1955 price (see figure C-12) has remained well below the longterm trends as estimated from the 1955 - 1977 data. In other words, we are currently within a negative price cycle. The third factor is that the additional data (post 1977) would significantly lower the computed longterm price trend and thus alter the above equation. For these reasons a value of R2 000 is selected as representing the 1980 longterm expected price. The choice of exactly R2 000 is somewhat arbitrary but deemed reasonable, as is the chosen 1980 price of R200 per kilogram for silver (Hunt brothers excluded!).

Although obviously extremely important (and significant), the absolute values of 1980 rand prices for copper and silver do not influence the ability of RISK\$ to demonstrate the influence of geological parameters on exploration / mining investment alternatives. In addition to this, any assessment of any prospects or mines producing the same commodity metals, must surely be based on identical commodity price estimates, and the relative attractiveness of investment alternatives would not alter. Note however, that deposits with markedly different copper / silver grade ratios would perform similarly for changed copper and silver prices ONLY if the price changes were RELATIVELY the same. Changes in the RELATIVE copper/silver price ratio would be expected to have a significant effect on such investment alternatives.

CASH FLOW CALCULATIONS.

This next section of RISK\$ handles the cash flow estimates. No additional input data other than the rate of taxation is required, the cash flows are all computed automatically from the already available data. Figure C-13 shows an example of a complete cash flow grid for one particular set of selected input data.

The cash flows are estimated by calculating all of the expected costs and revenues associated with any particular set of input data. Costs and revenues are conveniently expressed in units per tonne of ore milled, as this facilitates comparison and analysis of cost data.

In order to calculate the Realization charges (off mine costs) as well as the revenues, it is necessary to calculate the quantities of the different products produced, such as the tonnes of concentrates for example. To calculate the tonnes of concentrate that will be produced in a year the tonnes of copper produced is first calculated. This is simply

tonnage x copper grade x copper % recovery,

and yields the amount of copper produced. Dividing this value by the concentrate grade yields the tonnes of dry concentrate that would be produced, which needs to be known in order to calculate the smelting costs, which are quoted in prices/tonne concentrate.

The exact formula in each of the cells of the first period (year 1) cash flow are printed and displayed in figure C-14. Once all of the formulae for the year 1 cash flow have been established and entered, any required number of cash flow periods can be constructed using the COPY command. Developing the first cash flow took about one hour, expanding this to 10 cash flow periods took about 20 seconds.

The net total profits or losses in the final row of the cash flow grid constitute the production related portion of the total cash flow. The total capital investment forms the initial or year 0 investment portion of the cash flows. The initial investment and the production cash flows are then COPIED into the OUTPUT section of RISK\$ where they become the raw data for any investment criteria measures to be applied to.

E

104	YEAR 1	E104: ' YEAR 1
105	=====	
106	TONNES MILLED 850000	E106: (F0) +\$T
107	HEADGRADE % COPPER 1.98	E107: (F2) +\$CUGR
108	COPPER % RECOVERED 91.58	E108: (F2) +\$CUREC
109	TONNES COPPER PRODUCED 15388	E109: (F0) +E106*E107*E108/10000
110	HEADGRADE SILVER G/T 9.62	E110: (F2) +\$AGGR
111	SILVER % RECOVERED 89.70	E111: (F2) +\$AGREC
112	Kg SILVER PRODUCED 7337	E112: (F0) +E106*E110*E111/100000
113	COPPER CONCENTRATE GRADE 34.91	E113: (F2) +\$CONGR
114	DRY TONNES CONCENTRATE 44080	E114: (F0) +E109/E113*100
115	% MOISTURE OF CONCENTRATE 10.00	E115: (F2) 10
116	WET TONNES CONCENTRATE 48488	E116: (F0) +E114*(1+E115/100)
117	% SiO2 MORE THAN 10% 0.00	E117: (F2) 0
118	-----	
119	COPPER PRICE R/TONNE 2037	E119: (F0) +C90
120	SILVER PRICE R/Kg 185	E120: (F0) +C91
121	-----	
122	ON MINE COSTS/T ORE:	
123	MINING 10.54	E123: (F2) +\$F61
124	MILLING 2.61	E124: (F2) +\$F62
125	ADMINISTRATIVE 2.63	E125: (F2) +\$F63
126	TOTAL / FF 15.78	E126: (F2) +\$F65/\$FF
127		
128	OFF MINE COSTS:	
129	TRANSPORT: OF ORE 0.00	E129: (F2) +\$T*\$ORET
130	: OF CONCENTRATE 1.71	E130: (F2) +E116*\$CONT/\$T
131	:TOTAL 1.71	E131: (F2) +E129+E130
132	SMELTING: DIRECT CHARGE 3.89	E132: (F2) +E114*\$CONS/\$T
133	COPPER ALLOWANCE 0.32	E133: (F2) +E114*\$CUALL/100*E119/\$T
134	SILVER ALLOWANCE 0.34	E134: (F2) +E114*\$AGALL/1000*E120/\$T
135	PENALTIES 0.00	E135: (F2) +E117*E114/E106
136	TOTAL 4.54	E136: (F2) @SUM(E132..E135)
137	SHIPPING, HANDLING	
138	AND REFINING: 5.43	E138: (F2) +\$REF*E109/\$T
139	TOTAL OFF MINE COSTS R/T OR 11.68	E139: (F2) +E131+E136+E138
140	=====	
141	REVENUE / T ORE:	
142	FROM COPPER 36.89	E142: (F2) +E109*E119/\$T
143	FROM SILVER 1.60	E143: (F2) +E112*E120/\$T
144	TOTAL REVENUE R/T ORE 38.48	E144: (F2) +E142+E143
145	GROSS REVENUE 32709984	E145: (F0) +E144*\$T
146	TOTAL COST R/T ORE 27.47	E146: (F2) +E126+E139
147	GROSS PROFIT (LOSS) /T ORE 11.02	E147: (F2) +E144-E146
148	GROSS PROFIT (LOSS) 9363683	E148: (F0) +E147*\$T
149	CUMMULATIVE PROFIT/LOSS 9363683	E149: (F0) +E148
150	TAX @ FLAT 40% 0	E150: (F0) @IF(E149)>=\$TOTALCAP,@IF(D150>0,E148*\$TAX/100,
151	*****	(E149-\$TOTALCAP)*\$TAX/100),0)
152	NET PROFIT (LOSS) 9363683	E152: (F0) +E148-E150
	\$	

FIGURE C-14. Cell formulae as used in the RISK\$ CASH FLOW SECTION.

OUTPUT DATA.

The output data section of RISK\$ is infinitely flexible and user definable. Any desired record or tabulation in almost any desired format of any of the parameters that have formed any part of the input, cost or cash flow calculations may be instigated. The standard basic output data and format is shown in figure C-15. It includes a summary of the investment capital and cash flows, the calculated Internal Rate of Return (IRR) and Total Potential Revenue (TPR), and a selected listing of some of the more pertinent parameters as used in the generation of any particular set of cash flows. Figure C-15 shows a sample of such a block of data that was generated by ten automatic executions of RISK\$.

Ranges or tables of output data can be extracted from RISK\$ and exported to DATA storage files on a separate disc. It is from these data files, each generated from a particular set of input data characteristics, that the results may be summarized and analysed, and presented in the form of averages, frequency distributions, regressions or other desired forms (see Section D).

WORKING SECTION.

The last section of RISK\$, the working section, is tucked away to the right of the input data section (figure C-2). The section includes a portion where $N(0,1)$ random selections are made each time RISK\$ is executed, as well as a number of MACROS where various execution instructions are stored.

The working section as it stands in RISK\$ is shown in figure C-16. Macros \C and \E can be set to drive macros \A and \D for any chosen number of re-executions. Macro \A causes a full table of output parameters (as shown in figure C-15) to be entered at the position of the cursor, whereas macro \D generates an abbreviated table, recording only the IRR and TPR for each execution run. This is more convenient (and much faster) where a large number of runs is required.

This concludes the summary of the RISK\$ spreadsheet template and we turn to Section D for presentation and discussion of the results from selected examples.

 *** OUTPUT ***

CASH FLOW AND INTERNAL RATE OF RETURN ANALYSIS

SUMMARY OF CASH FLOWS.

INITIAL INVESTMENT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10
-3.2E+07	5723406	5628023	6984813	4932088	7857503	4173435	3860257	4996337	4287930	3113832

INTERNAL RATE OF RETURN = 11.04 ***
 MAXIMUM REVENUE (MILLIONS)= 279 ***

FORMULAE	IRR	TOTAL POTREV	CuGr	CuREC	AgGr	AgREC	CONGr	CuPr	AgPr	ON MINE COSTS/T	OFF MINE COSTS/T	TOTAL COSTS/T	TOTAL REVENUE/T	NET TPROFIT/T
NUMBERS:	11.04	279.01	1.69	92.68	10.40	89.14	38.68	1980.04	203.16	15.62	9.58	25.20	32.82	6.07
VALUES :	11.04	279.01	1.69	92.68	10.40	89.14	38.68	1980.04	203.16	15.62	9.58	25.20	32.82	6.07

CALCULATE RESULT TABLE BY POSITIONING CURSOR AT CELL A185 AND PRESSING ALT C. CALCULATE PARTIAL TABLE (IRR AND TPR) BY PRESSING ALT E.

***** RESULTS *****

IRR	TOTAL POTREV	CuGr	CuREC	AgGr	AgREC	CONGr	CuPr	AgPr	ON MINE COSTS/T	OFF MINE COSTS/T	TOTAL COSTS/T	TOTAL REVENUE/T	NET TPROFIT/T	T
19.86	325.48	1.97	90.37	10.04	89.41	36.31	2059.14	190.37	15.75	11.24	26.99	38.29	8.28	850000
18.33	336.19	2.09	92.00	9.95	90.58	29.88	1961.75	201.52	15.94	13.56	29.50	39.55	7.56	850000
22.38	335.41	2.06	91.89	10.97	90.33	34.61	1986.71	187.06	15.82	12.27	28.09	39.46	8.35	850000
21.73	325.99	2.09	92.71	9.68	90.32	34.82	2014.66	199.12	16.18	12.54	28.72	40.75	8.82	800000
19.40	338.64	2.09	91.10	10.17	88.21	34.82	1996.22	199.81	15.82	12.34	28.16	39.84	8.53	850000
17.24	321.93	1.99	90.72	11.00	90.66	34.93	1992.02	197.25	15.78	11.65	27.43	37.87	7.78	850000
18.52	323.18	1.99	92.24	10.26	90.32	33.46	1972.67	199.59	15.82	12.13	27.94	38.02	7.58	850000
21.96	330.73	2.21	92.30	10.03	89.85	36.72	1938.81	196.94	16.18	12.83	29.01	41.34	8.98	800000
16.18	303.19	2.07	89.56	10.39	89.90	35.02	1941.06	201.42	16.17	11.97	28.14	37.90	7.44	800000
11.04	279.01	1.69	92.68	10.40	89.14	38.68	1980.04	203.16	15.62	9.58	25.20	32.82	6.07	850000

 *** OUTPUT ***

FIGURE C-15. DATA OUTPUT SECTION of RISK\$. A "Block" of data generated by 10 iterations of RISK\$ is shown.

SECTION D : PRESENTATION AND DISCUSSION OF RESULTS.

INTRODUCTION.

This section is divided into three main parts.

* The first part deals briefly with the form and format of the output data from RISK\$, and how this data is configured, stored and analysed.

* The second part deals with the establishment of a BASECASE and output data generated by RISK\$ from the BASECASE parameter settings is discussed. In particular, the nature and distribution of the BASECASE output data is discussed and a SENSITIVITY ANALYSIS of the BASECASE is conducted. The BASECASE also provides an ideal vehicle to illustrate many of the effects of changing geological (as well as geographical and time dependent) factors on the economic outcome.

* The third part is in the form of an exploration - delineation simulation. Analyses are given of all of the key economic parameters starting with an initial geological 'model' and working through various phases of exploration and delineation of three different mineral prospects, all variants of the initial geological model. This in turn provides an ideal vehicle to illustrate the manner in which the risk element in particular is reduced as additional information becomes available. It also demonstrates how the relative economic attractiveness of different prospects may change significantly as more details become available, and at which stage (if any) particular prospects may be either rejected or selected for further evaluation leading up to the final production GO - NO GO decision.

OUTPUT DATA HANDLING.

Output data from RISK\$ is in the form of BLOCKS of data. The data is most conveniently and efficiently handled and analysed when the block-size is standardized. Different 'standard size' blocks may be specified in the OUTPUT section of RISK\$ and this method is employed. By using standard size blocks the analysis of the data is streamlined.

Each particular block of data generated by RISK\$ is EXTRACTED and SAVED as a data file on a separate data-handling disc. Data analysis 'templates' are constructed on the spreadsheet and saved on the same data storage disc. A particular analysis template is for example constructed for the standard output parameters as shown in figure C-15. A complete block in this case (figure C-15 shows a partial block) is 15 columns wide by 100 rows long (100 runs of RISK\$). The analysis template is set up in such a fashion that when such a block of data is COPIED into the template, the averages, standard deviations, frequency distributions, cumulative frequency plots, correlations and graphs of these distributions and correlations automatically portray the values derived from this particular block of data. COPYING in a new block means a new set of answers, relating to this new data.

In this fashion, standardized blocks of data generated from different sets of input variables can be rapidly and effectively analysed, and individual data templates can then be saved as separate discrete files.

ESTABLISHING A BASECASE.

INTRODUCTION.

As mentioned in Section B, in order to make a very broad topic tractable, a BASECASE situation specifically representing a base-metal sulphide copper-silver deposit type is used. To reiterate, the methods employed are applicable to any form of mineable commodity, but the objective of illustrating the influence of geological factors on investment alternatives is best achieved by confining the discussion to a particular deposit type.

BASECASE INPUT DATA SETTINGS.

The chosen BASECASE is meant to represent an actual, reasonably well delineated copper-silver deposit. As such, the various geological input parameters are fairly well constrained, and, more importantly, the Geographical and most of the Time Dependent variables are FIXED. In other words, when analysing the BASECASE, unless otherwise specified, factors such

as commodity prices, taxation policy etc. are CONSTANT (or at least have constant mean values) for any set of geological input parameters. Some of these are however varied (noting the changes), in order to fully assess the significance of certain of the geological factors, as it has been shown that there is considerable interplay and overlap of ALL of these factors (see figure B-3).

The BASECASE input parameters are shown in figure D-1. The delineation cost of R1 000 000 is taken to represent 20 borehole intersections (at an average of R50 000 per hole). The BASECASE orebody can be mined by open stoping

*****BASECASE*****			MEAN	STANDARD							
*****BASECASE*****			VALUE	DEVIATION							
*****BASECASE*****				(of the mean)							
DELINEATION COSTS	(DC)	R1 000 00		-							
MINEABLE RESERVES	(R)	10 000 00	100	000							
MINEABLE WIDTH	(W)	10		0.5							
COPPER GRADE	(CuGr)	2		0.1							
% RECOVERY	(CuREC)	92		1							
SILVER GRADE	(AgGr)	10		0.5							
% RECOVERY	(AgREC)	90		1							
CONCENTRATE GRADE	(ConGr)	35		2							
SHAFT DEPTH	(S)	400		10							
MINING METHOD	(MM)	1		-							
FAVOURABILITY FACTOR	(FF)	1		0							
=====											
REALIZATION COSTS:											
(per appropriate unit)											
ORE TRANSPORT	(OreT)	R0.00									
CONC. TRANSPORT	(ConT)	R30.00									
CONC. SMELTING	(ConS)	R75.00									
COPPER ALLOWANCE	(CuAll)	0.03									
SILVER ALLOWANCE	(AgAll)	35									
PENALTY UNIT	(PU)	2									
REFINING ETC.	(REF)	R300.00									
=====											
ESTIMATED METAL PRICES : MEANS AND DEVIATIONS											
=====											
COPPER :	R2 000.00	R100.00									
SILVER :	R200.00	R20.00									
=====											
LONG TERM PRICE CYCLE PREDICTION											
=====											
COMMODITY	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	AVERAGE
COPPER	1	1	1	1	1	1	1	1	1	1	1.00
SILVER	1	1	1	1	1	1	1	1	1	1	1.00
=====											
SET TAX VALUE % :	40										

FIGURE D-1. BASECASE INPUT parameter settings.

methods (MM = 1). In addition, the BASECASE deposit represents the 'ideal' average situation apropos the modified MacKenzie costing equations (see figure C-9) so that the Favourability Factor is set at 1 with zero variance.

All of the calculations are based on CONSTANT 1980 rands and the mean copper and silver prices are set at R2 000 and R200 as discussed in Section C. The copper and silver medium term price cycles are flat (1, 1 ... 1) and follow a long term growth trend that is also flat (no increase or decrease in CONSTANT Rand terms).

ANALYSING THE BASECASE.

OUTPUT DATA DISTRIBUTION.

In Section B it was assumed (for simplicity) that all of the risk input variables are normally distributed and RISK\$ is set up to randomly select risk variables from specified normal distributions. It was also claimed that the main output data, such as the Internal Rate of Return (IRR) and the Total Potential Revenue (TPR) would be NORMALLY DISTRIBUTED. Figure D-2 shows the frequency distribution histograms of the IRR's for BASECASE for 50, 100 and 1000 RISK\$ iterations. The IRR for the 1000 iterations case shows a near perfect normal distribution. Plotting this same data as cumulative frequency % curves (figure D-3) indicates that there is no significant improvement between the 100 and 1000 iteration cases, although the 50 iteration case deviates slightly. For this reason, all of the ensuing graphs are generated from 100 iterations of RISK\$, unless otherwise specified.

Notice in particular that the cumulative frequency % plot affords an easy means of assessing the risk associated with achieving any specified IRR. For example, one may read directly from the graph that there is only a 10% chance of realizing an IRR of less than 14%, or that there is likewise only a 10% chance of realizing an IRR in excess of 24%. The expected mean value and the standard deviation are also plotted and recorded on the graph.

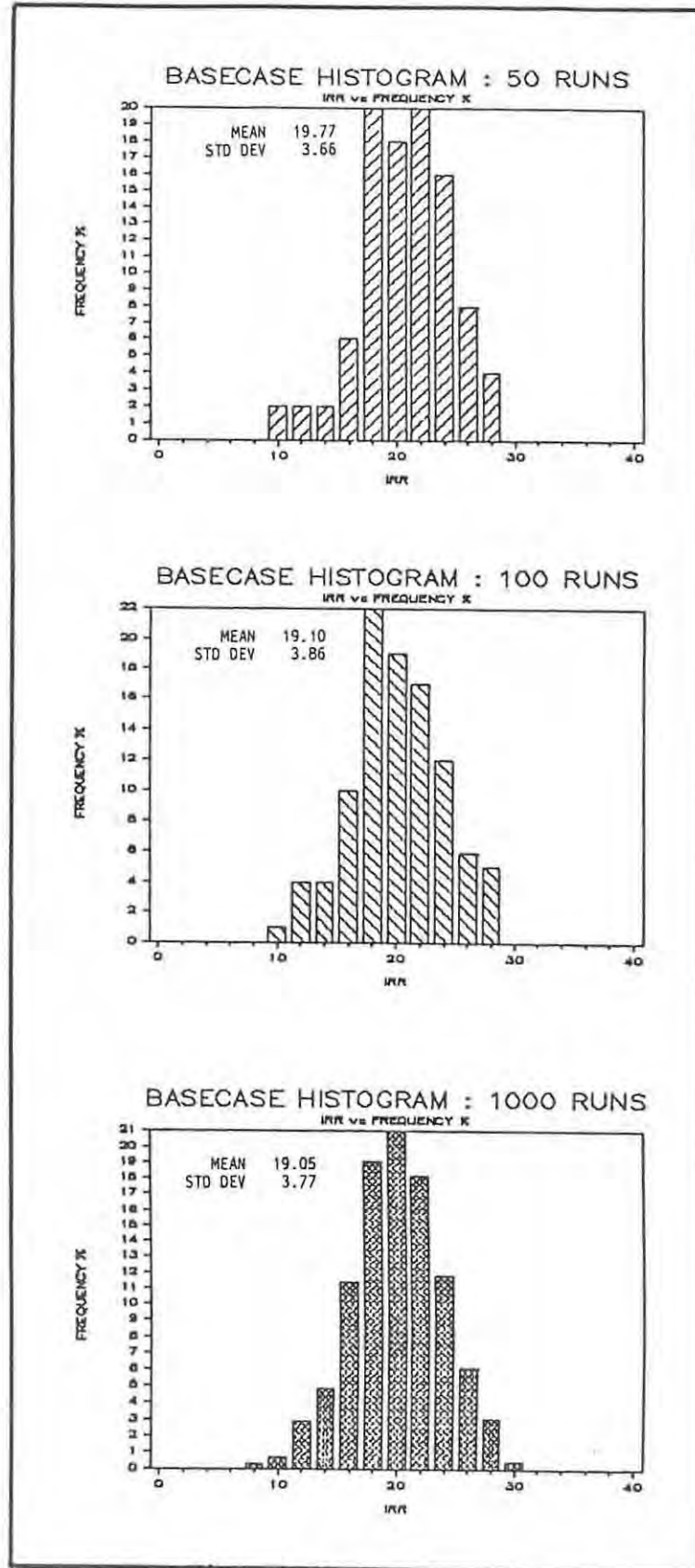


FIGURE D-2. BASECASE frequency percentage histograms for 50, 100 and 1000 iterations of RISK\$. Illustrates the improved form of the IRR distribution with more iterations.

Such cumulative frequency % curves are thus convenient forms of data presentation and allow for assessment of the risk of not achieving or the chance of overachieving any desired IRR. This data may also be plotted on probability paper where normally distributed populations plot as straight lines (not shown). Note also that the 'steepness' of the cumulative frequency % curve is a measure of the spread of the population, the more steep, the less the spread and the better defined and constrained are the results. Generally speaking, such curves would be expected to steepen as more data becomes available (see later), but would do so more and more sluggishly (for equal quantities of additional input data) and reach some maximum limit, representing the remaining NON-REMOVABLE RISK ELEMENTS.

Figure D-4 shows a similar frequency % histogram and a cumulative frequency % curve for the TPR of the BASECASE data.

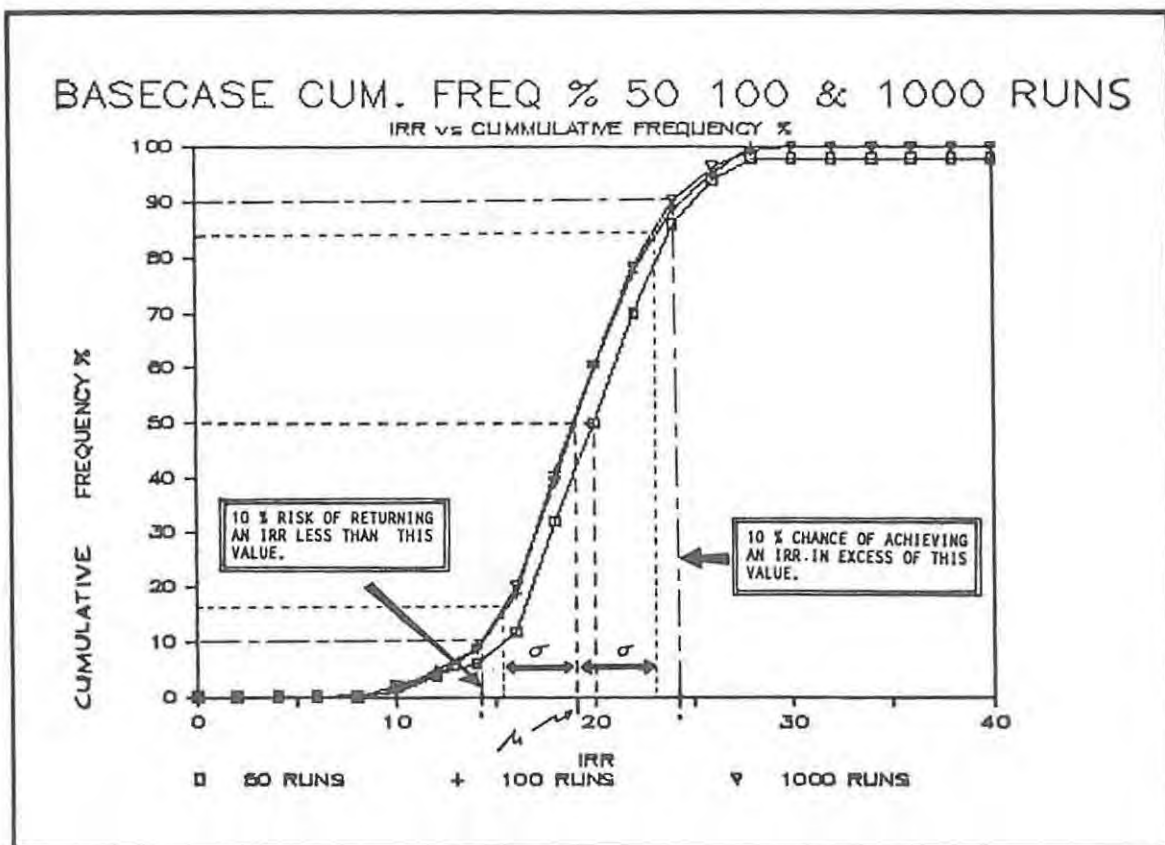


FIGURE D-3. BASECASE IRR results in the form of a cumulative frequency percentage curve.

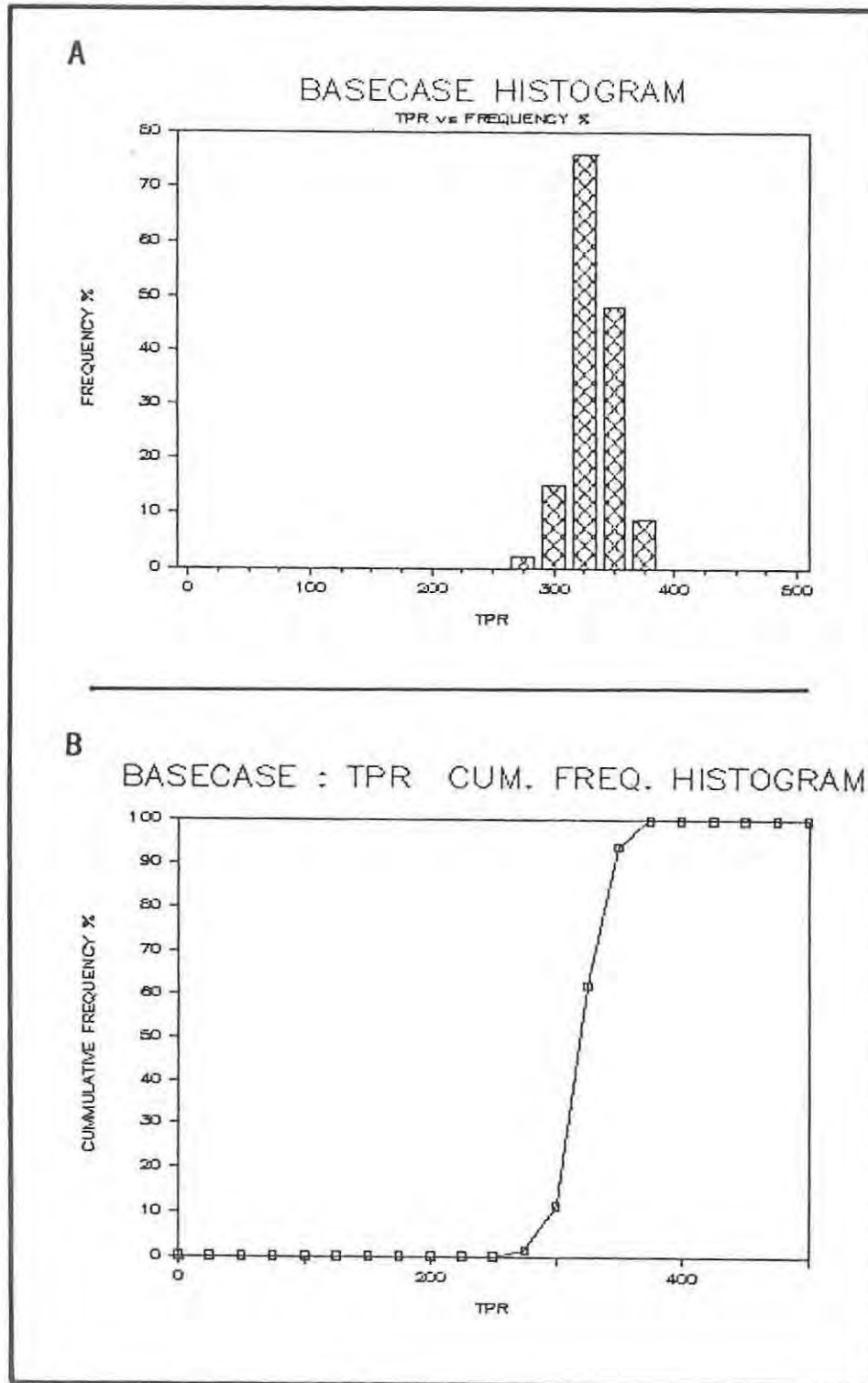


FIGURE D-4. A: BASECASE Total Potential Revenue frequency percentage histogram and B: cumulative frequency percentage curve.

SENSITIVITY ANALYSIS.

Introduction.

The object of a sensitivity analysis is to establish what effect the different variables have on the economic outcome of an investment alternative. At this point it is appropriate to note the following:

- * Changing any of the expected MEAN values of the input data will cause the MEAN IRR (and other measures) to change.
- * Narrowing the range or spread of the input values (MEAN constant, standard deviation reduced) will not alter the MEAN IRR, but only reduce the standard deviation of the IRR.
- * Changes in the variable that the investment is most sensitive to will cause the greatest change in the MEAN IRR outcome, whereas limiting the expected range of this same variable will be the most effective way of reducing the standard deviation of the MEAN IRR.

The idea then, having conducted a sensitivity analysis, would be to focus or direct attention to those factors that have been identified as the most important. If the project has already reached the mine planning/pre-production development phase, then every possible attempt should be made to incorporate FLEXIBILITY and ADAPTABILITY into the mine-plant design, in order to MINIMIZE the risks associated with the most important factors and to balance this by designing to MAXIMIZE the chances of increased returns if favourable situations unfold.

Methods.

As mentioned in Section B, a sensitivity analysis may be carried out either directly from normal risk analysis output data, or by fixing all but one of the risk variables at their expected mean values, and varying each of the variables one at a time within its expected range limits.

Using RISK\$ output data directly is an effective and rapid way to conduct sensitivity analyses. In a block of RISK\$ output data, for each particular

value of the IRR (and TPR), the particular set of input data (and any of the calculated data) is known. Because of this one-to-one correlation, any combination of results may be plotted against each other and regression slopes established. The method is most effective for those factors that have the greatest influence, as the correlations between the IRR and the less important factors may be swamped by the much higher risks associated with the more important risk variables.

Figures D-5 through D-9 show the results of plotting BASECASE IRR against concentrate grades, copper grades and prices, and silver grades and prices respectively. In each case the scale along the X-axis (risk variable axis) is set so that the expected mean value is central and 10% changes in each of the individual risk variables occupy equal distances on this axis.

The relative influence of the risk variables is directly proportional to the slope of the regression curve. Figures D-5 to D-9 show that BASECASE is most sensitive to copper price, followed by copper grade, concentrate grade, silver grade and silver price in this order.

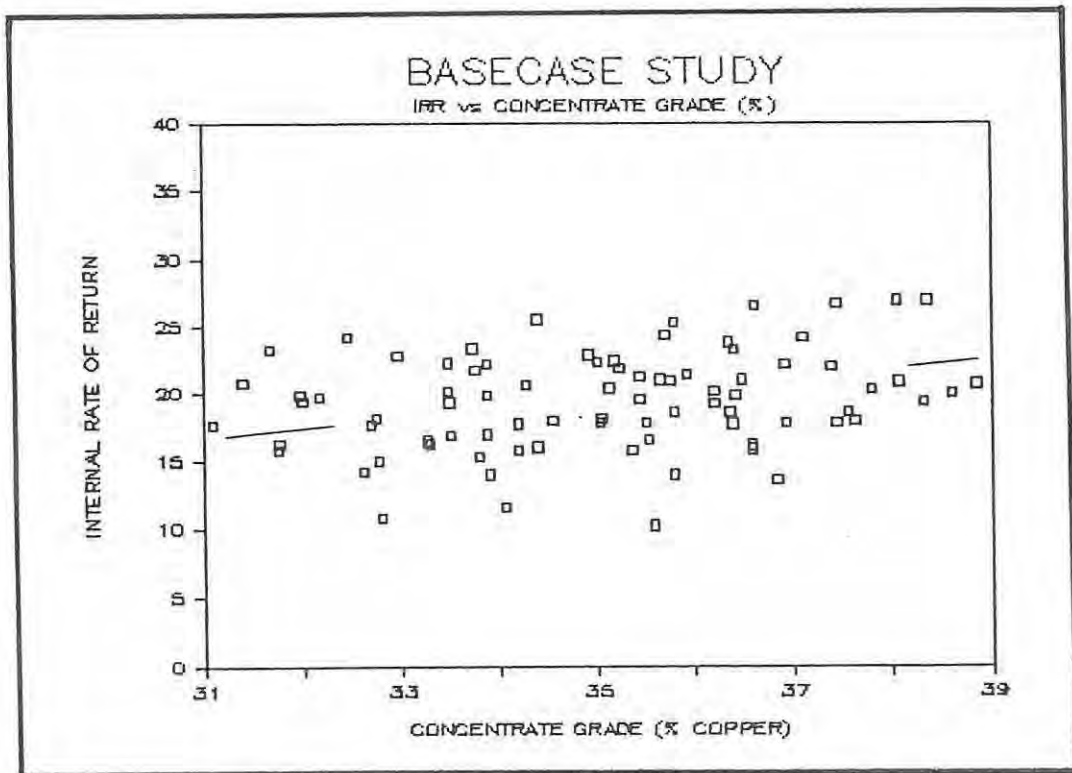


FIGURE D-5. BASECASE: IRR vs concentrate grade. Slight positive slope.

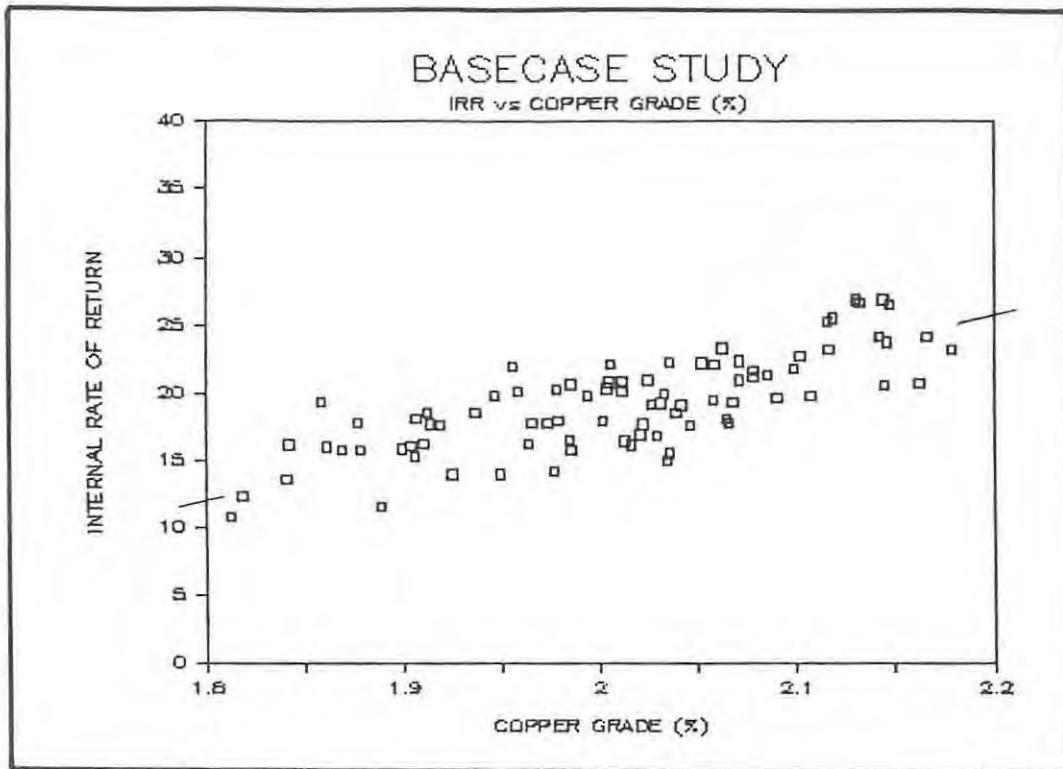


FIGURE D-6. BASECASE: IRR vs copper grade. Marked positive slope.

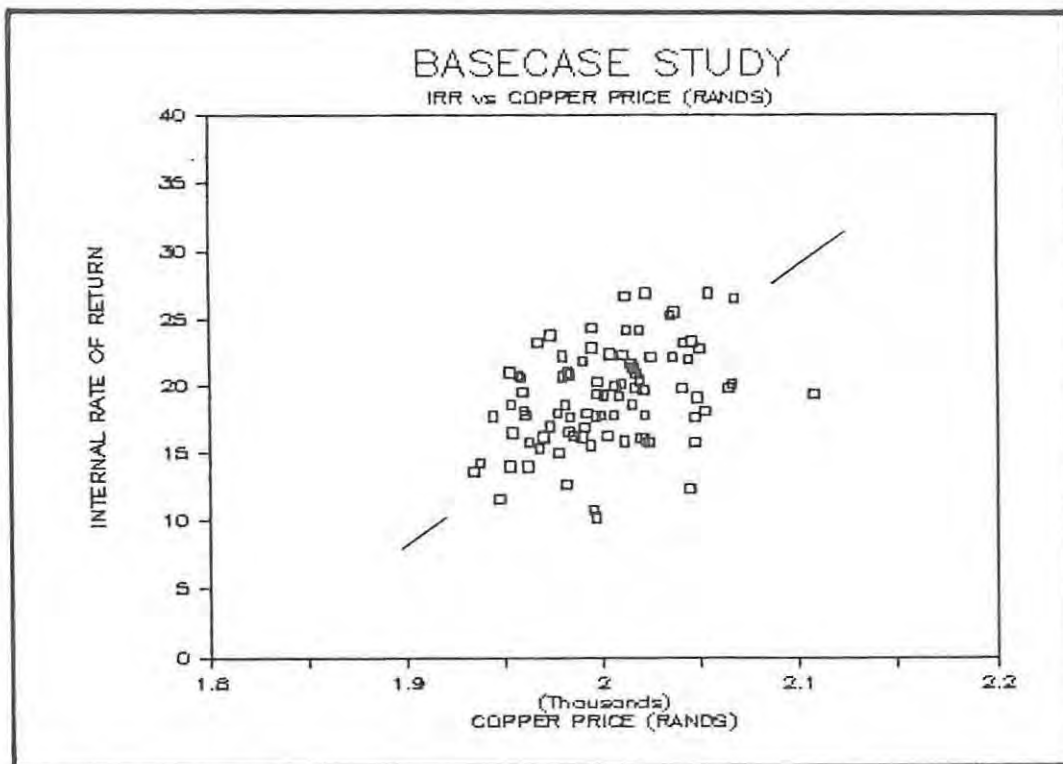


FIGURE D-7. BASECASE: IRR vs copper price. Steep positive slope.

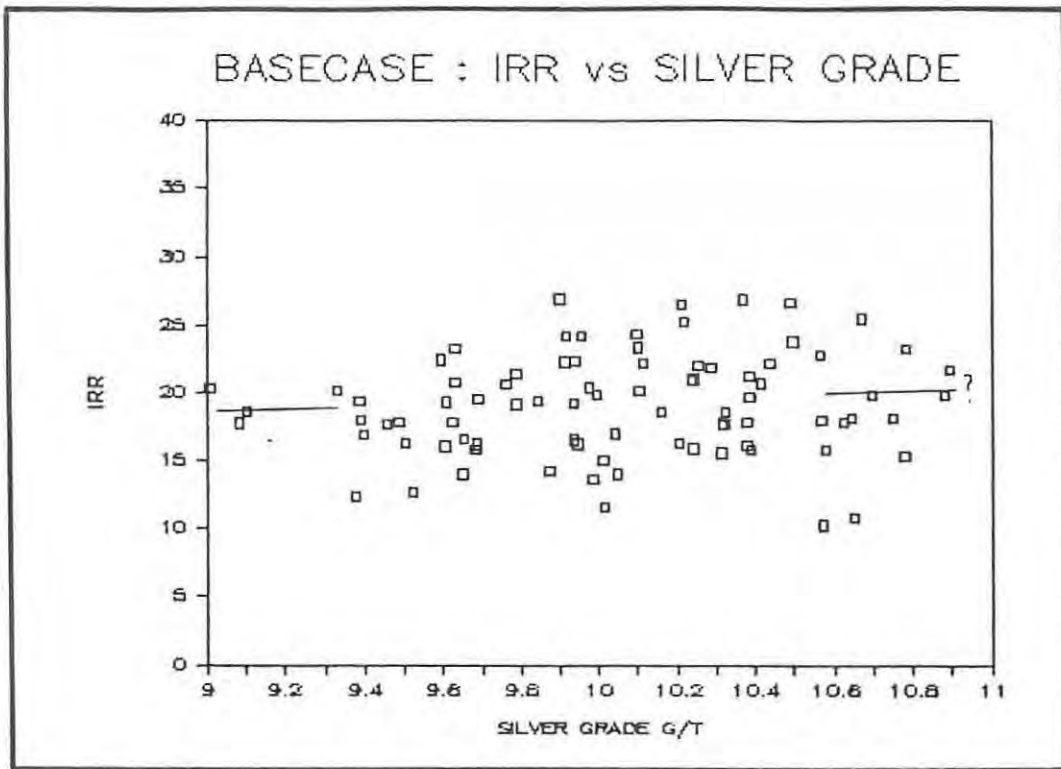


FIGURE D-8. BASECASE: IRR vs silver grade. Slope poorly defined (flat?).

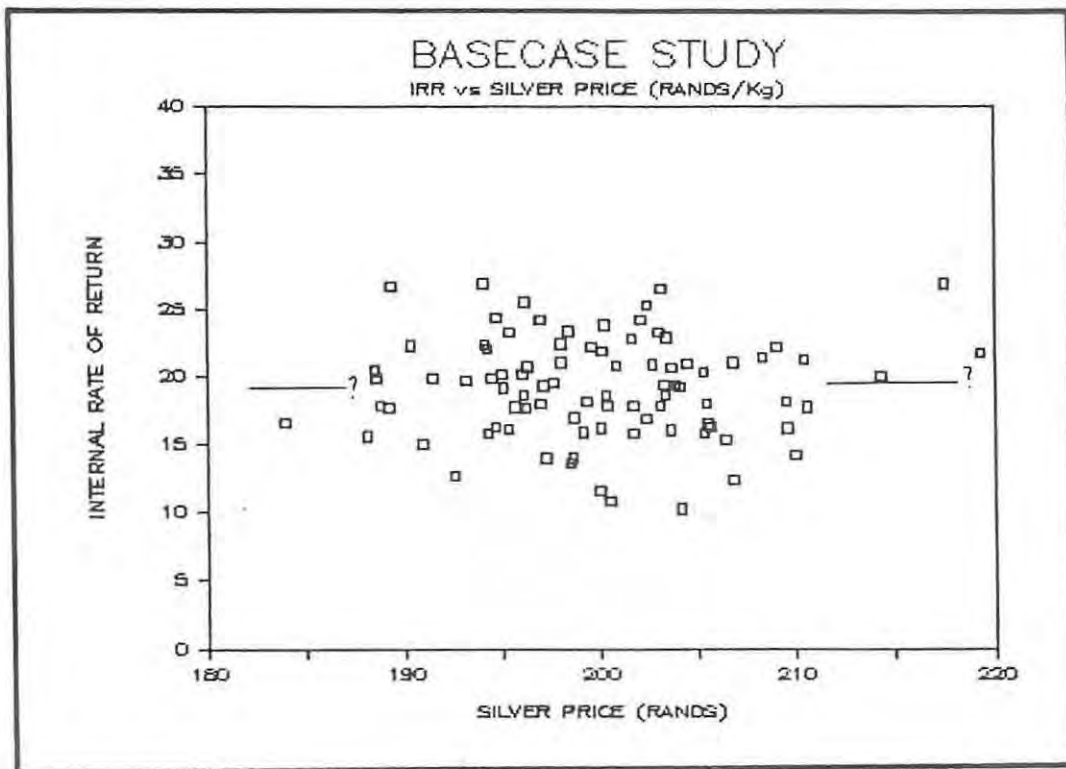


FIGURE D-9. BASECASE: IRR vs silver price. Slope poorly defined (flat).

We can also use the RISK\$ output data to compare for instance the correlation between IRR and net profit/tonne (figure D-10), or the relationship between off mine costs (realization charges) and the percentage of total ore sulphides (copper grade / concentrate grade, figure D-11).

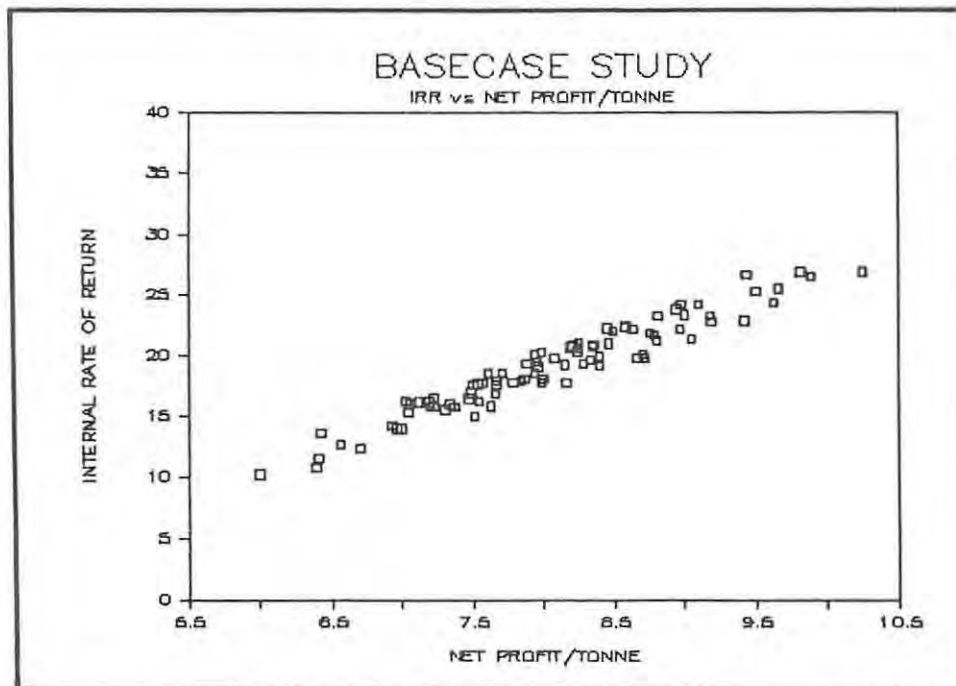


FIGURE D-10. BASECASE: IRR vs Net profit/tonne. Strong correlation.

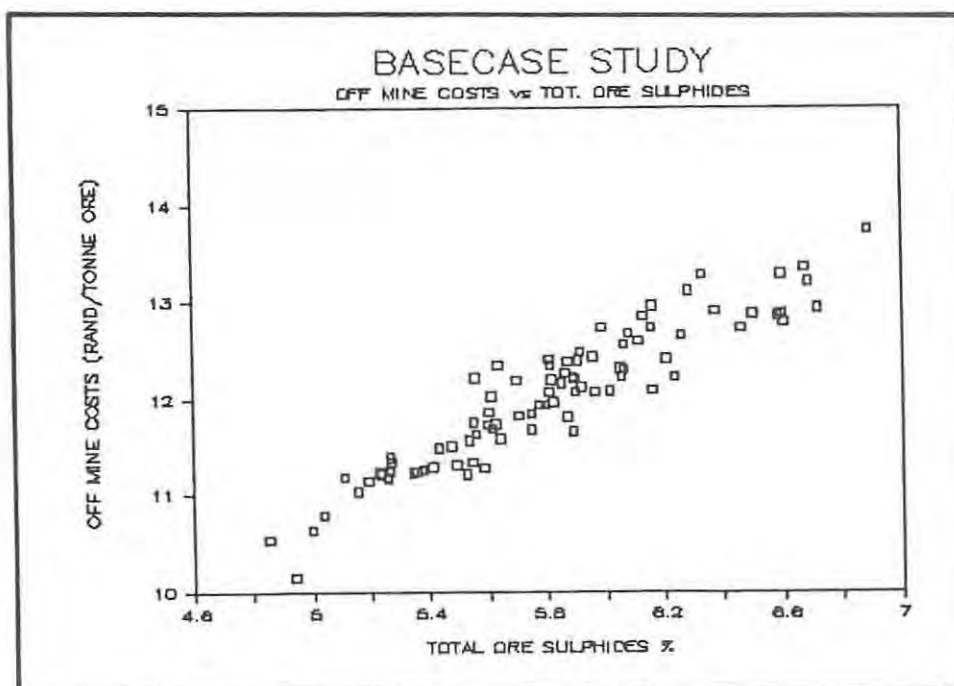


FIGURE D-11. BASECASE: OFF MINE REALIZATION COSTS vs TOTAL % ORE SULPHIDES. Strong correlation.

Figure D-12 shows a plot of IRR vs TPR and the spread or risks associated with the expected mean values are well portrayed. Contrast this type of result with the single point value of the type illustration in Figure B-1.

Also interesting is a plot of annual tonnage versus IRR (figure D-13). From the specified expected mean reserves (and standard deviations), RISK\$ selects two distinct possible annual capacity tonnages (800 000 and 850 000) as the random values are rounded to the nearest 50 000 (see equations in figure C-9). Notice that the spread of the likely IRR is greater for the lower tonnage, and that the most likely IRR is slightly higher for the higher tonnage.

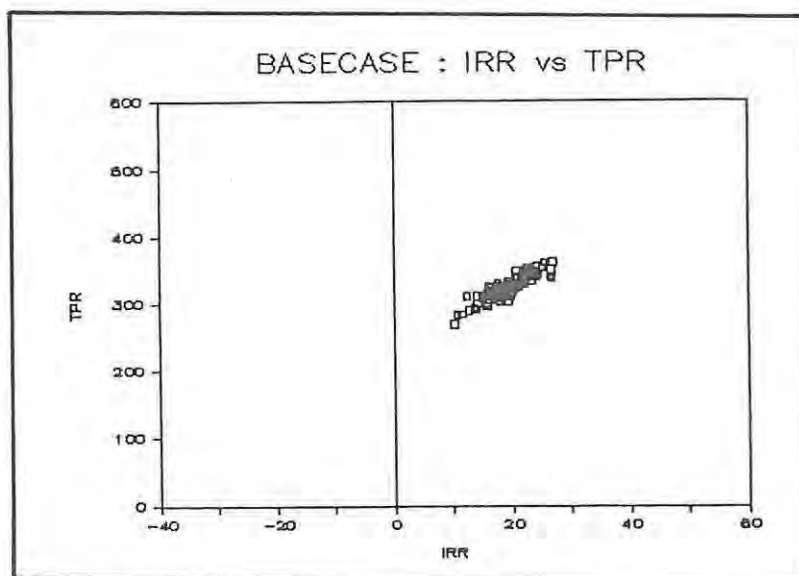


FIGURE D-12. BASECASE: TPR vs IRR. Contrast this type of result to that shown in figure B-1.

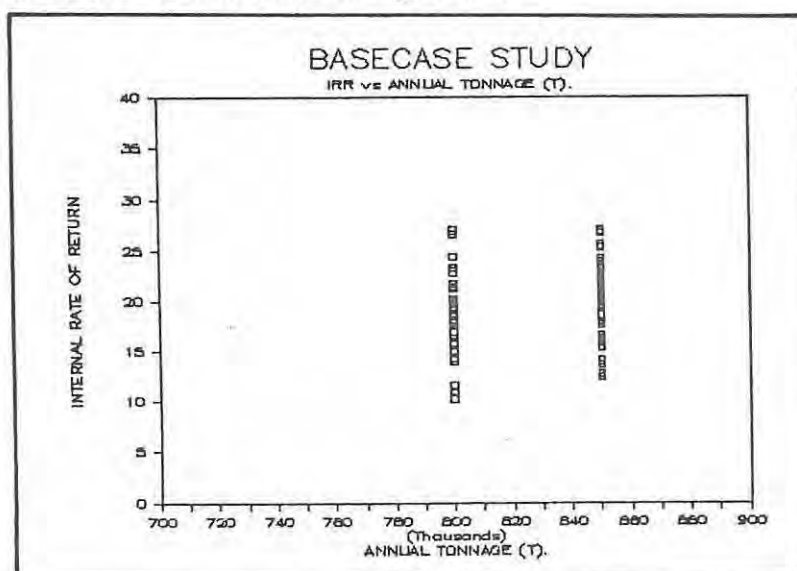


FIGURE D-13. BASECASE: IRR vs Annual capacity tonnage.

RISK\$ may also be used for conventional sensitivity analysis simply by setting the input data standard deviations all at zero, and then successively altering each of them in turn. Figure D-14 shows a listing of results from a number of such runs (referred to as BASESINGLE results, SINGLE parameter altered). These results are most conveniently expressed and summarized graphically (figure D-15) and once again BASECASE sensitivity is proportional to the slopes of the regression lines. Notice from figure D-15 that whereas BASECASE is in fact most sensitive to changes in the copper price, it is also extremely sensitive to the Favourability Factor (FF). A 10% change in the FF for example will influence the economic outcome of BASECASE more than a 10% change in ANY of the other specified risk variables, other than the copper price.

It is important to appreciate that the relative sensitivity and the degree of sensitivity to the different risk variables is deposit specific and would be different for different deposits. Note also that the degree of sensitivity and hence the sensitivity ranking for a single deposit may change in response to changing conditions, such as increased realization charges, increased commodity prices, increased energy bills etc, and that such internal changes would also be different for each and every individual ore deposit.

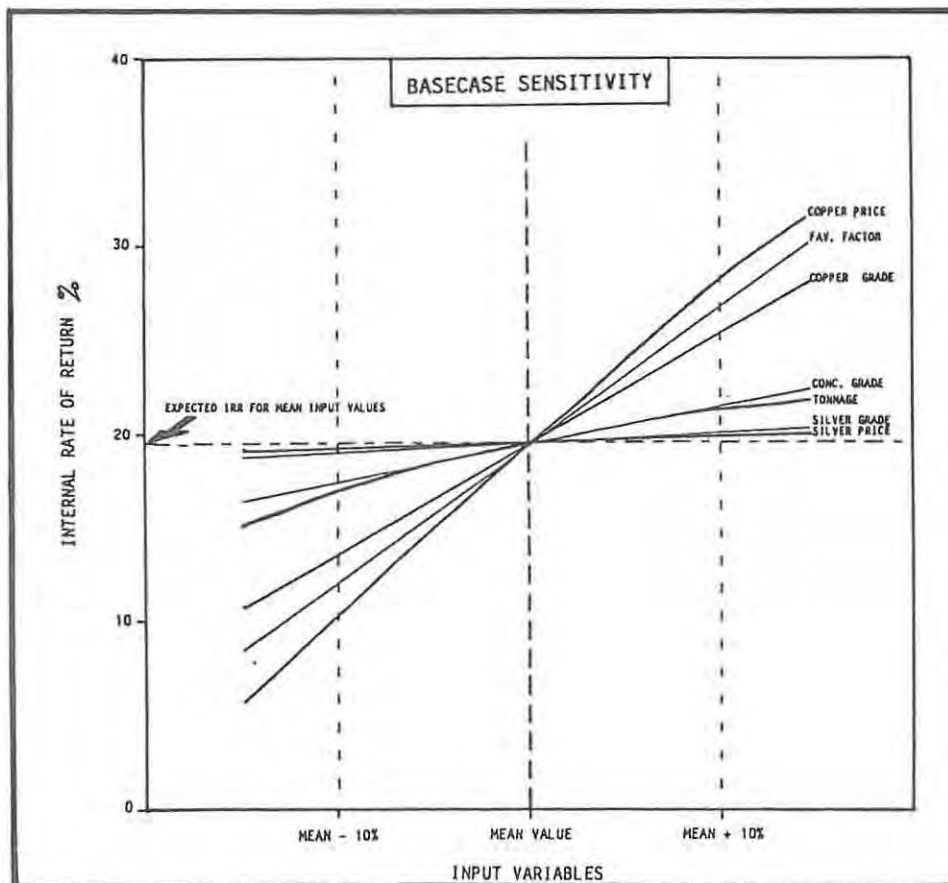


FIGURE D-15. BASECASE SENSITIVITY summary. IRR sensitivity to changes in prices and major geological factors indicated.

BASESINGLE RESULTS

Cu PRICE VARIABLE			COPPER GRADE VARIABLE					
CuPr	IRR	TOTAL POTREV	CuGr	IRR	TOTAL POTREV			
1800.00	10.08	296.82	1.80	13.57	296.82			
1900.00	15.00	312.46	1.90	16.68	312.46			
<u>2000.00</u>	<u>19.62</u>	<u>328.10</u>	<u>2.00</u>	<u>19.62</u>	<u>328.10</u>			
2100.00	24.09	343.74	2.10	22.49	343.74			
2200.00	28.27	359.38	2.20	25.25	359.38			
=====			=====					
Ag PRICE VARIABLE			SILVER GRADE VARIABLE					
AgPr	IRR	TOTAL POTREV	AgGr	IRR	TOTAL POTREV			
180.00	19.26	326.57	8.00	18.72	325.04			
190.00	19.44	327.34	9.00	19.17	326.57			
<u>200.00</u>	<u>19.62</u>	<u>328.10</u>	<u>10.00</u>	<u>19.62</u>	<u>328.10</u>			
210.00	19.79	328.87	11.00	20.06	329.63			
220.00	19.97	329.63	12.00	20.51	331.16			
=====			=====					
CONCENTRATE GRADE VARIABLE			CONCENTRATE TRANSPORT COSTS VARIABLE					
CONGr	IRR	TOTAL POTREV	INCREMENT FACTOR	IRR	TOTAL POTREV			
31.00	17.12	328.10	TIMES 0.5	21.75	328.10			
33.00	18.44	328.10	<u>TIMES 1</u>	<u>19.62</u>	<u>328.10</u>			
<u>35.00</u>	<u>19.62</u>	<u>328.10</u>	TIMES 1.5	17.46	328.10			
37.00	20.67	328.10	TIMES 2	15.23	328.10			
39.00	21.61	328.10	TIMES 5	0.50	328.10			
=====			=====					
DELINEATION VARIABLE			WIDTH VARIABLE			FAVOURABILITY FACTOR VARIABLE		
DELINEATION COSTS	IRR	TOTAL POTREV	WIDTH	IRR	TOTAL POTREV	FF	IRR	TOTAL POTREV
500000	20.04	328.10	8.00	18.81	328.10	0.80	3.87	328.10
<u>1000000</u>	<u>19.62</u>	<u>328.10</u>	9.00	19.25	328.10	0.90	12.12	328.10
2000000	18.81	328.10	<u>10.00</u>	<u>19.62</u>	<u>328.10</u>	<u>1.00</u>	<u>19.62</u>	<u>328.10</u>
5000000	16.62	328.10	11.00	19.93	328.10	1.10	26.77	328.10
=====			=====			=====		
TONNAGE VARIABLE			DEPTH VARIABLE					
TONNAGE (MILLIONS)	IRR	TOTAL POTREV	DEPTH	IRR	TOTAL POTREV			
8.00	14.62	270.20	320.00	19.95	328.10			
9.00	16.40	289.50	360.00	19.78	328.10			
<u>10.00</u>	<u>19.62</u>	<u>328.10</u>	<u>400.00</u>	<u>19.62</u>	<u>328.10</u>			
11.00	21.12	347.40	440.00	19.45	328.10			
12.00	22.57	366.70	480.00	19.29	328.10			
=====			=====					

FIGURE D-14. BASESINGLE results for a number of different variables.

EFFECTS OF MAJOR CHANGES.

Generally speaking, in a sensitivity analysis, the aim is to establish the sensitivity of a single investment alternative to limited changes in risk and other parameters. However, sensitivity analysis may also be used to assess the effects of major changes to these parameters. This analysis would normally be used to compare different deposits, although a need for major changes (and hence the need to evaluate the consequences) within individual deposits may become apparent as additional information comes to hand.

Mining Method (MM).

BASECASE is set up for open stoping mining methods (MM = 1). The effect of changing to say cut-and-fill mining (MM = 2) (necessitated for example by unfavourable orebody orientation, dimensions, regularity, continuity etc.) is dramatic as summarized below.

BASECASE RESULT - DIFFERENT MINING METHOD

MM	IRR	TPR
1	19.62	328.10
2	0.36	328.10

Notice that there is no change to the TPR, but that there is a dramatic negative shift in the expected mean IRR.

A Note on Taxation.

Taxation is briefly mentioned as it is important to note that different taxation policies will affect INDIVIDUAL DEPOSITS DIFFERENTLY, individual deposits being geologically different. A full discussion of taxation structures and all of the effects of such structures on geologically different ore deposits is beyond the scope of this dissertation and only a few of the key features are discussed.

Figure D-16 shows the effects of different tax rates on operating profits and tax rebates on capital expenditure for the BASECASE results. The latter has the more dramatic effect. If for example the capital necessary to float the BASECASE project was to come from existing pre-tax profits from some other venture within the same group or company, and was as such non-taxable, the resulting IRR changes from 19.62% to 40.96% ! This is a good example of the type alluded to in Section A with respect to present-day circumstances on the Witwatersrand goldfields. Note that the higher the rate of would-be taxation, the more dramatic the effects of financing projects with pre-taxed profits. In the extreme case of 100% taxation on existing projects, pre-tax investment of available funds in ANY project that yielded even the smallest of positive cash flows would result in INFINITE internal rates of return, there being effectively no capital investment cost to the company.

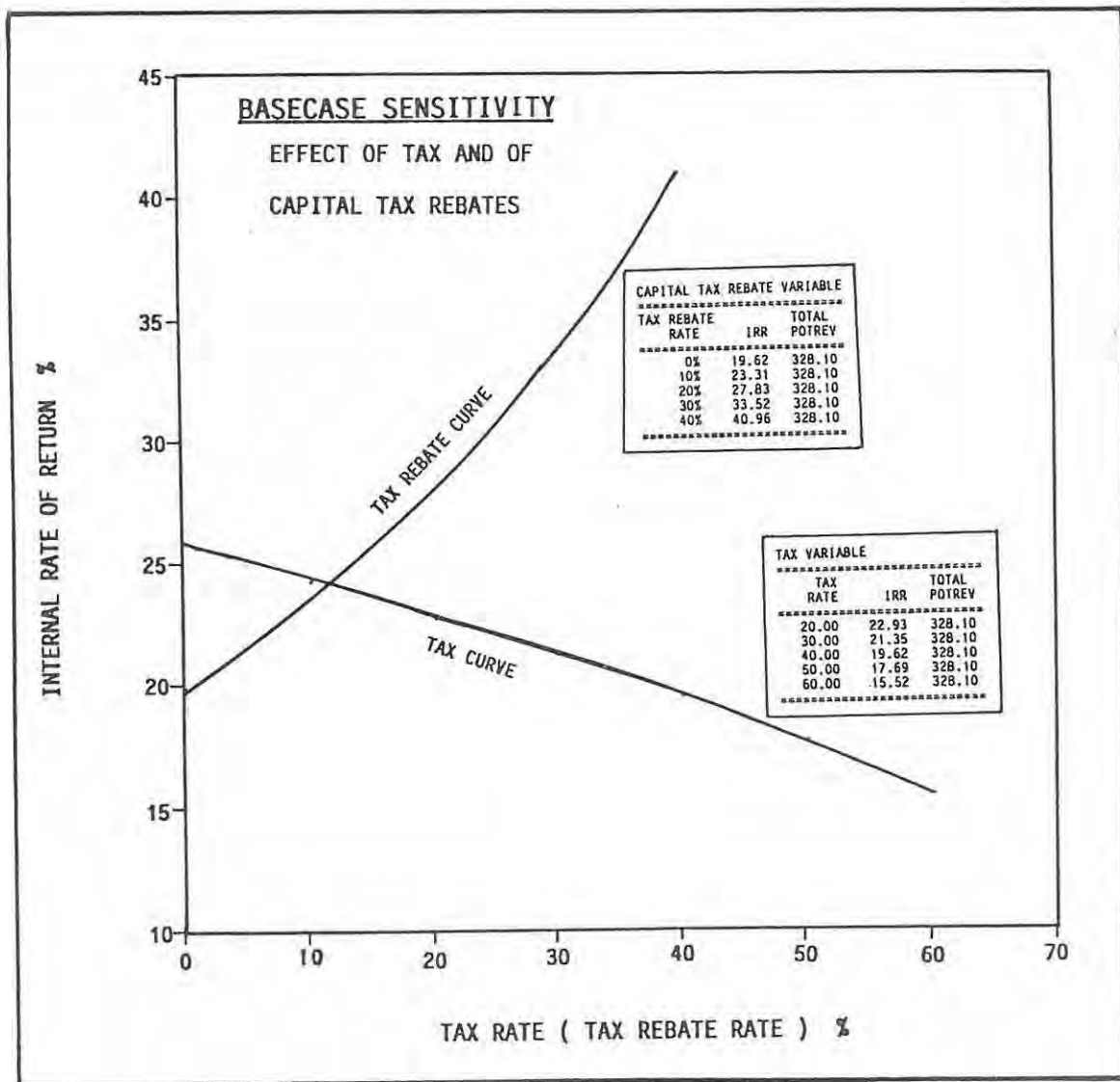


FIGURE D-16. BASECASE: Graph showing the effects of tax and capital tax rebates on BASECASE IRR.

Concentrate Grades.

Differences in concentrate grades provide perhaps the best example of the interplay of geological, geographical and time dependent factors. In the example used to illustrate this the following input parameters are used.

STANDARD CASE

	CASE A	CASE B
CONCENTRATE GRADE	25%	45%

All other parameters as per BASECASE.

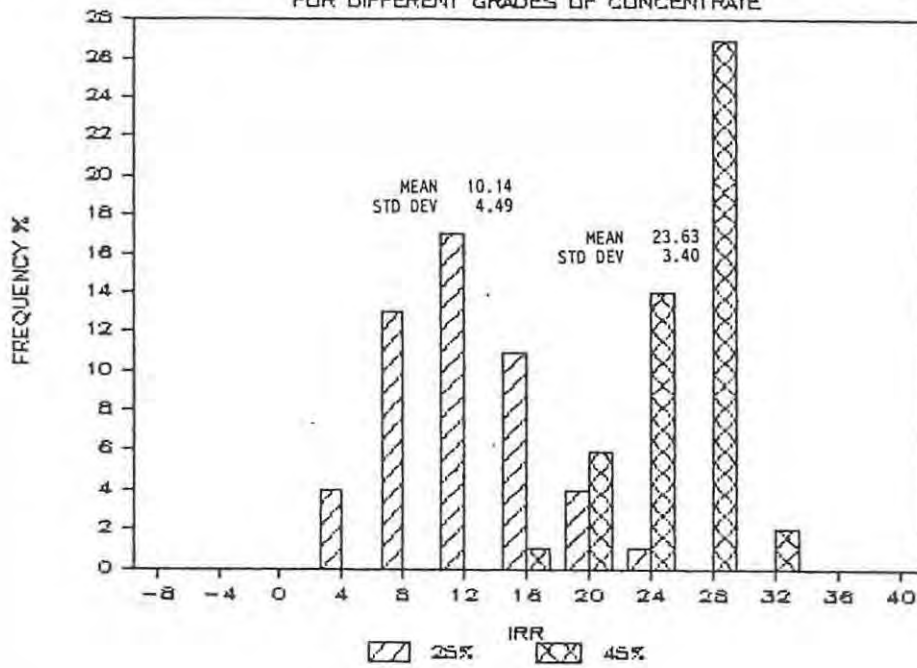
INCREASED CONCENTRATE TRANSPORT COST CASE

	CASE A	CASE B
CONCENTRATE GRADE	25%	45%
CONCENTRATE TRANSPORT COST	2 x BASECASE	2 x BASECASE

All other parameters as per BASECASE.

The results of running BASECASE RISK\$ with these altered concentrate grades and concentrate transport costs are summarized in figures D-17 and D-18. Notice in particular from the cumulative frequency curves of figure D-18B that the doubled concentrate transport costs affect the lower concentrate grade case (CASE A) more than they do the higher concentrate grade case (CASE B). Thus CASE A is MORE SENSITIVE to any changes in or changes affecting concentrate grades and related costs than is CASE B.

IRR HISTOGRAMS, STD. CONC. TRANS. COSTS FOR DIFFERENT GRADES OF CONCENTRATE



IRR HISTOGRAMS, DBL. CONC. TRANS. COSTS FOR DIFFERENT GRADES OF CONCENTRATE

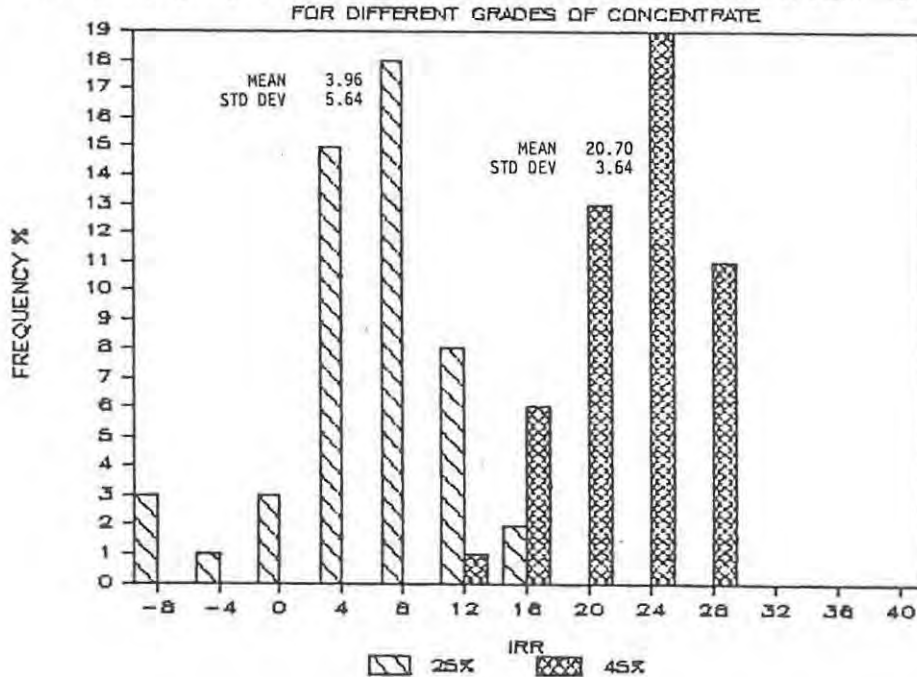
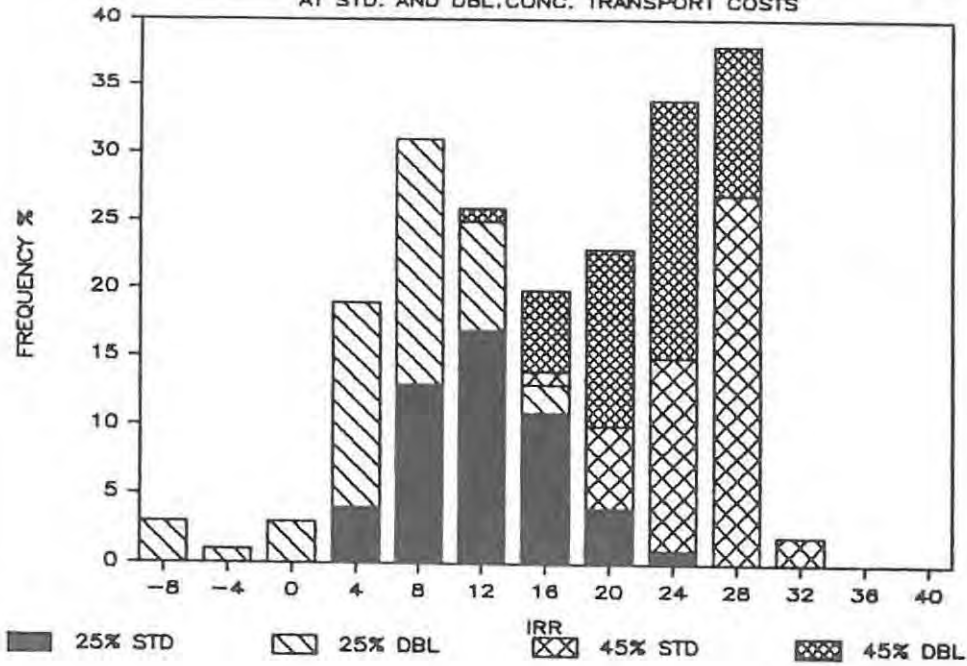


FIGURE D-17. IRR frequency percentage histograms for BASECASE data with different concentrate grades at standard and doubled BASECASE concentrate transport costs.

IRR HISTOGRAMS, DIFF. GRADES OF CONC.

AT STD. AND DBL. CONC. TRANSPORT COSTS



CUM. FREQ. % FOR DIFF. GRADES OF CONC.

AT STD. AND DBL. CONC. TRANSPORT COSTS

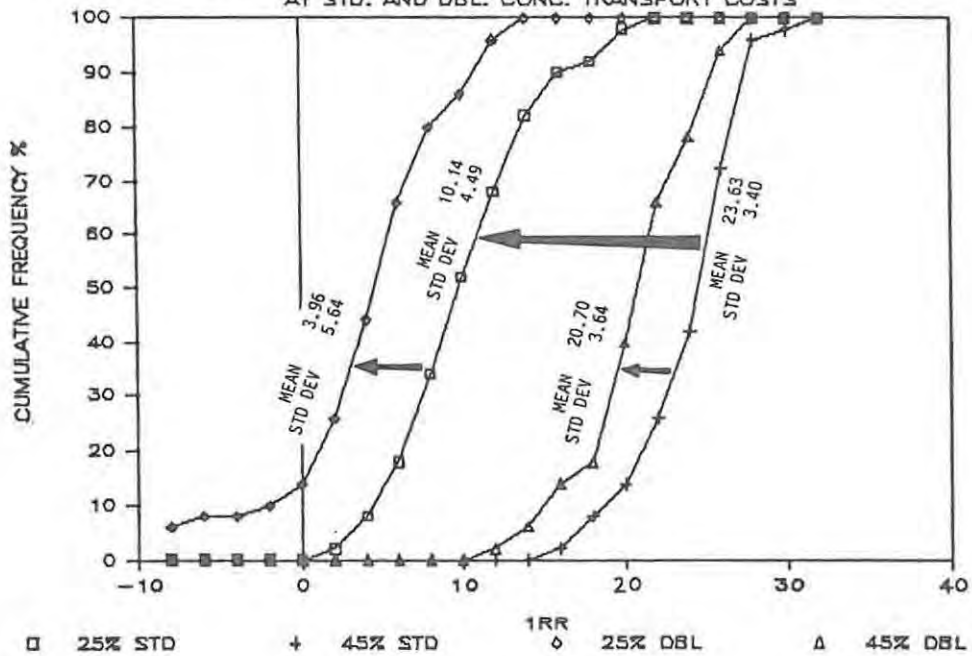


FIGURE D-18. Stacked IRR frequency percentage histograms and cumulative frequency percentage curves for BASECASE data with different concentrate grades at standard and doubled transport costs.

EFFECTS OF PRICE CYCLES AND GRADE SELECTION OPTIONS.

Although the correlation between the IRR and the average net profit per tonne for the BASECASE is excellent (see figure D-10), it is not exact. This is because the economic result in terms of the IRR is TIME DEPENDENT, whereas the average net profit per tonne is not. As a result of the time dependence of the IRR, the IRR will ALWAYS improve for the same average prices and grades if either the prices or the grades are higher than average to start with (in the early cash flow periods). Likewise, lower than average initial grades or prices or both will lead to a decrease in the IRR.

In many ore deposits, possibilities exist for increased grades in the early years followed by decreasing or constant but lower than average grades thereafter. The most common example of this would be the early extraction of material from a zone of secondary enrichment. Other examples might include zoned-orebodies or orebodies that are defined by cutoff grades. Note also that not all orebodies have sufficient variation of grade to enable early high grading to take place, nor can practical advantage always be taken of orebodies that may well be zoned, due to such factors as minimum mineable width, mining methods, minimum mine block sizes etc..

Medium term commodity prices generally follow a cyclical trend, average prices within cycles being either lower or higher than the long-term price trends (see figure B-4). An initial positive to negative medium term price cycle would increase the IRR of an investment alternative as compared to a flat trend, as would a negative to positive price cycle decrease the expected IRR.

In analysing the effects of both grade variations and price cycles on the BASECASE, the following cycles and grade variation multipliers were used.

FOR GRADE:

GRADE SEQUENCE	CASH FLOW YEAR.										AVERAGE
	1	2	3	4	5	6	7	8	9	10	
HIGH-LOW	1.2	1.2	.95	.95	.95	.95	.95	.95	.95	.95	1
FLAT	1	1	1	1	1	1	1	1	1	1	1
LOW-HIGH	.95	.95	.95	.95	.95	.95	.95	.95	1.2	1.2	1

FOR PRICES:

PRICE CYCLE	CASH FLOW YEAR.										AVERAGE
	1	2	3	4	5	6	7	8	9	10	
POS-NEG	1	1.1	1.2	1.2	1.1	0.9	0.8	0.8	0.9	1	1
FLAT	1	1	1	1	1	1	1	1	1	1	1
NEG-POS	1	0.9	0.8	0.8	0.9	1.1	1.2	1.2	1.1	1	1

Figure D-19 shows six examples of randomly chosen copper prices (short term fluctuations) within a medium term positive-negative price cycle. The data was generated by RISK\$.

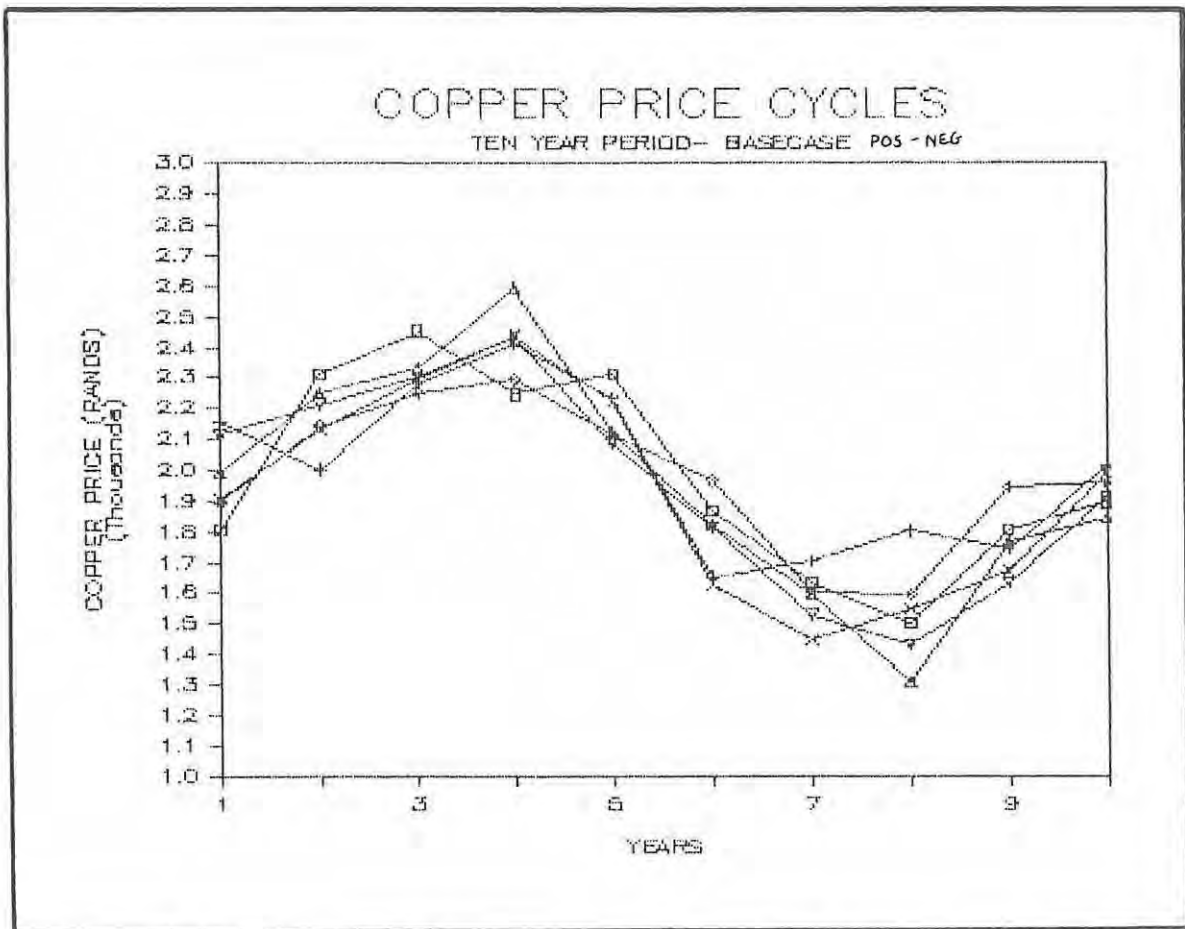


FIGURE D-19. Example of RISK\$ copper price simulations for BASECASE means and standard deviations and for a positive to negative medium term price cycle.

The results of grade sequencing and price cycle variations on BASECASE are displayed in figure D-20. As would be expected (from the sensitivity analysis of the BASECASE) silver grade sequencing and silver price cycle variations have little marked effect on the BASECASE IRR. However, both copper grade sequencing and copper price cycle alternatives have marked effects, whereas concentrate grade sequencing has a notable effect. The two extremes, copper, silver grades and concentrate grades high to low, prices positive to negative and the exact reverse of this, shift the mean IRR from 19.62% to 29.82% and 14.11% respectively, representing very significant shifts, particularly the upside potential shift.

CuGr SEQUENCE VARIABLE			Cu PRICE CYCLE VARIABLE		
Cu GRADE SEQUENCE	IRR	TOTAL POTREV	CuPr CYCLE	IRR	TOTAL POTREV
HIGH-LOW	23.17	328.10	POS-NEG	23.56	328.10
FLAT	19.62	328.10	FLAT	19.62	328.10
LOW-HIGH	17.84	328.10	NEG-POS	15.95	328.10

AgGr SEQUENCE VARIABLE			Ag PRICE CYCLE VARIABLE		
Ag GRADE SEQUENCE	IRR	TOTAL POTREV	AgPr CYCLE	IRR	TOTAL POTREV
HIGH-LOW	19.85	328.10	POS-NEG	19.76	328.10
FLAT	19.62	328.10	FLAT	19.62	328.10
LOW-HIGH	19.48	328.10	NEG-POS	19.47	328.10

CONGr SEQUENCE VARIABLE		
CON. GRADE SEQUENCE	IRR	TOTAL POTREV
HIGH-LOW	20.21	328.10
FLAT	19.62	328.10
LOW-HIGH	19.06	328.10

BASESINGLE RESULT - Ag,Cu & CONGr SEQUENCE VARIABLE							
Ag,Cu CONGr SEQUENCE	IRR	TOTAL POTREV	WITH:	POS-NEG PRICE CYCLE IRR	PRICE CYCLE TPR	NEG-POS PRICE CYCLE IRR	PRICE CYCLE TPR
HIGH-LOW	24.49	328.10		29.82	328.92	19.66	327.28
FLAT	19.62	328.10		23.72	328.10	15.82	328.10
LOW-HIGH	19.48	328.10		20.73	327.28	14.11	328.92

FIGURE D-20. Summary of results of grade sequencing and price cycle variations on BASECASE data. BASECASE settings outlined in diagram.

Whereas it is acknowledged that price cycle prediction may be rather hazardous, proper grade and ore mineral distribution delineation is less so, and one of the important aims of any delineation programme should be an assessment of the grades distributions (not only the average grade values!). This in turn should be translated into an estimate of the degree of mining FLEXIBILITY. High flexibility implies that adjustments can be made when it is known that prices are high, or low, during the actual production phase of the mine.

Unfortunately, the best techniques for assessing grade distributions and variances and mining block values (geostatistical techniques) are not particularly successful or applicable during the delineation stages, as they generally require large amounts of closely spaced samples, samples that must be spaced closely enough to be recognisably correlated. This is seldom the case during most borehole delineation phases.

For the above reasons, it is the geologist and ONLY the geologist who can make any sort of intelligent estimates from the total available data, based on his knowledge of ore deposit types and settings and mineralizing system characteristics, as well of course on the deposit specific conditions. As such, the borehole core material should most certainly not be seen as "grade or no grade", but rather as a collection of clues as to the likely dimensional configuration and internal detail of the embryonic "orebody".

A note on cutoff grades.

No attempt is made to discuss the general theory of cutoff grades as this goes way beyond the scope of this dissertation. There are however some observations and ideas that need stating, particularly as mine flexibility is often a function of cutoff flexibility. Note that an orebody envelope delineated by some or other cutoff grade boundary should be seen as a LIVING entity, continually changing its shape and dimensions as more information comes to hand and circumstances change. There should be no such thing as a cutoff decided on years ago (by someone?) and used for years into the future.

Note that a major distinction should be drawn between situations where material below the current cutoff grade is lost forever, or whether it will be extractable at a later date. Furthermore, any calculation of the optimum

cutoff grades must not only take into account which section of the production line is the limiting section (mine, mill or market (Taylor, 1972)), but it should be realized that changed cutoff grades affect both the revenues per tonne of ore mined AND the costs of mining each tonne of ore. As such, the "optimum" cutoff grade must not be derived without due consideration of changes to revenues and costs.

A risk analysis program such as RISK\$ will allow different grade-tonnage options to be compared and will take into account the changing production mining costs related to orebody dimension changes associated with different cutoff values. Ideally, a program such as RISK\$ in a modified form (modified to compute detailed monthly production cash flows) should be used to constantly update short, medium and long term mine planning, particularly if there are advantages to be had by selective mining and variable production tonnage and grade output.

AN EXPLORATION-DELINEATION SIMULATION.

INTRODUCTION.

A convenient way to illustrate the influence of geological factors on the risk element as well as the rate of return of investment alternatives is to use an exploration-delineation simulation, and to analyse the different stages of the simulation using RISK\$.

Throughout the exploration and delineation phases of the simulation, the non-geological risk variable settings are held constant. Any improvements in the spread of calculated IRR's and TPR's is thus entirely a function of more detailed and better constrained geological knowledge. Factors like the mean grade of the deposits are better defined when more samples (drillholes) are used in their determination. In this case for instance, the standard deviation of the mean is inversely proportional to the squareroot of the number of borehole samples.

SETTING UP THE SIMULATION.

The simulation is divided into four discrete stages: an exploration model stage, EXP_1; an exploration discovery stage, EXP_1ABC, representing the first phase of actual exploration; a delineation stage, divided into two delineation phases, DEL_1ABC and DEL_2ABC and a final flexibility evaluation stage, FLEX_1AB.

Note that throughout the simulation the chosen mean values are held constant and only standard deviations from these means are altered. Although this is obviously a simplification, it does not in any way distort the analysis, nor does it hamper the assessment of the influence of geological factors on investment alternatives.

Stage 1: EXP_1.

The initial stage, EXP_1 is an evaluation of a theoretical as yet undiscovered geological "model" deposit. The model deposit can be thought of as representing the result or outcome of an area resource appraisal (see Section A). The model parameters chosen to represent EXP_1 are in fact those of the BASECASE study, the difference being that the standard deviations are greater so that the model embraces a wide spectrum of similar deposit types that would average out to be the same as the BASECASE deposit, but may individually be quite dissimilar.

The input parameter settings for EXP_1 are listed in figure D-21.

Stage 2: EXP_1ABC

In the second stage of the simulation, three different prospects (A,B and C) are actually located and are defined by at least one cored drill hole intersection of a mineable width. Although limited core is available, at this stage intelligent estimates can be made as to the likely size and shape of the potential ore zone based on structural, surface geochemical and geophysical data.

The delineation costs for each of the prospects, A,B and C are R50 000 and represent the average cost of one borehole intersection. The expected mean

EXPLORATION MODEL		PARAMETER SETTINGS		FIRST EXPLORATION PHASE		FIRST DELINEATION PHASE		SECOND DELINEATION PHASE	
EXP_1				PROSPECT A : EXP_1A		PROSPECT A : DEL_1A		PROSPECT A : DEL_2A	
				MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION
THE VALUES IN THIS COLUMN ARE THE "LIKELY" RANGE OF VALUES BASED ON A GEOLOGICAL EXPLORATION MODEL FOR THE EXPECTED DEPOSIT TYPE AND SIZE. THE STANDARD DEVIATIONS REPRESENT THE ANTICIPATED RANGE OF DEPOSIT VARIATIONS.		1	1						
		1	1	R50 000	-	R500 000	-	R1 000 000	-
		1	1	6 000 000	1 000 000	6 000 000	450 000	6 000 000	100 000
		1	1	6	2	6	1	6	0.25
		1	1	1.5	0.3	1.5	0.15	1.5	0.1
		1	1	92	2	92	2	92	1
		1	1	30	5	30	3	30	0.75
		1	1	90	2	90	2	90	1
		1	1	45	3	45	2	45	1
		1	1	350	30	350	15	350	5
		1	1	1	-	1	-	1	-
		1	1	1	0.1	1.05	0.05	1.1	0.02
				PROSPECT B : EXP_1B		PROSPECT B : DEL_1B		PROSPECT B : DEL_2B	
				MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION
				R0.00	-	R500 000	-	R1 000 000	-
				10 000 000	2 000 000	10 000 000	600 000	10 000 000	100 000
				10	3	10	1	10	0.5
				2	0.5	2	0.15	2	0.1
				92	2	92	2	92	1
				10	3	10	1	10	0.5
				90	2	90	2	90	1
				35	5	35	2.5	35	2
				400	50	400	15	400	10
				1	-	1	-	1	-
				1	0.1	1	0.05	1	0.02
				PROSPECT C : EXP_1C		PROSPECT C : DEL_1C		PROSPECT C : DEL_2C	
FIXED REALIZATION COSTS AND COMMODITY PRICES USED AT ALL STAGES.				MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION
REALIZATION COSTS:									
(per appropriate unit)									
ORE TRANSPORT (OreT)	R0.00	1	1	R50 000	-	R500 000	-	R1 000 000	-
CONC. TRANSPORT (ConT)	R30.00	1	1	12 000 000	2 000 000	12 000 000	800 000	12 000 000	200 000
CONC. SMELTING (ConS)	R75.00	1	1	12	3	12	2	12	0.5
COPPER ALLOWANCE (CuAl1)	0.03	1	1	2.5	0.4	2.5	0.2	2.5	0.1
SILVER ALLOWANCE (AgAl1)	35	1	1	92	2	92	2	92	1
PENALTY UNIT (PU)	2	1	1	5	1	5	0.5	5	0.5
REFINING ETC. (REF)	R300.00	1	1	90	2	90	2	90	1
ESTIMATED METAL PRICES : MEANS AND DEVIATIONS									
COPPER :	R2 000.00 R100.00	1	1	25	3	25	2	25	1
SILVER :	R200.00 R20.00	1	1	600	40	600	15	600	10
		1	1	1	-	1	-	2	-
		1	1	1	0.1	0.85	0.05	0.95	0.03

FIGURE D-21. INPUT parameter settings as used in the EXP_1 TO DEL_2ABC simulation. Notice in particular the decreasing standard deviations on the geological data as well as the changes to the mining method (DEL_2C) and the changes to the Favourability Factor for Prospects A and C.

values (at this stage poorly defined - no real statistics possible) and their anticipated standard deviations are listed for each of the prospects in figure D-21.

Notice in particular that even at this very early stage of prospect evaluation, it may often be possible to begin assessing deposits in terms of the Favourability Factor (FF), where 1 = average, >1 = better than average and <1 = worse than average. Recall that the FF is a function of geological parameters that are liable to influence costs in a not easily definable way. Factors would include regularity, continuity, ground conditions, shearing, jointing and faulting etc., all of which can be intelligently assessed (by a geologist) as better or worse than average. The "AVERAGE" "better or worse" than average is then semi-quantified and expressed numerically as an amount more or less than unity. Uncertainty in the chosen FF is expressed in setting a mean expected FF and a standard deviation.

In this simulation, the FF at the EXP_1 and EXP_1ABC stages is set at unity with a standard deviation of 0.1.

Stage 3 - Delineation:

DEL_1ABC.

The first delineation phase represents the situation after 10 borehole intersections are available. At this stage classical statistics may start to be applied to the data. At this DEL_1ABC stage, the FF for PROSPECT A is adjusted to 1.05 (std dev = 0.05), that for PROSPECT B stays at 1 (0.05) and that for PROSPECT C, where conditions are considerably worse than average, to 0.85 (0.05). The first phase of delineation indicates that a second phase of delineation is warranted at each of the prospects.

DEL_2ABC.

The second phase of delineation represents the situation after 20 borehole intersections are available. The standard deviations of the means of the geological input parameters are now considerably reduced as contrasted with the EXP_1ABC phase and the expected IRR's and TPR's of each of the prospects are reasonably clearly resolved.

Notice in particular that the likely mining method for prospect C has been changed from open stoping (MM = 1) to cut-and-fill (MM = 2). This change was necessary when the additional information available after the second delineation phase drilling confirmed a lack of sufficient regularity and continuity for successful open stoping mining (figure D-21).

Stage 4 - Flexibility:

FLEX_1AB.

In this final stage of the simulation, an additional 20 boreholes (40 in total) are drilled at each of PROSPECTS A and B in order to test for possible flexibility, and to further constrain the geological data. Notice that PROSPECT C is rejected (or "put on ice") at the end of the second delineation stage when it is appreciated that cut-and-fill mining will be required (see summary of results).

The parameter settings for FLEX_1AB are given later after the summary of the results of the simulation from the EXP_1 stage through to the DEL_2ABC stage.

SUMMARY OF ASSUMPTIONS AND RESULTS: EXP_1 .. to .. DEL_2ABC.

The parameter settings for EXP_1 through to DEL_2ABC are summarized in figure D-21. The results of running RISK\$ with each set of parameters are summarized graphically in figures D-22 to D-25. The figures show the range of expected TPR's and IRR's in the form of scattergrams, histograms and IRR vs TPR plots. All of the figures illustrate that the range of expected TPR and IRR values is progressively reduced as more geological data becomes available.

Notice in particular that there is a dramatic negative shift in the expected mean IRR for PROSPECT C at the second delineation stage when the possibility of open-stoping mining is excluded.

The cumulative frequency percentage curves shown in figure D-26 illustrate the same changes in the narrowing down of the range of expected returns. As discussed in the BASECASE study, the slope of the cumulative frequency percentage curve steepens as the spread of the data is progressively

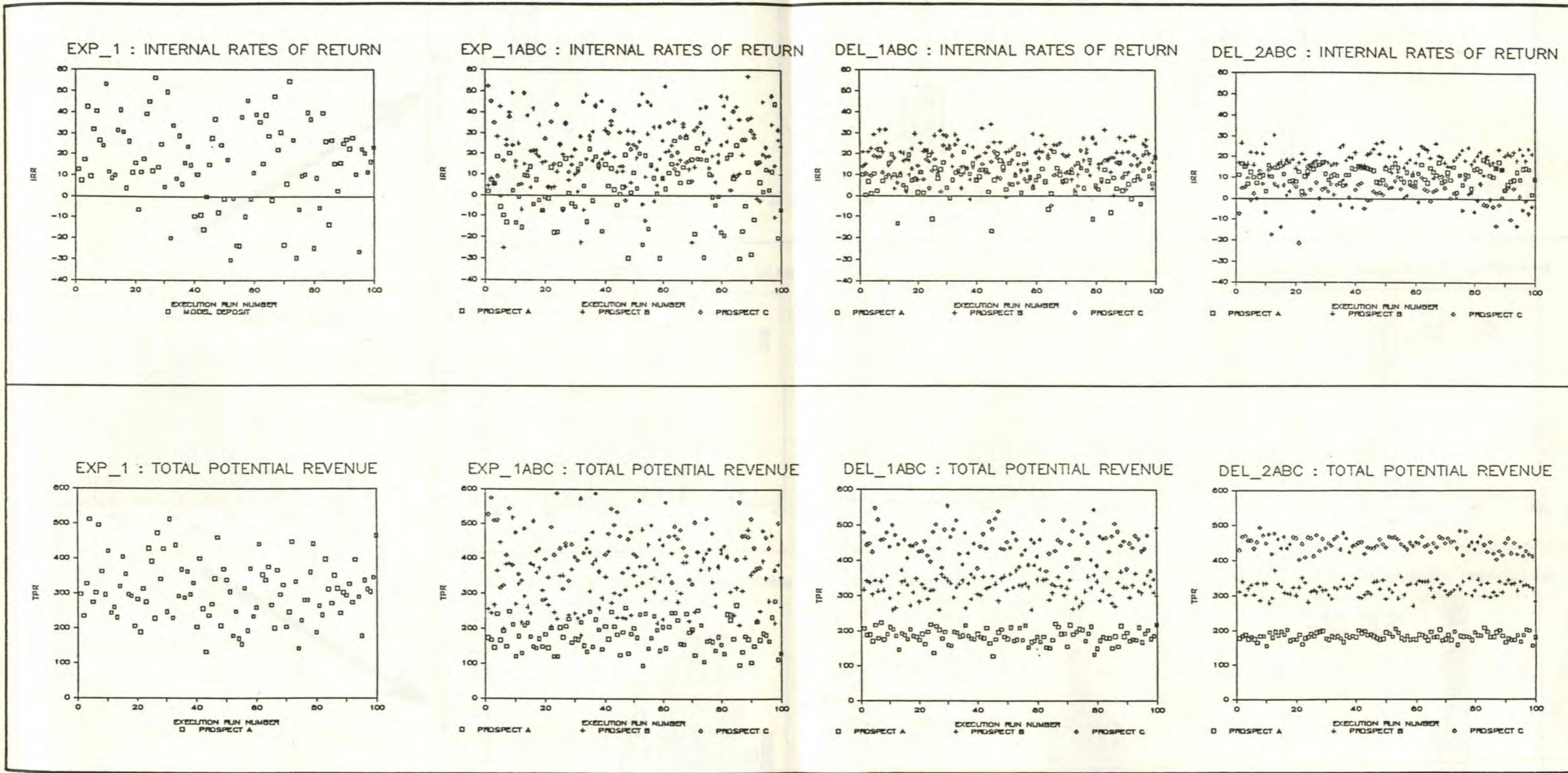


FIGURE D-22. EXP_1 - DEL_2ABC IRR and TPR scattergrams.

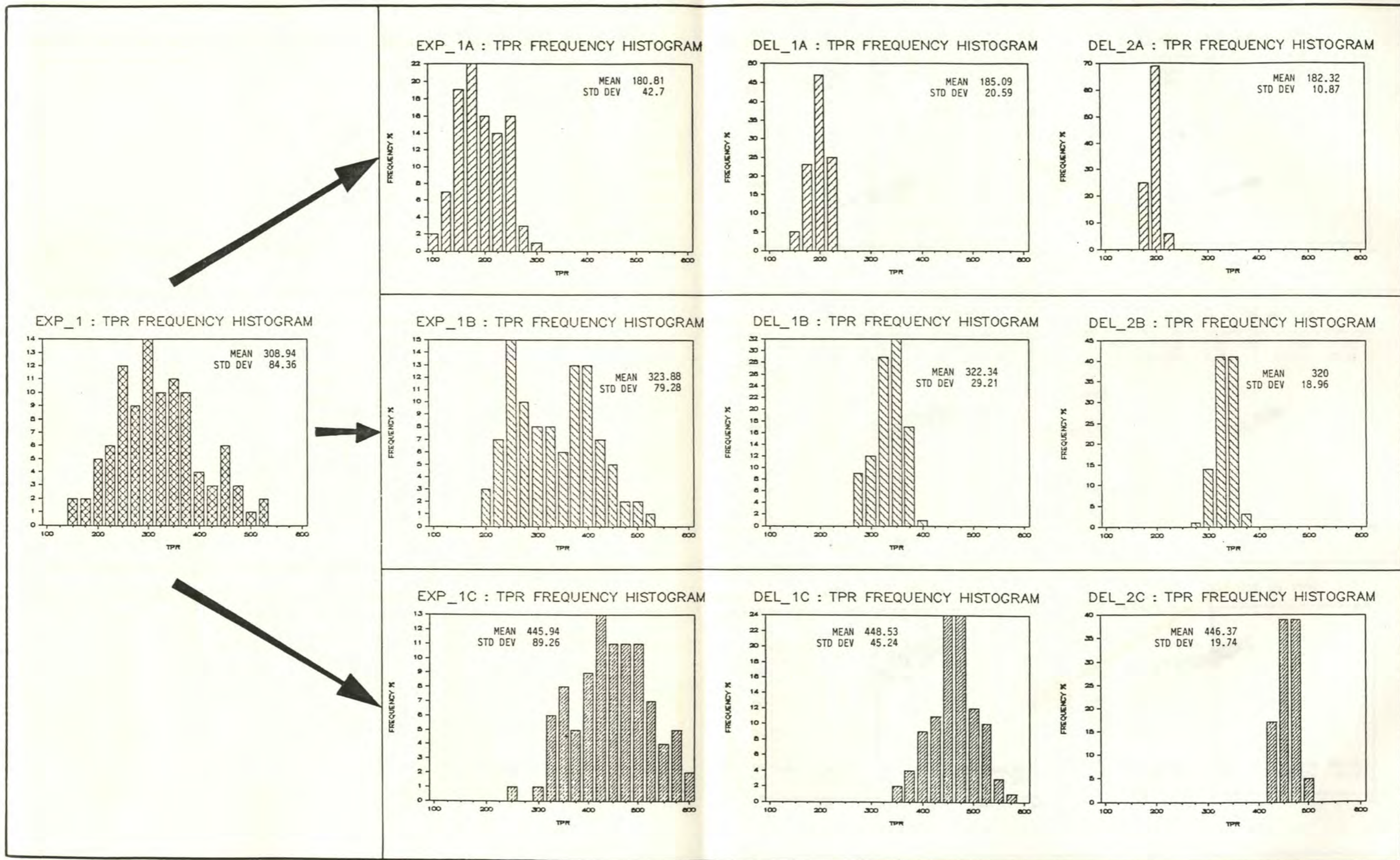


FIGURE D-23. EXP_1 - DEL_2ABC TPR frequency histograms.

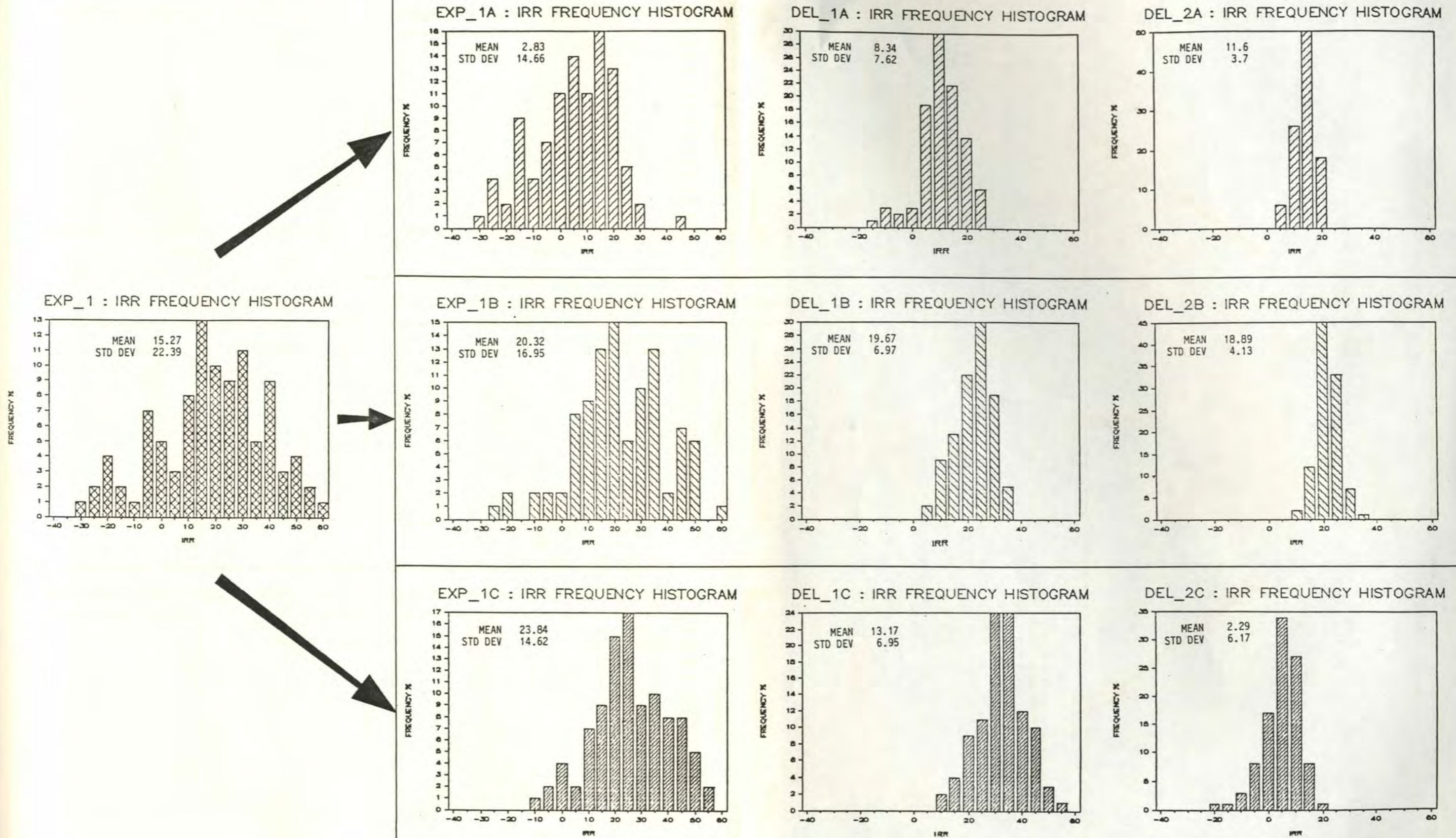


FIGURE D-24. EXP_1 - DEL_2ABC IRR frequency histograms.

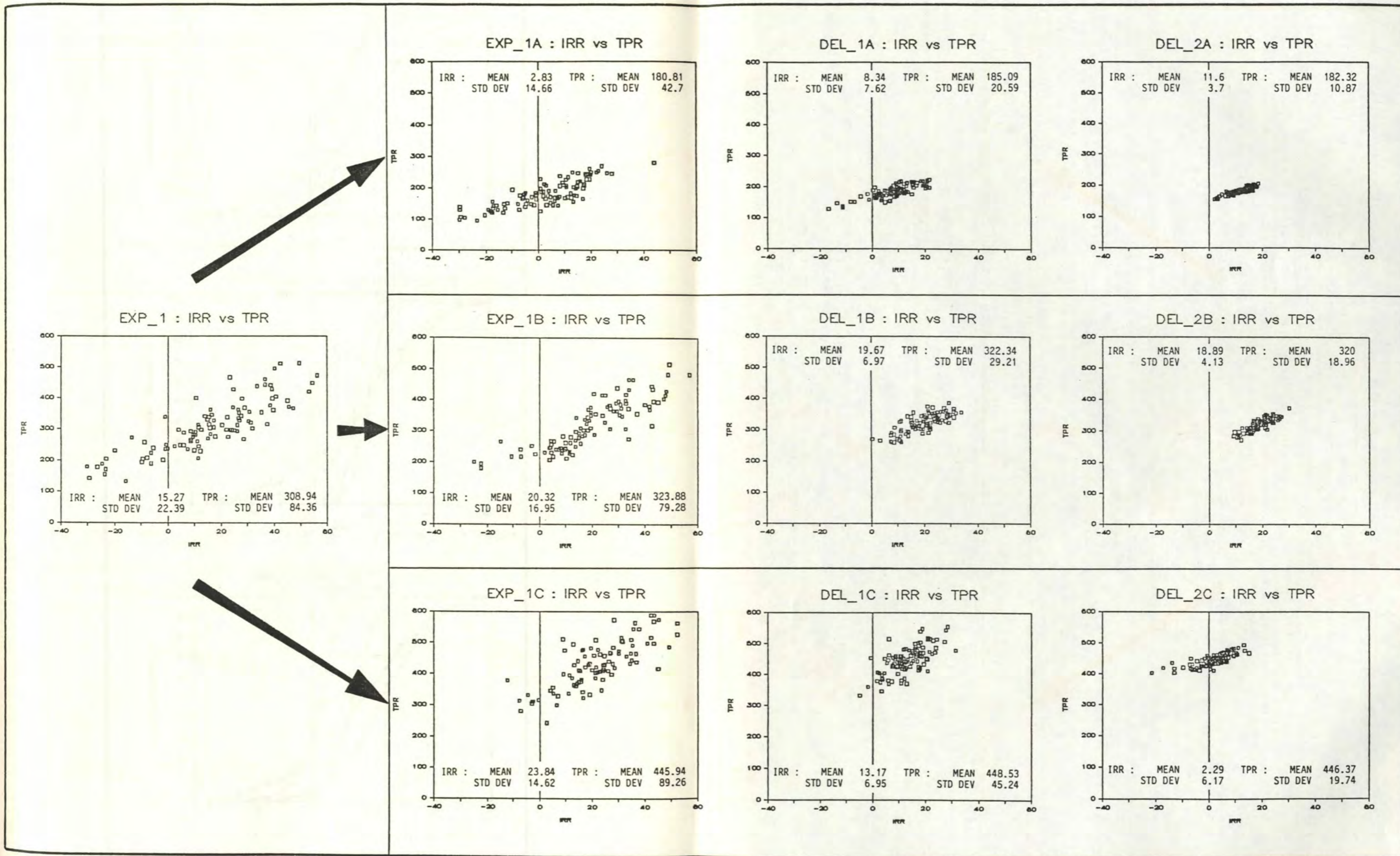


FIGURE D-25. EXP_1 - DEL_2ABC IRR vs TPR plots.

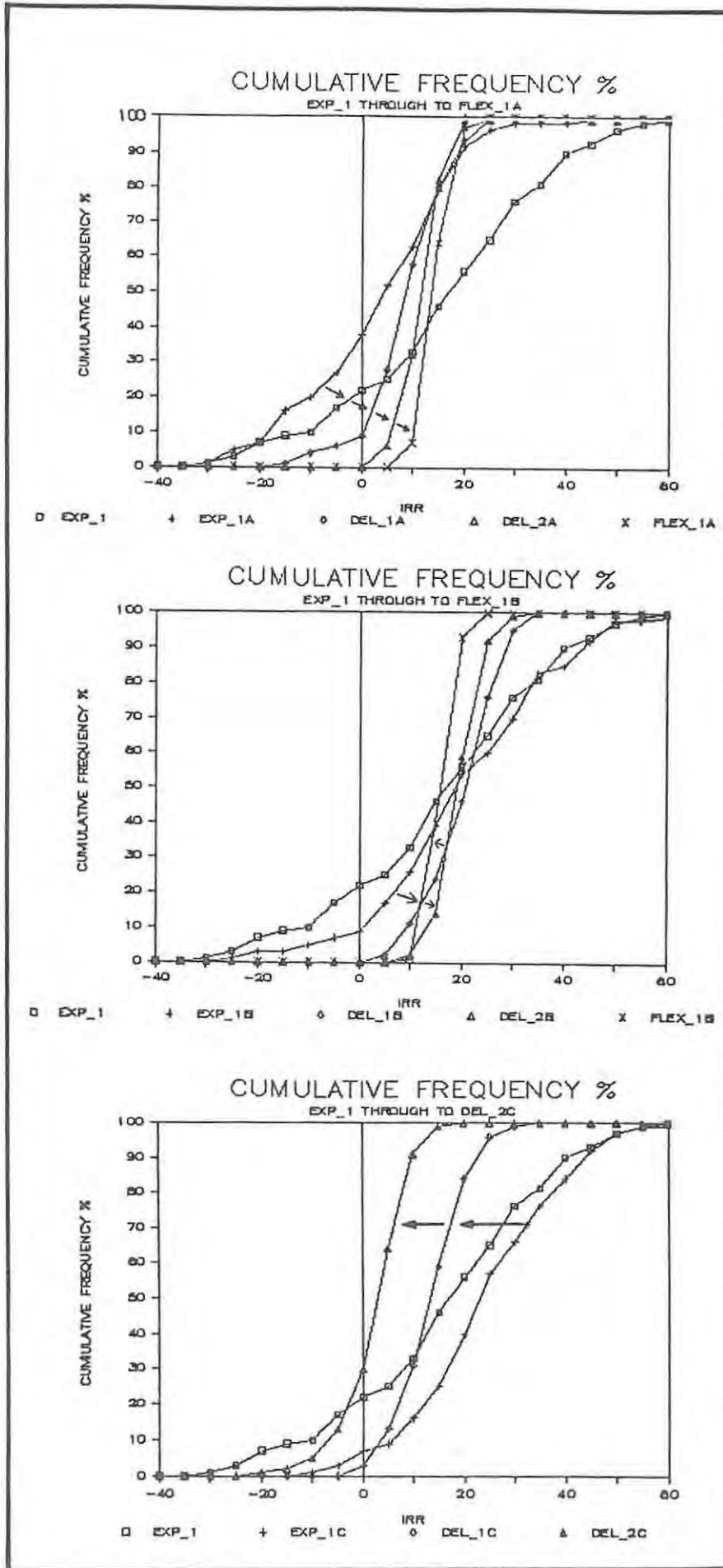


FIGURE D-26. EXP 1 - FLEX 1AB cumulative frequency percentage plots. Arrows indicate progressive shift in the curves for each of the prospects. See text for discussion.

restricted. This is well illustrated for each of the prospects. Notice that in this figure the changed mining method at the DEL_2C stage for PROSPECT C is represented by a negative translation of the cumulative frequency percentage curve.

SUMMARY OF ASSUMPTIONS AND RESULTS: DEL_2AB .. to .. FLEX_1AB.

Although post-DEL_2AB delineation further constrains the geological data and thus further limits the spread of the expected economic measures, it is subject to diminishing returns. Uncertainty associated with the non-geological risk variables, most noticeably the commodity prices, ensures that there will always be a healthy risk component to the expected returns no matter how well delineated and defined the orebody may happen to be.

The principal aim then of additional delineation drilling (and exploratory development) is (or should be) to test for FLEXIBILITY. At this advanced stage, details are required as to grade distributions and distribution variance with a view to directing the actual development planning. Can advantage be gained by different mining sequences? Is the proposed mining sequence viable or practical? Is the additional cost of a more complex mining sequence or technique warranted? etc.

In addition, this advanced stage of delineation is the time to seriously consider the effects of anticipated price cycles on the different prospects. Although this can easily be considered at earlier stages of the delineation programme, it is most useful at the stage of delineation when the grades and grade variations of the individual prospects are reasonably well constrained. Likewise, this is the appropriate stage to consider the details of FINANCING ALTERNATIVES, and the pros and cons of loans versus equity issues and the like.

In this simulation, PROSPECTS A and B are tested for flexibility, PROSPECT C having been rejected at the DEL_2C phase of the programme.

During the FLEX_1AB stage, an additional 20 boreholes are drilled into each of PROSPECTS A and B and the resultant expected improvements to the standard deviations are listed alongside the DEL_2AB results for comparison (figure D-27).

SECOND DELINEATION PHASE		FIRST FLEXIBILITY PHASE	
PROSPECT A : DEL_2A		PROSPECT A : FLEX_1A	
MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION
=====		=====	
R1 000 000	-	R2 000 000	-
6 000 000	100 000	6 000 000	70 000
6	0.25	6	0.18
1.5	0.1	1.5	0.07
92	1	92	1
30	0.75	30	0.53
90	1	90	1
45	1	45	1
350	5	350	0
1	-	1	-
1.1	0.02	1.1	0.01
+++++		+++++	
PROSPECT B : DEL_2B		PROSPECT B : FLEX_1B	
MEAN VALUE	STANDARD DEVIATION	MEAN VALUE	STANDARD DEVIATION
=====		=====	
R1 000 000	-	R2 000 000	-
10 000 000	100 000	10 000 000	70 000
10	0.5	10	0.35
2	0.1	2	0.07
92	1	92	1
10	0.5	10	0.35
90	1	90	1
35	2	35	1.5
400	10	400	0
1	-	1	-
1	0.02	1	0.01
+++++		+++++	
<p>COPPER AND SILVER GRADE SEQUENCES FOR FLEX_1A AS PER FIGURE D-28</p> <p>COPPER AND SILVER PRICE CYCLES FOR FLEX_1AB AS PER FIGURE D-29</p> <p>*****</p>			

FIGURE D-27. INPUT parameter settings as used in the DEL_2ABC to FLEX_1AB simulation.

Grades.

The FLEX_1AB delineation programme indicates that PROSPECT A has a zone of supergene enrichment and the possibility of highgrading for the first two years exists. PROSPECT B on the other hand has no such zone of enrichment nor does it have any significant grade zonation and it will have to be mined at approximately average grades throughout its working life.

In the case of PROSPECT A, the copper, silver and concentrate grades are all increased by a factor of 1.2 for the first two years, and thereafter reduced by a factor of 0.95 (figure D-28).

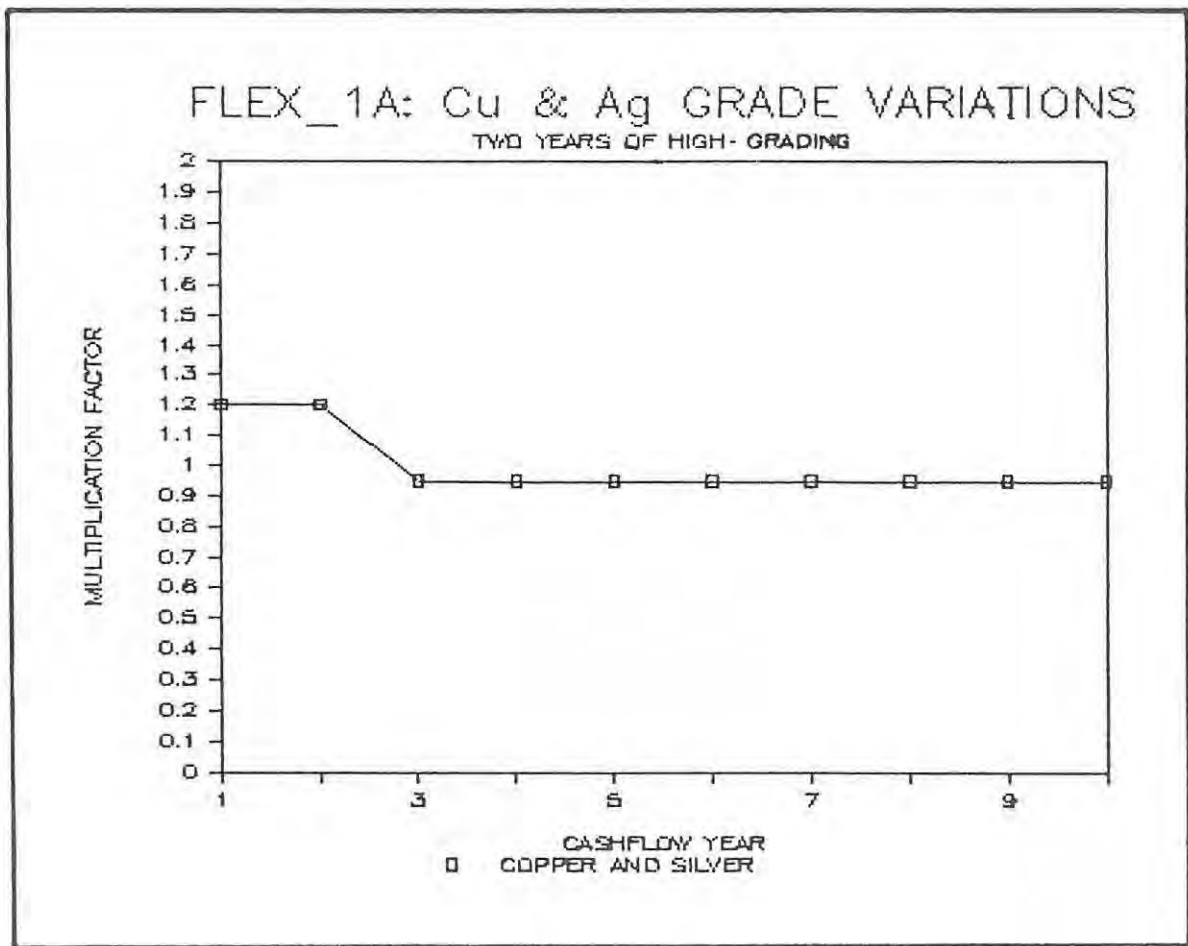


FIGURE D-28. PROSPECT A grade sequencing used in FLEX_1A simulation.

Prices.

The anticipated medium term price cycles for copper and silver are given in figure D-29. The anticipated copper price cycle is negative to positive and that for silver positive to neutral.

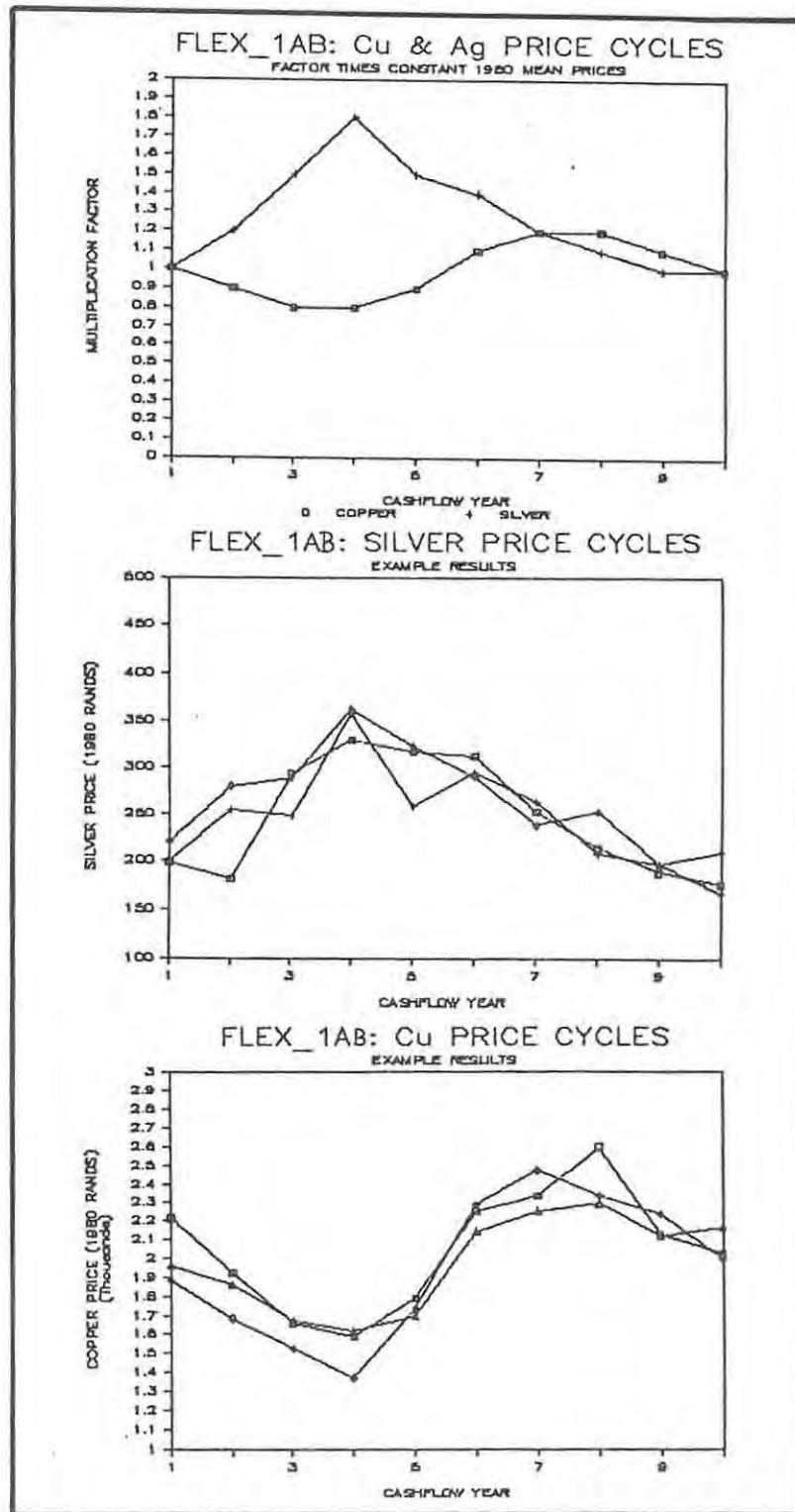


FIGURE D-29. Copper and silver price cycles used in FLEX_1AB simulation.

Results.

The results of the FLEX_1AB RISK\$ simulation are summarized in figures D-30 to D-34 that portray both the DEL_2ABC and the FLEX_1AB results in the same form as the EXP_1 to DEL_2ABC results but use an expanded IRR scale and a smaller class interval for the histograms to provide increased definition.

The main results are perhaps best summarized in the IRR cumulative frequency percentage curves shown in figure D-34. The form and smoothness of the curves improves both for prospect A and B as compared to the DEL_2AB phase. Both curves are also steeper, indicating the reduced spread in the geological data at the FLEX_1AB stage. Notice in particular that for the assumed medium term price cycles, and for the established grade sequencing possibilities, PROSPECT A shows an improvement in the IRR despite the anticipated negative to positive copper price cycle whereas PROSPECT B shows a decrease in the expected IRR.

The examples and assumed price cycles illustrate a number of features. One of these is the advantage of initial high grading enjoyed by PROSPECT A. Another is the advantage of a healthy by-product, also enjoyed by PROSPECT A. Both prospects have silver as a by-product but PROSPECT A has five times the silver grade of PROSPECT B, and as it has an appreciably lower copper grade, it is LESS SENSITIVE to copper price changes than PROSPECT B, and in many ways more FLEXIBLE.

If a choice had to be made between PROSPECTS A and B at this stage, at least that choice could now be made on well defined parameters. PROSPECT B would require a larger capital investment (bigger deposit) and has the larger TPR. The FLEX_1AB study indicates that there is little to choose between the prospects in terms of expected IRR's.

Detailed sensitivity studies should now be conducted on each of the prospects and these studies, together with the anticipated commodity price variations and all of the various expected cost escalations would enable a final EDUCATED DECISION to be taken, a decision that could be based on company or group requirements of minimum RISK AND REWARD criteria.

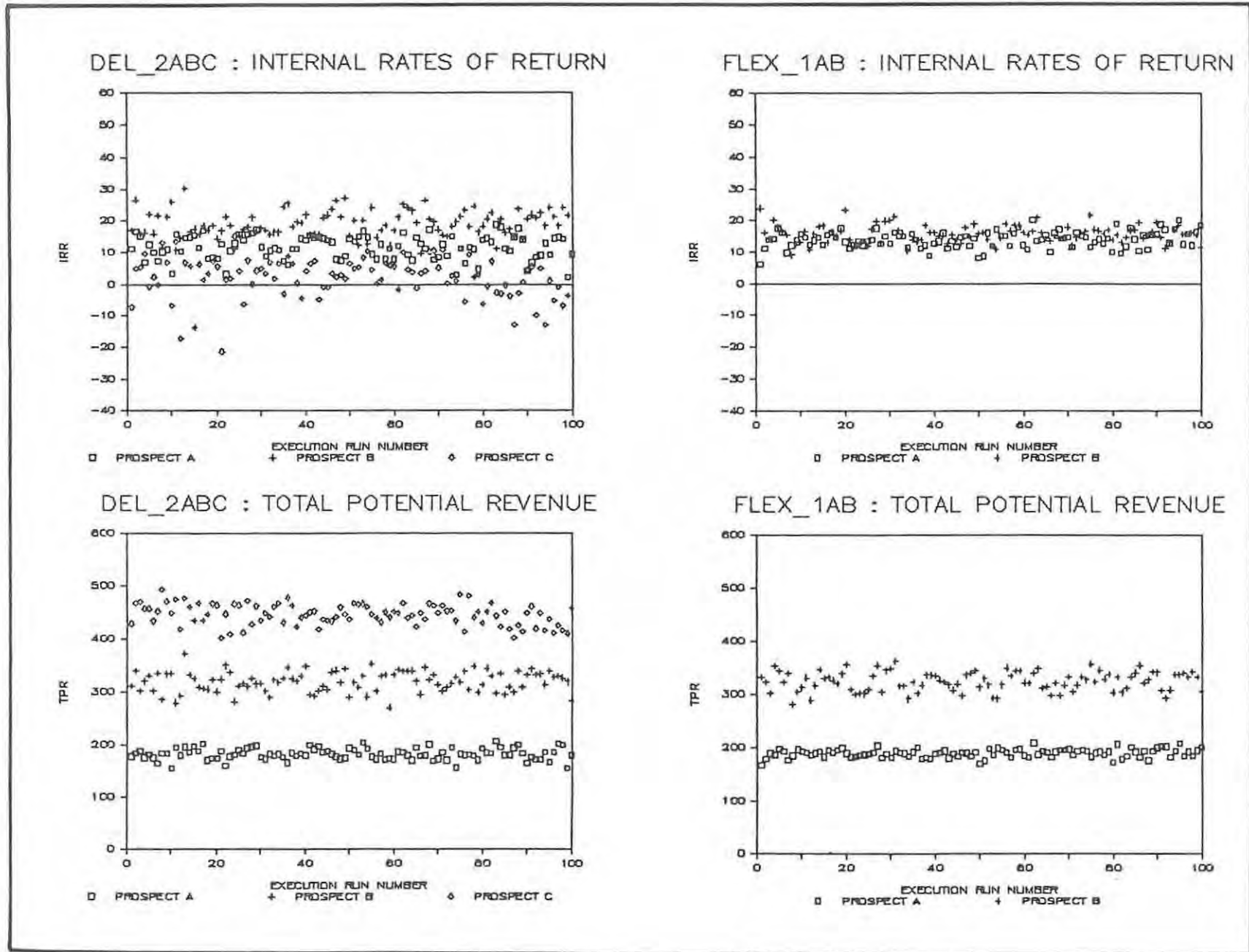
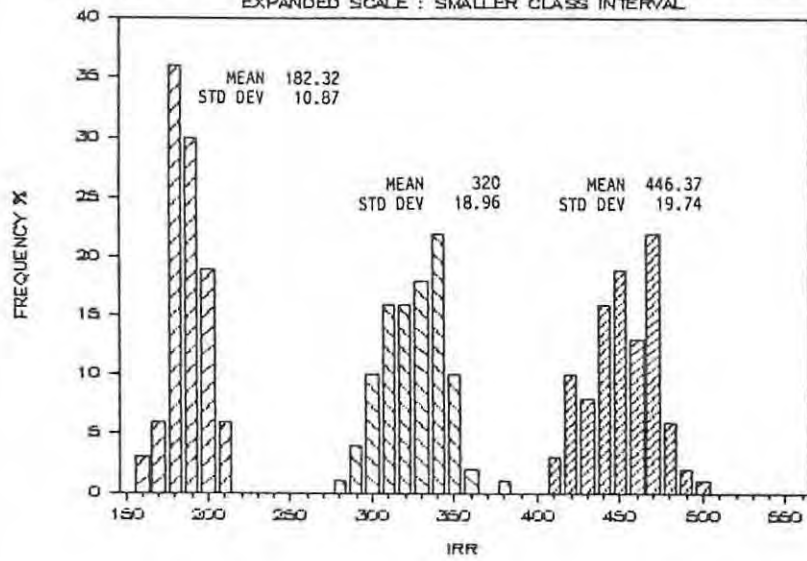


FIGURE D-30. DEL_2ABC - FLEX_1AB IRR and TPR scattergrams.

DEL_2ABC : TPR FREQUENCY HISTOGRAMS

EXPANDED SCALE : SMALLER CLASS INTERVAL



FLEX_1AB : TPR FREQUENCY HISTOGRAMS

EXPANDED SCALE : SMALLER CLASS INTERVAL

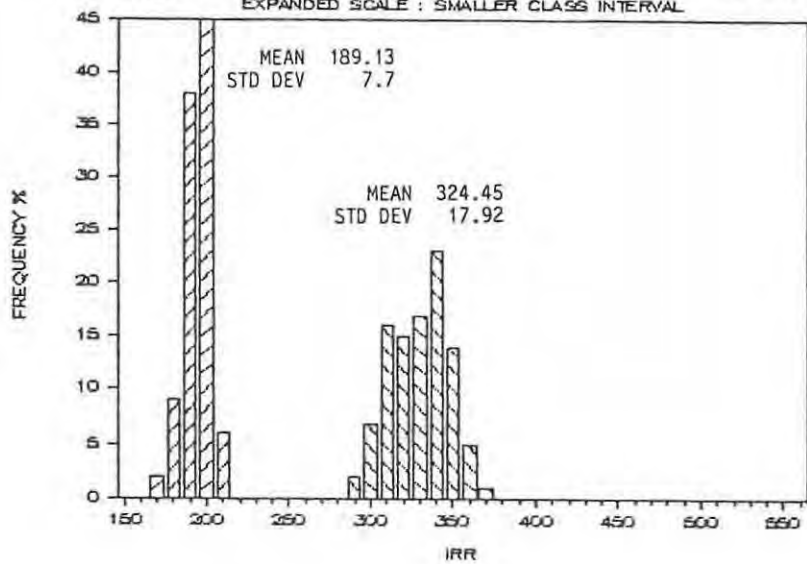
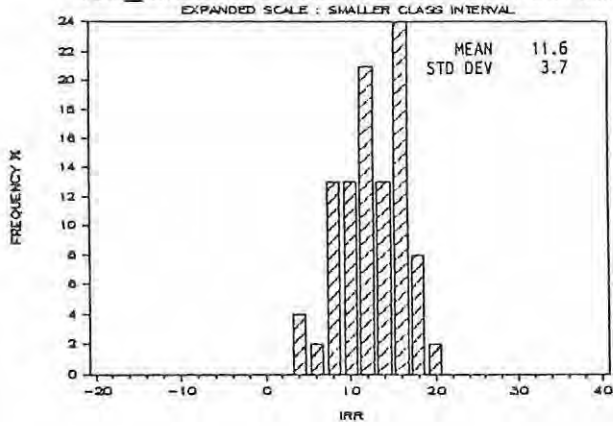
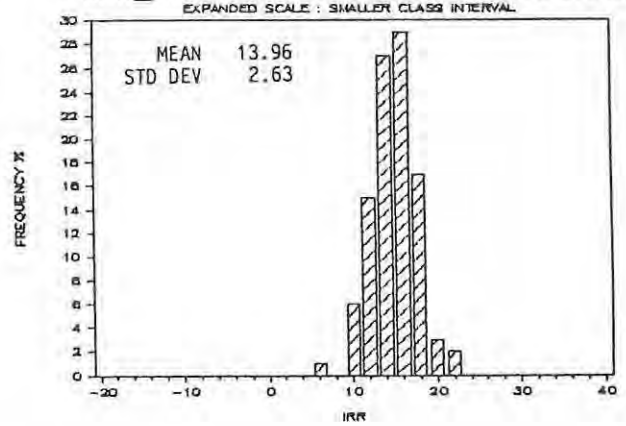


FIGURE D-31. DEL_2ABC - FLEX_1AB TPR frequency histograms.

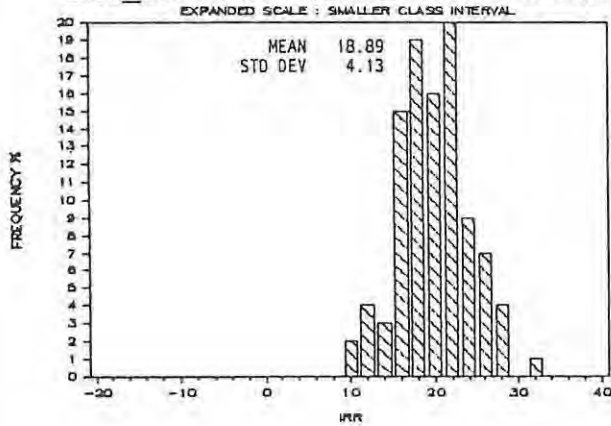
DEL_2A : IRR FREQUENCY HISTOGRAM



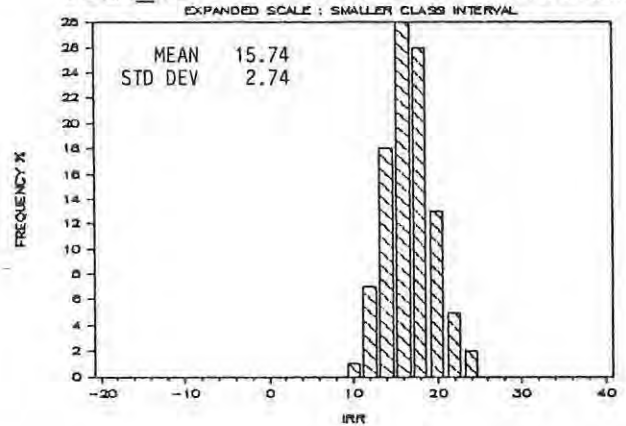
FLEX_1A : IRR FREQUENCY HISTOGRAM



DEL_2B : IRR FREQUENCY HISTOGRAM



FLEX_1B : IRR FREQUENCY HISTOGRAM



DEL_2C : IRR FREQUENCY HISTOGRAM

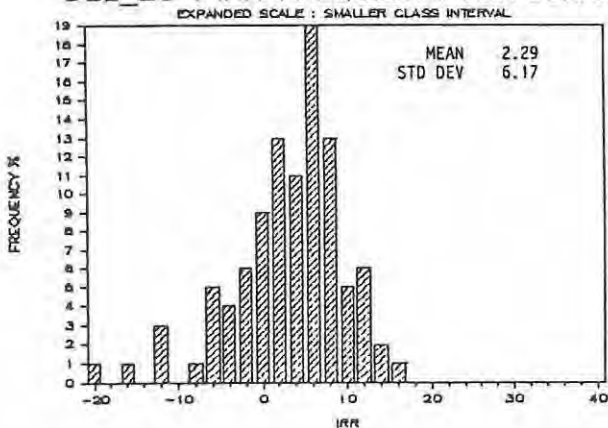


FIGURE D-32. DEL_2ABC - FLEX_1AB IRR frequency histograms.

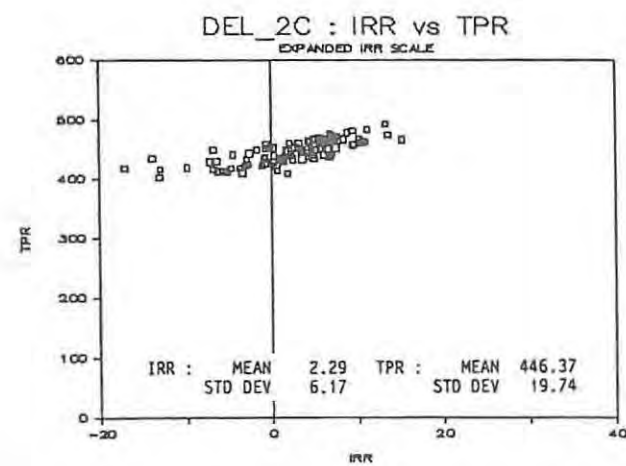
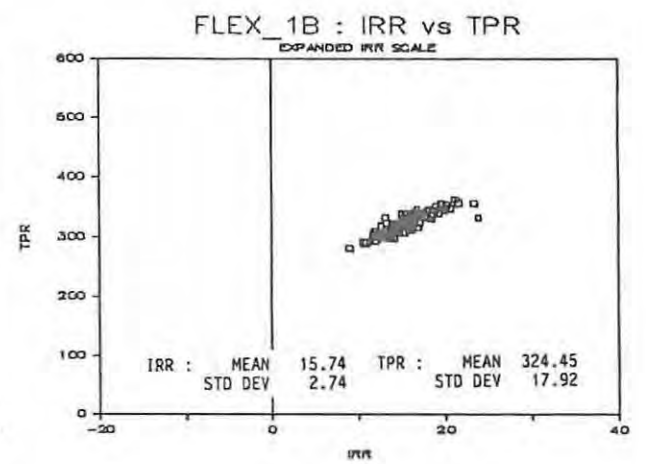
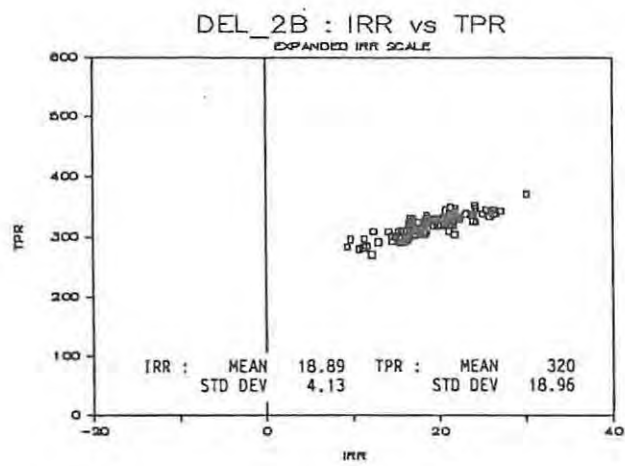
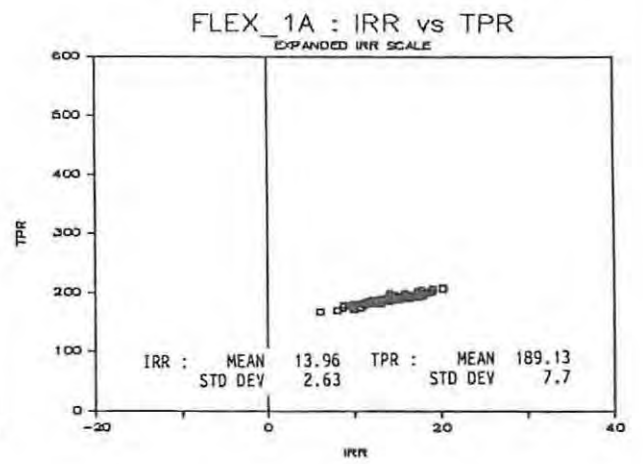
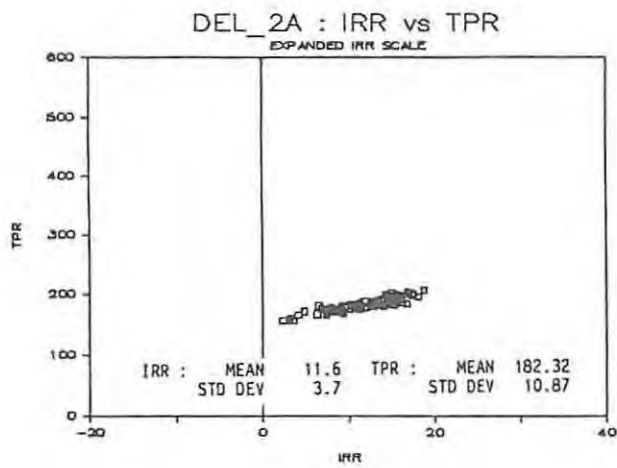
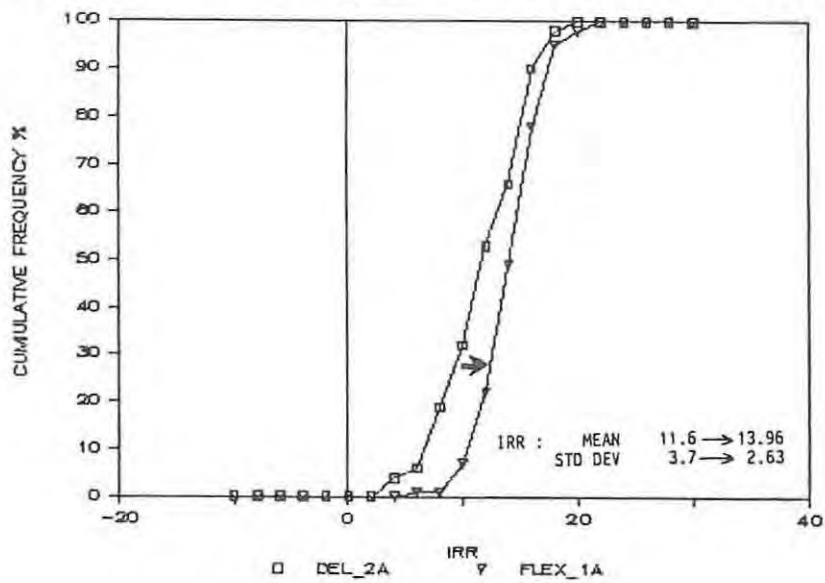


FIGURE D-33. DEL_2ABC - FLEX_1AB IRR vs TPR plots.

DEL_2A & FLEX_1A : CUMULATIVE FREQ. %



DEL_2B & FLEX_1B : CUMULATIVE FREQ. %

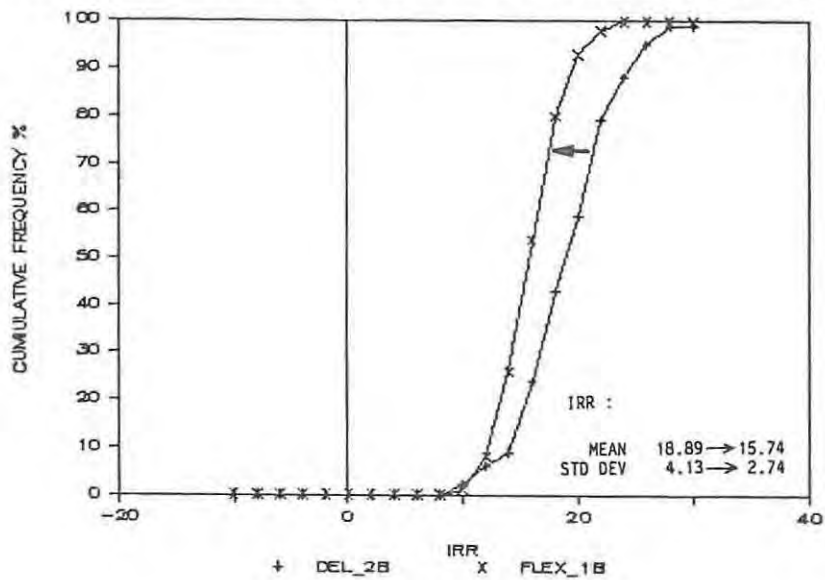


FIGURE D-34. DEL 2AB - FLEX 1AB cumulative frequency percentage plots. Note improvement in PROSPECT A IRR and decline in PROSPECT B IRR.

GENERAL CONCLUDING REMARKS.

Through the use of a risk analysis simulation, the effects of geological factors on exploration and mining investment alternatives have been demonstrated. The rôle of the geologist is critical. Until such time as the geological uncertainty is satisfactorily resolved or reduced the geologist is the key person to co-ordinate decision making. It has been demonstrated that premature exclusion of geological input to prospect and mine evaluation can have disastrous consequences. There is little to be gained by ultra-detailed production planning, or carefully balanced financing arrangements and cash flow schedules if the geological data is tenuous or unacceptably uncertain.

Finally, it is suggested that the use of a simulation program such as RISK\$ in a LIVE SIMULATION offers scope as a teaching aid, specifically to heighten the awareness of geologists to the influences of geological factors on the assessment of exploration - mining investment alternatives. It is they who must determine the geological factors, ascertain their significance and finally shoulder the responsibility of all the GO - NO GO decisions leading up to that final nod or shake of the head. One might well say:

KNOWN RISKS KNOWN REWARDS!

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Finally, to my wife Brenda, who not only found herself operating as a single parent on many occasions over these two years, but was coerced into pretending an interest in subjects as diverse as the level of the Archaean oceans to the finer points of equity vs. loan financing, a special thanks. Brenda also typed all of my written work and did so extremely efficiently, under often harrowing conditions.

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