EXPLOITATION OF MINERAL POTENTIALS OF CAMEROON

by

EMMANUEL OJONG

(Licence en Geologie, Yaounde)
(MSc. Mining Geology, Leicester)

A thesis submitted for the degree
of Doctor of Philosophy and the
Diploma of Imperial College

Department of Mineral Resources Engineering,
Imperial College of Science and Technology
LONDON SW7 2BP

JANUARY, 1984

Acknowledgments

The assistance of many unnamed persons is inevitable in work of this nature; it is duly acknowledged.

I would like to express special gratitude to my research supervisor, Dr. Ralph Spencer, for use of the financial computer package MECON, for the many suggestions he made and other assistance he readily gave to enable me complete this research work.

I am thankful to Professor C.T. Shaw, Head of the Mineral Resources Engineering Department for arranging my field exercises and providing me financial assistance on many occasions.

My thanks go to Dr. Smith of the Mineral Technology section for assistance in the formulation of Mineral Beneficiation Schemes used in this thesis; thanks to Dr. Gouchin for many suggestions he made and to Dr Dudeney for giving me some useful references in mineral beneficiation. I thank Mr. John Watson of the Mining Engineering section for many suggestions and literature he readily gave in the construction of excavation models used in this thesis.

Many thanks are also due to the Mineral Resources Engineering Library Staff, Margaret Burr, Adrian Clark and Ruth Cooper for their tireless cooperation in the search for literature. Thanks to Trevor Allen , Barry Holt and his assistants for their technical help; and to Christine Knight, I owe special thanks for the professional attention she gave the typing and correction of this thesis. I thank also Ella Ng Chieng Hin of the Geology Department for proof reading this work.

I like to give many thanks to Mr. H.E.K. Allen, Director of the Mineral Production Management MSc course and members of his 1981/82, 1982/83 M.P.M. course groups, for useful interchange of ideas. I thank Mr. Grier son and Dr. Ayres for their administrative help.

And finally I pay posthumous homage to Professor Pryor who admitted this research proposition but never lived to see it reach completion.

N.B.

This research work was sponsored, through a scholarship award, by the Ministry of Education of Cameroon (MINEDUC); I am very grateful for their cooperation.

Abstract

A quantitative appraisal of the exploitability (amenability to economic exploitation) of four selected mineral cases in Cameroon (cassiterite porphyry case of Mayo-Darle, bauxite case of Fongu-Tungong, rutile heavy sands case of Douala and Kribi iron-ore case) has been carried out in this thesis. This has been realized through geoeconomic modelling of the exploitation of each case-study and analysis of possible economic rewards to the investors and the government. Results obtained from the study have been used as a basis for suggesting a minerals policy formula for the exploitation of these and similar mineral potentials in Cameroon.

The MINEX model (Integrated Mineral Exploitation and Evaluation System) has been developed, tested and used in quantitative appraisal of the exploitability of the selected mineral cases; MINEX is also being proposed as a suitable scheme for fulfilling Cameroonian mineral policy objective (2).

Profitability outcome of the mineral base-case studies indicates Mayo-Darle cassiterite porphyry case to hold the most attractive economic potential with a Net Present Value (NPV) of $368.8 million dollars, an Internal Rate of Return of 30.6%, the shortest payback period of 3.4 years and a fair Investor-Government ratio (NPV/GVS) of 35%. The Fongu-Tungong bauxite alumina and the Douala rutile heavy sands cases are marginal with IRRs, NPVs and NPV/GVS ratios of 11.3%, $25.5 million dollars, 6%; 8.1%, $-6.2 million dollars, - 13% respectively. The Kribi iron-ore case is clearly submarginal with an IRR of 1.9%, NPV of $-57.1 million dollars and an NPV/GVS ratio of -52%. Government equity participation was observed to be a crucial determinant in case-study profitability; lower GVSs than the 50%

base-share, improved base-case profitabilities significantly.

1,200 potential Cameroonian jobs at lucrative salary scales will be provided with 300 expatriate jobs as a consequence of implementing the exploitation of these mineral potentials. Elaborate infrastructures of roads, rail lines, mining town sites and restored mined land for agricultural use, will be derived from the implementation of these mineral cases.

It is recommended that Government equity participation be negotiated below the <50% ceiling; <45% for Mayo-Darle, <40% for Fongu-Tungong and <30% for Douala. Government should hold little or no equity interest in the Kribi case, even in the event of constructing an iron and steel complex. The Fongu-Tungong bauxite-alumina case should be reinvestigated first and the Mayo-Darle case reinvestigated after completion of the 50 kilometers rail link to the anticipated Bafousam rail terminus. These mineral cases indicate good potential for economic viability. The Douala and Kribi cases need further pre-project reappraisal.

Future research study should be directed at the analysis of pre-project economics of mineral beneficiation; this will be helpful in elucidating on the economics of mineral exploitation for use in mineral management and policy formulating.

Collection and collation of data and creation of a Cameroonian mineral data base is also recommended as a sound basis upon which a comprehensive appraisal of the exploitability of national mineral potentials can be undertaken.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

CHAPTER 1. INTRODUCTION

1.1. Prelude

Emerging nation-states together with the entire international community are preoccupied with the development of global natural resources potentials for the purpose of improving economic achievement. The importance of minerals in this general economic endeavour, and the role which mineral exploitation has played and will play in the process of economic development have unfortunately not yet been properly understood.

Mineral material is undeniably of such importance in industrialization and the sustenance of industrial society that its availability features foremost in the policy framework of industrialized and less industrialized countries. Crowson (31 32) and Schmidt (127) share similar views on this matter while Kursten (74) is unequivocal in asserting that,

> "In the development of modern industrialized societies mineral resources have invariably played a decisive role."

This appears to be the case because industrial development or evolution is unimaginable without assured access and availability of energy and mineral material. Perhaps the incidence of iron and coal upon the industrial revolution of Western Europe and America, justifies this assertion ; and as Bertrand Russell (14) acknowledges,

> "It is to steel, oil and uranium not martial ardor that modern nations must look for victory in wars."

The usefulness of mineral material (raw, refined and manufactured) is often taken for granted; the much prized jewellery in our coffers is unmistakably of a large mineral essence, the motor cars in our garages, aeroplanes, food and drink cans in our supermarkets, the iron and steel on our rail lines and cars, the portland and asphalt we use in paving our roads, harbours and airports; infact the whole transportation system including oil and gas platforms and pipelines, are based on minerals and fuels. Toombs and Andrew (139) have noted that;

> "...every country is continually dealing with problems of mineral supply and demand."

McDivitt and Jeffery (90) hold the opinion that minerals are one aspect of land resources and the presence of reasonable exploitable occurrences can give a country significant moment during the early stages of development because as Malcolm Gillis (46) has observed,

> "Mineral resources can be likened essentially to unfinished capital goods or alternatively a ready made stock of natural capital that can be used to short cut the normally slow and painful process of capital accumulation",

which is accepted to be prime-mover in the process of economic development.

The United States, Canada and Australia, Kuwait, Algeria and Venezuela are quoted off-hand (39) as being paradigms of economic achievement via mineral exploitation. However was the process by which these achievements were realized, it is perhaps plausible to say that successful mineral exploitation schemes and projects need to be well studied and planned through consistent strategies, by private and public organizations such as governments or multinationals.

The objective of producing wealth out of rock for improving well being has not however been without problems, failures, misfortunes and conflicts. The business of mineral exploitation is now acclaimed by the informed and the lay as being highly capital intensive, financially precarious and abnormally politicized. As a result, the execution of a mineral exploitation plan needs to be subjected to much study, analysis and rehearsals over long periods of time; from the early project-conception phases, to the project phase and throughout the project life, in order for there to be positive economic or welfare outcomes from the enterprise; and this prescription holds true whether the project be undertaken by public, private or joint interest.

Misfortunes of certain mineral rich countries such as Zaire, Ghana and Nigeria, to mention but a few, are no doubt, not easy to explain. But when results from the exploitation of their minerals are set upon a background of the results obtained in some of the success cases mentioned earlier on, it would be right to allude much of the fiasco to the absence of a well studied and planned mineral exploitation strategy. A mineral legislation cannot play the role of a minerals policy plan, neither is it a substitute for a policy formula. This is so because mineral legislation, though of much assistance to administrators in implementing and executing the law as pertains minerals, is of little assistance in an atmosphere of competing and conflicting ends and sub-aims of private, public and social interest.

A minerals policy is more of a comprehensive instrument to assist administration in these matters because it should comprise of and take into account all national goals and objectives, those of investors (foreign and indigenous) and society, welding them up into a harmonious overall economic plan.

C.W.M. Court's observations (30) of the potential contributions of a mineral industry are stated in conclusion:

> "Given the opportunity, the mining industry can do more to alleviate world poverty, hunger and human misery more to close the widening gap between advanced and developing nations and more to achieve international peace and understanding than any other single force in the world."

1.2 AIM

The main aim of this research work as was originally conceived and proposed by the author, is to express in quantitative as opposed to qualitative terms, the "exploitability" (see definitiion under section 2.2) of four selected mineral cases in Cameroon (cassiterite porphyry ore of Mayo-Darle, iron ore of Kribi, bauxite ore of Fongu-Tungong and rutile heavy sands of the Douala region.), in order to measure their economic worth in terms of possible economic rewards to the investors and the government; thereafter to use the results obtained from the study in suggesting a minerals policy formula for the exploitation of these and similar mineral potentials in Cameroon.

In pursuing this aim, it was deemed useful to carry out a preliminary study of the mineral circumstances of Cameroon with the aim of identifying her mineral potential and also the possible causes for the collapse of the non-fuel minerals industry.

Development of a suitable tool and methodology for systematic quantitative appraisal of the selected mineral cases was inevitably another important objective in accomplishing the main aim stated above.

1.3 Mineral Policy Problem

Metalliferous and non-fuel minerals exploitation no longer exists as an organised activity of any economic consequence in Cameroon. Apart from a negligible total annual non-fuel minerals output of less than 20 tonnes of alluvial cassiterite concentrate grading about 56% -70% SnO_2, plus some 7 kilograms or so of gold (Au) metal, both of which are mostly collections from "Scavengers" and treasure hunters, most of the colonial mining sites are now either defunct (Colmin) or reduced to artisannal operations (Mayo-Darle). Fig 1 is a location map of these old mining sites. Though processing of alumina at Edea remains quite important (50,000 tons per annum aluminium ingot), the total non-fuel mineral produce in Cameroon still accounts for less than 0.5% of the G.N.P.

However mineral endowment to Cameroon seems to be quite substantial . varied and comparable only to those of some proximate and adjacent mineral rich territories such as Nigeria, Gabon and the Congo which lie equally on the same geotectonic substratum (76); (87) (98) (143). Iron ore, bauxite, rutile heavy sands and cassiterite porphyry ores have long been known to occur in significant quantities to warrant immediate exploitation or further detailed economic and geological appraisal (24), (147) (148).

The absence of a non-fuel mineral industry in Cameroon can justifiably be blamed on none other than the absence of a studied minerals policy-plan. The problem and need for formulating such a minerals policy for Cameroon is not new; the question was eloquently spelt out in the summary minutes of a national select committee deliberating the state and future of minerals exploitation in Cameroon (115).

> "Aussi le probléme resumé dans tout sa brutalité est de savoir si l'on doit denier tout avenir minière au Cameroon ou s'il est encore temps de réflechir aux moyens d'orienter cet avenir par l' établissement d'un plan de réchèrche minerale ."

A paraphrase of the passage would state the problem as a blunt official statement of misgiving and concern over the state and future of the minerals industry; an expression of doubt as to whether there was any hope left in formulating a minerals policy which could then be used to orient the future of minerals in Cameroon.

1.4 History

The policy problem has been fundamental and central to the collapse of the non-fuel minerals industry in Cameroon. This problem can be traced from the end of the last world war up until the present time. By the end of that war when allied war-demands and exigencies became relaxed in about 1945, there was a steady and consistent decline in mineral output and activity from three of the chief production centers in Cameroon - see fig.1

At that time, mineral activities were supervised from abroad by the British and French overseas geological and mineral departments of government (O.G.S. and B.R.G.M.). In the following 15 fifteen years before independence, these responsibilities became more and more relinquished to indigenous hands and were at the same time foreshadowed by political campaign and activities leading to independence.

By 1960, responsibility for managing investment into exploration and mineral development fell more directly and prematurely into indigenous, private and public hands. Not only was government ill-prepared for these tasks with no independent ministry set to take charge of these mineral matters, but it had to rely yet on the same overseas bureaux for mineral and geological research in Britain and France, because at the time, concepts of commercial large-scale mining, exploration and geological surveys were entirely foreign to both English and French speaking Cameroonian authorities.

An important factor which is often overlooked but which seems to the author to have played negatively in favour of progress in mineral development activities, especially in Cameroon is what can be termed the "phenomenon of multiple-colonialism"; whereby colonial allegiance was paid successively to the Portuguese and Germans and simultaneously to the English and French administrations. This not only caused inconsistencies in the adoption and emulation of a unique mining legislation, but also produced severe disconformities in the inheritance of wholesome colonial mineral archives and valuable records dealing with geology and mineral surveys. Reasons are several. Most valuable material was either taken by the outgoing colonial authorities or discarded by the incoming masters in cases where major linguistic dissimilarities or ideological differences existed.

During the short German colonial era which spanned 1884 - 1918, mining activity is known to have been diversified in terms of mineral species, territorial location and scale of operations. Mica and salt were won by medium scale methods from Mbiofon and Mbakang in

Mamfe, gold was mined in Betare-oya and rutile from Yaounde area; production of the latter was second only to Australian output for many years before the wars. Most mineral sites were initiated by the German administration and their geological records form the basis of most information now available on mineral matters in Cameroon. Unfortunately most of the literature is lost and hard to come by and what is left requires translation.

After the partition of Cameroon between the French and English at the end of the German era, new mining concerns that inherited available mineral reserves from ongoing mining projects failed to carry out organised exploration and investment work in these areas or elsewhere, especially after the war. This negligence in exploration - disinvestment, coupled with the absence of an independent ministry for mineral matters both rapidly culminated in exhaustion of available ore in the three mining sites at Mayo-Darle (Tin), Colmin (Gold) and Yaounde (Rutile). Creation of a separate government ministry for mineral matters in the early seventies (1971/72) did not improve matters much because what was amiss is the absence of a mining policy and mineral exploitation strategy.

Meantime, reconnaissance missions, pathfinding expeditions and other mineral exploration errands by friendly foreign bodies and governments were not only sporadic but almost haphazard since they were neither planned nor geared towards the accomplishment of set goals drawn-up in a mineral exploitation plan. The Germano-Cameroonian cooperative mission on mineral resources, of the 26th November to the 15th Decmeber 1977, the Roumanian pathfinding mission of January 1981 were very much in the same spirit; Marmo V (87) had this to say in his summary report of a field symposium organised by the United Nations for the study of

West-African granites:

> "Documentation provided by Cameroonian authorities was heterogenous and greatly hampered those members who sought mainly to understand the terrain.
> The outcome of that experience set against the results achieved in the Ivory Coast and Nigeria shows that it is better to provide visitors with documentation which is self-consistent and complete even if the interpretations it comprises are arbitrary and open to criticism."

There are currently two geological survey missions operating in the South of latitude 4° North and separated by longitude 13° East (South-West and South-East mineral inventory projects). The projects are jointly funded by the European development fund and the French fund for Cooperation. These missions are traditionally geological and general in their character and like their predecessors are not aimed at the appraisal (qualitative or quantitative) of any particular mineralization or geotectonic structures holding possible mineral potential. Instead they seek to catalogue minerals occurring within those regions according to conventional geological classification systems. The only exception to this rule seems to be the I.A.E.C.* sponsored project which is investigating Uranium mineralization at Goble within the Poli-rift of the Benoue Aulacogene.

*I.A.E.C.: International Atomic Energy Commission

1.5 Profile of Cameroon

Cameroon like most of its sub-Saharan neighbours is a typical post colonial developing country with a dualistic* economy. It occupies an almost triangular piece of territory (see map fig 1) measuring approximately 475,000 square kilometers in surface area and lies within the 2^{o} North latitude and 13^{o} North latitude. It is fringed by a short Atlantic coastline (270 kilometres) on the South-Western edge of the country. The Western frontier is marked off by Nigeria, the rest of the territorial boundary runs in a clockwise direction between Tchad, Central African republic, Congo and Gabon.

Recent population estimates put the figure at about 8.5 million inhabitants, 55% of whom are between 15-65 years of age; 40% are below 15 years of age. About 82% of the population is engaged in subsistence agriculture and related activities, dwelling in rural communities with little or no social or public amenities. Urban population migration at a rate of 7% per annum exceeds the overall growth rate of 3%. This has resulted so far in an urban population of 30% of the whole populace. Douala and Yaounde are respectively the economic and administrative capital towns.

* Dualism:

The development of a modern commercialized industrial sector alongside a traditional subsistence agricultural sector resulting in what is termed a dual economy. The introduction of a developed modern sector was the result of colonial contact and its sustainance by colonial entrepreneurs or surrogates most of which administer and orient the exports of their countries to these colonial parent countries by trade which results only in the introduction of western elements into what may still be a purely traditional economic society (3), (8).

1.5.1 Infrastructures

Fig 1 shows the network of main roads linking principal demographic and economic centers in the country. Bituminised all-weather roads are few in the country. There is an axis of truck roads linking Tibati to Meidougou in an East-West direction within the heartland of the Adamawa plateau. The same class of road system runs from Kouserri in the North passing through Maroua, Garoua and terminates at Ngaoundere in the heart of the enormous bauxite fields of the Adamawa region.

The most important of these trunk roads runs through Douala, Limbe, Kumba, Nkongsamba, Foumban and Bamenda in the North-Western tip of the country.

Railway lines are undergoing extension and improvement. The new rail system (TRANSCAM) links Belabo, Yaounde and Douala (934 kilometers). Douala and Nkongsamba are linked by a 172 kilometer old narrow gauge line while Kumba and Mbanga are connected by the TRANSCAM system. Freight on these transportation routes is about 20 cents (U.S.) per kilometer per tonne for cement, aluminium ingot, steel and crude mineral produce. The principal maritime port is Douala, which has a capacity of six million tonnes per annum and handles up to 90% of total trade by volume. Other smaller port-towns located along the Atlantic coastline are Limbe, Tiko, Kribi and Campo; their development in the future will result in a maritime port complex capable of serving Cameroon and her landlocked Central-African neighbours.

Fig 1 Location map of Mineral Cases and Infrastructures

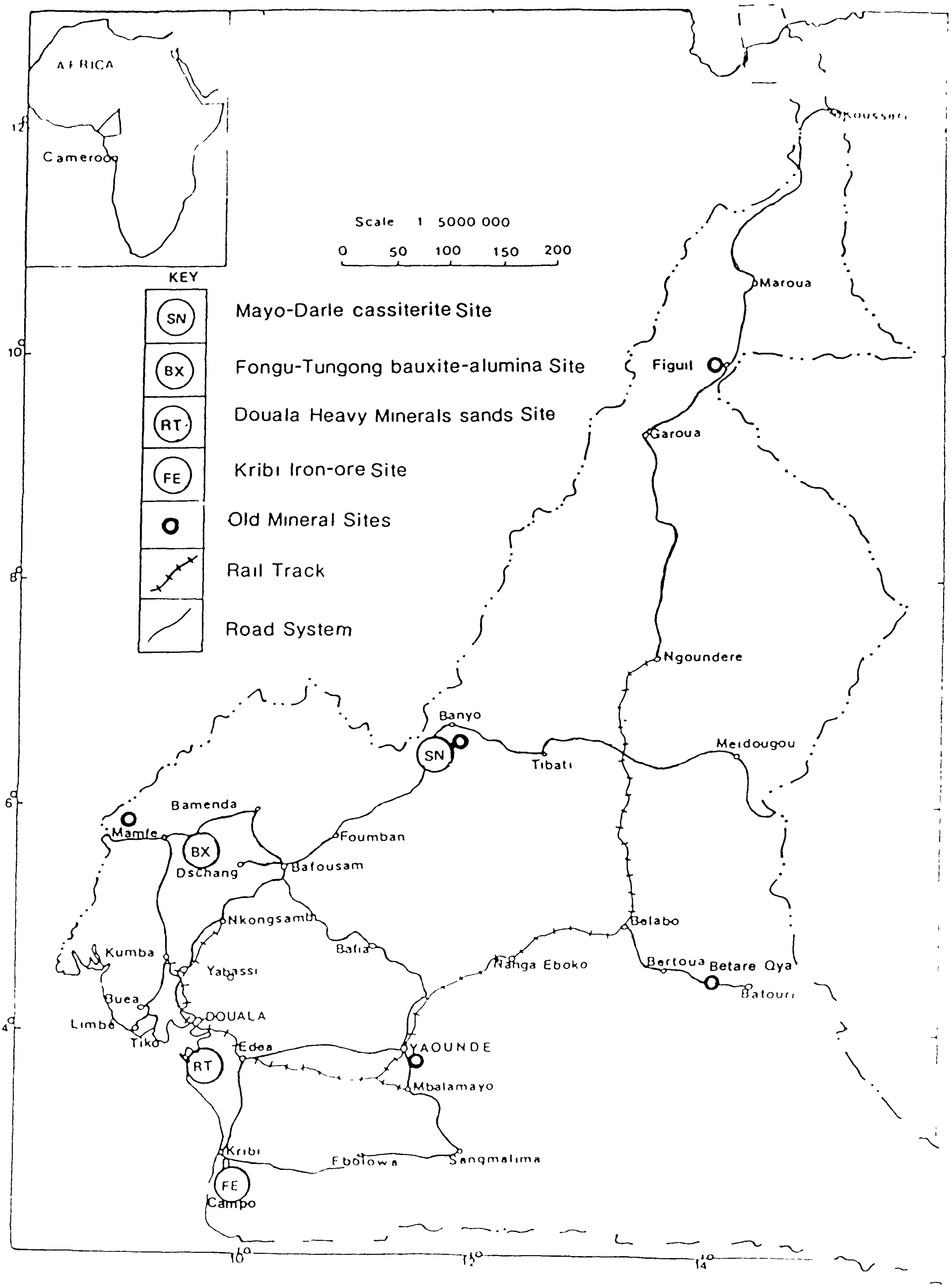


Fig 2 Growth Profile of Gross Domestic Production (1963-1982)

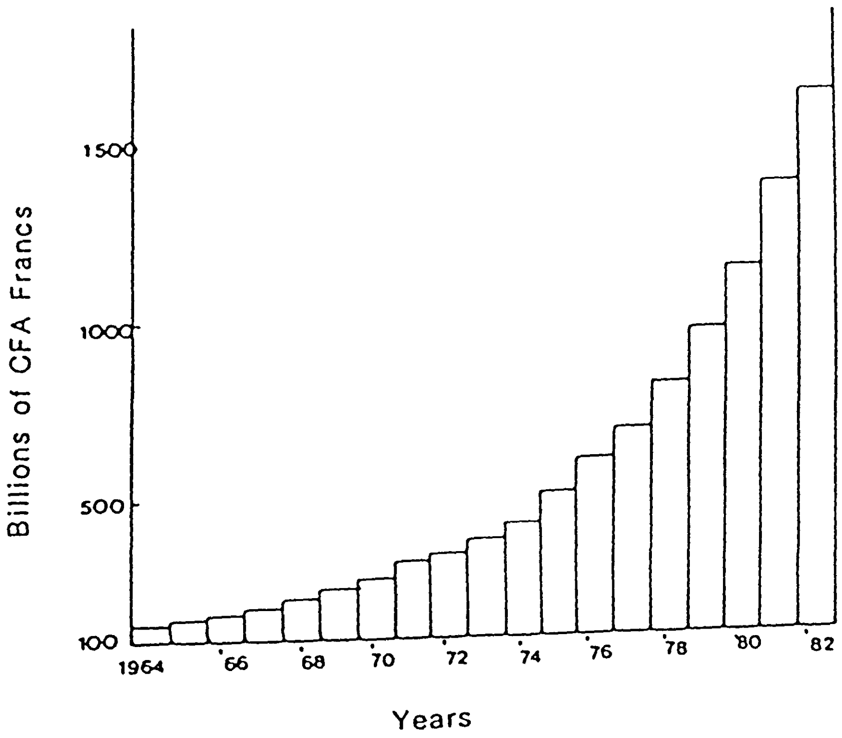
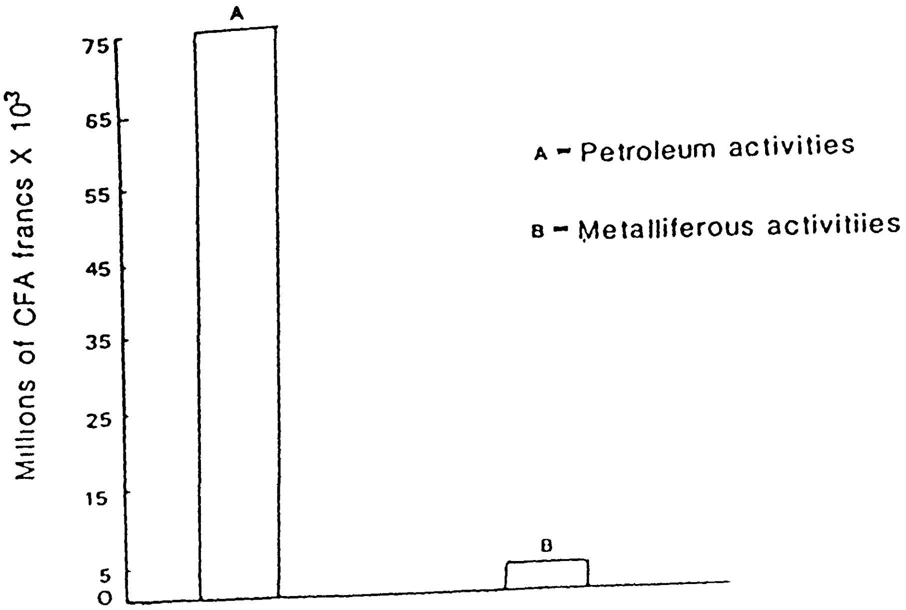


Fig 3 Investment Plan for mining activities in Cameroon (1975-1981)



Hydroelectric power plants have been located on the maps in reference(7) current hydroelectric power output (268 mega watt) is generated from dea where ALUCAM (Aluminium Cameroon) has an aluminium smelter, with an annual capacity of 120,000 tonnes. The construction of hydroelectric units at Njock Songloulou and Nachtigal will increase capacity in the short-run to about 731 mega watts.

1.5.2 Agriculture and International trade:

Agriculture is the basis of the Cameroonian economy, generating up to 35% of the G.N.P. and providing more than 80% of the population with occupations and income. Coffee, cocoa, bananas, rubber, tobacco and cotton are the major cash crops; they form more than 70% of exports from Cameroon. Subsistence farming is the main drawback in agriculture because of its low productivity and intensity. Only 10% of total arable land has been put to agricultural use. The importance of this sector of the economy is asserted by the fact that secondary processing industries are based on the manufacture of consumer foods such as cocoa-butter, chocolates, textiles and beverages from these very primary agricultural products.

Trade with the outside world depends on the export of primary and semi-primary semi-processed agricultural and forestry products. These are exported to the industrial nations of Western Europe and America, Japan and Canada. About 44% of total exports are directed to France while the U.S.A. and the E.E.C. combined receive 60% of her exports. The rest of the exports go to Japan, the planned economies of Eastern Europe and U.D.E.A.C. countries (Economic and Customs Union of Central African Countries).

In return, Cameroon imports fuel products, lubricants, petrochemical products, heavy machinery and their spares, beverages and general consumer goods, beverages and general consumer goods.

1.6 Prevailing Minerals Legislation and Policy.

The Cameroonian minerals legislation is a multi-colonial inheritance which has albeit undergone superficial modifications to keep apace with the conflicting contradictions engendered by increasing demands for fiscal earnings and the maintenance of a viable minerals industry for economic prosperity through substantial foreign investment.

This legislation (109) is preoccupied with asserting state title to minerals in the ground including statutes for regulating mineral operations and administering fiscal matters. New elements in the legislation include government participation in large-scale mining organizations for granting them special fiscal advantages.

There are two laws which regulate mineral activities in Cameroon, the one concerns hydrocarbon minerals and the other regards metalliferous minerals. The latter is detailed in seven parts, the first part deals with formalities for the acquisition of exploration rights issuable in the form of processing licences, exploration permits and mining concessions. These rights are renewable but can be withdrawn at ministerial discretion. A specified minimum of work is expected to be done on a lease area for it to remain valid to the holder. A special minerals category (uranium, thorium, beryllium, helium with liquid and gaseous hydrocarbons) has been defined as strategic* to the state.

The rest of the law deals with ownership of surface and subsurface rights together with government supervision and arbitration of mineral matters and disputes.

Fiscal obligations are stipulated in a separate law (109). wherein fees and payments for holding exploration and mining permits are detailed on a per kilometer basis for each mineral species. Royalty charge is 5% on metalliferous mineral substances at their gross F.O.B. value. All mining companies are liable to a 30% tax on their taxable earnings plus a 27.5% tax on their net profit or dividends occurring to shareholders (Withholding tax).

The mineral policy objectives of government are not entirely new or unique to Cameroon.

Some of the main policy objectives of government regarding minerals are:

(1) "Mineral activity must become a driving force in the economic development of this country ... "

* Strategic

The term strategic mineral has military overtones, and suggests that these minerals are considered essential in maintaining defensive readiness in times of war; in recent times the term has taken on an extended connotation to cover the vital requirements of an industrial economy as a whole.

(2) "It is necessary to quantify the mineral resources of the whole country, conduct profitability studies based on quantitatively good data."

(3) "... the State will intervene more vigorously to render national mineral and geological research structures more operational and more dynamic ... "

(4) "In order to provide the sector with a sufficient number of qualified national senior staff and technicians, the fifth plan will give priority to training mineral research and prospection personnel."

These objectives are not entirely new ones, they have been recanted in almost every economic plan since independence (102) (138). What remains to be fulfilled is the carrying out of a quantitative study which models these mineral objectives in a mineral exploitation scheme within the Cameroonian context; so as to give reasonable answers and guidelines on how best their implementation can be wrought.

1.7 Thesis Layout

The thesis is separated into three sections, exclusive of the preliminaries, references and appendices.

In the first part of the thesis which is an introduction to the study, the first chapter introduces the global objective of mineral exploitation and the aim of the research study. The minerals policy problem of Cameroon is identified and presented in this chapter. The rest of chapter one comprises a profile of Cameroon and the layout of the thesis.

Chapter two deals with the philosphy of appraising the "exploitability" of mineral potentials and the methodology adopted in th appraisal. The second part of the chapter presents the information base of the appraisal together with a summary outlook of the mineral market position of the selected case-studies.

The last chapter of part one is a literature survey of three main areas of the work - cost economic modelling of mine systems, mineral exploitation modelling and Cameroonian mineral policy issues.

Part two of the thesis is devoted to development and verification of an integrated mineral valuation model (MINEX), the basis of which is an open-pit shovel-truck excavation system, which is a suitable system for mining the selected mineral cases. This part of the thesis is composed of four chapters; the first two sections of chapter four contain a descriptive analysis of the open-pit subsystem model parameters. (Section 4.1 geotechnical input parameters, Section 4.2 equipment and job input parameters, section 4.3 quantitative derivation of open-pit output parameters). Chapter five considers the modelling of exploration and beneficiation sub-systems (sections 5.1 and 5.2). Chapter six treats cost modelling of the total MINEX system according to the three sub-system models. Costing procedures of the four main cost sources are presented and compared. The rest of the chapter is reserved for revenue derivation and cash flow modelling. Chapter seven experiments and verifies the MINEX model.

In the third part of the thesis (Part 3), the application of MINEX to the appraisal of the "exploitability" of the selected mineral cases in Cameroon has been presented in chapters eight and nine. In the former, equipment and job requirements and anticipated geotechnical set-up capital and operating cost summaries of each mineral case are presented and discussed.

Ranking of significant case-study parameters together with the profitability of the four mineral base-cases is also done in chapter nine including economic appraisal of the mineral cases studied. Investor rewards and government take are measured for each project.

Taxation impact, transportation impact, inflation impact and foreign exchange issues are studied here. Possible economic benefits and project alternatives are deduced and studied by a sensitivity analysis of the base-case appraisal.

Chapter ten is the last chapter and is made up of a policy proposal. The fiscal question, timing of mineral development strategies, exploration investment are all considered here. There are references included at the end of the work, most of which have been discussed in the literature survey or alluded to in certain parts of the thesis. There are four appendices; a summary geology of Cameroon, an open-pit model, break-even methods and a specimen output from the MINEX model.

CHAPTER 2 Mineral Exploitability

2.1 Preamble

The conflict-strewn arena of global mineral resources exploitation is the result of inadequate understanding of the phenomena we term mineral occurrence. This lack of understanding is often greater in the case of third world host-country governments of mineral projects than to international mineral entrepreneurs who, more often than not, command an appreciable measure of knowledge on the geo-economic worth of the mineral resources in question.

Consequences of such unequal bargaining positions especially in negotiating mineral exploitation, has been the creation of complex and unprofitable conflict potentials for confrontation during the course of mineral resources exploitation:-

(a) Changing fiscal laws to track irregularities in revenue accrual from mineral operations.

(b) Loss of confidence by parties concerned in the mineral project

(c) Abnormal practices by operators to maintain profitability (high-grading, transfer pricing) and excessive taxation by government authorities to maximize fiscal revenues in the short term.

(d) Disinvestment.

(e) Expropriation.

(f) Premature closure of operations.

These host-country governments versus private foreign interest conflicts are typified by the Chilean experience between 1964-1969/1971 during which copper mines were partially nationalized and then completely taken over by government, the nationalizations, in Zaire, Zambia and Peru, to name but a few.
Radetsky (19) Carman (20) .

Improving the understanding of interested parties on the true nature and geo-economic worth of a mineral occurrence is the sure means of mitigating the well known conflicts and misunderstandings that litter this province of economic development.

It is the opinion of the author that such an ambitious task can be initiated at the commencement and early conception stages of the mineral project, through quantitative as opposed to quasi-quantitative descriptive appraisals of the mineral accumulation in question. The necessity for quantitative knowledge about the geo-economic circumstance of a mineral occurrence is apparent in both private and public circles. In the former, this knowledge has been provided traditionally by the so-called feasibility or pre-feasibility analysis of established consultancies or within in-house facilities; the U.S.B.M. is an example of a government agency interested in this kind of knowledge. It has set up an organ responsible for such work - Minerals Availability System (M.A.S.).[150].

In the latter examples, quantitative appraisals have been based upon abundant, expensive and confidential data, available at late stages of project study and design. Such data includes sub-surface bore-hole logs and geostatistical data on cut-off grades or ultimate pit-slopes and limits, development openings and slope design; plus actual cost data of equipment and services. These appraisals are usually complex involving a large team of cost and design engineers, geologists and economists including several consulting groups. Such feasibility studies often cost as much as 6% - 10% of total estimated capital costs of the envisaged project.

But the special nature of mineral exploitation involving long lead times requires much planning, study and rehearsals because of the high risk nature of mineral ventures. The commissioning of huge sums of investment capital in mineral undertakings also needs justification through continued quantity appraisals from early days of project conception up to and until project implementation. Most governments are interested in being able to obtain quantitative information about the "exploitability" of a mineral occurrence so that this can be intergrated or studied together within a common economic development plan. These and several more reasons have heightened the demand for early stage pre-project quantitative appraisals of mineral occurrences.

At such early pre-project conception phases of mineral exploitation, information concerning mineral occurrence is not plentiful and is limited in amount of detail. However, modelling can proceed upon the basis of a minimum or rudiment of geo-economic data. Such a data-base would include all facts and figures obtainable from a wide variety of sources - mineral surveys and archives, mine bureaux; all the information would necessarily be limited to pre-drilling stages of exploration activities.

The results of such early-stage pre-project quantitative appraisals will be to inform on the "exploitability" of the mineral potential in question. "Exploitability" will imply the amenability to exploitation (using a particular mining beneficiation system) plus economic viability (duly tested by a formal profitability analysis). Exploitation will also involve all stages of mineral activity from exploration to beneficiation. Mineral resources potential will refer to naturally occurring accumulations on or within the accessible part of the earth's crust in such a manner that it can be of usable value to man when rationally exploited.

This system of study falls right at the heart of mineral economics, a field of study which Brooks D.B. (15) describes thus:-

> "Mineral economics as a discipline finds its justification in what might be called "economics of specialization" that is, it rests less on any special theory than on the fact that the application of general theory to minerals is best accomplished by blending physical and social science most specifically, Geology and Economics and this requires a considerable knowledge of mineral occurrence and mineral technology."

2.2 Philosophy of appraisal

The absence of an established science or engineering discipline directed at the appraisal of mineral resources potential is regrettable in the academic field of study. This observation is not quite obvious to many who are involved in the broad field of pure and applied earth science, it has however been remarked by some eminent authorities in this field of study. Cohen's (28) comments below say it all in a few words:

> "It is a historic accident that the subject of geology was on the whole developed as an art or pure science, rather than as an engineering discipline directed at the appraisal of mineral resources. Even today the basic outlook of many field geologists is heavily biased towards the niceties of genetic history or classification of rocks and the mineral resources potential is not always given its deserved priority."

The philosphy of appraisal as originally conceived by the author is centered around the use of a rudiment of geological and economic data on the mineral occurrences being studied, (iron-ore, rutile heavy sands, cassiterite porphyry ore and bauxite ore see table 1 , fig 1) together with knowledge of geo-technical and economic principles, in carrying out quantitative analysis of the exploitability of the mineral potential in question.

Throughout the work, the philosophy of appraisal relies on substantial insight into and mastery of the principles of pure and applied earth science, especially the nature and substance of mineral occurrence, as recommended by Brooks. This knowledge is deployed in deriving or formulating additional information and data from the original minimum (pre-exploration drilling data base) for use in further appraisal. Excavation principles are considered by the author to be the province of applied earth science, they have been used substantially in the calculations to estimate equipment requirements and material supplies. Beneficiation principles are based inescapably on the mineralogical content of the ore mineral suite in question. Mineralogy and physico-chemistry of minerals fall within the same province of applied earth science. It becomes quite clear why Brooks D.B. stresses the importance of a "considerable" knowledge of mineral occurrence and "mineral technology" in studies dealing with mineral economics and why Cohen ([28]) laments the historical error of maldevelopment and misdirection of geology into areas of study other than "the appraisal of mineral resources" and or mineral resources potential. Translation of all mineral exploitation results into costs, revenues, and estimation of the profitability and economic effectiveness of the mineral case-study is possible only through economic analysis. As a result, methods of costing and sources of cost values in the process of capital investment appraisal become equally important and complementary in realising a sound appraisal of the exploitability of the mineral potential.

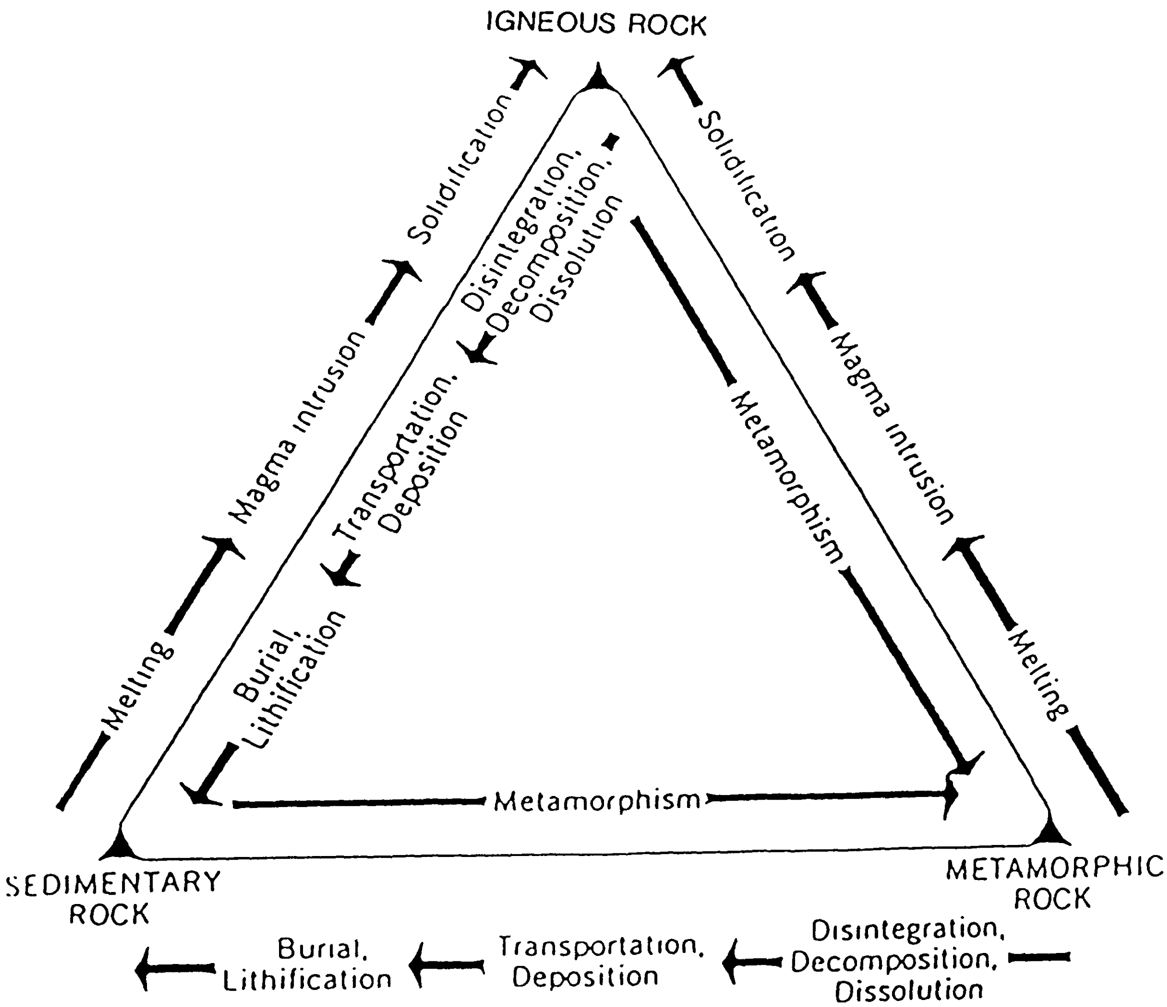
2.2.1 Nature of mineral occurrence and exploitability

A mineral is formally defined as any homogeneous naturally occurring inorganic substance, usually having a definite chemical composition and a characteristic atomic structure e.g. quartz (SiO_2), Kyanite (Al_2SiO_5), Ilmenite ($FeTiO_3$) or Hematite (Fe_2O_3). Mineral substances are the constitution of rocks which make up the accessible crustal and sub-crustal parts of our planet. This material (mineral) and hence rock (broad sense) originates as a consequence of geological phenomena - magmatism, metamorphism and sedimentation or a combination of these natural processes.

Within this very simplified geodynamic scheme, (see fig 4), there is perpetual mobilization and demobilization of rock and its inherent mineral constituents in geological time and space; the result of which is an anomalous formation or concentration of minerals and or their native elemental constituents within set geological space and time where physico-chemical geological setting permits or satisfies their precipitation or accretion from magma, sediment and or mineral laden fluids.

Samples of these anomalous concentrations often show above normal values for some minerals or elements, when compared with their average crustal abundance (CLARKE); both of which are measured in the same units - parts per million (PPM) or percentage metal. The occurrence of minerals within set geotectonic milieu has been demonstrated by mineral geneticists as being a scientific reality.

Fig 4 The Rock Cycle (After Strahler)



Bateman (66), A. Evans (40), Mitchelson and Garson (98). Mineral occurrence cannot therefore be spoken of as being erratic within geological space or time. There is however no correlation between mineral distribution and territorial demarcation. As such search and location of mineralization within given territorial domains should proceed along the lines of systematic prospection and exploration guides using suitable scientific techniques. Only in this way can the mineral potential or "resources base" within territorial boundaries increasingly be established and progressively ascertained.

These observations lead to the question of mineral inventory and resource classification. The U.S. Bureau of Mines and the Federal Institute for geophysical sciences and raw materials (B.G.R.) Hanover define a resource as "a concentration of naturally occurring liquid or gaseous materials in or on the earth's crust in such form that economic extraction of commodity is currently or potentially feasible". Reserves are defined as "identified resources from which the usable mineral or energy commodity can be economically and legally extracted at the time of determination." Reserves can be developed or undeveloped the difference being in the availability for immediate use.

These classifications are mainly based upon the long term and the short term inventory concepts. The reserves category includes material that can be mined under current techniques and market conditions. The resource concept takes account of the total materials contained within a given volume of the earth's crust - resources base, regardless of the feasibility or not of exploiting it.

Until quantitative value can be fixed upon a mineral occurrence through mineral economic appraisal the basis of classifying any mineral occurrence under set resources categories becomes doubtful and arbitrary for economic man. For example, for a mineral to qualify as a "reserve", it is necessary that its geological and economic definitions be quantified and is amenable to profitable exploitation.

"Technology" becomes a prime determinant in this regard, and in its largest sense encompasses all methods, procedures, investment resources and operational expertise available or employed in the exploitation of mineral or non-mineral resources. If this concept of technology were equated to costs of all factor inputs used in the investment production process, then the outcome, when measured in equivalent units (price), the profitability and economic feasibility of the project become determinable by NPV or IRR. Quantitative profitability measures such as these are then fixed upon the mineral occurrence as a criterion and measuring standard for categorization of resources into economic, marginally economic and then into whatever classification there is.

Infact because "technology" and technological innovation and costs are uncertain economic variables mineral and economic classification concepts must of necessity take regard of this fact. The U.S.B.M. and G.D.R. reserves and resources concepts must be explicitly dynamic.

The use of either capital-intensive or labour-intensive mining methods in the exploitation of a given mineral occurrence could be the final determinant as to whether that prospect qualifies as a reserve or not; even though it is denominated as a resource.

In summary it should be said that the economic worth, and benefits derivable from the exploitation of a mineral occurrence depend almost entirely upon "technology" applied in the exploitation, when other conditions are held constant (Price-market scenario).

And for there to be any meaningful and significant economic benefits from the exploitation of any mineral prospect, it is almost imperative for "technology" employed to be compatible and congruent with the mineral prospect and societal context within which exploitation takes place. Mineral development and economic achievement do not take place in vacuo but within human societal entities or systems having differing cultural settings. There is no sense in employing cumbersome capital intensive and fuel consuming machinery in exploiting modest mineral resources that may be found in some labour surplus, fuel-energy-scarce environment of remote underdeveloped areas of the world.

2.3 Methodology and Scope of appraisal

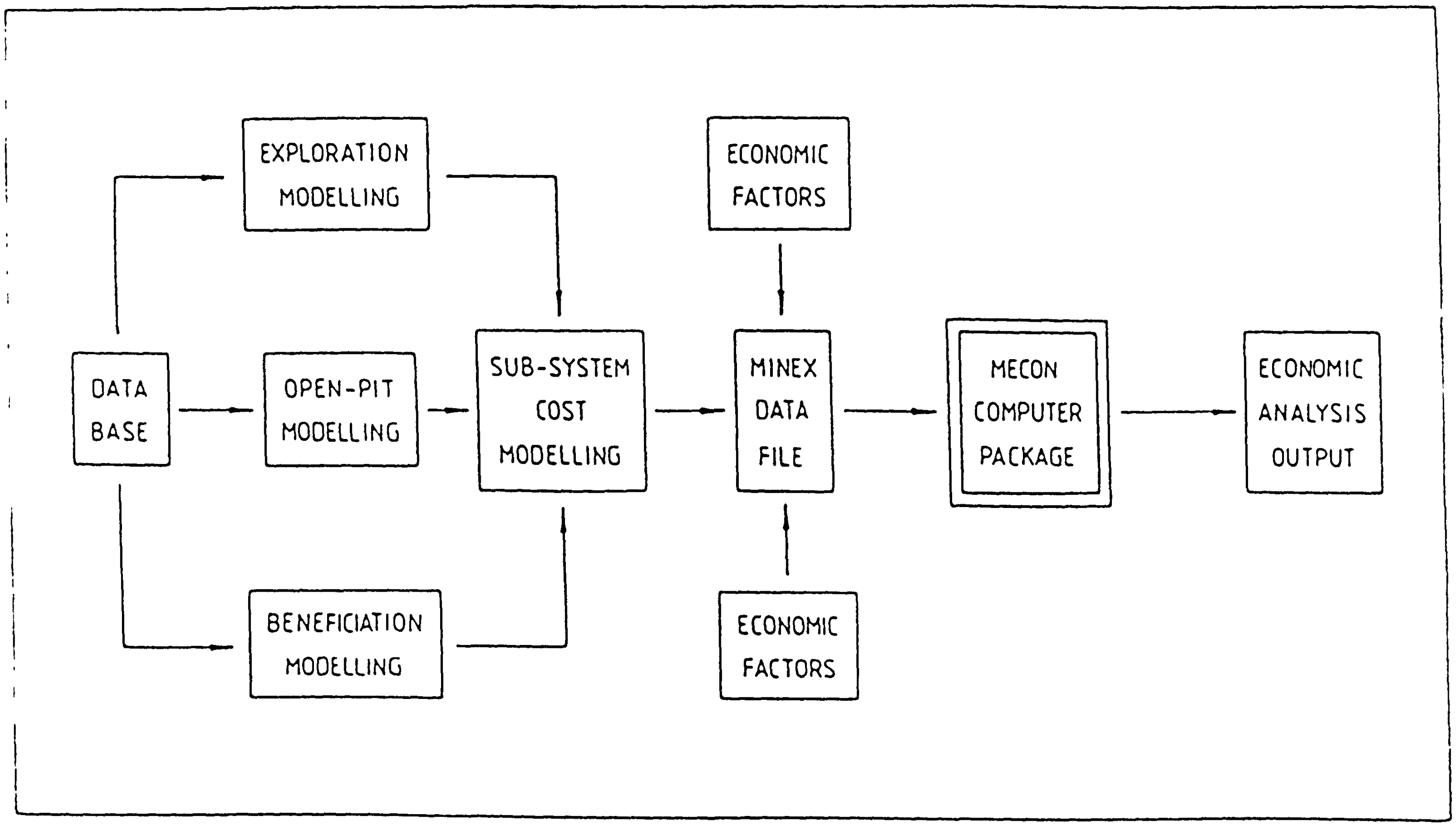
2.3.1 Methodology

Fig. 5 is an idealized schema depicting the methodology adopted in carrying out appraisal of the exploitability of any given mineral potential or occurrence by the **MINEX** system.

The methodology relies on the information base accessible to the appraiser. The information is used in modelling the three sub-systems that are considered by the author to be vital members of any given total mineral exploitation system - exploration, excavation and beneficiation. The excavation sub-system chosen for this appraisal is an open-pit, shovel and truck subsystem.

At the modelling stage the methodology aims to create a mathematical representation of the exploitation

Fig 5 MINERAL EXPLOITATION/EVALUATION SYSTEM (MINEX); FLOW SCHEME



of the actual mineral occurrence from data, such that the basic components or parameters of the model can be expressed in quantifiable terms. Such modelling requires an understanding of the basic components of each subsystem.

In the exploration subsystem model the aim is to reduce the sub-system into quantifiable parameters such as exploration area, number of exploration drill-holes and number of profile lines to be traversed during geophysical and geochemical surveys. The ultimate aim of the modelling stage is therefore to produce sub-system parameters that are amenable to regular costing.

The information data-base contains figures on amounts of anticipated and/or hypothetical ore in-place say (To) tonnes bank ore which dips at an angle (θ) degrees with a mean density of say (Do) tonnes/meter ; with mean thickness (H) meters. With knowledge of an acceptable drill-hole influence (exploration-drilling) the total exploration area and number of holes are computed. All exploration items and parameters are expressed in quantities per unit of area. The same methodology applies with the modelling of the shovel - truck subsystem. Here calculations are made to match capacities of shovels and trucks during a specified time shift for a given production schedule of ore waste. Number of trucks, shovels and drills are estimated using all information available from their manufacturers including geo-technical properties of the materials to be excavated - mean density ore waste, swell factors, performance data - shovel and truck cycle times, average truck speed, mechanical availability and engineering equipment efficiency. In the excavation of ore waste, estimates of number of blastholes and explosive requirements are made using data such as material densities, powder-factors of explosives and daily

Table 1 Basic data on mineral case-studies

ORE-TYPE	GRADE-TONNAGE	MINERAL-SUITE	LOCATION	COMMENTS
Bauxite gibbsitic trihydrite ore	44%-45% Al_2O_3 250 million tons	$Al_2O_3.3H_2O$ Fe_2O_3 TiO_2 SiO_2	FonguTungong Dschang 2000 m O.D.	Poor roads to Dschang Virgin Prospect 100 km to rail terminus
Cassiterite porphery tin, hard rock ore	0.5% - 1% Sn 80 million tons	SnO_2 $(Fe, Mn)Wo_4$ SiO_2 Fe_3O_4 Tourmaline	Mayo-Darle 1500 m O.D. $06^o27'N, 11^o31E$	Old Mine Site remote, 200 km to rail terminus
Banded iron formation, metamorphic ore West-Africa type	35%-50% Fe 150 million tons	Fe_3O_4 Fe_2O_3 $FeO(OH)$ SiO_2	Les Mammelles Kribi 500 m O.D. $10^oO'E, 2^o40'N$	Poor roads Virgin prospect 20 km to Port Abundant Energy
Rutile Beach sands Beach sands heavy mineral ore	2% - 5% Heavy-sands Heavy-sands 500,000 tons	SiO_2 TiO_2 $FeTiO_3$ Al_2SiO_5 $ArSiO_4$ $(Ce, Ye\ th)Po_4$	Douala Coastal localities 500 m O.D.	Widespread occurrence in heavy mineral Polygon 50 km to Port

production schedules. Drill-bit consumption is estimated using the annual blasthole schedule, bit-life, abrasiveness factor and drill performance engineering specifications.

All equipment and supplies are estimated in terms of daily or annual schedules.

The beneficiation sub-system is modelled according to standard processing sequence formulated for separating desired end-mineral species from the rest of the minerals in the paragenetic ore-suite. This is done on the basis of the physico-chemical properties and mineralogy of the ore suite in question. It should be stressed that the level of detail implied in the selection of the process routes is the "process-cost-center". All sub-systems are then costed according to capital and operating headings per annum; this is done by means of assigning costs to parameters or elements of the sub-system models. Wages, maintenance and administrative costs are estimated for a postulated workforce. Working Capital, royalties, taxes and depreciation allowances for capital costs are worked out and the whole cost model established as a data-file for the MECON computer package which (1), translates results into cashflows and then into profitability terms. Inflation and escalation factors with currency conversions are included in the MECON data-file. Revenue is estimated by using a reasonable mineral produce price at the year of sales and thereafter adjusted for annual inflation. The resulting profitability measures and economic results are finally used in an economic appraisal.

The range-approach, is adopted throughout the work to show outcome for alternative cases vis-a-vis the base-case at all levels and for all aspects of MINEX system appraisal. This is realised by the sensitivity-analysis technique. This alternative outcome method is reasonable in informing on the "what-if" aspects of the appraisal. For example a range of project-outcomes expressed as a percentage of the base-case (+50%, +30%, +20%) have been used to generate corresponding information on other project economic options. Results obtained from the appraisal are used as the basis of recommending a minerals policy formula for Cameroon.

2.3.2 Scope of appraisal

Appraisal of the exploitability of four selected mineral cases in Cameroon via the MiNEX model will be the main task of this work. This will be carried out as explained in the methodology according to the philosophy of appraisal discussed above. The data-base upon which the appraisal is carried is limited in volume and detail (see section 2.3.1, table 1) because of the constraints imposed by lack of proximity to Cameroon and financial resources; as such details on full-scale engineering design at the open-pit, beneficiation and exploration subsystems have not been produced in these models and are beyond the scope of this work.

Exploration activities are supposed to refer to and include only definitive stages of ore-body delineation by drilling, backed up by complementary geophysical, geochemical and geological methods; at which time certainty about mineral occurrence has been resolved. As a consequence, mineral target search and detection problems are beyond the scope of this work. Also the formulation of beneficiation routes for the recovery of end-minerals which has been done according to standard and conventional flowsheeting methods is not based on mineral processing criteria such as material balances and equipment selection and design via pilot and laboratory plant verification procedures. These procedures are beyond the scope of this work.

Cost data have been collected essentially from four main sources - Straam Engineers of

Santa Clara (132), Mular (101), Church (25) and O'Hara (106). These costs are mostly for North American mines in the U.S.A. and Canada. Translation of these costs to reflect Cameroonian circumstances has been made possible by the use of subjective escalation factors in some cost centers - e.g. equipment and expatriatewages.

In the economic appraisal, the profitability will be measured by three criteria - Net present value (N.P.V.), Internal Rate of Return (IRR) and payback. Effects due to changes in MINEX system parameters will be monitored in these profitability criteria by sensitivity analysis.

Estimates of economic outcome of projects will be limited to calculations of government-take (joint venture plus fiscal revenues) and investor share of revenue. The rest will be a descriptive appraisal of other possible non-quantifiable benefits from these hypothetical factor input modifications. Suggestion will be made on mineral policy formulation based on the results obtained from the study.

2.4 Minerals Position

2.4.1 Mineral Potentials

The geotectonic setting of Cameroon is a natural endowment holding much potential for mineral occurrence (fuel and non-fuel). The territory lies adjacent to the plate-tectonic focus (the Triple-Junction); a prominent geotectonic feature which is thought by geologists to be the focus of continental separation in the mid-Triassic, about some 200 million years ago (5). Up until now, successful exploration programmes within this broad geological zone have struck oil in sedimentary formations of the continental margin. Salt and evaporite domes associated with these oil deposits lead-zinc deposits of the Benoue trough and Uranium occurrences in the Poli rift, attest to mineral potential in this area.

A semi-dormant volcanic region cuts diagonally across Cameroon (see fig 62) from the off-shore Islands of Principe, Annobon, Sao-Tome and Fernando Po; they culminate in a volcanic mount (4070 meters high) at Buea in Cameroon and extend inland into what is popularly called the Adamawa highlands. Weathering of these volcanic products has resulted in the formation of enormous bauxite deposits in this part of the country (1.2 billion tonnes, 44% Al_2O_3). These deposits are located at Minimartap in Ngaoundal (Adamawa Province) and Fongu-Tungong in Dschang, in the Western Province (see figure 1). Both deposits are of the trihydrite ore type (Al_2O_3-$3H_2O$). Only the Dschang bauxite occurrence (250 million tonnes, 43% Al_2O_3) has been studied in this work.

The rest of the country is made up of stable geotectonic ensembles which are extensions of the great West and Central African cratons with a purely metamorphic constitution (see fig 62). Here, soils (laterites) eluvial and alluvial concentrates originating from the metamorphic basement - complex through weathering, are common place. Concentrations of heavy-minerals in stream beds or in beach localities (rutile heavy-sands), or in ancient depositional environments, are potential sources of mineral accumulations (127). Occurrence of banded iron-ores within this precambrian basement rocks is widespread in adjacent territories (Gabon, Nigeria); some significant finds have been made at Kribi (see fig 1) where 150 million tonnes of $35\% < Fe < 50\%$ have been located.

The rutile heavy-sands occurrences are important in Cameroon within a distinct heavy minerals polygone between the urban towns of Yabassi - Bafia - Nanga Eboko - Sangmalima - Kribi - Yabassi (see fig 1). This heavy minerals polygone was discussed by Mercier, Guillemin and Tymen in an appraisal of industrial development in Cameroon (95). Laplaine (76) discusses these occurrences in his mineral occurrences and resources of Cameroon. Shei (127) reports of important occurrences of these heavy mineral sands in Nanga - Eboko. Other accumulations particularly of rutile and some ilmenite sands (56% TiO_2) are given by the United Nations Secretariat (143) as 400,000 tons near Douala. Infact production of rutile in Cameroon was important during the wars (109). Causes for a post-war decline have been suggested in the historical profile of the Cameroonian minerals policy problem in section 1.1.4.

Cameroonian tin occurrences are principally tin porphyries of the primary hard-rock type. Proximity with the plateau tin-fields of Nigeria (87) is a hint to their common origin which is postulated by Mitchelson and Garson (98) to be the consequence of continental hot-spots.

Some secondary cassiterite accumulations are still being worked at Mayo-Darle (<20 tons per annum) by local artisanal operators. These types are commonly found in ancient stream channels, adjacent to their primary sources. The bulk of world tin produce excluding the Bolivian output, comes from these secondary cassiterite accumulations such as in Indonesia, Malaysia and Thailand. K.F.G. Hoskin (59), has given an exhaustive appraisal of global occurrences of tin mineralization.

As mentioned earlier on, not only are there conducive paleogeographic environments and domains in which the search for mineral occurrence can be oriented, but the variety of such domains and the results obtained so far attest to the existence of an important potential in minerals. A discussion of the market scenario of the selected case studies has been done below.

2.4.2 Market Scenario

2.4.2-1 Generalities

A study of market trends for the minerals industry (3), (11), (120), (92), (145) indicates that demand for raw and processed mineral materials was down in the developed market economies (which account for 70% of total world non-ferrous metals consumption, by 11% to 12% for tin and aluminium, 5% for iron and steel. Growth rates for consumption of iron, tin and aluminium were also down from 5% annually (1951 - 1975) to less than 2% annually (1974 - 1979). This overall decline in mineral demand is blamed on a global recession which has hit the construction industry, civil and military aviation industry causing reduction in demand capacity. Infact though 1980 appears to be relatively a boom year in market activity (high demand and prices realized for all mineral produce - (120), (145)) this was short-lived, as another important demand trough occurred in the first three years of the 1980s. Prices for minerals and allied produce were depressed to unprecedented levels

2.4.2-2 Tin position

The tin market has been relatively a stable one in terms of global mine production and consumption, in the decade spanning 1972 to 1982 (see table 4.5). World mine production has been maintained between about 218,000 metric tons to about 238,000 metric tons. Consumption has matched these output figures during the same period, the former being maintained at about 210,000 metric tons the latter at about 239,000 metric tons.

Market stability seems to have been ensured to a large extent on the regulating influence of the (ITC) International Tin Council which has stockpiling facilities and a prerogative to price stipulation during its tin agreements. With these facilities, it has buffered the tin market in order to realize its producer - consumer objectives.

Mine production is controlled by South-East Asian countries such as Malaysia (26.1%), Thailand (14.3%), Indonesia (13.8%) and Bolivia (11.7%) - average 1980 statistics. Australia (4.4%), Nigeria (1.1%) and Zaire (1.4%) are other producer countries that together with the major 4, make up the International Tin Council. These countries command also greater than 60% of known tin reserves in the world. The U.S.S.R. with 7.2% of world production, the Peoples Republic of China (6.8%) and the United Kingdom (1.3%) are the other significant producers. 1980 consumption of tin (Table 5) shows that 168,700 tonnes of world consumption went to western countries, principally the United-States , Europe , and Japan . The Eastern countries consumed about 55,300 tonnes. Global consumption for that year was 224,000 metric tons of processed tin.

Since the year 1977, statistics table 4 , 5 show that production has superseded consumption. This has resulted in a glut situation (aggravated by the recession) in which current ITC agreements have imposed a 36% production cut-back in order to restore demand and supply balance. These actions have time and again conflicted with United States General Administration stockpile releases creating as it did 27,000 tons of surplus tin in 1981 and 10400 tons of tin in 1982 (121).

2.4.2- 3 Bauxite Position

In the last 3 decades, bauxite production increases were dramatic, from a meagre 8,418 X 10^3 tons (dry basis) in 1950 to 37,291 X 10^3 tons in 1965. By 1970 it had reached 60,710 X 10^3 tons, peaking to 92,623 X 10^3 tons in 1980. Since then there has been a downward trend in production 85, 729 X 10^3 tons and 75,800 x 10^3 in 1981 and 1982 respectively. World production has not shifted a lot from traditional tropical regions such as Australia (29.3%), Guinea (14.4%), Jamaica (13%), Surinam (5.3%), Brazil (4.5%) and Guyana (3.3%) (125). These countries with the exception of Brazil make up the International Bauxite Association (IBA) which includes the Dominican Republic, Haiti, Sierra-leone, Ghana and Indonesia. Greece, Yugoslavia, U.S.S.R. and Hungary are significant developed country producers. Known bauxite reserves are found principally in Guinea (27.8%), Australia (19.7%), Brazil (10.7%) and Jamaica (8.5%).

Production of primary aluminium (table 3) has been between 11744.2 X 10^3 metric tons and 16045.2 X 10^3 metric tons during the decade spanning 1972 - 1982. Production of primary-aluminium has been in the industrialized countries of Europe (3759) X 10^3 metric tons), America (6536.4), Asia (1566.8) and the Eastern block (3285.9) as opposed to the less developed bauxite producing nations. The consumption pattern corresponds to the production pattern, for primary aluminium (see table 2,3).

During the same period, consumption of primary aluminium has stood above production except for the recent slump in industrial activity (the recession). However, new demand from China (60,000 - 70,000 tonnes) ( ), 10,000 tonnes per month Iranian demand plus growing Japanese, Korean and Taiwanese demand, will clear present stocks and relax the contracting production capacity (82% in 1981) and (70% in 1982) ( ); thereby restoring some balance in the market in the near future.

2.4.2-4 Iron-ore position

Distribution of world production of iron-ore is almost uniform, 38% in the western industrialized countries 28% for developing countries and 34% for planned economy countries (125). Production trends have been steady from 1965 (324 million tons by weight of iron content) to 498 million tons in 1975 (50% up). Since 1979, because of weakening demand for iron and steel products as a result of substitutability by wood, plastics and concrete materials, production has declined steadily since then. These production cuts have amounted to almost 50%, especially in the U.S.A. where rising costs of production coupled with market surplus from E.E.C. sources have necessitated government intervention. Some taconite mines have closed in the Lake Superior area for a short time.

The U.S.A. (8.8%), Canada (60%), Australia (11.8%), Brazil (13.5%), India (6.0%) and Sweden (3.3%) are the main producers in the west (1980 basis). The U.S.S.R. however stands as the major producer (26.0%) with China (7.4%) and South Africa (3.3%).

2.4.2-5 Rutile position

The rutile market is temporarily depressed as a result of shrinking demand occasioned by a cut-back in the amounts demanded from the aircraft industry. Rutile however seems to have a bright future because it is a vital industrial metal (161) whose demand would augment as soon as industrial activity is resumed at the end of the recession.

Almost 79% of world production stems from Australia (65.7%) and South Africa (10.7%) alone. There is also a growing potential in production from Sierra Leone (11.8%), Sri Lanka (3.4%), India (2.7%) and other tropical countries. Infact Cameroon has been a principal producer of this commodity; having been second to Australia in 1944 with (3300 tons of rutile concentrate).

2.4.2-6 Mineral Prices

Producer prices for most of the minerals and metals have consequently dropped with the demand. In 1980, aluminium prices which stood above >85 cents per pound had fallen to 74.1 cents per pound and then to 64.5 cents per pound by 1981 and 1982 respectively. Tin prices at similarly high levels in 1980 average yearly prices of 761.4 cents per pound (L.M.E. spot cash quotes) attained an apparent rise to 784.8 cents per pound in September. By December of the same year they had plummetted to 666.8 cents per pound. The following year (1981), prices stayed generally depressed at 606 cents per pound in July (642.1 cents per pound yearly average). Continuously they dropped to 502.9 cents per pound in June 1982 (581.7 cents per pound, yearly average). Figs (9) and figs (8) show the annual average prices of aluminium and tin in real dollar terms between 1972 and 1982 inclusive.

65% Fe iron-ore concentrates (C.I.F. North sea ports) from Lake Jeanine and Mt. Wright Canada had unstable prices in the last 3 years. At $28.9 per ton they dropped to $25.9 per ton in 1981 and rose to $27.1 per ton in 1982. Apparent drop in prices was due to market contraction while mine closures and production costs increases pushed up prices. (see fig 7). Rutile producer prices were also in the downturn. Price trends in real terms (fig 6) showed a consistent decline from 1975 ($1,423/ton to $548/ton) to 1980. A depressed market for civil and military aviation aircrafts production was an important cause.

General price decline indicated by actual mineral commodity market quotes and market indices prepared by the United Nations for minerals, ores and metals was negative (-12%) between 1980-1982. Producer countries especially those dependent upon mineral export as a principal source of foreign exchange have suffered most from these negative price trends (91). The future however, is alleged to be optimistic (145) for minerals, ores and metals; this outlook will be dependent upon higher demand growth rates anticipated in developed market economies.

The world bank index of metals and mineral prices covering copper, tin, nickel, bauxite, aluminium, iron-ore, manganese ore, lead, zinc and phosphate rock is expected to rise 17% between 1982-1983 (145). Chase world information's estimate of aluminium prices (3) gives a price of $1,720 per ton by 1985 or 78 cents per pound; this forecast seems to accord with Marian Radetzky's (120) which projects prices of 80 cents per pound in the year 1985. Metal week (93) postulate between 73-80 cents per pound of aluminium.

Alumina prices predicted by ALUTERV-FKI (3) are in the region of $242 - $342 per ton for 1985. These are all producer price forecasts in the short term.

Strong market control by ITC will ensure a regular price trend for tin in the short term. Metals week (122) have forecast tin metal prices in the short term to fluctuate between the $650 per pound floor price and $700 per pound ceiling price range.

Cassiterite concentrate prices are put at between £275 - £325 per ton for 70% Sn to 75% Sn concentrate grades or £207 - £235 per ton of 30% Sn - 65% Sn concentrate grade.

Rutile prices (95% - 97% TiO_2) vary between \$450 - \$475 per ton. The iron ore market is unlikely to change its price structure a great deal because of the counter balancing effect of cost increases on production surpluses. \$28 per ton of 65% Fe ore (C.I.F.) concentrate delivered to European ports from Canadian sources will be used as the short term price.

Conclusion:

This market scenario, especially the projected prices for the four mineral cases, (3) (122) (120) has been used in the economic valuation. Other market issues, such as existence of markets for sale of mineral produce are not considered necessary in this work. It is hoped that economic recovery in industrial Nations will create sufficient demand to warrant the development of many new mineral projects, especially in traditional mineral producing countries in Africa. Other economic factors such as inflation, mineral freight, foreign exchange and cost of capital have been considered fully in Chapter 6.

Fig 6 Rutile Ore(96%TiO2) Real Price Trends (U S dollars per ton) ( )

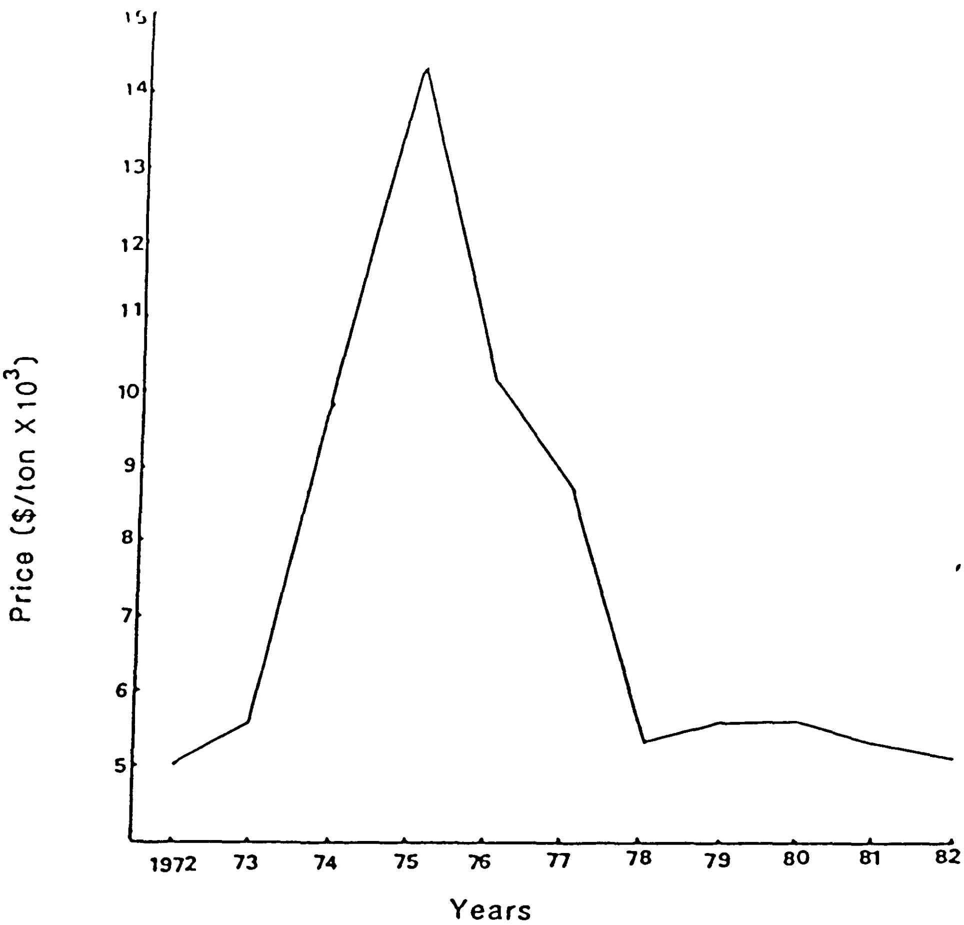
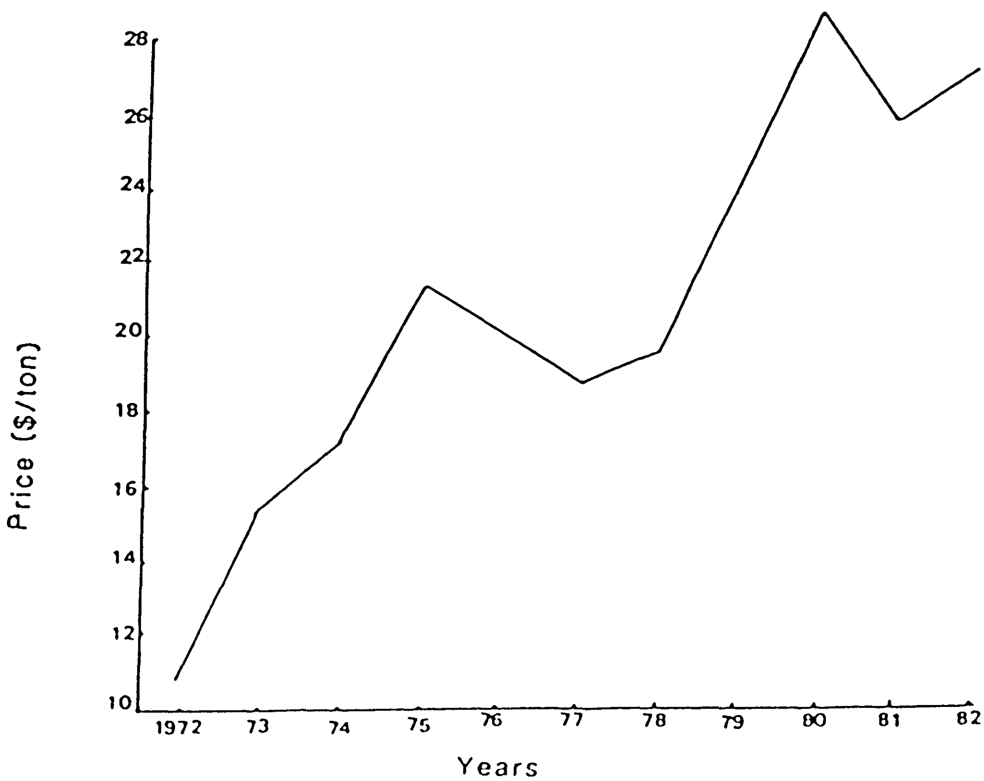
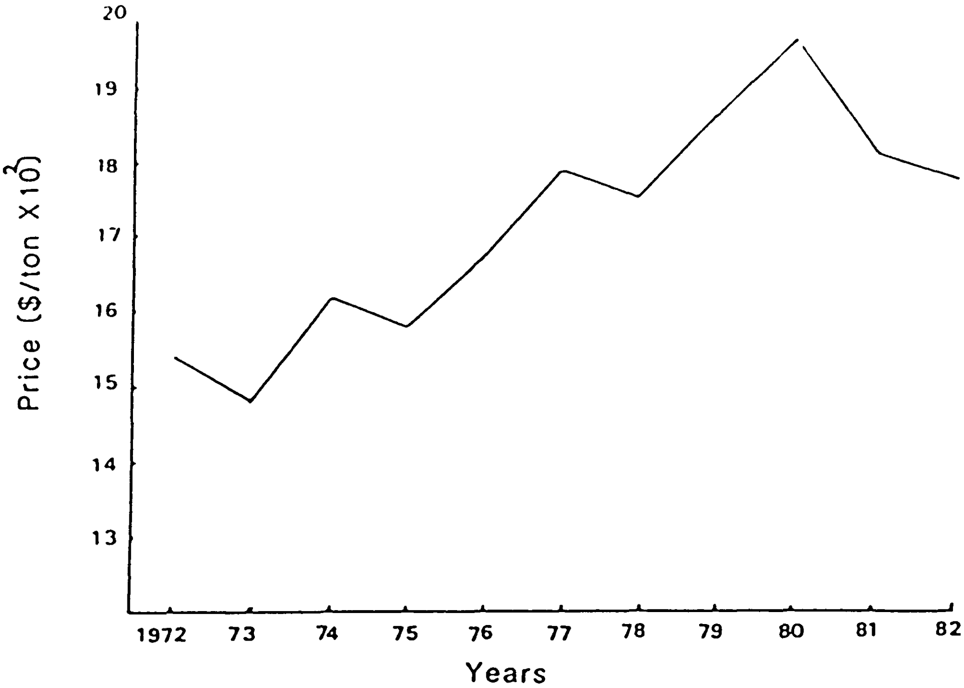


Fig 7 Iron-Ore (65%Fe) Price Trends Real US dollars per ton



Aluminium World Producer

Fig 8 Price Trends Real(99 5%) U.S dollars per ton



Tin Real Price Trend

Fig 9 (LME Cash Standard Grade) U S dollars per ton ( )

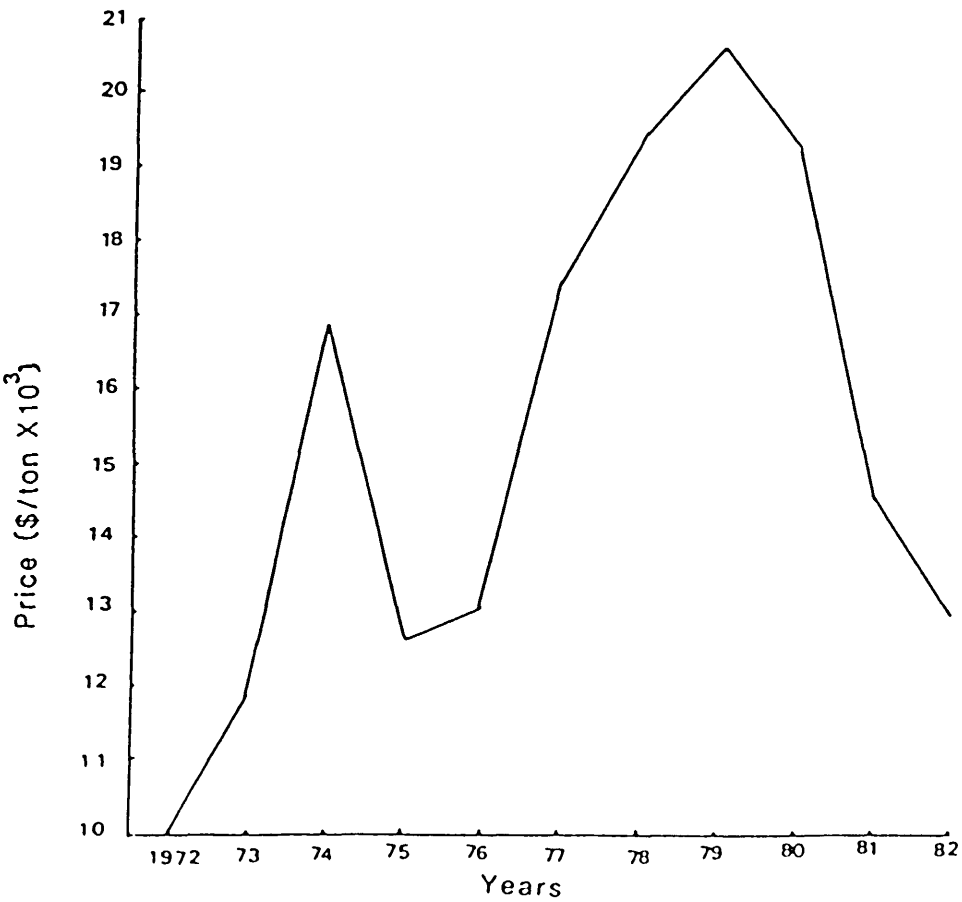


Table 2

PRODUCTION OF PRIMARY ALUMINIUM (1972-1982)

10^3 Metric tons	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	198
Europe	2518.1	2850.7	3297.0	3230.9	3334.2	3469.3	3523.5	3592.5	3759.0	3724.6	352
Asia	1304.7	1439.1	1464.4	1406.0	1362.2	1612.9	1511.8	1464.3	1566.8	20.9	101
Africa	231.9	249.1	279.0	273.0	337.2	368.3	336.3	400.7	437.4	483.2	50
America	4856.5	5266.2	5726.2	4665.0	4802.1	5449.7	5826.0	6085.0	6547.2	6395.2	514
Australia-Oceana	293.5	323.9	329.4	322.8	372.1	392.7	414.5	423.7	459.7	534.8	54
West Block	9204.7	1012.9	1109.6	9897.7	10207.8	11292.9	11612.1	11938.9	12759.3	12471.5	1077
East Block	2539.5	2708.3	2857.8	2940.6	3006.5	3045.8	3156.4	3202.4	3285.9	3235.2	1324
World Total	11744.2	12837.3	13953.8	12953.8	12939.3	13214.3	14338.7	14768.5	15191.3	16045.2	1398

Table 3

CONSUMPTION OF PRIMARY ALUMINIUM (1972-1982)

Europe	2791.8	3208.2	3389.7	2806.6	3466.0	3494.5	3553.1	3835.2	3881.9	3527.3	366
Asia	1546.8	1978.3	1704.0	1548.2	2115.2	2033.1	2330.8	2496.2	2344.2	2326.3	241
Africa	97.6	110.1	115.9	113.6	11.3	130.8	133.1	136.6	171.1	177.8	18
America	4881.8	5716.1	5881.6	3953.9	5209.3	5511.1	5804.9	5910.8	5342.8	4974.9	439
Australia-Oceana	133.1	175.1	202.6	153.3	186.5	193.5	206.9	240.3	244.9	261.6	24
West Block	9450.1	11187.8	11293.8	8611.6	11095.3	11363.0	12028.8	12619.1	11984.9	11240.4	1089
East Block	2397.3	2578.1	2762.8	2840.0	3003.6	3153.8	3302.6	3374.2	3322.3	3310.4	33
World Total	11847.4	13765.9	14056.6	11451.6	14098.9	14516.8	15331.4	15993.3	15307.2	14538.7	1425

Table 4

TIN - MINE PRODUCTION (1972-1982)

10^3 x metric tons	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982
Europe	4.7	4.9	4.5	4.5	4.6	4.8	3.9	3.1	3.7	5.0	5.
Asia	123.1	117.9	117.1	113.6	113.7	118.0	129.4	136.6	136.8	135.3	123.
Africa	18.1	16.9	15.9	15.6	13.8	13.7	12.9	12.3	11.9	11.3	10.
America	36.6	36.0	35.4	38.1	37.6	40.7	39.4	36.2	36.4	39.4	37.
Australia-Oceana	12.0	10.8	10.5	9.6	10.6	10.6	11.9	12.6	10.4	12.1	12.
West-Total	194.5	186.5	183.4	181.4	180.3	187.8	197.5	200.8	199.2	202.9	188.
East-Total	35.5	36.5	38.4	36.6	37.8	36.8	38.2	37.0	35.4	34.2	34.
World-Total	230.0	223.0	221.8	218.0	218.1	224.6	235.7	237.8	234.6	237.5	222.

Table 5

TIN - CONSUMPTION (1972-1982)

Europe	69.3	72.6	73.4	61.0	64.4	62.4	62.7	61.5		4.4	51.
Asia	39.8	49.0	43.1	37.3	45.1	40.9	43.4	44.0	41.5	42.1	39.
Africa	3.9	4.0	4.3	3.8	3.9	3.6	4.0	4.0	3.6	2.8	2.
America	67.3	73.9	67.4	57.1	66.2	62.9	64.0	64.8	61.5	55.5	49.
Australia-Oceana	3.8	4.6	4.7	3.7	4.0	4.1	3.8	3.7	3.4	3.4	2.
West-Total	184.1	204.1	192.9	162.9	183.6	173.9	177.9	178.6	168.7	5.9	146.
East-Total	47.9	50.3	51.9	53.3	55.4	54.6	56.2	54.0	55.3	53.1	54.
World-Total	232.0	254.4	244.8	216.2	239.0	228.5	234.1	232.6	224.0	209.0	200.

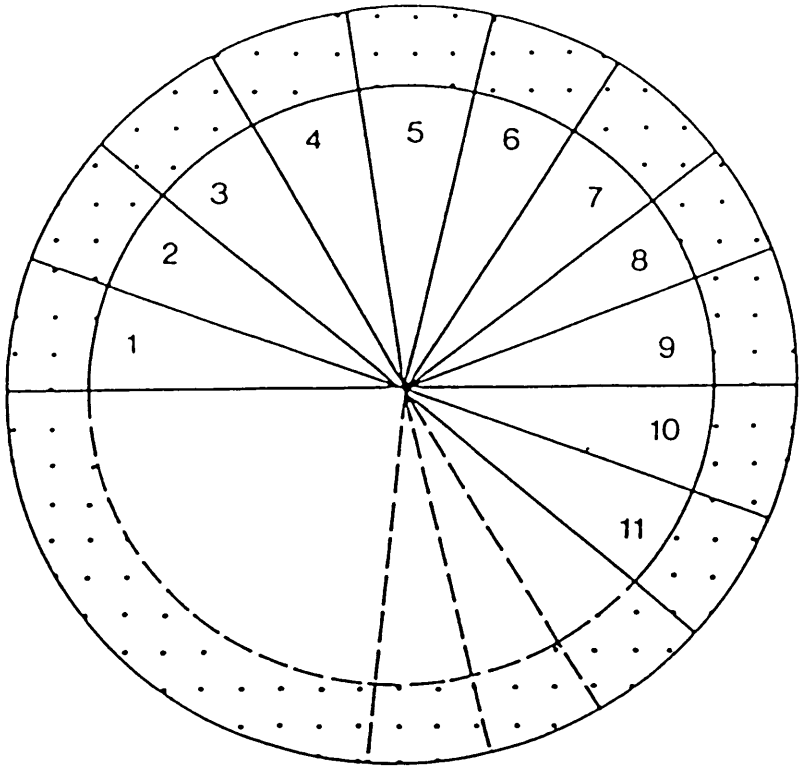
CHAPTER 3 LITERATURE SURVEY

The interdisciplinary nature of mineral economics (see fig 10) and the multifaceted aspect of mineral exploitation necessitated a broadly based search for literature pertaining to this research work. Though much literature was acquired during the period of study, search was initially directed mainly at retrieving material relevant to cost economic modelling of mineral exploitation by open-pit systems, side-by-side information dealing with mineral policy issues of Cameroon.

An abridged list of this literature has been presented as references to this work; discussions of specific references has been done in those sections of the work dealing with similar concepts. Exclusive discussion of literature dealing with cost and economic modelling, and mineral policy issues of Cameroon, has been considered necessary in this chapter. Modelling of mining systems as carried-out over the years has been written and published by Armstrong (4), Wilson and Motley (159). Chamberlain and Leo Borasio (22) have also carried out elaborate treatments of this subject. Scale-modelling was and still is a device to represent the complex mine system in miniature either pictorially or as a solid-three dimensional construct of the physical environment of the mining system being modelled. Their use by mine engineers and managers for designing and planning complicated work schemes in the mine complex is still common.

Conceptual modelling pertains to the theoretical representation of real physical phenomena; such representation being in the form of flow sheets, engineering drawings or mathematical relationships or, where the latter are translated into computer codes, computer models.

Fig10 Multidisciplinary Schema of Mineral Exploitation



1 Engineering Geology

2 Geochemistry

3 Geophysics

4 Rock Mechanics

5 Geostatistics

6 Mine Geology

7 Mine Engineering

8 Mineral Technology

9 Mine Environmental Science

10 Mine Surveying/Design

11 Mineral Management

Mineral Economics

Cost and economic modelling of mineral exploitation systems falls under the set of conceptual models wherein, the mineral exploitation system is modelled into cost relationships which can further be used for economic analysis. Cost modelling of mineral exploitation as an art is said to be quite recent (89) and its introduction in the mining industry occurred about three decades ago. Its importance as a means of deducing the financial implications of mineral enterprise has been accentuated because of the growing capital intensity and financial riskiness of mineral exploitation over the years.

Cost and economic models are now accepted to be invaluable tools for good project overseeing and management. Most of the cost models that have so far appeared in industry are rather cumbersome, too sophisticated or far too expensive to manipulate especially for academic purposes, such as in the solution of this research problem. The master design simulator (86) with eleven sub-systems which themselves are simulators, is an example of such models.

It is worth mentioning that predominance of mine design engineering studies has resulted in a natural evolution of mining system modelling into the development of so-called mine simulators and replicative models which are actually computerised expressions of the mine design; in which the aim of the simulator is to mimic the real dynamic mining system, so that system performance and efficiency can be studied experimentally. The elaborateness of the resources input and data needed for such simulators and their construction is often tremendous in terms of cost, detail and expertise (see Manula Rivell[86,87]). Inflexibility of these simulators in the face of a small data base (such as at project conception phases) is if anything, their principal shortcoming.

This has occasioned a new drive into the area of pre-project modelling of mineral exploitation systems; pre-project modelling itself becoming more and more desirable for studying minerals availability in the face of global political and economic uncertainty (150) provides better leverage for mineral planning both for private and public organizations.

The U.S.B.M. has spear-headed this quest with the award of a large package of contracts (13) (35) (68) (133), for the study of minerals exploitability under the caption Minerals Availability Systems, with the intention of obtaining tools for the appraisal of her mineral position in order to insure against critical shortages of materials. One such contract awarded to the Straam Engineers of Santa Clara California (133) has produced a mines (surface, underground and beneficiation) costing system for the evaluation of mineral occurrences where it is unknown if they can be mined and/or beneficiated at a profit using state-of-the-art technology.

That work (133) involved the collection of field data by specially commissioning another company (Dolbear and Company) to study 66 other mining organizations mainly in the U.S.A. The data was principally capital and operating costs of mining and primary beneficiation (see section 6.1.1).

Using this data has a great number of hazards. Firstly they are modelled to reflect U.S. mining circumstances only and so need to be adjusted with some chosen factors to reflect different country circumstances.

Costs are presented as averages from regression analyses with selected mine parameters. Because the cost data is given for 1975, updating of costs needs the use of acceptable escalation indices such as the Marshall and Swift Indices. Another important contribution in this area of pre-project modelling of mineral exploitation systems is O'Hara's work (106) which incidentally is identical to the Straam Engineers macro-cost models, except perhaps that his is more robust, using Canadian instead of U.S. dollars at their 1978 value.

Discussion with O'Hara (105) on his models showed that his pre-occupation was to produce a simplified generalized cost model which can be used for cost-estimating ore-bodies at their pre-project phases, with the degree of detail limited to operating centers or cost centers.

> "Detailed quantity and cost breakdown of unit operations of each stoping method was not given in my paper, because the variability of costs and quantities of unit operations would be typically much greater than the variability of total costs and quantities for a given mining method."

Michael Bertoldi (13), Frank, Peters, Paul, Johnson, Ralph and Kirkby (41) have been involved in U.S.B.M. contract work of a similar kind but with the objective of carrying out mine-system study and/or economic analysis of specific problems. Katell and Hemingway (68) carried out work for the U.S.B.M. on estimating the capital and operating cost for strip coal mines. In these latter works procedures for modelling of mine systems has not been demonstrated, though the output is shown and economic valuation results presented.

It seems that models are split between those that tend to be general for given mineral exploitation systems and those that are specific to particular mineral systems as is explained by O'Hara.

Further communications with John Murphy (99) of the U.S. Department of the Interior Pittsburg research center in Pennsylvania showed that much of the U.S.B.M. interest was currently geared towards this area and more of the work is still being contracted for study. McClay's study of industry practice (89) shows that mining organizations tend to use fairly unique methods of cost evaluation.

These organizations also seem to rely very much on the use of historical data and experience on past projects, in formulating nouvel cost reports and data for use in future valuations. These are generally aggregated costs expressed per ton of output and presented per cost heading e.g. labour cost per ton or supply costs per ton as the case may be. Some consulting firms tend to produce cost models such as in the case of Golder Moffits and Associates (45). Actual cost values are not however divulged by firms. Ketron Inc. of Wayne in Pennsylvania (116) also adopt the procedure of modelling mining operations before subjecting them to profitability tests.

It is probably worth adding that problems that persist in the area of pre-project modelling of mineral exploitation systems are not very much those of valuation facilities (computer packages - see directory of computer packages 116) but of geo-economic concepts and procedures for solution of general or specific problems. This tends to be so because those working in the minerals industry have traditionally been segregated into miners, geologists mineral technicians, economists or computer specialists; whereas the study of mineral exploitation is inevitably interdisciplinary in nature (see fig 4) and as James Cobbe quotes Theodore Morgan as alluding to Tobin,

> "The best cross fertilization of ideas normally takes place in one head."

It will be discovered that literature pertaining to mineral and policy issues of Cameroon is not only hard to come by but often scant when one chances on them. A trip conducted to Cameroon in January 1980 to April 1981 enabled the acquisition of some useful information on these matters. This information has formed much of the data basis for this appraisal (see section 2.4).

Much of the information on Cameroonian mineral matters is of a purely geological nature, describing the subsurface on a large scale -

Jeune Afrique (67). The latter are geological works (mostly maps) with very useful explanatory notes which unfortunately make only brief allusions to potential mineralizations and their paleographic provenance.

Laplaine's work (76) is the working handbook on mineral matters in the ministry of mines in Cameroon, though this document does no more than inform on the well known mineral sites (defunct and potential in the country. Morawietz (100) however carried out a study of the cassiterite mineralization of Mayo-Darle. Again the study was geological in nature with parts of it describing a detailed mineralogy of both hard rock cassiterite porphyries and alluvial deposits. A summary of the old mine works is also presented.

Most of the rest of what one comes across in the mines archives is in Marmo's words either "heterogeneous" or without coherence in terms of substance and the interpretations of the materials it purports to bear. This sort of material was commonly found as short geological or geochemical field reports of "itinerary" instead of geological, geochemical findings and interpretations of the field exercise in question.

The nearest documents to an appraisal of the mineral position of Cameroon is one prepared in French by Mercier, Guillemin and Tymen of the society for social and economic study (S.E.D.E.C.) this document treats the industrial development of Cameroon on the eve of her independence, in an appraisal of various aspects of her natural resources exploitation. With regard to mineral resources exploitation, their work was quite interesting indeed because it not only pin-pointed mineral targets but suggested those that were suitable for immediate exploitation and possible economic extraction. This work has formed an important element in the minerals documentation of most mineral undertakers in Cameroon. The authors propose advantages and strategies of synchronizing mineral and non-mineral development. If there is a shortcoming in that work which dates 22 years, it lies in its being a descriptive appraisal.

Other data was obtained from U.S.B.M. literature - Mineral facts and problems (146), Minerals Commodity Summaries (149), some publications by the Ministry of Mines and Energy Cameroon (97) inform on the annual position of minerals. Work reports on exploitation sites especially those of the limestone works at Figuil, old tin mines in Mayo-Darle and sand quarry works of Douala are commonly featured in them. The rest of their information now focuses on the expanding energy sector of petroleum and gas exploitation.

Articles from the French periodical chronique de la réchèrche minièré, (23), (24) have been invaluable in confirming facts on the minerals situation in Cameroon before and after Independence. Bauxite and iron ore operations have been dealt with respectively in mineral trade notes (148) and (147) while the exploitation of tin and rutile deposits appear in (24), (143).

The mineral legislation of Cameroon (109) and the fiscal code pertaining to minerals (109) have been used to study overall policy matters of the minerals industry. Pierre Legoux has studied the mining legislation of French speaking countries. Ely Northcutt's summary of mining and petroleum laws of the world (36) touches upon Cameroonian issues. Because these works are surveys of mineral codes, their sources and contents are often identical.

Brown, Rowland and Faber Mike's appraisal (17) of Legal and Policy issues affecting mineral legislation and agreements in African Commonwealth countries is possibly more of an appraisal than the former ones, though Cameroonian issues are missed out. Personal communications with Tanka (138) and Ngombe (103) have also been invaluable material for confirmation of data obtained from diverse sources.

CHAPTER 4 OPEN-PIT SUB-SYSTEM MODELLING

4.1 Generalities

Operations of an open-pit mine is dependent upon geo-technical parameters such as the over all stripping ratio, the burden, blasthole diameter, powder factor and weight of explosives used in blasting rock mass. Ore and waste densities, swell factors, indices of drillability and abrasiveness are also important in determining quantities of loose material to be handled from the bank or insitu rock mass. These parameters are crucial in dictating the amount of waste and ore to be moved during eventual operations.

Basic mine engineering calculations have been used to establish an input-output model for use in the analysis or appraisal of mineral cases that are amenable to this mining method. Input parameters have been discussed in the first part of Chapter four under two subheadings:

Geotechnical parameters Section 4.1.1
Equipment and Job parameters Section 4.2.1

Output parameters have been derived from these geo-technical and job input parameters according to first principles. Quantitative relationships used in their derivation have been presented below for each output parameter. The absence of standard engineering notations in mine engineering calculation has necessitated improvision of arbitrary notations to circumvent the shortcoming. (See pages 69 and 71).

4.1.1 Geotechnical parameters

4.1.1-1 Burden

Burden refers to the perpendicular distance between the row of blastholes and the nearest free-face. Burden is important in rock blasting; empirical knowledge acquired from field and laboratory work has shown that too small a burden results in loss of blasting efficiency (132) because of venting of explosion gases and increased drilling costs. Too large a burden will choke the blast because of a decrease in the powder factor (83), giving rise to poor fragmentation and a similar loss of blasting efficiency.

The estimation of a suitable burden is generally carried out during the designing phase of the project, after on site trial tests. The inclination and pattern of blastholes is also an important consideration in this regard. Estimates involving this degree of detail are beyond the scope of this work.

For our purpose, the estimation of an effective burden (B_e) will be done using the relationship

$$Be = Kb\ d$$

where

Be = Effective burden

d = drill-hole diameter in inches

Kb = Burden constant = 45

4.1.1-2 Spacing

Spacing is the distance between adjacent blastholes measured along the row of holes and perpendicular to the burden. In the process of rock fragmentation the gas pressure entering cracks parallel to the free-face fragments the rock by exerting an outward force. Putting sufficient blastholes in a row at an effective spacing will even out the force exerted by the explosive gases causing uniform fragmentation results. Hoek and Bray ([54]), have recommended a spacing of 1.25 times the effective Burden as the normal practice. Spacing dimensions are normally expressed as a factor of the Burden according to the relationship:

$$Se = Be\ Ks$$

where

Se = Effective Spacing in feet

Be = Effective Burden in feet

Ks = Burden to Spacing constant (1.2)

Normally the Burden to Spacing constant varies according to the blasthole design ([83]); Hoek and Bray have suggested typical values of Spacing as ranging between 18 feet and 33 feet while Church ([25]) has given a table for selecting effective burden and spacing for drilling.

4.1.1-3 Bench Height

Bench height corresponds to the effective working height of the pit-face from the rock-floor. It is the third dimension in the calculation of the total volume of rock per blasthole. This height includes the excess depth below grade to which the blasthole is drilled in order to enhance the breakage and removal of rock from the bench toe. This sub-drilling depth is often denoted by the lift-of-rock factor:

$$h = H(1-Fl)$$

where

H = Bench height

h = sub-drilling

Fl = lift-of-rock factor

Dimensions of the loading equipment in the open-pit are also determinants of the actual bench-height. Values ranging between 35 feet and 50 feet have been recommended by Hoek and Bray ([54]). A survey of open-pit iron mines ([37]) shows 40 feet for benches in

Adams mine in Kirkland Lake Ontario, 55 feet for the Marmoraton Mining Co. Ontario, 45 feet for the National Steel Corporation Moose Mountain Capreol Ontario and 37 feet for the Steep Rock Iron mines Hogarth, Atikokan in Ontario. These bench heights conform to the range suggested by Hoek, though one of the mines, Hilton mines, Bristol, Quebec, has a bench height of 99 feet.

The amount of subdrilling required depends as many other geo-technical parameters upon the rock-mass characteristics. Since the lift-of-rock factor (fl) is given to vary between:

$$0.75 < fl < 0.8$$

it follows that subdrilling depth can vary between 0.3H and 0.2H.

$$0.2H < h < 0.3H$$

4.1.1-4 Blasthole diameter

The overall economics of an open-pit excavation depends to a large extent upon the choice of a suitable blasthole diameter; this is primarily because most other excavation parameters are directly or indirectly related to the blasthole diameter.

Relationships for the calculation of burden,

$$B = Kb.d$$

Spacing $S = B.Ks$

are all a function of the blasthole diameter (d). Estimation of explosives requirements via the hole diameter (see output parameters section 4.2.3) and tonnage of ore and waste output, are all done using the burden, spacing and powder factor (see section) which are all functions of the blasthole diameter. It now becomes clear why the blasthole diameter features as a controlling parameter in excavation of rock. Empirical evidence now shows that greater hole diameters result in better blasting economies (83), (26).

The blasthole diameter also dictates the type of drill-rig to be employed. A demonstration of the theoretical drill-hole effect on rock excavation has been shown in section 7.1 and 7.2.

4.1.1-5 Excavation ratios and material densities

A host of ratios exist in the literature relating to the relative quantities of overburden, waste rock and/or country-rock to the actual quantities of ore (15) . As far as appraisal of open-pit mineral pre-project economics is concerned, the most vital and relevant question to be answered is that of total material quantities to be moved or handled in the mineral exploitation process (mining, ore-processing, product sales transportation). The crucial parameter that dictates what these material quantities would be is invariably the overall waste-to-ore ratio:

$$N = Vow / Vo$$

where

Vo = Volume of ore

Vow = Volume of ore and waste

N = Waste-to-ore ratio.

By means of preliminary map-work analysis of the mineralization, it is possible to determine the geometry and morphology (depth of emplacement, dip, thickness and volume) of the mineral prospect. With additional information on the rock densities which can be obtained by direct measurement with gravitymeters or approximation with similar mineral occurrences, the actual mass of the mineral prospect can be worked out using relationships such as:

$$To = Vo.\ Do.$$

$$Tow = To(N+1)$$

where

Do = Mean ore and waste density

To = Mass of Ore

Tow = Mass of ore and waste

In these calculations, the mean density of ore and waste (Do) is expressed as the mass per unit volume or as is usually the case, pounds per cubic yard or kilograms per cubic meter.

The rock swell factor (fs) expressed as:

$$fs = \frac{100}{100 + \%\ Swell}$$

i.e. the volumetric increase resulting from fragmenting rock (change from insitu or bank conditions to the loose state).

Brealey and Atkinson (15) have discussed the importance of material ratios in open-pit economics. In their discussions, cut-off and stripping ratio concepts are raised; these ratios are said to be volumetric and as such cannot be expressed at a single point within the pit limits.

Richard Stewart and Bruce Kennedy (132) have also dealt in depth with these ratios - stripping ratio, overburden ratio and economic stripping ratio. Much confusion has occurred in the usage and application of these concepts .

An open-pit model has therefore been studied to contribute to the understanding of these material excavation ratios (Appendix II) It has been suggested that the ratios be reserved for material excavation in the process of ore recovery. Other preliminary excavation ratios (overburden ratios) should be referred to only in the pre-production stages of work. The case of stratiform deposits with a uniform overburden have not been considered neither have cases of dipping stratified deposits been considered.

4.1.1-6 Ore-grade

The grade of ore is the assay value of valuable material (metal or mineral) expressed as a percentage of the ore. Another common ratio in which grade is expressed is in parts per million (ppm). Natural concentration of valuable mineral material in the earth's crust is measured in the same units; it thus becomes possible to compare abundance of minerals in the rocks and hence to identify areas having anomalous concentrations. These are termed deposits, prospects, accumulations or what should rightfully be called mineral occurrences.

The concept of cut-off grades has also been elaborately dealt with by many authors (77), (15). This concept seems to find much relevance in mine designs because it serves as a geo-economic marker between valuable and sterile portions of the mineral occurrence.

As far as open-pit pre-project economics is concerned, the cut-off grade problem is foreshadowed by materials excavation ratios, which are themselves determined almost entirely by geo-morphology of the mineral prospect and occurrence, and the ultimate pit angles (see Appendix II This is so because the open-pit mining system is often used to excavate mineral material below the minimum grade acceptable for normal underground operations, such as porphyry orebodies, highly altered and disseminated mineral accumulations which are accessible to this method. It is paradoxically for the latter ore types that cut-off concepts require much emphasis; this paradox has been expressed differently by Crowson (32):

> "It is commonly supposed that higher grade deposits are mined first and that there is a gradual movement towards the exploitation of even leaner deposits. In reality, the orebodies exploited are those that are most accessible and grade is only one aspect of accessibility."

When uniform mineralizations of the Zambian copper type (stratiform sedimentary orebodies) or bauxite strata of the Guinean type and laminar limestone strata are encountered, cut-off issues become mitigated; the concept has much relevance no doubt in underground excavations where stope walls and hence area is designed to represent the optimum between what is regarded as ore and waste.

4.1.1-7 Production Schedule

Of all MINEX parameters, the output rate (mill and mine) is paramount and the principal determinant in project economics. Output is often expressed in tonnes or tons per day (short or long tons) or per year for ore and/or waste.

Output is fixed by stipulated market demand or projected or anticipated demand by contract. Sometimes this rate is fixed by quota imposed by Cartel production terms. In the absence of market criteria, the output rate could be determined by domestic government policy, especially for operations in which government holds vested interests.

In this work the mineral potential of each case-study has been used as the basis for estimating a suitable output rate which will suffice a 38 years project life, without exceeding most average producer outputs for that mineral. Producer outputs considered are mainly those of sub-Saharan countries - Nigeria, for tin (< 2,500 ton/annum), Ghana for aluminium (< 200,000 ton/annum), Sierra leone for rutile (<55,000 ton/annum) and Nigeria for iron ore (<2.5 million ton/annum). Other criteria used in fixing possible output schedules are existing secondary beneficiation facilities such as the 180,000 tpy aluminium smelter at Edea; or past production schedules (rutile).

4.2.1 Equipment and Job parameters

Engineering stipulations and detail on mechanical functions of shovels, trucks and rotary - drills have been described below. The shovel-truck model accepts inputs from any standard source and as such it is possible to match equipment from differing manufacturers with differing engineering specifications.

4.2.1-1 Shovel parameters

The shovel bucket capacity rating is the prime equipment parameter which, determines the productivity of the loading machine and hence the number of complementary truck units needed to meet the production schedule. The bucket capacity has been used by Mular (101) to relate shovel capital costs. The bucket capacity is rated by multiplying the heaped capacity by the loose density of the rock material being loaded.

Because of the bucket geometry, nature of the material to be loaded, and the skill of the shovel operator, only a fraction of the rated capacity will be met in actual operations. The shovel fillability is thus the percentage of a bucket's rated capacity that can be filled by a specific material in a specific form.

$Ff = 0.90$

where

Ff = fillability factor

In loading operations, a shovel will have to do a number of passes to fill up the truck because the truck pay-load is greaer than the shovel pay-load for the same material. The time required for a shovel to load up a truck to capacity is called the truck-load-time (Lt) and it is equal to the time taken per load-cycle times the number of passes:

$$Lt = Pn \cdot Lc$$

where

Pn = number of shovel passes

Lc = load cycle time in minutes

Lt = truck-load-time in minutes

4.2.1-2 Truck parameters

The truck capacity is often rated in tons and this is related to the shovel capacity by the capacity-ratio which expresses the truck capacity in tons over the shovel capacity in cubic yards or some other units (106). Another factor used in correcting the actual loads of material that enter the truck from the shovel is the fill-factor (fo). This often approximates to the fillability factor (Ff).

In truck and shovel operations, the truck hauls its load over a distance to the ore depot. During that time, it turns corners, spots ore and crusher bins before releasing its load. These operations expend time which is very crucial in estimating the overall productivity of a shovel-truck system. More time is used to return the truck to the shovel where it waits for another load. These time intervals are usually read off performance charts for vehicles supplied by their manufacturers and by relating these factors to the actual pit design parameters.

4.2.1-3 Job efficiency factors

Job efficiency is the estimated proportion of the scheduled work hour during which the machine is actually applied to the work cycle. It depends on the skill and experience of the personnel. Job efficiency factors of 80% to 90% or between 40 minutes and 50 minutes of the work hour are commonly used for estimating the effective working time per shift.

Machine availability is the proportion of the scheduled working hour during which the piece of equipment is mechanically able or available for its job. Operating conditions such as conditions of terrain, preventative maintenance all affect the machine availability. This parameter is expressed as a percentage of available time:

$$0.5 < Ma < 0.9$$

where Ma = machine availability.

Other delays during the work cycle shift are caused in setting up and demobilizing equipment (shovels, trucks and drills). Most modern rotary drill systems do not require rod manipulation such as pulling and inserting, they are becoming fully hydraulically automated performing their tasks with a minimum of assistance from the drill man. Other delays are caused by alarms and official break time. Generally the delay time for a normal 8 hours work shift is estimated at about 1.5 hours.

Table 6 Nomenclature of notations

Be	=	Burden (equals to stemming (St) feet)
Se	=	Spacing (feet)
Kb	=	Burden constant
Ks	=	Burden to spacing constant
d	=	Drill-hole diameter (inches)
H	=	Total drilling depth per blasthole (feet)
h	=	Sub-drilling depth (feet)
Vo	=	Volume of ore (cubic yards)
Vow	=	Volume of ore and waste (cubic yards)
N	=	Excavation ratio, waste-to-ore
Do	=	Mean ore-waste density (insitu or bank lb/cu.yd)
To	=	Average ore mass (tons; 2,000 lbs)
Tow	=	Average ore-waste mass (tons)
Fs	=	Swell factor
Fl	=	Lift of rock factor
Pr	=	Penetration rate, feet per minute
	=	Explosives charging density (tons per cubic yard)
Qn	=	Weight of explosive per blasthole
Cs	=	Estimated specific charge
Ce	=	Empirical specific charge
Q	=	Total weight of explosives required per production schedule
St	=	Stemming, which equals to burden

4.2.1-4 Penetration rate

The penetration of a drill into a rock for the purpose of creating blastholes, is affected by rock resistance which is termed the drillability. Wear of the drill-bit on the other hand is effected by abrasiveness. High drillability factors enhance the penetration rate of a drill into a rock while elevated abrasiveness factors blunt the cutting edges of the bit, thereby reducing its life (bit-life), which is measured in number of feet of rock penetrated by the drill-bit.

C. Church (25) and C.G. White (158) have recommended drillability, abrasiveness and bit-life factors for varying rock species and drill-types.

Table 7 Nomenclature of notations

Nb	=	Number of blastholes per production schedule
Nd	=	Number of drills required per production schedule
No	=	Total number of drills required for job
Tb	=	Tons of material per blasthole
T	=	Tons of material required per day
Tf	=	Tons of material per foot drilled
Tl	=	Tons of material per truck load
Lts	=	Truck loads per shift
Ts	=	Tons of material per shift
Lss	=	Shovel loads per shift
Dt	=	Drill-cycle time per blasthole
t	=	Effective shift time in minutes.
Sc	=	Shovel capacity
Tc	=	Truck capacity
tc	=	Shovel loading cycle per truck load, minutes
nt	=	Number of shovel loading cycles per truck
Ma	=	Mechanical availability
Je	=	Job efficiency
Ns	=	Required number of shovels
Nt	=	Required number of trucks
Tt	=	Total truck cycle time per haul-dump.
fs	=	Stand-by factor for pit requirements
Ab	=	Annual drill-bit requirements
An	=	Annual production schedule (days)
bl	=	Bit life
bi	=	Abrasive index
di	=	Drillability factor
ff	=	fillability factor
fo	=	fill factor
ND	=	Number of drills required

4.2.2 Open-pit output parameters

Engineering appraisal of mine systems are aimed at resolving pertinent quantitative questions about future project requirements from computations involving data on mine-system parameters. In the case of open-pit mine systems, the kind of knowledge required about a future operation is in the form of estimates of number of mine equipment (shovels, trucks, drills and compressors) and associated quantities of supplies and ancillary material requirements (explosives, drill-bits, spares, energy) needed in performing the job.

This open-pit subsystem has been modelled to generate the output parameters discussed below from calculations involving the input parameters just discussed above. Derivation of these output parameters have been demonstrated below each heading; they have been presented in their conventional arithmetic form, with a listing of their nomenclature.

4.2.2-1 Tons of material per blasthole

Each blasthole normally has an area of influence comprising of the product of the burden (b) and spacing (s). When the area of influence is multiplied by the blasthole length, the volume of anticipated rock per blasthole is obtained. This volume is adjusted by any of the mass-volume conversion factors (densities) to obtain the tonnage of material per blasthole (Tb).

$$Tb = Be. \ Se. \ H. \ Do$$

where

Be = Burden (feet)

Se = Spacing (feet)

H = Blasthole length (feet)

Do = Mean ore and waste density (lbs/cu-yds)

The daily blasthole schedule can thus be worked out from the overall production schedule:

$$Nb = T/Tb$$

where

T = Overall production schedule

Nb = Number of blastholes per production schedule

The number of tons per foot drilled (material) can be estimated by dividing the output per blasthole by the blasthole length (H).

$$Tf = Tb/H$$

where

Tf = Tons per foot drilled

All estimates at this stage are made on the basis of bank measure of material.

4.2.2-2 Equipment requirements

Drill-rig requirements are computed by means of dividing the number of blastholes required (Nb) by the number of blastholes sunk per drill (Nd) during the production shift using the drill cycle time per blasthole (Dt).

$$Nd = \frac{t \cdot Ma \cdot Je}{Dt}$$

where Ma = Mechanical availability factor

Je = Job efficiency factor

$$ND = Nb/Nd$$

and ND = number of drills required.

It is assumed generally that the number of drills is equal to the number of tandem-compressors where these are used.

The required number of power loaders is estimated by matching the quantities of material to be transported by truck for a given production schedule with the shovel capacities. Noting that mineral material occurs in bank or insitu condition and that equipment container capacities are given in heaped units, we first of all convert all heaped measure to bank units or vice-versa by dividing or multiplying by the swell factor and converting the new capacity into tons by multiplying by the material density. Shovel capacity is adjusted by the fillability factor.

$$Tl = \frac{Tc \cdot Ff \cdot Do}{fs}$$

where

Tl = Tons of material per truck load

Tc = Truck capacity - cubic yards.

Ff = Fillability factor

Do = Mean waste-ore density pounds per cubic yard

Fs = Swell factor

$$Lts = T/T1$$

Lts = Truck loads per shift

T = Tons of material required per day

Tl = Tons of material per truck load

The effective work-time divided by the shovel cycle time will give the number of shovel loads per truck per shift which are in turn adjusted by the mechanical availability of the shovel and the job efficiency of the shovel operator:

$$Lss = \frac{t \cdot Ma \cdot Je}{tc \cdot nt}$$

where

Lss = Shovel loads per shift

Je = Job efficiency

tc = Shovel loading cycle per truck load in minutes

nt = Number of shovel cycles per truck

t = Effective shift time in minutes

Ma = Mechanical Availability

By dividing the number of truck loads per shift

(Lts) by the shovel loads per truck per shift (Lss) we obtain the required number of shovels needed to keep the trucks busy. Fractional parts of a unit greater than > 0.3 should be rounded up as a complete unit. Increased operating efficiency should be able to eliminate the need for fractional units:

$$Ns = \frac{Lts \cdot Fs}{Lss}$$

where

Ns = Required number of shovels

Fs = Stand-by factor for shovels and trucks

Truck haulage requirements are met by dividing the total truck cycle time for loading, hauling, returning to shovel side and waiting for next load, by the unit truck loading time per shovel. This gives the number of trucks per shovel. The truck fleet requirements would be obtained by multiplying the number of trucks per shovel by the number of shovels doing the loading operations. This fleet number would keep the shovels busy throughout the work shift.

$$Nt = \frac{Ns \cdot Tt}{Tc}$$

where

Nt = Required number of trucks

Tc = Shovel loading cycle per truck load in minutes

Tt = Total truck cycle time per haul-dump

Extra shovel and truck units are provided for to allow for a production guarantee, whereby spare equipment units are available in case of machine breakdown during the work cycle. Spare units of equipment are inexpensive because they do not incur production expenses, on the other hand, lack of spare units can result in disruption of the production schedule.

4.2.2-3 Supplies and production requirements

The quantity of explosives requirements necessary to fragment a blasthole tonnage of rock is estimated by first calculating the length of charged hole, which is the difference between the bench height plus sub-drilling height and the stemming*. Knowing the charging density of the explosive, the weight of explosive used per length of hole can be calculated and the weight of explosive required per blasthole computed by multiplying the former by the length of charged hole.

$$Qn = \frac{(H - St) \cdot \pi \cdot d^2}{4} \cdot \rho$$

where

Qn = weight of explosives per blasthole

H = Blasthole length feet

St = Stemming (equals burden = B)

d = Drillhole diameter inches

ρ = Explosives charging density

A comparison of the estimated specific charge and an empirically selected specific charge will tell upon the suitability of the explosive density in use; generally both values should be made to agree.

$$Cs = \frac{Qn \cdot Do}{Tb}$$

where

$Cs = Ce$

Cs = Calculated specific charge

Ce = Empirical specific charge

Explosive requirements per production schedule are then calculated by multiplying the blasthole requirements per production schedule by the weight of explosive used per blasthole.

$$Q = Qn \cdot Nb$$

where

Q = Total explosives requirements per production schedule

Nb = Number of blastholes per production schedule

* = stemming is the uncharged top length of a blasthole.

Estimating the explosives accessories (detonator fuses, primers and electric caps) would be done by using an accessories consumption factor expressed as a fraction of the explosives cost per blasthole.

Drill-bit requirements are obtained by calculating the total blasthole length to be penetrated per production schedule and then dividing by the bit-life being used. This is adjusted by the abrasiveness of the rock (abrasive index) in question.

$$Ab = \frac{Nb \cdot bi}{be}$$

where

Ab = Scheduled drill-bit requirements

bi = Abrasive index

be = bit-life (feet)

Mine equipment and power requirements are estimated from engineering stipulations on equipment efficiencies and capacities, from their manufacturers. These include such details as the motor horse-power, wattage for power shovels and compressor motors. The total open-pit installed power capacity requirements are simply estimated from the overall equipment power capacities, converted into equivalent units (kilowatt or horsepower).

Operating power requirements are calculated from the equipment fuel consumption rates or from Churches (25) operating supplies estimates.

Estimating the open-pit and general mine work-force requirements is a very inexact exercise indeed; the basic workforce is derived from the number of equipment items and back-up requirements - drivers, explosives men, drill-men, supervisory staff, maintenance and blastmen. This is the case because there are no simple guides to facilitate the estimation. In most cases company policy and objectives of various interested parties determines the size of the workforce both for the open-pit and the process section, where such mine and process plant integration exists.

The workforce in most mineral developing countries is composed of an expatriate and an indigenous component. Again, decision as to what the ratio of the two components would be is dependent upon non-scientific rules such as company - government agreements. The expatriate fraction comprises of skilled experts and machine operators on hire or from parent operating sites plus company officials at top-management. The local fraction is sometimes larger in size and is made up of semi-skilled labour and middle-class engineers and government representatives. Bertoldi's labour list (13) and Alutev's work (2) were used in estimating a minimum work force for each case-study.

CHAPTER 5 Exploration and Beneficiation Subsystem Modelling

5.1 Exploration Subsystem Modelling

5.1.1 Generalities

With reference to the mining industry, exploration includes all work or activities necessary to locate and define new mineral occurrences for the purpose of acquisition and economic extraction.

The development of this exploration subsystem is intended to comprise of a final tactical phase of orebody delineation and quantitative definition of the ore target. Information obtainable from such an exploration model will include data on:

- shape and size in 3-dimensions;
- depth to ore zone;
- overall mean grade of ore mineral;
- detailed mineralogy and ore paragenesis;

such information would resolve all the unkowns about the subsurface geological characteristics of the ore-prospect.

Of the four main stages or types of exploration programs outlined by P.A. Baily (6), this model can be said to be an amalgamation of the last two stages. It comprises of a conclusive exploration drilling exercise complemented by geophysical and geochemical methods. Other back-up methods such as seismic reflection, self potential, induced-polarization, radiometry, electrical and electromagnetic methods or gravimetric techniques are included. Preliminary stages in an exploration program enable the acquisition and analysis of information for the search and location of mineral prospects; these stages differ from the definitive stage in that they deal with random search for mineral targets (64).

At the end of these stages of target search, the occurrence or presence of mineral accumulations is often ascertained. Hence the function of the definitive stage of any exploration program would be the retrieval of subsurface data through drilling and core-analysis, after detailed geochemical and geophysical appraisal and mapping.

The importance of exploration, particularly this phase of the activity, in increasing mineral potentials, creating new ore and extending project life cannot be overemphasised. The topographic target area of the mineral prospect is the controlling parameter in exploration economics because all of the other tangible exploration and cost parameters are expressed in terms of the exploration area to be covered (see table 10).

The target area of an ore prospect of desired size can be derived from first principles by means of projecting such a hypothetical or conjectured ore-prospect in two dimensions at the topographic surface. Derivation of the target area has been demonstrated under output parameters in section 5.1.1-2.

5.1.1-1 Exploration input parameters

Where the in situ tonnage of a mineral prospect is not available, this parameter can be estimated from preliminary geological and prospection work. Estimation by mapwork analysis (contour method) is a common geological procedure, which H. Church discusses elaborately (25). It is full of geological intuition as well as being cheap. Early stage estimation of ore prospect parameters (Dip, Volume, Density grade) in the absence of detailed subsurface information which is very expensive and available only at late stages during project design, is invaluable in this work. Calculation of material tonnages using preliminary data such as these has been described under section (4.1.8) with other material excavation parameters.

The dip angle of a mineral prospect is a well known geological parameter accompanying geological features on map-sheets. It is the angle contained within the line drawn perpendicular to the strike of an ore-body and a horizontal plane. In the area of projection calculations, the cosine of the dip angle is used to resolve the dipping ore-body on plan. Horizontal stratiform mineral beds have a zero angle of dip with a cosine of one. Vertically dipping orebodies such as dykes and pipes have zero cosine of dip and theoretically no target area on plan, except of course, their mineral outcrop.

Variability of orebody morphology produces great problems in the determination of orebody dimensions. The mean thickness of an ore-prospect can also be calculated by taking many values along its long axis. This can be done during preliminary mapwork. Depth to ore zone is approximately equal to the depth at which a vertical exploratory borehole makes contact with the orebody at subsurface. This parameter is important in informing the exploration planners on how deep below surface to drill and as such to equip and make appropriate estimates and preparations for a drilling exercise which often takes place in the remote country-side.

Because the target area of the ore-prospect is the main exploration parameter and the basis of costing and estimating all other exploration parameters, many other exploration activities and their costs are expressed as cost per unit of area. These have been presented in Table 8

An exploration schedule is often organized to coincide with good weather conditions. Durations of four calender months are normal practice. The estimation of some exploration items is subjective because they are actually intangible - number of vehicles to be used during the exploration programme, number of drilling men and some lump sum items of expenditure such as public relations and administration.

5.1.1-2 Exploration output parameters

As mentioned earlier on in section 5.1.1-1 the ore-body target area is a prime exploration parameter. If the ore model was dipping at (θ), with a mean thickness of (H), density (Do) and estimated mass (To), then the area of influence of the ore target (Ao) is given as:

$$Ao = \frac{To \cdot \cos\theta}{Do \cdot H}$$

Conversion factors are used in adjusting (Ao) into the desired units.

The number of exploration drill-holes to be put down during the campaign depends on management decision which is based most often on past experience (wild cat ventures) and on availability of data on the field to be drilled. Alternatively, other criteria can be used in deriving the economic optimum number of drill-holes and depth necessary to intercept a prospect at surface (64).

If the optimum or selected grid pattern be (X) and (Y) in linear units of measure, then the corresponding number of boreholes required to strike the orebody at sub-surface will be given by:

$$N = \frac{Ao}{X.Y}$$

where:-

Ao = Topographic area of projection of ore-prospect
To = Insitu ore mass
H = Mean thickness of ore-prospect
Do = Mean ore-waste density
θ = Dip angle of ore prospect
N = Number of exploratory drill-holes
X = Grid interval on one rectangular axis
Y = Grid interval on second rectangular axis

The importance of exploration-drilling has been summarised by the contract enginers of Dravo Corporation (35) in their analysis of large-scale mining methods for the U.S.B.M.

> "Drilling is the deep orebody exploration tool of final resort."

Because of its high cost and the requirement for actual penetration of the mineralized target, all supplementary tools (geological and geophysical) should be employed in advance to determine drill-hole locations.

This statement stresses two facts. That ore-targets need to and should be identified by preliminary exploration techniques before the ultimate exploration technique is deployed in delineating and defining details on the remaining subsurface geotechnical parameters such as:-

grade variations within the mineralized area; distribution patterns of minerals; physico-chemical characteristics of ore and waste, ground-water distribution.

The statement also highlights the question of high exploration drilling costs (diamond-drilling) within the total cost of the exploration campaign.

Once this cost heading is accounted for in the estimate, a good part of the overall exploration program would have been accounted for. Development drilling especially for the purpose of obtaining supplementary core for metallurgical test work (ore beneficiation tests) does not enter into this exploration model.

5.2 Beneficiation Sub-system parameters

5.2.1 Generalities

Resort has been made to the Straam engineers mineral beneficiation model (132) which is constructed on contract for use by the U.S.B.M. Minerals Availability System (M.A.S.), for costing mineral occurrences where it is unknown if they can be mined and or beneficiated at a profit. O'Hara's mineral process model (106) has also been used for comparison with the former. Both beneficiation models are macro-cost models in which the cost of beneficiation processes have been statistically regressed against process plant capacity (milled head grade) for each unit-operation of the beneficiation scheme. These models are expressed directly as functional relationships of the form:

$C = a(X)$

C = cost

x = Milled head capacity

a, b = regression constants

Development and design of mineral processing schemes from test work involving pilot-plants and small-scale mineral dressing methods has foreshadowed the use and application of simplified conventional mineral processing flow-schemes, in pre-project studies such as in the appraisal of economic feasibility of mineral prospects, even for academic purposes.

Development of simplified mineral processing and beneficiation models based on a minimum of information obtainable at post-reconnaissance stages of mineral exploitation activities has been done in chapter five section 5.2.1. This is carried out in accordance with, and on the basis of conventional mineral beneficiation schemes.

5.2.1-1 Idealized Mineral Beneficiation Schemes

The performance of the beneficiation subsystem is expressed as a ratio (recovery) of the weight of a desired component in the concentrate (end-mineral) to the weight of the same component in the feed (137).

$$R = \frac{Cc}{Ff}$$

where

R = recovery of end-mineral

C = weight of concentrate

F = weight of ore-feed

f = assay of ore-feed

c = assay of concentrate

a,b = constants

By manipulating this basic relationship approximations for the concentrate weight or the ore-feed can be estimated on the basis of one of these parameters together with a knowledge of their corresponding assay values or grades (expressed as metal or end mineral).

The ore-feed weight would thus be estimated from the relationship:

$$F = \frac{c \ . \ 1 \ . \ C}{f \ . \ R}$$

while the concentrate weight will be calculated from

$$C = \frac{f \ . \ R \ . \ F}{c}$$

O'Hara (106), has also provided expressions for the recovery factor of some base metal ores in terms of their head grades for various beneficiation methods.

These expressions are of the form

$R = 100\% \ (1 - af^{b})$. Copper recovery from chalcopyrite ($CuFeS_2$), when effected by the flotation method, is given as:

$R = 100\% \ (1 - 0.07Cu^{-0.8})$ while the recovery of iron-ore by gravity and magnetic methods have an expression of the form

$R = 100\% \ (1 - 1.5Fe^{-0.6})$ More of these expressions can be found in reference (106).

Impact of fluctuating recovery factors for mineral beneficiation have been studied in chapter 7 and 9 by means of sensitivity analysis, which will indicate the most suitable recovery factors necessary to produce favourable project economics.

Schemes depicting the beneficiation of four mineral cases have been depicted below in fig 11, fig 12 13, and fig 14, according to established practices. The choice of beneficiation routes is based on a minimum of data summarised in table 1 . This information comprises of the gross-mineralogical composition of the paragenetic suite of minerals, grade of the desired end-mineral and known amounts of ore in place plus data on locations of the mineral occurrences.

Flow-sheets have been formulated for the beneficiation of metamorphic iron-ore deposits, cassiterite porphyry ores of the hard rock type, rutile beach sands and bauxite beneficiation via the Bayer process. Details of process selection are limited to the level of identifiable process centers for which cost models are available (138).

Finer details of process selection such as the calculation of material balances within the beneficiation models are beyond the scope of this work. As such this section entails a description of the process-flow from the run-of-mine ore to the end-mineral through the different process centers of the beneficiation model.

Costing of the beneficiation models by means of their cost-centers has been considered separately in chapter six.

5.2.1-2 Rutile heavy-sands beneficiation scheme

Fig (11) shows the beneficiation scheme for rutile heavy-sands. Contemporary heavy-sands beneficiation in Australia and Sierra Leone (161), both of which produce about 78% of total world out-put of rutile concentrate (96% - TiO_2 heavy-sands), depends on heavy-sands grades of about 0.4% to 2% heavy minerals - rutile, ilmenite and others. These processes normally yield concentrate grades of between 90% to 95% heavy minerals (161).

Mill concentrate to ore feed ratios of about 1:220 have been reported by Mineral Deposits Limited of New South Wales and Queensland, which accounts for up to 1/3 of Australian rutile production (161). Here, 100,000 ton of concentrate are beneficiated from 22 million tons of ore per annum. Using a recovery ratio of R = 0.90, a concentrate grade of about 95% heavy minerals and an ore feed grade of say 0.5% heavy minerals, the ore feed to concentrate ratio comes out to be about 211 which corresponds to the Australian data.

As depicted in figs (11), the beneficiation procedure exploits the magnetic property of ilmenite which is recovered after washing, blending and screening of the ore-feed. The non-magnetic portion which is made-up of quartz, rutile, zircon and the remaining non-magnetic portion of the paragenetic ore-suite minerals is classified in spirals and wet tables, filtered and then dried. Concentrate from the wet plant is then sent to the dry-separation plant where conductor minerals are separated from non-conductor minerals in a high tension separator. Both streams are subjected to magnetic separation. The magnetic fraction of the conductor minerals is mainly ilmenite concentrate. The non-magnetic fraction includes rutile. This fraction is subsequently subjected to electrostatic separation to recover rutile.

Production capacity would very much depend on project profitability and available ore reserves and also on potential market demand. Present measured reserves of about 400,000 tons of rutile ([143]), indicate the existence of about 84.4 million tons of heavy minerals ore plus waste, using an ore to concentrate factor of 211, as derived above using a base-case concentrate production schedule of 10,000 tons per (96% heavy minerals - TiO_2), the daily ore feed capacity would be about 2.1 million tons, which would completely exhaust the existing reserves in a scheduled production life of 38 years. Upon this base-case, capacity, other production capacity scenarios have been studied in relation to their effect on overall project economics, by means of a sensitivity analysis chapter 9

5.2.1-3 Cassiterite beneficiation scheme

Only the hard-rock beneficiation scheme for cassiterite processing has been discussed hereunder. Run-of-mine ore is first crushed and then cassiterite bearing ore fractions separated out of the crushed ore by heavy-media. The underflow is ground and treated for sulphides by means of sulphide flotation; that is if sulphide minerals occur in much quantity in the ore mineral suite. The non-sulphide fraction which is made-up of cassiterite and wolframite (principal ore minerals) and the rest of the accessory ore minerals is subjected to gravity separation to recover the principal minerals which have a high specific gravity (see table 1). These two minerals (cassiterite and wolframite) are separated by means of their magnetic susceptibilities, in magnetic separation stages. Middlings produced at the gravity separation stage are reground and reclassified. The overflow is recycled into the gravity separation stage and the underflow is discarded as waste. Concentrate obtainable should be about 50% Sn. The wolframite fraction can be marketed if it occurs in marketable quantity. Waste including fines and all forms of reject ore fractions are dumped at a convenient lieu for possible future re-use.

At feed grades of about 1% Sn and concentrate grades in the order of 50% Sn, the feed to concentrate ratio at a recovery of 80% is about 63. This means that a mill output schedule of 1000 tons of 50% Sn cassiterite will necessitate 63,000 tons of crude ore feed from the mine. If mining recovery has an extraction ratio of about 90%, then the mine ore capacity will have to be about 70 tons per ton of concentrate.

Lower ore grades would necessitate a greater ore feed to meet the mill output schedule; according to the equations set in section 5.2.1.

For a start, the production schedule will be envisaged to tally with existing producer capacities of most sub-Saharan countries:

Nigeria (2000 t/y)
Zaire (2500 t/y)

A base-case schedule of 2000 tons/annum of 50% Sn concentrate has been adopted. This will require in the neighbourhood of 6 million tons of 1% Sn cassiterite ore for a scheduled project life of about 38 years.

5.2.1-4 Bauxite beneficiation scheme

Fig (14) depicts the process flow in the beneficiation of alumina from bauxite ore of the gibbsitic-trihydrite type ($Al_2O_3 \cdot 3H_2O$). In the end-use specifications for alumina production, a high grade of 99.5% alumina (Al_2O_3), low in impurities of the noble oxides (MgO, CaO), is a requirement because both oxides increase the energy requirements of aluminium refining when they occur (21). In the Bayer process, crude bauxite containing a head grade of about 35% < Al_2O_3 < 45% with variable iron oxide 10% < Fe_2O_3 < 30%, quartz 4% < SiO_2 < 18% and rutile 2% < TiO_2 < 5% is dried, ground and reacted with soda ash (NaOH) and lime (CaO) in steel digesters (21):

$$Al_2O_3 + 6NaOH = 2Na_3AlO_3 + 3H_2O$$
$$Al_2O_3 + 2NaOH = 2NaAlO_2 + H_2O$$

Most of the alumina goes into solution as an aluminate. Some silica enters solution as an insoluable sodium aluminium silicate ($Na_2O.Al_2O_3.3SiO_2$) which consumes caustic soda. A red-mud also forms as a mixture of the unreacted titanium oxide (TiO_2), iron-oxide (Fe_2O_3) and some quartz (SiO_2). Both components of the reaction are separated by thickening, filtration and precipitation. The coarse precipitate is calcined at high temperatures to produce 99.6% Al_2O_3 (alumina) with minimal impurities of: iron-oxide

0.04% < Fe_2O_3 (iron-oxide)

0.05% < SiO_2 (quartz)

0.08% < Na_2O (soda)

Generally, 1 unit of alumina is said to require about 2.5 units of crude trihydrite bauxite ore (21); if allowance is given for dilution or incomplete extraction by 90%, then up to 3 units of crude bauxite ore will be needed to produce 1 unit of alumina at a recovery ratio of about R = 85%, ore-feed grade of f = 43% Al_2O_3 and a concentrate grade of 99.5% Al_2O_3. As stated in section 6.2.4-2, it is envisaged that this alumina concentrate will supply the smelter at Edea which currently depends on foreign sources for its alumina input.

5.2.1-5 Iron-ore beneficiation scheme

Beneficiation practice in upgrading low-grade metamorphic banded iron-ores of the West-African type (112), (143), (160) is similar to the practice in Australian cases (161) and ores of the Canadian shield·

The run-of-the-mine ore at a grade between 35% <Fe < 45% is fed into cone crushers to reduce feed to size suitable for grinding. Crushed ore is wet ground autogenously in closed circuit with a trommel screen (111). Undersize passes to spirals for classification. Middlings and fine magnetite undersize passes to the dewatering stage. The product is passed through a magnetic separator and the final concentrate filtered and transported to storage depot. Tailing is thickened before disposal.

Fig 11 Idealized beneficiation scheme (Rutile Heavy Sands)

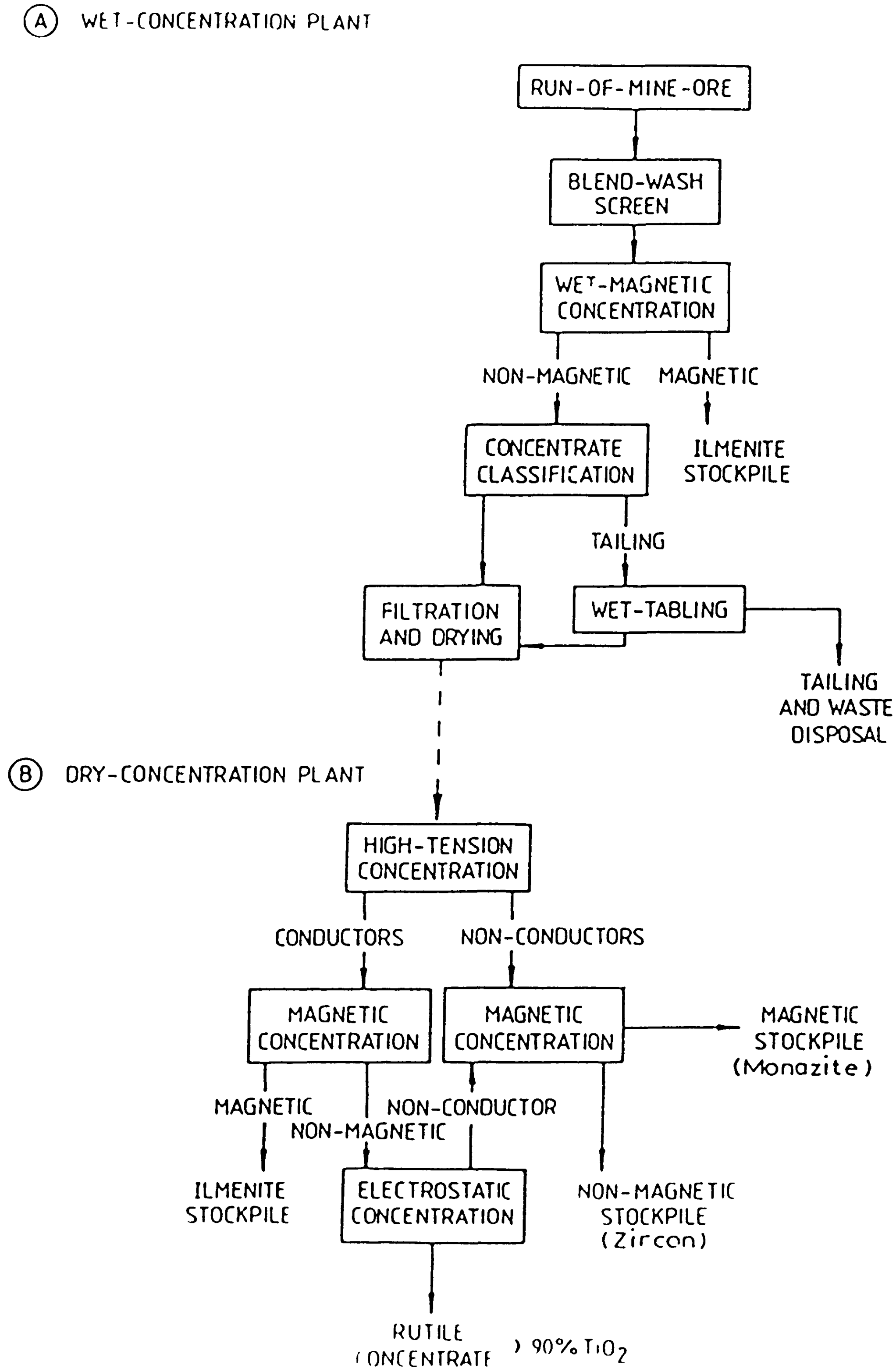


Fig 12 Idealized beneficiation scheme (Iron-ore)

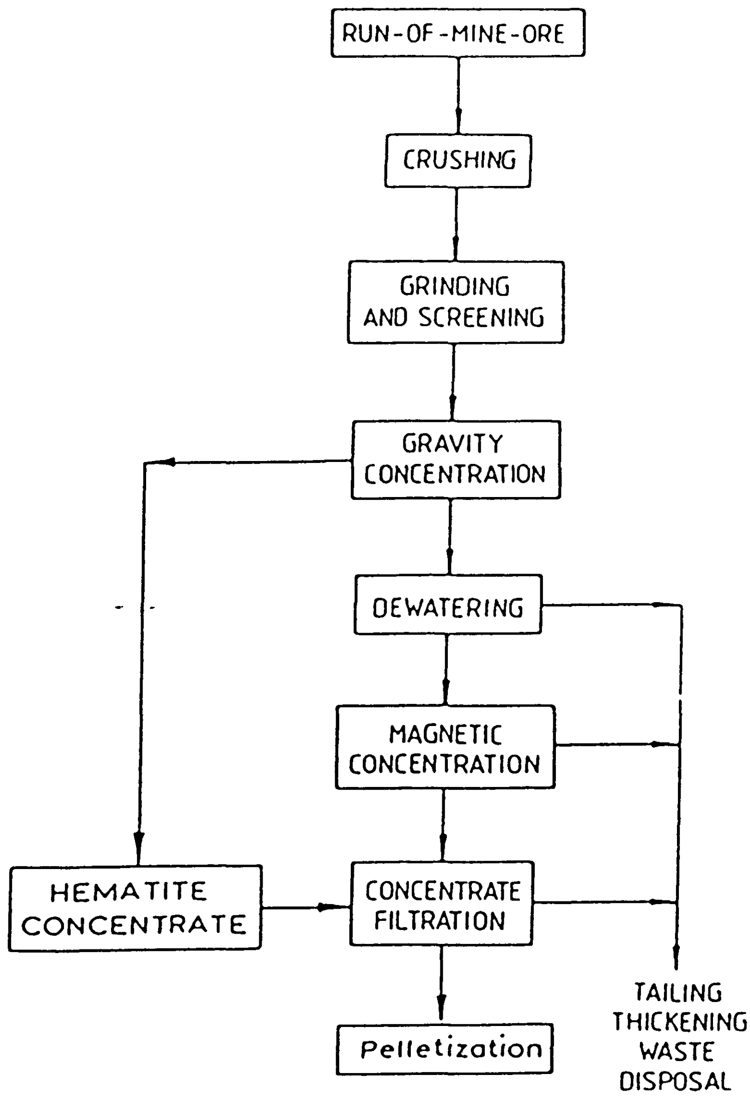


Fig 13 Idealized beneficiation scheme (Porphery-tin)

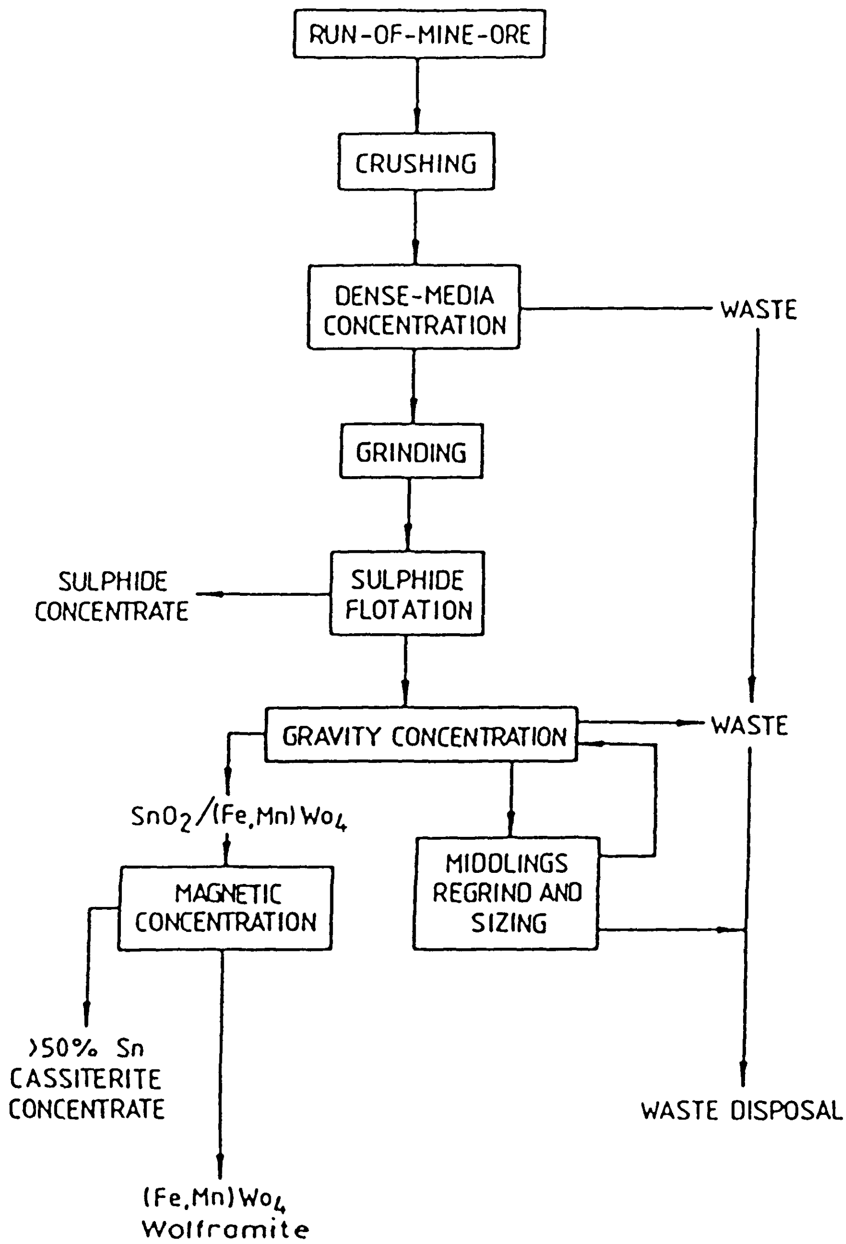


Fig 14 Idealized beneficiationn Schemo(Bayor process)

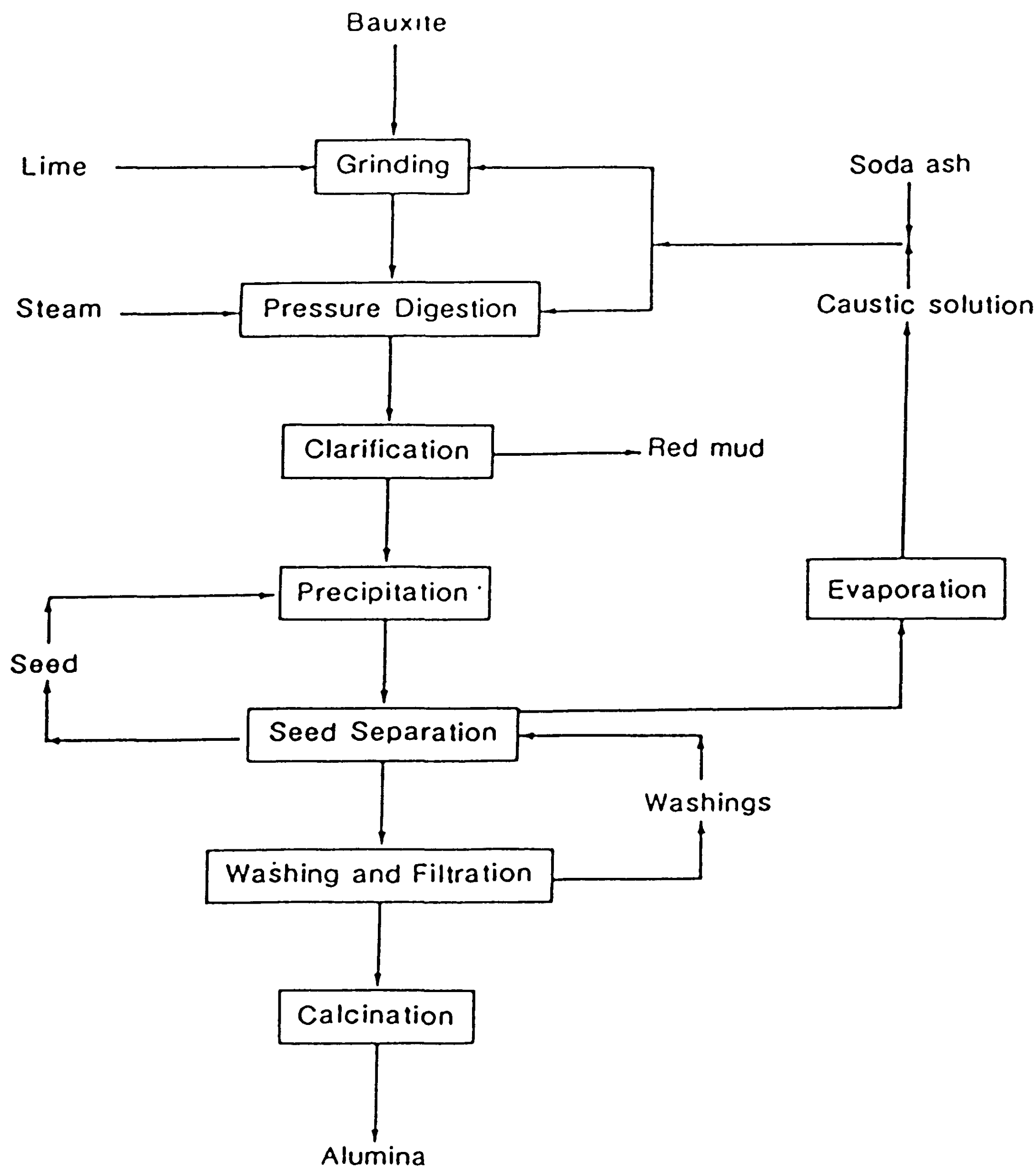


Table 8 Exploration schemes and costs for Case-Studies

TECHNIQUE	BAUXITE	RUTILE	IRON-ORE	TIN-ORE	COST
Aerial photo interpret Black and white Coloured	A A A	A A A	A A A	A A A	$2000/print $40/sq. km. $60/sq. km.
Surface mapping Geological inference	A A	A A	A A	A A	$150 sq. km. $400/Day
Soil sampling Stream sediments	A A	A A	A A	A A	$2500/sq.km. $30/sq. km.
Magnetic Surveys (Air) Magnetic Surveys (ground)	NA NA	A A	A A	A A	$30/sq.km. $100/sq.km.
Electromagnetic (Air) Electromagnetic (ground)	NA NA	A A	A A	A A	$30/km.Prof. $100/km.Prof
Gravity Surveys	NA	A	A	A	$600/km.Prof
Resistivity Surveys	NA	A	A	A	$60/depth probe
Induced Polarization	NA	A	A	A	$40/sq. km.
Drilling	A	A	A	A	$100/meter
Drill-hole logging	A	A	A	A	$20/meter
Seismic Surveys	NA	NA	NA	NA	$150/depth P.

A = Applicable

NA = Not applicable

Table 9

EQUIPMENT/JOB PARAMETERS (MINEX model)

1 Shovel capacity in cubic yards = 6 cu. yds.
2 Truck capacity in cubic yards = 35 cu. yds.
3 General horse-power engine load-factor = 0.67
4 Shovel fillability factor = 0.9
5 Truck load time in minutes = 3 mins.
6 Total truck cycle time per haul-dump (min) = 10
7 Mechanical availability (equipment) = 0.85
8 Effective shift time in minutes = 1.5 hours
9 Drill-penetration rate feet per minute = 0.85 ft/min.
10 Bit-life in feet = 2000 ft.
11 Stand-by factor for equipment = 1.2
12 Job efficiency = 50 min/60 min hour

Table 10

GEOTECHNICAL PARAMETERS (MINEX model)

1 Burden constant = 45
2 Bench Heights in feet = 30 ft.
3 Lift of rock factor (sub-drill) = 0.7
4 Burden to Spacing constant = 1.25
5 Drill-diameter in inches = 5 inches
6 Powder factor ANFO lbs per cu. yd. = 2.6 lb/ton
7 Drillability factor ore and waste = 1.2
8 Abrasiveness = 1.2
9 Stripping ratio = 3
10 Mean ore and waste density lbs/cu. yd. = 4420 lb/cu. yd.
11 Swell factor ore and waste = 1.2
12 Charging density of explosive = 0.8
13 Daily production capacity = 20,000 tons.

CHAPTER 6 Cost-Modelling of MINEX

6.1 Generalities

Conventional methodology in investment appraisal requires costing to be carried out in the first instance according to capital and operating headings. Capital costs comprise lump-sum disbursements such as those incurred at the initial stages of project-life for the procurement of "real capital" (excavation machinery, processing plants, utilities and facilities plus a stock of material inputs for start-up of operations). Operating costs include all expenditures made on a recurring basis to enable project to function (energy costs, employee wages, equipment maintenance costs, general supplies and communication bills). These costs are estimated over the project life together with anticipated gross revenues (quantity of product times the produce price) per annum.

The difference between revenues and costs per annum (cashflow), equals to the annual net incremental change in cash level. When account is taken of fiscal costs and allowances (taxes, royalties, depreciation and loan allowances) within the cashflow calculations, the result becomes known as net cashflow.

Project profitability and economic feasibility are conventionally tested by the Net Present Value (NPV), Discounted Cashflow rate of return (DCFRR) criteria (157) and informally by the payback criterion; all of which have been briefly discussed below.

The type of costing undertaken within this work is intended to fall within the order-of-magnitude category of cost estimations as described by Brian Lawrence of Dravo Corporation (61). This is so because of the nature of the data base and also because this project has a purely academic essence. The costs used are mostly historic and academic; access to real world cost values is an intriguing experience because most of these even when they do exist are held within confidential company confines.

6.1.1 Costing procedures

Straam Engineers (132), O'Hara (106), Church (25), Mular (101) and Bertoldi (13) are some important sources from which most of the cost data has been elicited. The first four have formed the basis of formulating an "element by element" procedure for costing the mineral exploitation model (MINEX).

Mular has used prime mine equipment parameters such as shovel or truck capacities, drill weight, compressor horsepower to relate equipment costs. These parameters are assumed to bear quantitative relationships to their equipment costs, within certain valid ranges. These relationships are of the form

$$\text{Cost (C\$)} = a\,(X)^{b}$$

X = equipment parameters
a, b = constants
C$ = Canadian dollar

His basic data were obtained from many manufacturer sources and have been updated by him to reflect 1982 cost values. The cost of haulage trucks with capacity range 22 tons < X < 90 tons is given in 1982 Canadian dollar terms as

$$\text{Cost (C\$)} = 13420\,(X)^{0.8773}$$

These cost relationships have been given for production drill-rigs, bulldozers, front-end loaders, shovels and air compressors (101).

H. Church (25) uses a similar principle assuming that all costs are approximately proportional to the total capital cost of the equipment. He has translated capital costs for machinery and their operating costs (ex-operator wages and fringe benefits) into hourly costs as a percentage of thousands of dollars (US$1000s) of their capital costs. These cost figures are for the United States and include average capital costs of machine assembled and erected f.o.b job in 1978 terms. Costs of machine operation have also been presented by Church as a percentage of thousands of dollars and are to include consumable items such as tyres for trucks, cutting edges for bull-dozers, bits for drills and other accessory equipment consumables. The hourly cost of operating a large drill of capital cost say $600,000 (overburden drill) is given as 2% of the $1000s of its cost. That is

$$\frac{2 \quad X \ \$600}{100} = \$12/\text{hour}$$

and for an annual work schedule of 2000 hours this gives $24,000 per annum, including rods, bits and other drill accessories. His cost data base was gathered over a period of 50 years from contractors, equipment manuals, construction and mining companies, magazines, books and various kinds of publications.

Straam Engineers and O'Hara have developed cost relationships based on mine-project parameters such as plant capacities, ore and waste schedules for various cost centers of mining systems (132), (106). Cost expressions were developed from a computerized statistical analysis of the best fit of the cost data to an expression of the form:

$$Q = KT^X$$

Q = actual data on quantities required or cost

T = represents the tonnage rate, milled head grade or some other physical condition causing change in quantities or costs.

X,K = constants

Total capital costs of an open-pit and mill project in 1982 Canadian dollars has been given by O'Hara as

$$\text{Cost (1982 C\$)} = 566{,}372\ Tm^{0.6}$$

where Tm = Tons of ore mined per day

Straam Engineer's relationship for the capital cost of the drainage system of a surface mine operation is:

$$\text{Cost (U.S.\$1983)} = 1.654\ (X)^{0.831}$$

where X = drainage capacity expressed as metre - cubic metres per day.
metre represents the total pumping head.

The modelling of a mine system into its basic components enables such a system to be costed according to first principles by simply summing up these system components of the cost model. A system modelling exercise (chapters 4 and 5) has enabled costing of MINEX according to this procedure.

Element by element costing procedures are extensively used in budgeting schedules; Bertoldi (13) uses this method in his work while McClay (89), adopts the same methodology via the Master Simulator (86) in the evaluation of mine systems. Mular (101) recommends this procedure for the costing of itemized quantities of engineering projects such as these.

6.2 Capital Expenditures

6.2.1 Exploration costs

Exploration costs have been estimated to include the minimum cost of acquiring mineral rights as detailed in law No. 78/24 of the 29th December 1978 (Mining taxation code).(109)

The rest of the capital expenditures in the exploration cost heading are due to geological surveys, photogeological interpretation, geochemical prospecting and geophysical exploration for the sake of delineating orebodies of a pre-determined size.

Exploration schedules have been detailed for each case study in table 10. Generally the use of exploration techniques for delineating orebodies, depends partly on the mineralogical characteristics (chemical and physical) of the inherent minerals. Basically, surface mapping ($150/sq.km) of all mineral occurrences is an indispensable first step to applying other techniques·Aerial photos ($40/sq.km black and white photos, $60/sq.km. coloured photos) are also very useful as exploration aids in exploring uncharted and remote mineral sites.

Geological inference costs about $400/day, while aerial photo interpretation is estimated to cost around $2000 per print. These cost estimates are obtained from (35).

Soil sampling and stream sediment analysis ($2500/sq.km. and $30/sq.km.) are almost always used as surface methods to complement aerial or remote sensing techniques in mineral exploration. The use of gravity techniques, resistivity and induced polarization methods is dependent upon the chemical and physical identity of the mineral occurrence in question . These methods cost $600/kilometer profile, $60/depth probe and $40/square kilometer respectively. These three techniques have been used for estimating the cost of delineating rutile heavy mineral sands, Iron ore and the hard rock cassiterite occurrence. Air and ground magnetic surveys ($30/km. profile and $100/km. profile respectively) have been used in the above mentioned three mineral cases. The same applies for electromagnetic survey (ground $100/km. profile; air $30/km. profile). Drilling ($100/meter) and drill hole logging ($20/meter) have been used in all four mineral cases. Seismic surveys ($180/depth probe) have not been considered in the exploration schedules of any of the mineral cases.

The cost of obtaining a prospecting licence is equivalent to $7000 at an approximate conversion rate of 300 CFA Francs to $1 U.S. The fee charged for issue and first year renewal of a mining permit approximates to about $10,500 for issue (3,000,000 CFA francs) and $14,000 for first renewal (4,000,000 CFA francs). Concession granting costs a minimum of $175,000 (50,000,000 CFA francs), while annual rents for the concession cost $7000 (4,000,000 CFA francs) and mining $3500 (2,000,000 CFA francs). The total minimum cost of acquiring mineral rights, exclusive of initial exploration costs which are based on the total initial exploration area at a cost of $1.0 per sq. kilometer, are estimated at about $227,000 or 74,4000,000 CFA francs. A summary of the exploration capital costs has been presented for each case study (tables 13, 17, 21 and 25).

Note

CFA francs = African Financial Community francs. Local currency of the country equivalent to 1/50th of a French franc. Approximately 300 CFA francs equal $1 U.S. (1982 estimates).

6.2.2 Site-development and Utility Costs

6.2.2-1 Site development

It comprises of all initial work directed at developing the mineral site such as access road construction, clearing and stripping operations in preparation for production work. This includes construction of town-site for workers, basic infrastructure of road and or rail lines necessary for transportation of inputs for the project and for movement of mineral produce to depot for eventual marketing. Utilities comprise of offices, laboratories, storehouses, electrical power facilities, water supply drainage and fueling stations.

The cost of mine access and site preparation has been obtained from O'Hara (106) who expresses the variation of costs as a function of the square root of the size of the daily output capacity (ore and waste):

$$\text{Cost (U.S.\$1983)} = 7100\ (Tp)^{0.5}$$

where Tp = output capacity for ore and waste. The cost of site stripping is given as:

$$\text{Cost (U.S.\$1983)} = 12070\ (X)^{0.5}$$

where X = overburden mass in millions of tons. This cost relates to stripping in heavy vegetation and difficult work conditions such as in the tropical rainforests of the Kribi area or the high savannah woodlands of Mayo Darle and Dschang.

O'Hara's cost relationships for plant site clearing, excavation and foundation concreting have been used:

$$\text{Cost (U.S.\$1983)} = 56800\ (X)^{0.3}$$

where X = mill output per day in tons; cost referring to clearing and foundation excavation only. Costs for concrete foundation construction is given as:

$$\text{Cost (U.S.\$1983)} = 28400\ (X)^{0.5}$$

Throughout the cost modelling exercise, costs have been updated to their 1983 value by escalation factors. A factor of 1.8 has been used to update all United States cost figures for 1975 ; which represents the Marshall and Swift chemical plant and equipment cost index ratio between 1983 and 1975 (88). Canadian costs have been converted to their United States dollar equivalents and then escalated to their 1983 values. A factor of 1.60 has been used to update O'Hara's and Mular's costs.

The cost of townsite construction for employees (Cameroonian and expatriate) has been done using an approximate figure of $10,000 (U.S.) per house, capable of housing two grade 3 or 4 workers and $20,000 (U.S.) per house containing one grade one or two worker. Company chiefs and top executives would be housed in single bungalows costing approximately $30,000 (U.S.) each. Abundance of local raw materials such as wood, will be an inexpensive input to cut down on these costs.

Construction of mine and mill offices, laboratories, storehouses and change rooms, including repair shops and garages has been given by Straam Engineers (133):

Offices and laboratories for mine -

$$\text{Cost (1983 U.S.\$)} = 1736.8\ (X)^{0.485}$$

repair shops, warehouses, and garages for mine -

$$\text{Cost (1983 U.S.\$)} = 5544\ (X)^{0.576}$$

other surface buildings for mine -

$$\text{Cost (1983 U.S.\$)} = 4089.6\ (X)^{0.375}$$

where X is ore and waste output tonnage per day.

Mill offices and laboratories:

$$\text{Cost (1983 U.S.\$)} = 29282.4\ (X)^{0.35}$$

where X is the mill feed, tons per day.

6.2.2-2 Mine utilities and costs

Capital costs under this heading include the cost of acquiring and installing a drainage system, water system, communication, electricity and fueling facilities. Costs are obtained from Straam Engineers (133).

Drainage system:

$$\text{Cost (U S \$1983)} = 27.97\ (X)^{0.738}$$

This will include construction of ditches, culverts, sumps and drainage facilities needed to drain the immediate mine vicinity. The cost varies as a function of (X) which is the quantity of water to be drained from the open-pit, expressed as meter-cubic meter of water, with the first meter representing the pumping water head. Drainage of pits during the rainy season would be quite an important factor in determining the continuity of work efficiency at the assumed rate of about 85%.

A water head of about 2 meters for a material density 1.689 Cubic meters per ton of bauxite (ore and waste), would necessitate the drainage of about 4 meter - cubic meters of water. Cassiterite ore and waste assumed to weigh 1 ton per 4.35 cubic meters, iron ore and waste weighing say 1 ton per 3.27 cubic meters would respectively require 9 and 7 meter-cubic meters of water to be pumped. The heavy mineral sands operations will not create pits and mine faces, since no blasting or vertical excavation would occur. Exploitation would be mainly lateral and will not create water ponds requiring pumping. Cost of pumping is given by Straam Engineers as:

$$\text{Cost (1983 U.S.\$)} = 1.654\ (X)^{0.831}$$

where X = meters - cubic meters per day.

Communications system:

$$\text{Cost (U.S.\$1983)} = 447.8\ (X)^{0.495}$$

Fueling system:

$$\text{Cost (U.S.\$1983)} = 65.9\ (X)^{0.846}$$

Electrical system:

$$\text{Cost (U.S.\$1983)} = 63.1\ (X)^{0.846}$$

In the three capital cost relationships, (X) represents the total daily ore and waste output capacity in tons.

The availability of abundant water resources and hydroelectric potentials in most Cameroonian localities is a great asset to these potential projects and could be harnessed to cut down on expensive commercial purchases.

6.2.2-3 Mine equipment capital costs

Derivation of basic mine equipment requirements from first principles (section 4.2) will depend on size and number of drills plus their tandem compressors, number of shovels and trucks whose fleet sizes depend mainly on the mine production schedule (see figs 15,19,20), equipment capacities, job and equipment parameters (section 4.3).

Mular's (101) and O'Hara's cost relationships in 1978 Canadian dollars for the estimation of shovel costs according to their capacities are given as:

$$\text{Cost (C\$1978)} = 105{,}916\ (X)^{1.09} \quad \text{(Mular)}$$

$$\text{Cost (C\$1978)} = 230{,}000\ (X)^{0.73} \quad \text{(O'Hara)}$$

where (X) = shovel capacity in cubic yards.

Using an 8 cubic yard power shovel, estimates come out to be about C$ 1,050,000 and C$ 1,049,000 with a discrepancy of about $3000 which is due to averaging and regression (+0.3% error).

Dump truck costs obtained from Bertoldi (13) and cost estimates using O'Hara's relationship:

$$\text{Cost (C\$1977)} = 8370\ t^{0.85}$$

for an 85 ton dump truck were in close agreement using a conversion factor of 0.93 U.S.$ to the Canadian dollar in 1977. (U.S.$370,000 O'Hara), (U.S.$380,000 Bertoldi). Horace Church's costs for the same year are slightly lower at U.S.$340,000 per unit 85 ton truck. The discrepancy in cost may be due to lieu of purchase - United States for Church, Canada for O'Hara, but a disparity of $40,000 (U.S.) between Bertoldi's costs and Church's costs exists.

Costs for drills have been obtained from Mular using his cost relationship.

$$\text{Cost (1983 U.S.\$)} = 40.6\ (X)^{0.861}$$

where X = is the drill pull-down force in lbs.

For a 59,000 lbs.(pull-down force) drill having diameters ranging from 5 inches to 8 inches, the estimate gives about $520,000. This figure accords with the cost range provided in Bertoldi's work (about U.S.$400,000 to U.S.$1,000,000 for small and large overburden drills in their 1983 costs).

In general it can be said that cost sources for mine equipment are in close agreement because they reflect North American cases. Differences in costs over the years are due to currency exchange rate fluctuations and differential inflation between American and Canadian markets.

Ancillary mine equipment such as cranes, fork-lifts, graders, caterpillars, scrapers, front-end-loaders, personnel trucks, explosive trucks and ambulance units have been considered in addition to the basic shovel-truck-drill fleet requirements. Because this heading has an optional character about it, one unit each has been included as ancillary equipment requirement including 5 personnel trucks of medium size. The costs of these items can be found in reference

(13). A factor of 1.5 has been used to account for on site delivery of equipment in Cameroon, from a North American or Western European market.

6.2.3 Beneficiation plant and utility costs

6.2.3-1 General costs:

Beneficiation processes have been separated into identifiable cost-centers each of which has been depicted in the process scheme of the mineral cases considered (figs,11 ,12,13,14). These costs include the acquisition and installation of the main pieces of plants plus any accessories required for plant functioning. Cost relationships have been taken from Straam Engineers (132). Utility costs for provision of a water system and an electrical system have also been obtained from the Straam Engineers.

The crushing section includes crushers, conveyors, screens and feeders as major items of equipment which will enable the reduction of run of mine ore to a fine ore size suitable for grinding:

$$\text{Cost (U.S.\$1983)} = 8249.4\ (X)^{0.749}$$

(X) = mill feed capacity per day

Flotation section covers items such as slurry pumps, piping, flot cells and agitators or conditioning tanks. Only the single product conditioning costs have been considered:

$$\text{Cost (U.S.\$1983)} = 24193.8\ (X)^{0.512}$$

Concentrate thickening and filtration costs are given respectively as:

$$\text{Cost (U.S.\$1983)} = 8247.6\ (X)^{0.514}$$

$$\text{Cost (U.S.\$1983)} = 24969.6\ (X)^{0.558}$$

Magnetic concentration requires such major items as screens, magnetic separators and pumps. Costs are given as:

$$\text{Cost (U.S.\$1983)} = 2246.4\ (X)^{0.841}$$

Gravity concentration includes major equipment items such as jigs, screens, conveyors and pumps. The cost relationship is:

$$\text{Cost (U.S.\$1983)} = 3668.4\ (X)^{0.841}$$

Concentrate dewatering, concentrate drying and tailing disposal are respectively given as:

$$\text{Cost (U.S.\$1983)} = 1000.6\ (X)^{0.808}$$

$$\text{Cost (U.S.\$1983)} = 60190.2\ (X)^{0.339}$$

$$\text{Cost (U.S.\$1983)} = 4590\ (X)^{0.489}$$

6.2.3-2 Costs for the Bayer Scheme

Costs for the Bayer proces scheme selected for bauxite beneficiation were adapted from two sources - P. Christie and R. Derry (21) and the United States report of investigation RI6730 (21). These costs were escalated to their 1983 values using a factor 1.8.

The costs represent capital for acquisition of a 350,000 ton per annum alumina plant. A relationship has been established relating the capital cost of processing plants with their operating capacities (the six-tenth rule) (101):

$$\frac{\text{Cost (1)}}{\text{Cost (2)}} = \left[\frac{\text{Capacity (1)}}{\text{Capacity (2)}}\right]^{0.6}$$

where cost (1) and Cost (2) represent capital costs corresponding to capacities (1) and (2).
The relationship becomes:

$$\text{Cost } (T_m) = \text{Cost } (350{,}000) \left[\frac{T_m}{350{,}000}\right]^{0.6}$$

where = mill feed capacity per annum.
Estimates of costs by Christie and Derry ( ) updated to 1983 U.S.$ values come up to about $100 million for a

350,000 tons per annum Bayer plant for alumina production.

Hence the relationship becomes:

$$\text{Cost (Tm; U.S.\$1983)} = 100 \left[\frac{Tm}{350{,}000}\right]^{0.6}$$

The plant would generally include a steam section, air compressors, plant facilities such as buildings, laboratories, shops, fences, grinding and digestion section, clarification and precipitation section, spent liquor recovery and lime calcination plant plus an alumina calcination section. 15% of the costs will cover engineering and construction expenses (21).

Ancillary equipment and vehicles costs for the mill plant include such items as service vehicles, emergency lighting, standby generators and other special purpose equipment normally needed in the mill section. This cost heading is expressed as a cost relationship of the form:

$$\text{Cost (U.S.\$1983)} = 11723.4\ (X)^{0.640}$$

where X = mill feed, daily capacity in tons.

Costs of equipment and plant replacements amount to about 5% (132), these are calculated annually and are depreciable.

The alumina plant will be aimed at feeding the 180,000 tons per annum aluminium smelter at Edea (see fig. 1). Edea enjoys abundant hydroelectric energy facilities and was one of the first aluminium smelters installed in Africa . but input for the smelter originates abroad from Guinea and Europe despite abundant bauxite ocurrences in Cameroon (4% of world known reserves (127)).

6.2.4 Infrastructures, Agro-restoration Engineering Construction and feasibility capital costs

6.2.4-1 Infrastructures:

Capital cost centers have been allocated for setting up of roads and rail links between the mine or beneficiation plant and the produce depot where the final concentrate will be stored pending export to overseas markets or as the case may require, internal consumption (alumina for Edea smelter). The cost per kilometer of road construction as given by O'Hara (106) is:

$$\text{Cost (1983 U.S.\$)} = 284{,}000 \text{ per kilometer}$$

This cost includes provision for adequate drainage curvature and gradients to permit satisfactory concentrate haulage conditions. The road type is 30 feet wide and gravelled with material coming from local sources. Provision for a major bridge has been made in each case to span a major river or creek:

$$\text{Cost (1983 U.S.\$)} = 184.6\ (X)^{1.5}$$

where X is the bridge width in feet.

Road construction in the iron-ore prospect will amount to about 30 kilometers and provide an all weather road network to the port at Kribi. The road from Fongu-Tungong about 50 kilometers to Dschang will be reconstructed. Bafousam would then serve as a railine terminus with the construction of a 100 kilometer rail-link with Nkongsamba, the present rail terminus (see fig 1). Fifty (50) kilometers of this rail-link will be attributed to the alumina model while the remaining (50) kilometers to the cassiterite model at Mayo-Darle. The remaining road distance from Bafousam could then be redeveloped into a full-scale highway; which incidentally happens to fit in with long-term planning for a trans-African highway through this road .
The rutile heavy sands model will not require much

infrastructure development because the littoral and central south provinces have a network of motorable roadways and rail lines. However about 50 kilometers of roads will be constructed within the heavy minerals polygone wherever and whenever it becomes necessary to translocate the operations to obtain new ore.

Straam Engineers' relationships for rail-road construction have been used.

$$\text{Cost (1983 U.S.\$)} = 22.61\ (X)^{1.0} \cdot 10^4$$

where X = kilometers of rail line.

6.2.4-2 Restoration cost of mined land for agricultural use

Restoration of mined-out areas and adjoining waste dumps to arable and recreational land-strips could be one way of creating positive impacts on the environment and the community ([38]). This exercise though lofty in concept could be actually less exacting in terms of costs. The estimated costs of restoring an hectare of land, fit for mechanized agricultural use in the United States (133) is about $9000 (U.S.$1983). This amount can be used to level up disused areas of the mine with less rigorous restoration requirements - minimum fertilizers, scrub and tree growth. The cost per kilometer has been estimated at $10,000 (US$1983) using cheap local labour, water and fertilizers from Douala, lime from Figuil at less exorbitant costs. It is possible by so doing for mining to have positive impacts on the Cameroonian economy, 80% of which is directly dependent on agriculture.

6.2. -3 Engineering, feasibility and start-up costs:

The cost of engineering construction, feasibility study and project design are said to bear a relationship to the overall cost of acquisition and installation of mine and mill plants, buildings, townsite, mine and mill equipment, access roads, pre-production development and restoration (133). O'Hara states that about 4% to 6% of direct capital costs are attributable to feasibility studies while 6% to 8% of all other mine costs (capital) are due to design, engineering and technical planning.

Straam Engineers relate these costs to the net constructing costs (\$X) in the equation:

$$\text{Costs (U.S\$1983)} = 0.529\,(\$X)^{0.922}$$

Start-up and working capital costs are estimated at about one month of full capacity operating expenses for administration, mine and mill operations. It will be the capital expenditure required to meet payrolls and pay bills for material and product inventory, and to carry accounts receivable until ore sales produce revenue to pay for these items.

6.3 Operating costs

6.3.1 Labour costs

The total labour force engaged in project work is often determinable only during project implementation and is subject to government and company negotiated policy. In these base-cases the work force for each of the four case-studies has been maintained above 200 people. This includes an expatriate fraction of up to 20% of the total workforce, the rest of whom are local employees and government representatives. Expatriate experts are divided into 2 groups; 30% fall in category one and the rest in category two. The local labour force is in four categories.

Category four has an annual salary of about U.S.\$4,000, category three a salary of U.S.\$6,000, category two U.S.\$15,000 and category one has U.S.\$24,000 per annum. The expatriate category two is expected to earn a basic salary of U.S.$24,000 per annum while the top company executives receive U.S.$30,000 per annum. Expatriate and local allowances have been included in these salaries.

6.3.2 Supplies, maintenance and repair costs

The cost of consumables needed for the operation of the mill section have been presented in reference (132). These are separated into supplies and equipment operation and are related to the daily mill feed tonnage. These costs have been computed for each day of the work year (350 days) against each of the cost centers of the process schemes (figures 11,12,13,14). These cost relationships are identical in format to the capital cost relationships presented above .

Costs of power and fuel needed to run mine equipment (shovels, trucks, drills and ancillary vehicles) have been estimated using Church's procedures (25), whereby the hourly operating costs are calculated as a percentage of the $1000s of the cost of an equipment piece (see section 6·1·1). Hourly operating cost percentage of the $1000s of major equipment items is on the average about 2% (25). An annual schedule of 2000 working hours has been assumed. These costs are contrived to include the cost of spares such as drill-bits, tyres and drill stems.

The cost of explosives consumption per annum has been estimated from the blasting requirements which have been derived in section 4·2·2 · Cost of explosives per ton (ANFO) is about $260 1983 U.S. value (61), (155), while explosives accessories expressed per ton of ANFO are estimated at about U.S.$2 to $3. A factor of 15% has been used to account for secondary blasting where this might be prevalent as in the fragmentation of porphyry tin granitoids and weathered country rocks and metamorphic iron rocks. Rutile operations and bauxite mining would not require in some cases any drilling or explosives consumption. Examples of this non-usage of explosives in bauxite mining can be found in Weipa (161), Western Australia. However, a modest amount of fragmentation (<50% of full capacity estimate) has been used in the bauxite model and one drill rig would also be necessary to accomplish drilling requirements.

Like operating costs, maintenance and repairs have been estimated to be related to the percentage of the U.S.$1000s dollars of the equipment cost (25). The cost of maintenance and repairs has therefore been estimated using 5% of the $1000s of the equipment costs.

In the Bayer process, the costs of steam, fuel oil, electricity, starch, lime and soda ash, plus other unconceived accessories and consumable items have been estimated (21) at about $117 (U.S.1983) - per ton of alumina processed.

6.3.3 Overseas transportation costs; inland transportation of supplies and concentrate

Mineral produce markets are traditionally located in Western Europe, for most West-African producers. This outlook does not appear to change in the long run. As such a cost center for overseas transportation of mineral concentrate (unsmelted concentrate) from mines in Cameroon, via a suitable port depot in Douala or Kribi has been assigned to each case-study. Alumina will be sold directly to the smelters in Edea from the alumina process plant in Fongu-Tungong near Dschange at a distance of about 500 kms. Cost of inland transportation will be about 8 cents per ton kilometer (surface, road or rail).

The cost of concentrate delivery from Douala or Kribi (West African ports) to ports within the European Economic Community have been put at about $5 U.S. 1983 per ton.

These operating costs have been estimated and separated into four subheadings:

- mining operating costs
- milling and concentrate transportation costs
- salaries and wages
- overseas transportation costs

Both capital and operating cost estimates have been factored by a contingency figure of 10% to take care of unforeseen cost elements within the total MINEX model.

6.4 Cash-flow modelling

Cash-flow constitutes a monetary representation of an investment transaction wherein the annual cash disbursements and revenues are computed in the light of fiscal constraints, to produce net cash flows due to the investment.

Cash-flow appraisal is at the heart of capital investment studies and has to do with the occurrence of cash as a function of time and interest. The effects of inflation and depreciation schedules on these cash impulses and financial leverage are also used in determining the profitability of any investment. Some of these topics have been discussed below.

Project life has been envisaged for 38 years for each case-study, during which time, explortion activities for the retrieval of subsurface data would be carried out over the space of two years. A definitive feasibility study conducted at the beginning of the third year could be integrated with commencement of construction and carried out by the engineering contractors of the mine and mill plants and buildings. Mine equipment will be commissioned in the third year and delivery and installation completed in the fourth year before production commences in year five. Infrastructures are installed preferably two years before commencement of production, since market efficiency will be dependent almost entirely on the timely delivery of both mineral produce and operational inputs. As a result, the first four years of project life have been considered as the pre-project development period, at the end of which time, production would begin.

6.4.1 Economic factors affecting cash-flow

The effects of inflation, time value of money, taxation and depreciation have been discussed below in relation to their impact on the estimated cash-flow and project profitability.

6.4.1-1 Taxation and other financial questions

The fiscal regime of non-fuel mining in Cameroon (109) has not been studied with regard to its impact on non-fuel mineral exploitation. This appears to be due partly to the absence of a non-fuels minerals industry and secondly because the current tax regime is new and (1978) comparatively fair (36, 79).

Fiscal issues seem to be the bases on which the profitability and success of mineral exploitation are assessed. This view seems to be shared by many mineral men especially those involved in mineral taxation and policy formulation. Carman (20), Krige (73), Lentz (82), Keith Buck (69) and Gillis (46 47) have written exhaustively on mineral taxation and policy matters.

Gillis states that:

"mineral taxation is at the backbone of policy formulation."

Fiscal obligations of mineral investors are generally stipulated in the form of royalties, corporate tax, rents, fees and duties. These charges do cost the investors a lot of money and must be viewed as costs to their operations; especially as these fiscal charges are not productive input costs. Fiscal payments go into government funds as remuneration for exploiting her natural resources by profit seeking entities.

Fiscal laws of the mining world (36) vary from country to country and are styled to reflect individual country government attitudes towards her mineral resources. Carman (20) has noted that each country should formulate her fiscal laws and overall policy to reflect individual outlook and priorities, instead of blind emulation of foreign laws, which often bear little compatibility in terms of economic aspirations or policy towards mineral endowment .

The fiscal requirements for acquiring mineral rights have been presented in the exploration cost model (section 6.4.1). Corporate taxation for non-fuel mining organizations is made up of a 30% standard corporate taxation rate and a 27.5% withholding tax rate. The royalty rate is 5% of gross f.o.b. value of mineral produce. Government participation is not stipulated but various scenarios have been studied - 30%, 50% and 70% equity participation in investment. It is unlikely that the basic tax regulations will be changed drastically or even moderately in the short-run.

As such base-case studies have adopted the present stipulations in law No. 78/24 of the 29th of December 1978 (106

Depreciation of capital expenditures has been scheduled to take place over seven years using the straight-line depreciation procedure (151).

An initial allowance of 20% on the capital costs is assumed to be allowed as a fiscal advantage to encourage more capital investment. The rate applied in the depreciation schedule is 12%.

The foreign exchange component of most capital cost headings is estimated at 90%, except in the case of rail and road construction because in the former, most of the capital equipment and machinery would normally have foreign provenance and would be financed from abroad, while local gravel and timber would form a major part of their costs. All mining and milling operating costs have been estimated as having about 50% foreign exchange component because much of it originates when production commences, from operating revenues and is recycled domestically in the form of wages for labour and power supply. Overseas transportation of concentrates has been assumed to have a 90% foreign exchange component because the costs are paid mainly to foreign cargo ships.

Exchange rate fluctuations have depressed the value of the French franc and consequently the CFA franc to which it is tied:

50 CFA francs = 1 F franc

A weak CFA franc would facilitate the export of mineral produce though not necessarily helping in the net income position. The United States dollar has been used throughout the calculations as the main currency of exchange at a conversion rate of:

300 CFA francs = 1 $ U.S.

Financing methods, effects of financial leverage and or gearing have not been considered in this work because the study of financial details of that nature are unwarranted at pre-project phases of mineral exploitation (conception phase). Stermole (131) also cautions on the use of zero leverage for base-case economic analysis, because leverage effects could often cause misleading results in economic analysis. Much study of this topic has been carried out by Eugene Pfleider (43) and Claus Freyberger, who have demonstrated the effects of long-term debt and equity capital (various combinations of financing) on project profitability. Financial leverage results from the increase in total finances of an investor with a decrease in the overall cost of capital; which is often due to a lower after tax cost of borrowed money than the cash investment DCFRR (151). Generally if zero leverge cash investment analysis results look good, leverage results would look even better.

6.4.1-2 Inflation and time value of money

Inflation decreases the purchasing power of goods over time causing new project inputs to be charged at high prices. The presence of inflation in an economic set-up has the effet therefore of distorting cost schedules, cash-flow profiles and profitability indices within such an economic environment.

Cash-flows are therefore corrected to their current purchasing power in year zero by discounting the cash-flows by an acceptable inflation rate (55):

$$NPVpp = Acf + \frac{Acf}{(1+i)(1+d)} + \frac{Acf}{(1+i)\ (1+d)} + \quad + \frac{Acf}{(1+i)\ (1+d)}$$

Similiarly, the impact of inflation is to decrease the Discounted Cash-flow Rate of Return (I) to its effective or constant purchasing power i^* according to:

$$Acf(1+i^*) = Acf \frac{(1+i)}{(1+d)}$$

$$i^* = \frac{i - d}{1 + d}$$

Negligence of inflation in cash-flow estimates normally results in overestimation of the net present value (profitability of the project).

The grievousness of inflation negligence in cash-flow and profitability calculations is still very contentious. Freidenfelds and Kennedy (42) have demonstrated that the positive relationship between the cost of capital (see below) and price inflation causes less sensitivity to the real values of costs assumed, than is generally accepted.

Because the value of money is a function of time (151), (126), it is conventional practice to weight cash occurrences over differing periods of time during project life by a "discount factor", so that all cash flows have a common time referent and equivalent money value. This device is called discounting.

The question of selecting an adequate discount factor by which profitability estimates are going to be made, is still being given much attention in the literature (18), (157). The rate of interest at which money is hired or bought is a guide as to what the discount rate for estimating an investment outcome should be. This rate of interest is usually estimated as the average cost of Capital-ACC (151), (18):

$$i = \frac{D_1(1+g)}{V_1} + g$$

where

i = cost of capital

g = constant rate of dividend growth

V_1 = market value of asset at year zero

D_1 = Dividend at year zero

Carsberg (18) demonstrates a growth model in which the growth rate (g) can be derived or estimated from the reinvestment rate ( ) using a reinvestment of (b):

$$i = \frac{D_o(1+rb)}{V_o} + rb$$

In the usage of both debt and equity capital, the weighted average cost of capital becomes the most suitable measure of the overall cost of capital-ACC:

$$ACC = \frac{V_e i + V_e i}{V_e + V_d}$$

where V_e, V_d = market values of equity and debt

i_e, i_d = cost of equity and debt.

More rigorous estimation procedures for the cost of capital (capitalization rate - rate of interest which induces investors to hold or buy securities in a venture) can be found in the references (151), (33), (51). Essentially the cost of capital is the sum of a selected base-rate of interest for investing in a risk free venture (Government securities) plus a risk premium, derivable from the "Capital Market Line" of the security in question. The risk premium is obtained by calculating the Beta Coefficient of the (CML), which is a cartesian plot of expected rates of return versus the standard deviations of the expected returns for the same security.

This can be estimated using the Capital Asset Pricing Model[15]

Four discounting and inflation scenarios have been assumed in these models. Discount rates of 0%, 5%, 10% and 15% with an average inflation rate of 10% have been applied.

CHAPTER 7 MINEX Verification

7.1 Generalities

Verification of the MINEX model has been limited to studying the influence of select model parameters on overall mineral exploitability. The exercise has been accomplished according to exploitability test procedure, (see definition in section 2.1) in two stages.

In the first stage, input and output responses due to MINEX parameter variations are verified to see how their interplay conforms to empirical predictions and observations, then to further select controlling MINEX parameters through an economic sensitivity analysis. Here, the economic significance of these parameters is analysed. Assumptions have been made especially in choosing values for certain base-case parameters which have been discussed in chapters 4,5 and 6, and inset in most of the figures. Most experiments were performed with open-pit shovel, truck, drill subsystem model see table 9/10

7.1.1 Open-pit subsystem model tests

Here, three input parameter effects (Drill-diameter effect, Bench-height effect and material density effect) were studied against four output parameters (tons of material per blasthole, number of blastholes per day, number of drills and compressors required and annual drill-bit requirements). Shovel and truck equipment capacities were studied against scheduled output requirements.

7.1.1-1 Bench-height effect:

The open-pit production schedule assumed a daily output schedule of 20,000 short tons of ore having a waste to ore ratio of about 3:1. The average ore and waste bank density in pounds per cubic yard (lb/cu.yd) was taken to be about 4420 lb/cu.yd, corresponding to the density of an earthly bauxitic orebody (see figs 15 21 23).

Variation of bench heigh from 20 feet to 50 feet showed a corresponding increase in the estimated tonnage of material per blasthole (Tb), as the drill-diameter (d inches) was increased (see fig 21). This is due to the fact that bench-height, as explained in section 4.1.1-3, is simply a third-dimension in the estimation or calculation of the total yardage of blasted rock. The consequence of increasing it (bench-height) is to cause a sympathetic decrease in the required number of blastholes (see fig 21 per day (Nb). This would in turn necessitate a relatively fewer number of drills per day per production schedule as shown in fig (15). The tonnage per foot drilled (tf) is a constant for each drill-diameter because, the bench height and tons per blasthole bear direct proportionality. 23

7.1.1-2 <u>Material density effect:</u>

The effect of material density in excavation has not been given the signification it deserves. Its effect has been shown to be quantitatively identical to the bench height effect in that it tends to increase the tons per blasthole as does the bench-height. (Compare figs 23 , and fig 24 ; fig 15 16 and fig 21 22). However, variability of material densities within a mineral prospect is often not so pronounced as to produce the observed density contrast being explained. Each mineral occurrence at the pre-project phase is assumed to have an average ore material density (Do).

7.1.1-3 Drill-diameter and density effects

Density and drill-diameter effects when combined tend to produce amplified trends in the output parameters, when both bench-height and drill-diameter are increased. 29 32 33

Large diameter drills have the effect of reducing the number of blastholes, relative number of drill operating units and their compressors by increasing the blasthole capacity (see fig 25 , fig 27 , and fig 29) in tons.

Generally, these effects are very desirable in excavation economics; because capital costs of equipment such as drills, compressors and their operating costs (drill accessories) are reduced substantially, thereby improving project profitability (see fig 30 ,

7.1.1-4 Shovel-Truck-Drill selection

Material density and capacity ratio (C·R) between trucks (tons) and shovels (cubic yards), are two primary determinants used in working out shovel and truck combinations.

Material densities ranging between 1.5 tons per cubic yard and 3 tons per cubic yard and for capacity ratios ranging between 3 and 6 were considered. As such each material type (density) has a corresponding shovel truck capacity combination from which to choose.

Fig (19) shows the shovel requirements per production schedule (tons of material per day). Each of the graph's branches corresponds to a particular shovel and truck capacity ratio (C.R); from which to derive the one from the other.

Basically, the loading time per truck is assumed to be 3 minutes, load-haul-spot-pump-return time taken to be 10 minutes, shift time taken as 8 hours with a 1.5 hours overall break time. Efficiency of operations is taken to be 50 minutes per 60 minute hour, with a mechanical availability of 80%, and fillability of 90%.

Final equipment selection depends upon actual pit design characteristics such as berm width, shovel loading height and other shovel-truck specifications which are determinable only at the design stages of project implementation.

The model (shovel, truck, drill) was verified with case examples from (155, 61) John Soma and John Watson. In the first example, John Soma's data comprised of rock material having a swell factor (1.2) and a density of 2.15 tons per cubic yard. Six-cubic-yard capacity power shovels having a fillability factor of 0.9, and a cycle time of 0.4 were used to load 37 cubic yard trucks, at a rate of 6 cycles per truck load. With a shift time of 8 hours, an overall break time of 1.5 hours the effective workshift amounted to 6.5 hours, for a single work shift per day. Overall job efficiency of 50 minutes per 60 minutes hour was assumed by John Sama. An additional 30 minutes was assumed for truck clean-up time. These data were fed into the model and produced, a truck fleet requirement of 4 trucks per power shovel. The result corresponds exactly with Soma's estimates (61).

Soma's drill-blast model assumes a 250 days annual operating schedule, during which 3,125,000 tons of material are produced. A blasthole diameter of 6" inches penetration rate of 0.85 ft/minute or 42.5 ft per 50 minute work hour are assumed. A bench height of

25 feet, hole depth of 30 feet, 15 feet burden and 12 feet spacing are assumed. 2.3 minutes of drill manipulation and an overall break time of 1.25 hours are considered. Estimates conformed with Soma's calculations producing 35 blastholes per one day-shift, 9 holes per shift per drill and approximately 4 drills per shift per day (61).

The second example (155) with a stripping ratio of 19:1 or 20 units of waste and ore per unit of ore (see section 4) was also verified. Altogether, 380,000 tonnes of muck is excavated per 20,320 tonnes of ore in 5 days of the work week. This averages to about 80,000 tonnes of rock per day. Blastholes were about 30 feet deep and were perforated by 6 inches drills having a penetration rate of 4.3 feet per minute. Drill requirements were derived using fig (15) (5 drills). This corresponds to the actual number of operating drills on the field (155).

7.1.2 Profitability tests

Second set of tests were carried out using selected MINEX parameters (see fig 36) whose impact on profitability were observed using a sensitivity analysis.

The procedure adopted was to vary base-parameter values about a chosen value range (see fig 36).

The MINEX model used was a bauxite exploitation model with an output capacity of 5000 tons of ore (Trihydrite ore, 45% Al_2O_3 grade) per one shift per work day, with a waste-to-ore ratio of 2. Operations in the mine section were scheduled for 340 days, those in the mill for 300 days in the work year. Mining was envisaged to be effected by shovels and trucks in an open-pit excavation. Mine operations were assumed to take place over 38 years.

An exploration venture to delineate 250 million tons of in situ ore having a stratiform disposition was assumed to be undertaken by a 250 meter by 100 meter drilling program.

Beneficiation of bauxite to alumina was assumed to be effected by the Bayer process, at a recovery ratio of 85% and alumina grade of 99.5% Al_2O_3. Alumina was assumed to be delivered to a buyer in a Western European market at a market price of $300 (U.S.) dollars per ton.

Concentrate transportation to such an overseas market would cost about $5 dollars per ton. A local rail line (50 kilometers) and a local road 50 kilometers were to be constructed to enable the ore and alumina to be moved from Fongu-Tungong to Dschang and from Dschang to Douala and also to enable the transportation of supplies from Douala to Dschang and Fongu-Tungong.

A total operating workforce of 450 men was assumed, 20% of whom are expatriate with two salary grades of $24,000 (70%) dollars and $30,000 (30%) dollars. A mining town was considered necessary for the workers; 80% of whom are locals. Restoration of mined-out areas for agricultural purposes was envisaged.

Production and sales of produce were envisaged to take place in 1987 four years after pre-production development of the mine and mill site. Taxation dues, 5% royalties (Ad valorem) and 30% standard taxation rate plus a 27.5% withholding tax, were taken into account; plus a 50% government share in the investment.

The sensitivity analysis was carried out by flexing select base-parameters being studied by +50%, +30% and +20% from their base values.

The changes that occurred were monitored on the project profitability, measured as Net Present Value (NPV) and percentage NPV.

7.1.2-1 Discussion of results;

Concentrate produce-price produced the greatest sensitivity causing a profitability change of +587% NPV as a result of a base-case variation of +50% (see Fig 36). Mill recovery changed by -261% and +292% of NPV for a variation of -50% and +50% change in the base-value. fig 36 shows the sensitivity or significance of geo-technical parameters over all other MINEX model parameters except of course produce price. (Note high sensitivities due to proximate value of IRR 10.88% and discount rate 10%, whereby denominator is always very small.)

From the first revenue year (1987) the bauxite exploitation model indicated a payback period of 5.6 years during which time all expenditures would have been recouped and exploitation break-even attained. This value was for the undiscounted cash-flow stream. The internal rate of return (IRR) was approximately 11%, while the Net Present Value at 10% discount rate was estimated at $17.5 milliion U.S. dollars. Total Government take was $849.9 million U.S. dollars; 53% from taxes and 47% from equity participation.

Estimation of a produce price for this mineral model was carried out by extrapolating break-even values from base-case parameter values (see appendix III). A producer price for alumina mined and beneficiated at Fongu-Tungong near Dschang and marketed to Western Europe via the Douala port, at a market price of $300 U.S. dollars per ton including a $5 per ton freight charge for the concentrate, was estimated at $270 U.S. dollars per ton. This leaves the mine a 9.8% producer price margin with which to produce its profits. It was also estimated that mine output could be brought down by about 13.5% of its base-case operating schedule (4325TPD) before NPV equals zero. Ore-grade could also be mined below 10.1% of its present grade, (40% Al_2O_3 cut-off grade). Mill recovery at 85% can be lowered down to 76% without operations resulting in negative NPV. Both capital and operating total costs had an absorptive capacity of about 10.1% of their base-values; both cost headings can be increased by 10.1% of their base-values before the break-even point is attained.

Operating costs per ton of bauxite exploitation (bauxite mining and transformation to alumina via the Bayer process) were estimated at $21.2 per ton; $52 per ton of alumina concentrate. These costs are distributed as follows;

Mining operating costs/annum	= $2.17 million (6%)
Milling and concentrate transport cost/annum	= $24.02 million (66.6%)
Salaries and wages/annum	= $6.04 million (17%)
Overseas transportation/annum	= $3.8 million (10.4%)

Capital costs per ton were in the order of $389 of concentrate with a break even cost of $428 per ton of concentrate; this gives the project an absorptive capital cost capacity of about $39 per ton of concentrate. Capital costs are distributed as follows:

Exploration capital cost	= $2.3 million
Plants, development cost	= $3.8 million
Replacement capital cost	=
Mill capital cost	= $163.5 million
Rail and road capital cost	= $28.0 million
Feasibility, Eng. Construction	= $23.5 million
Mining equipment capital cost	= $10.2 million
Agro-restoration capex	= $0.1 million
Start-up and work capex	= $0.4 million

The overall viability or profitability of a commercial mineral exploitation undertaking, depends upon the "technology" applied and deployed in its exploitation. If as shown in the modelling sections (chapters 4 and 5, the "technical inputs-shovels, trucks, drills, vehicles, skills (expatriate and local engineers and craftsmen), were convertible into factor costs such as has been accomplished in chapter 6; and these measured against potential revenues from the sale of produce, it becomes evident that the profitability test of a mineral investment venture is actually an interplay between economic, geo-technical and job parameters.

High output schedules produce positive changes and conversely depress the profitability at lower schedules (see fig 36). Increase in the output schedules indeed produces positive profitability indicators (IRR, NPV, PAYBACK) if factor costs are sympathetically reduced, even though the volume of absolute cash returns would fall (NPV) (see figs 36 ,). Increase in operating and capital expenses and outlays for a given output schedule results in low profitability outcomes and a cut-back in expenditures either through manpower or energy savings or optimizing operating costs would result in improved profitability (see fig 36).

As a model, commercial exploitability of a mineral occurrence should best be regarded as a cost and price relationship (16), whereby the natural tendency of any exploiter, be it government, multinational or both, would be to seek to find a cost-price mix that would satisfy her profitability objectives. This can be achieved by altering costs if, as often is the case, price is unalterable; by reducing overall project costs whether they be fiscal, operating or capital. And as Beasley and Pfleider have noted in their paper (43) "profitability, sensitivity analysis of a mining venture",

"this feature is contingent upon the investor's philosophy" of appraisal.

As such, increase in cost intensity of a given mineral exploitation model predictably produces lower profitability results, as compared to lowering the cost intensity, for the same price and market scenario.

Fig 15 Bench-Height Influence on Drill-Compressor Requirements

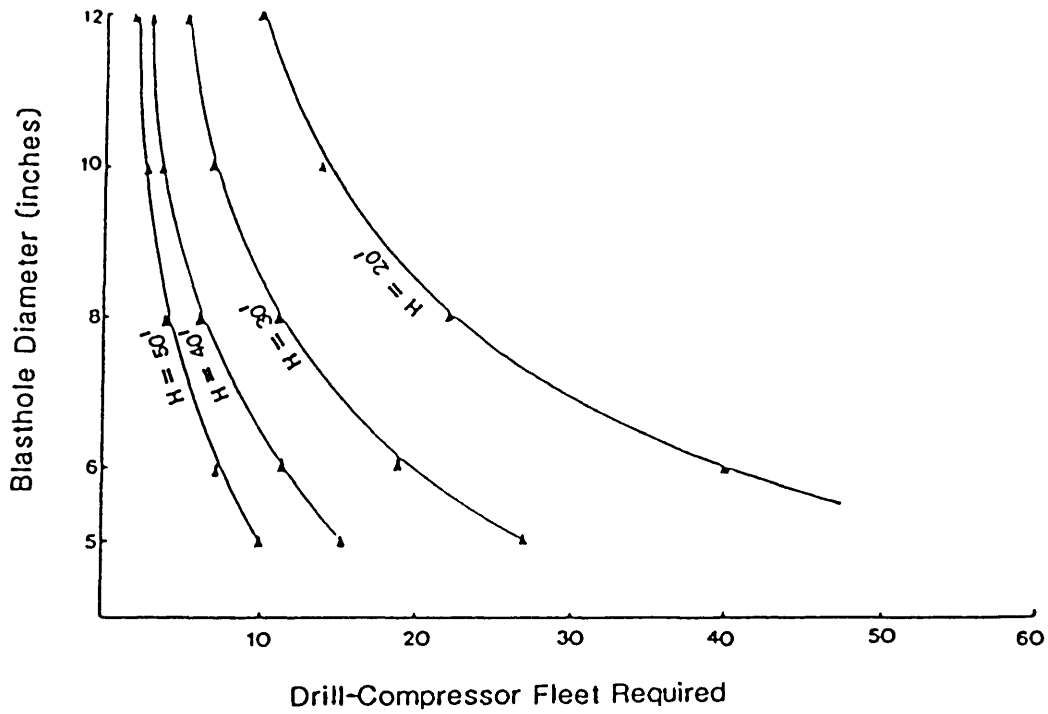


Fig 16 Density Effect on Drill-Compressor Requirements

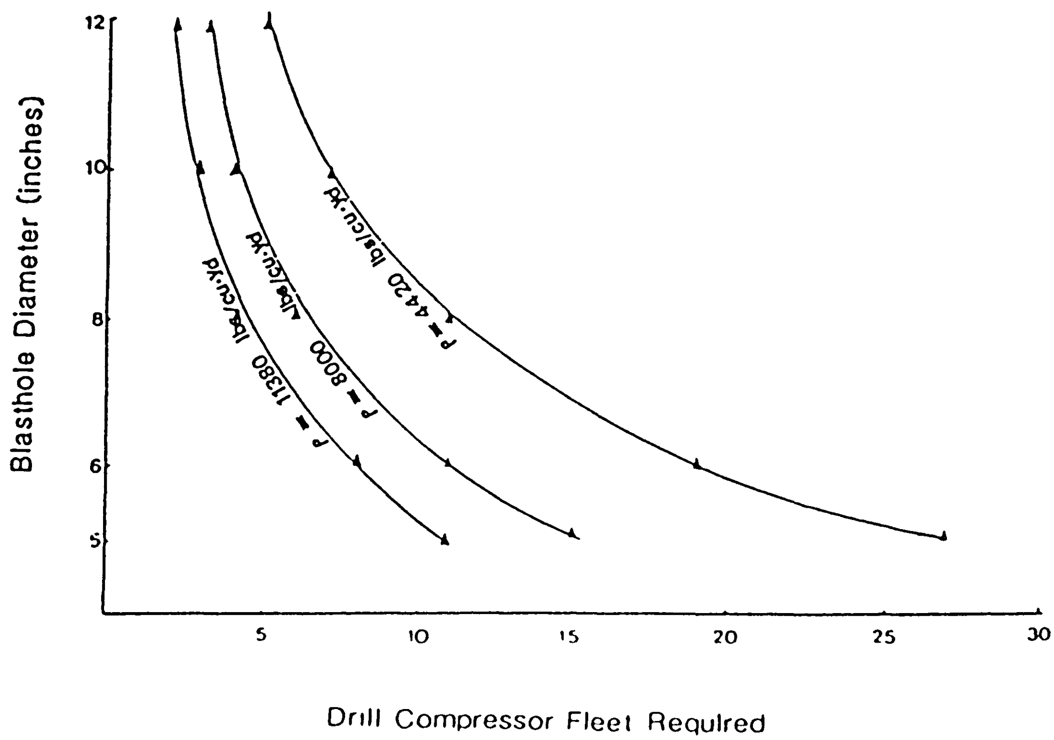


Fig 17 Density-Effect on Tons of Waste-Ore Per Foot Drilled

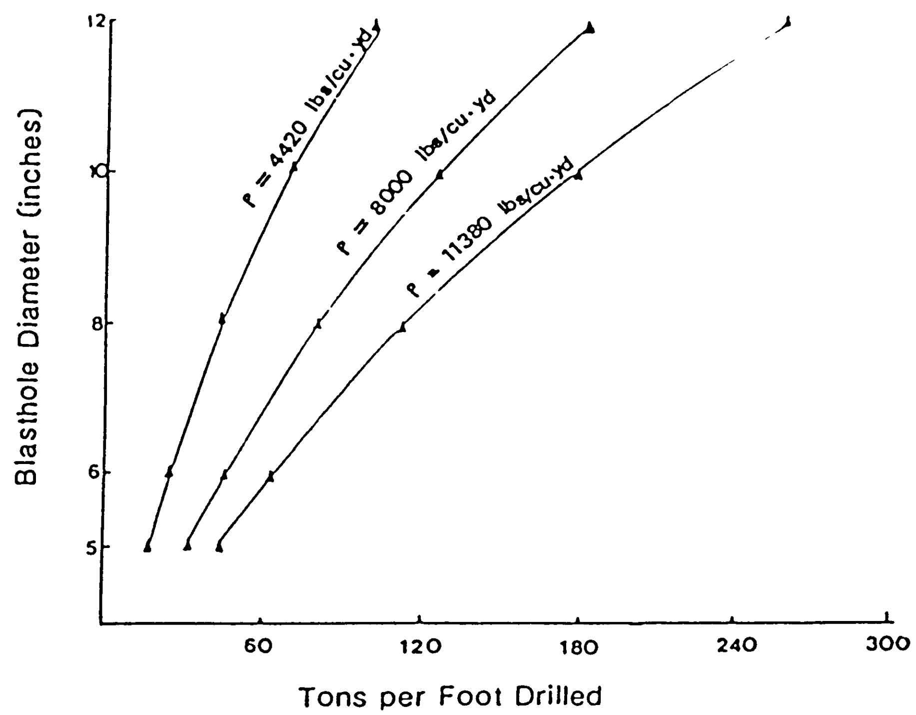


Fig 18 Density Effect on Annual Drill-Bit Requirements

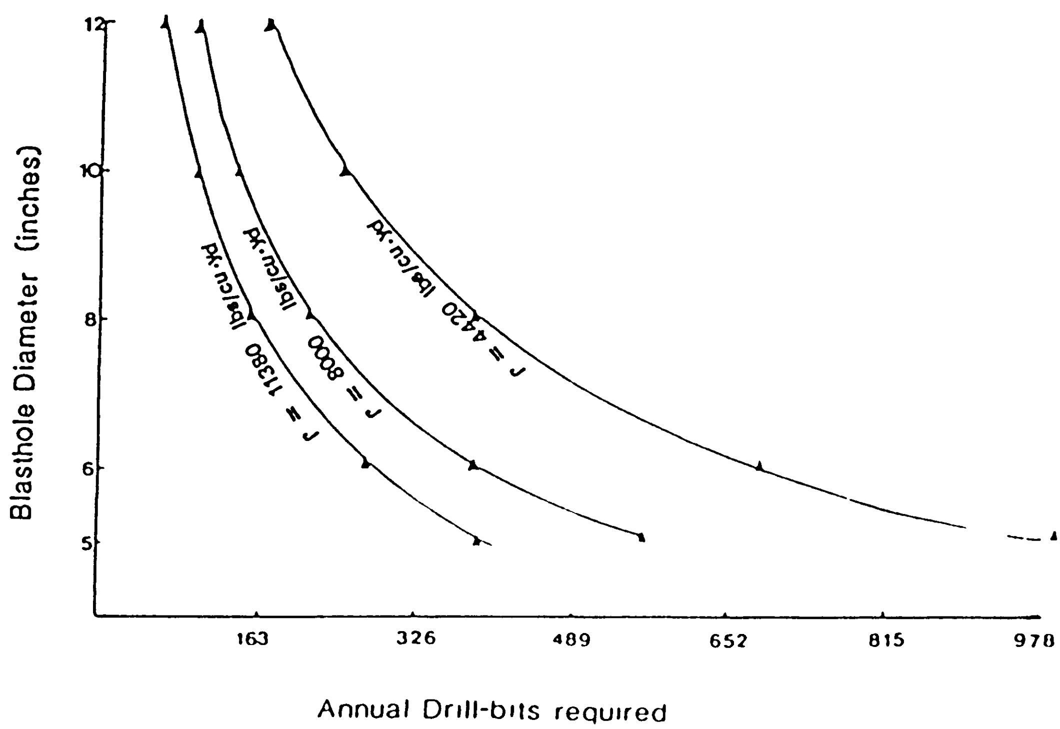


Fig 19 Shovel-Fleet Requirements per Mine Output Schedule

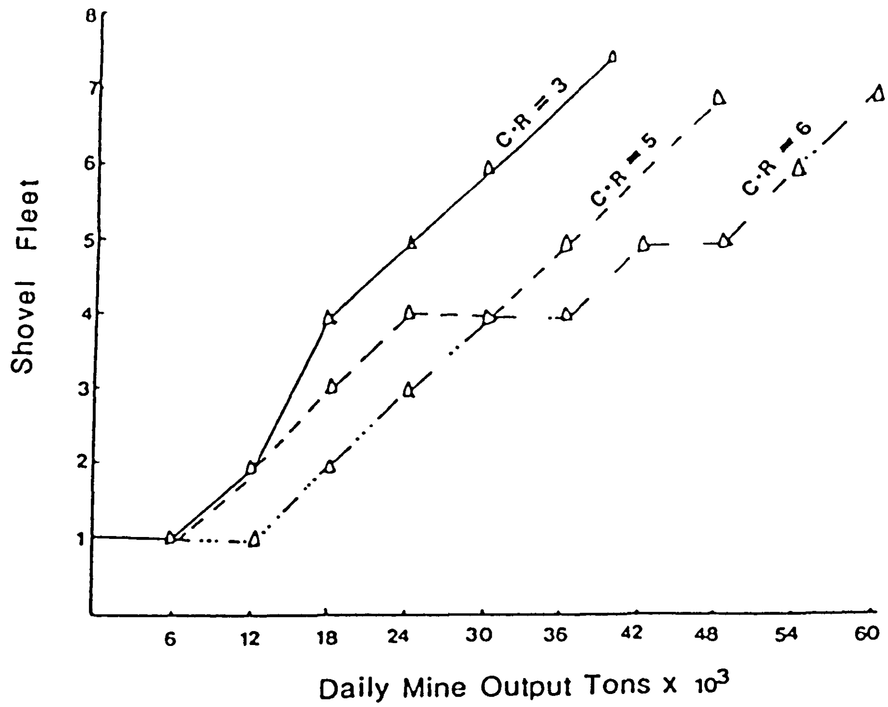


Fig 20 Shovel-Truck Fleet Requirements per Mine Output Schedule

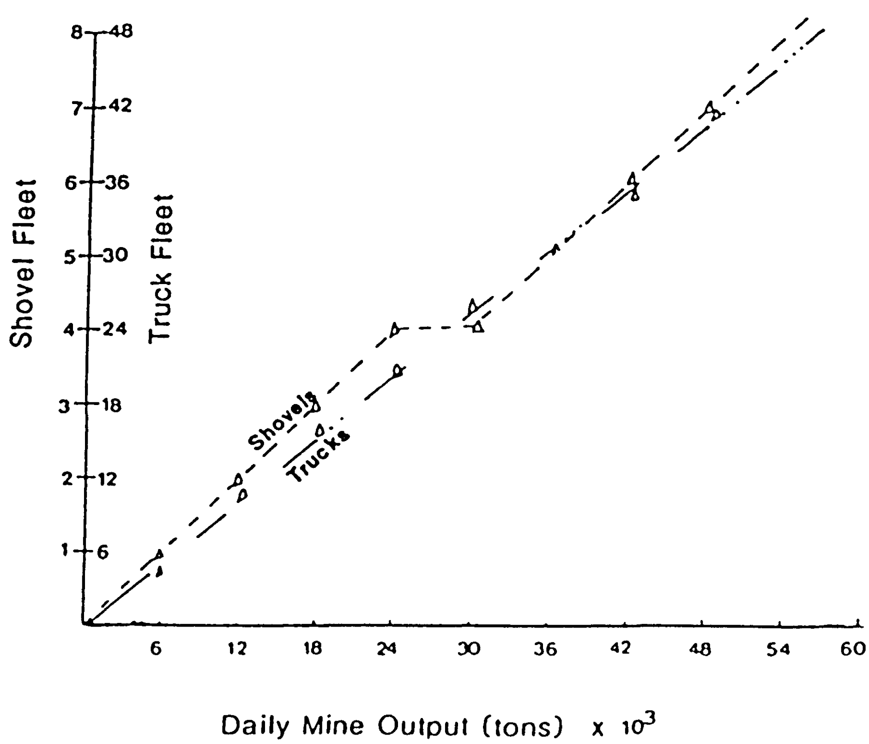


Fig 21 Bench-Height Influence on Daily Blasthole Requirements

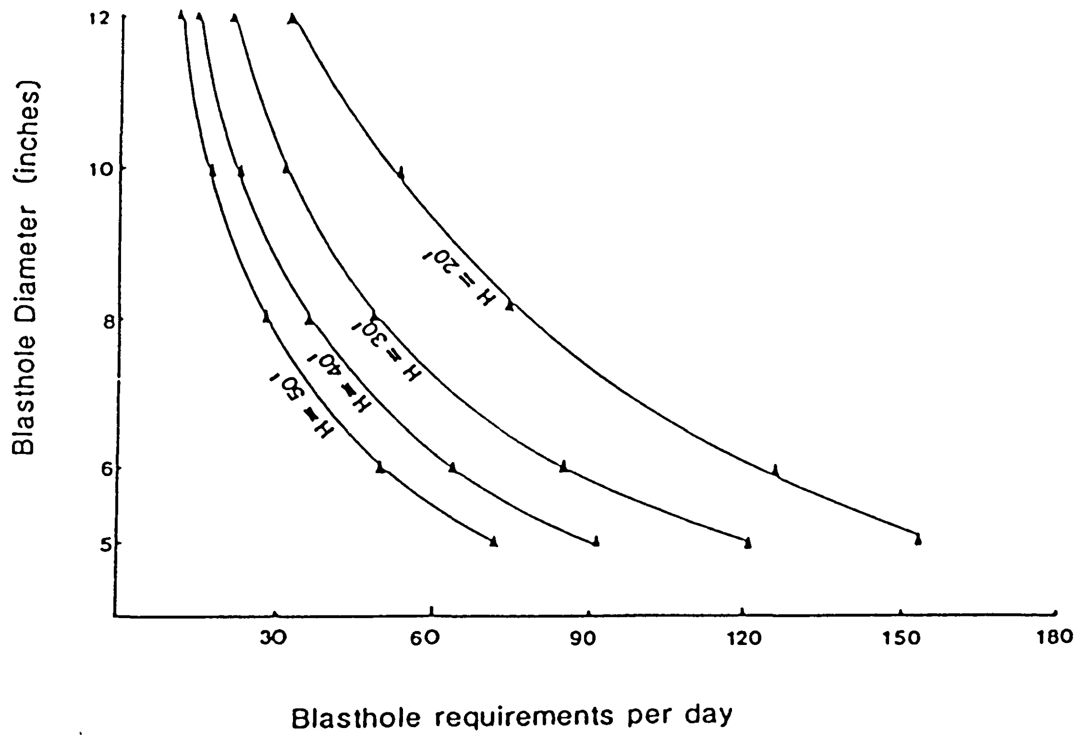


Fig 22 Density Effect on Daily Blasthole Requirements

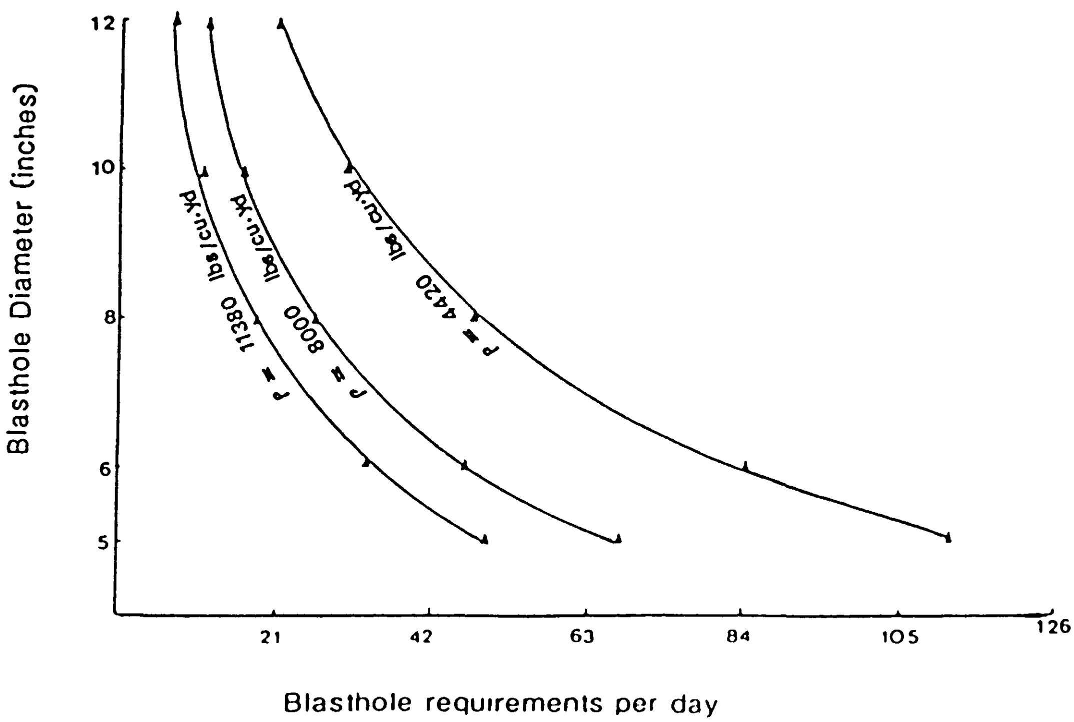


Fig 23 Bench Height Influence on Blasthole Capacity (tons)

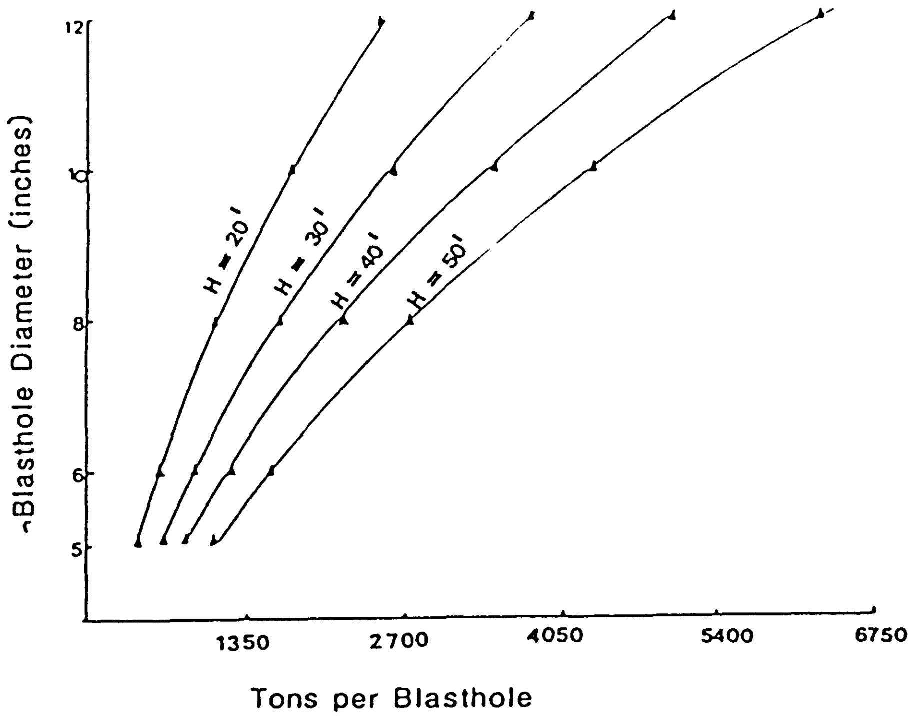


Fig 24 Density effect on Blasthole Capacity (tons)

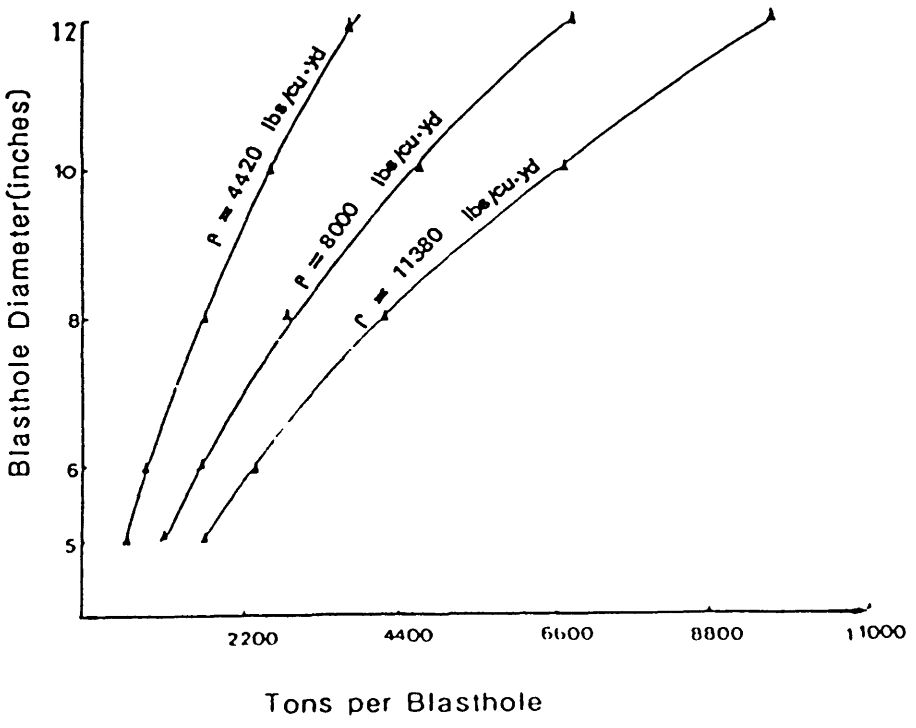


Fig 25 Influence of of Drill-Diameteron Blasthole Capacity(tons)

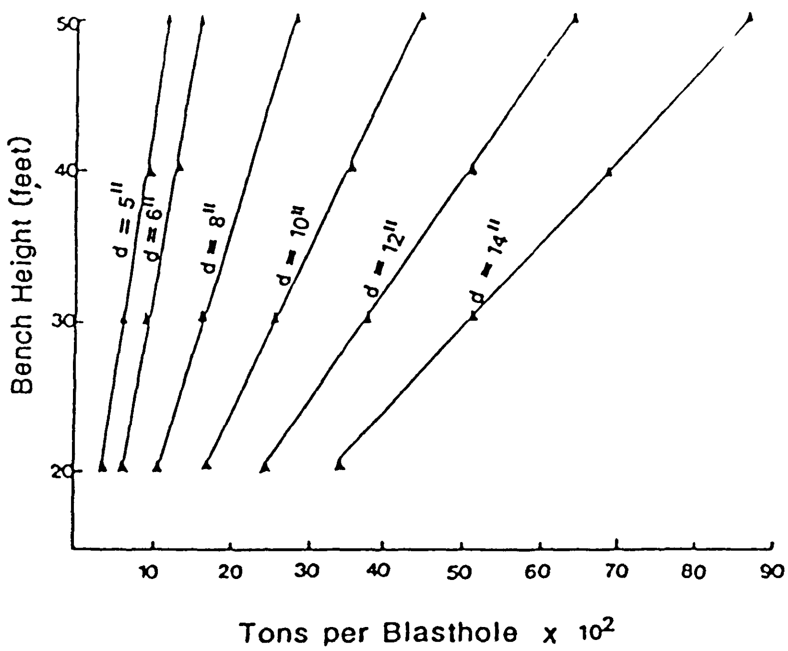


Fig 26 Density Effect on Blasthole Capacity (tons)

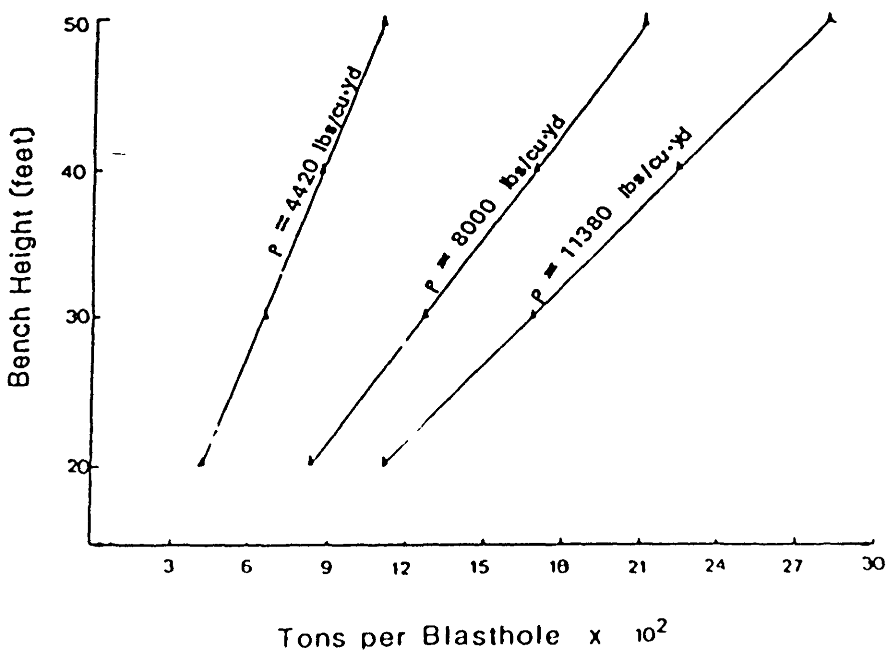


Fig 27 Influence of Blasthole Diameter on Blasthole requirements

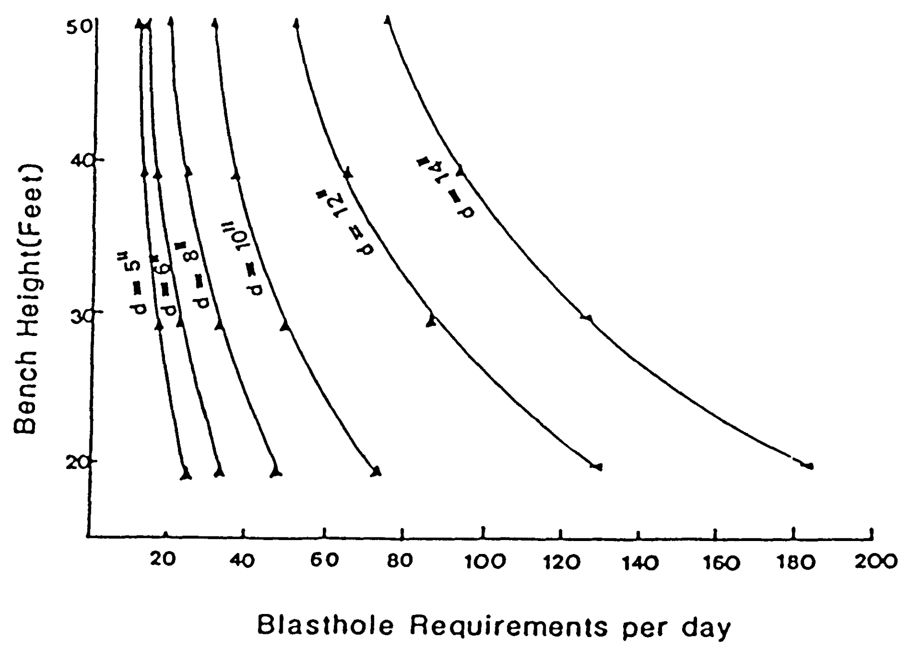


Fig 28 Density Effect on Blasthole Requirements

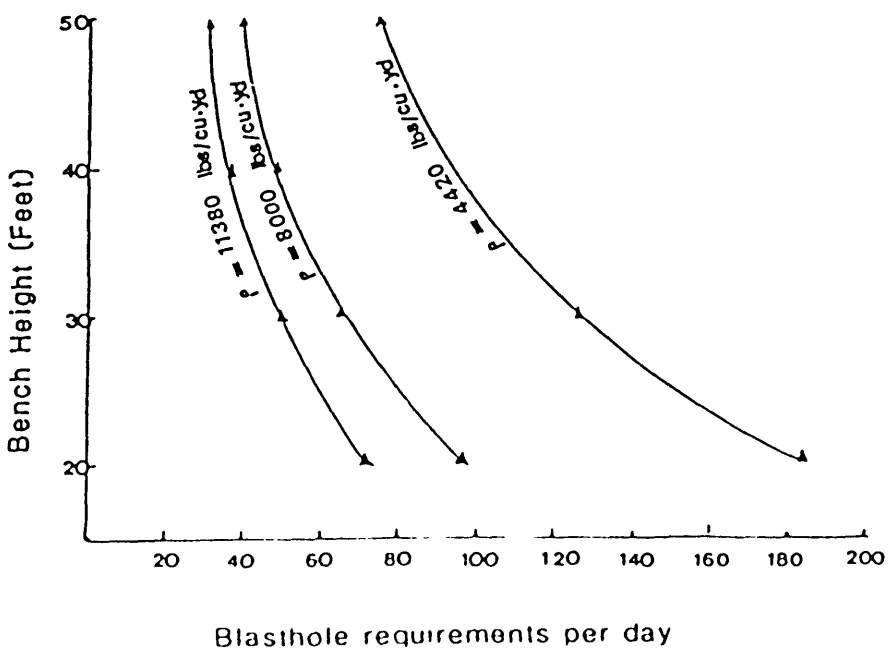


Fig 29 Blasthole Diameter and Density Effects on Blasthole Capacity (tons)

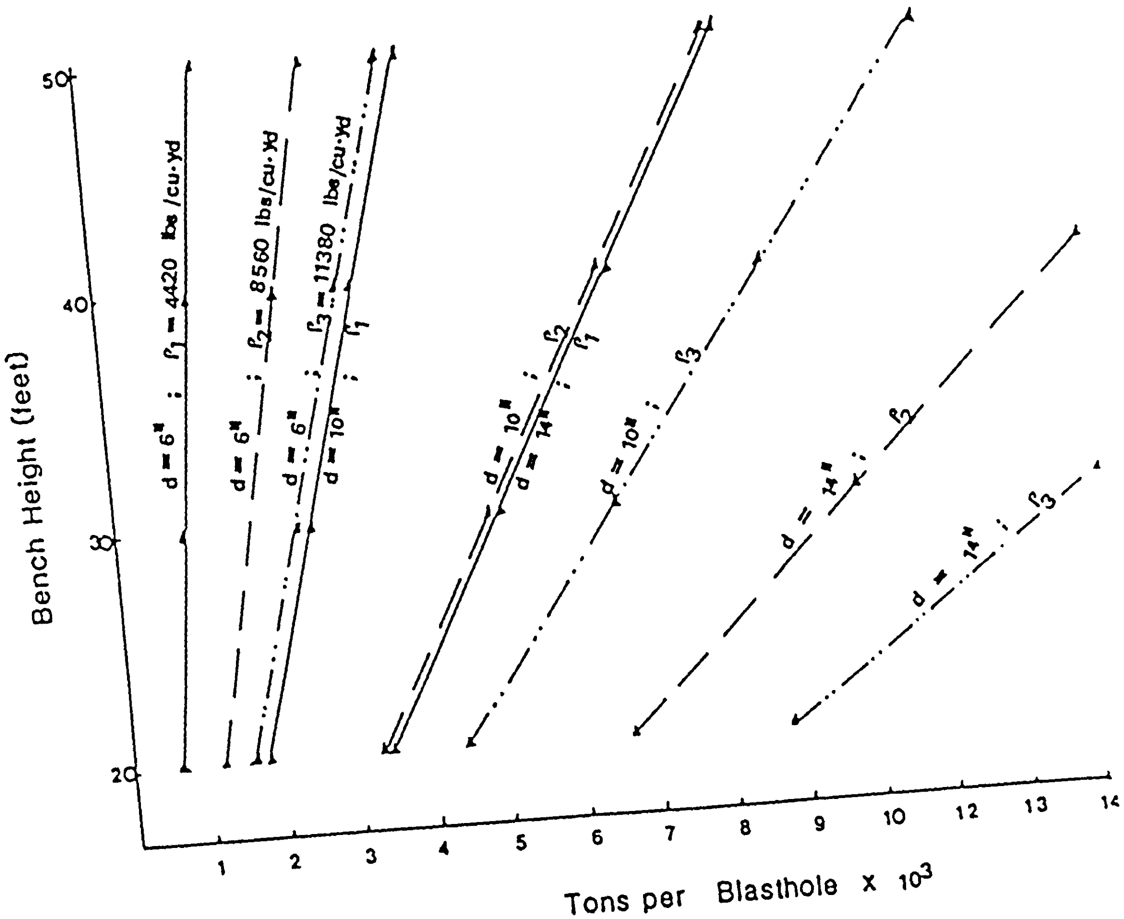


Fig 30 Influence of Blasthole Diameter on Drill-Compressor Requirements

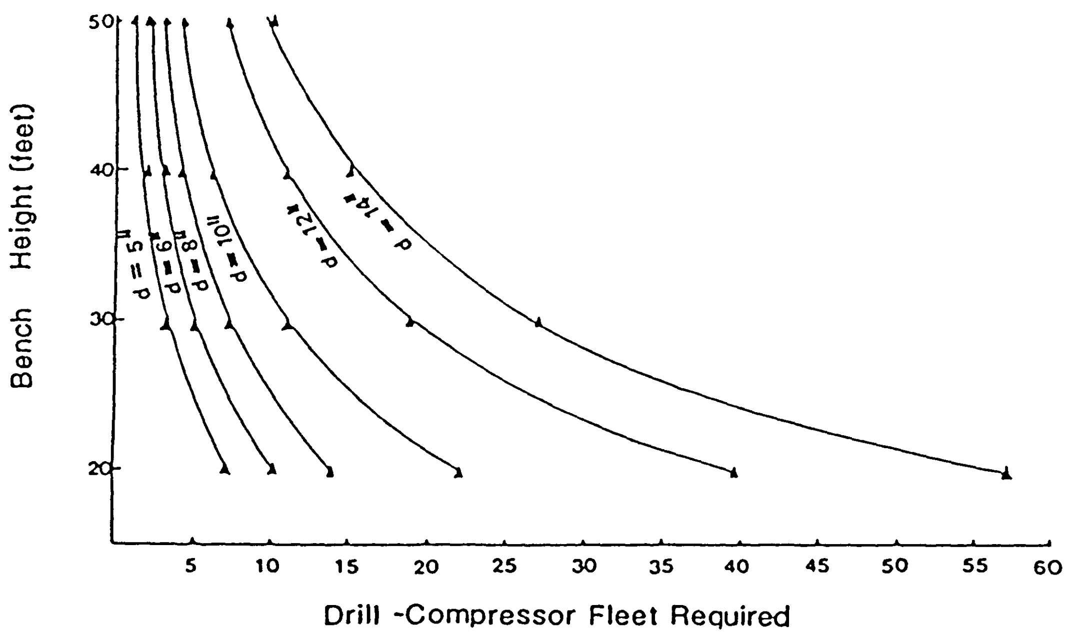


Fig 31 Density Effect on Drill-Compressor Requirements

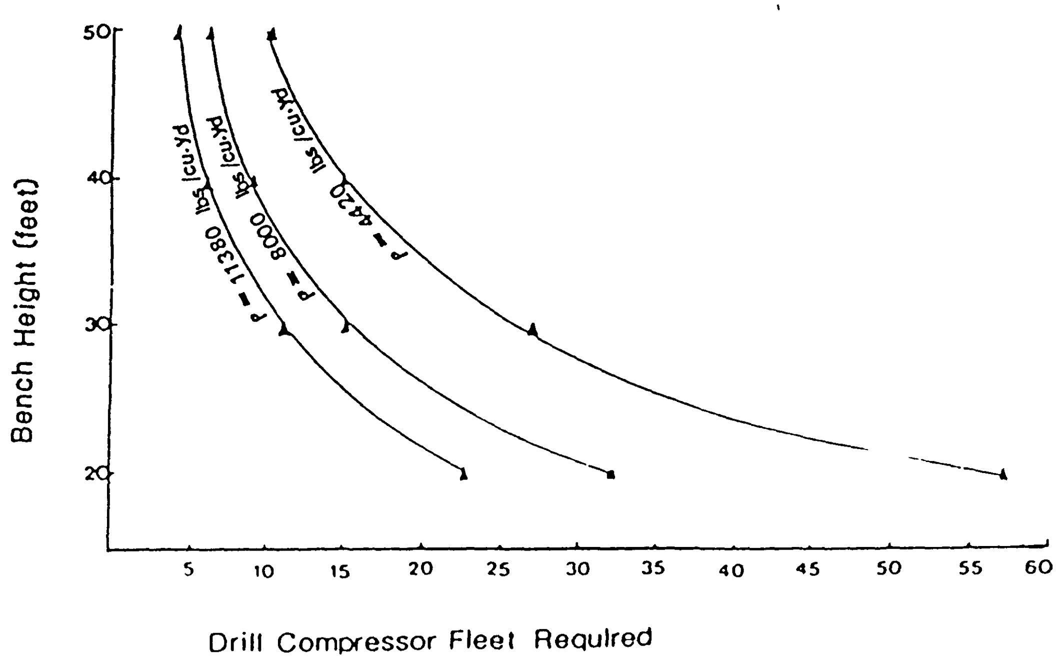


Fig 32 Combined Blasthole Diameter and Density Effects on Blasthole Requirements

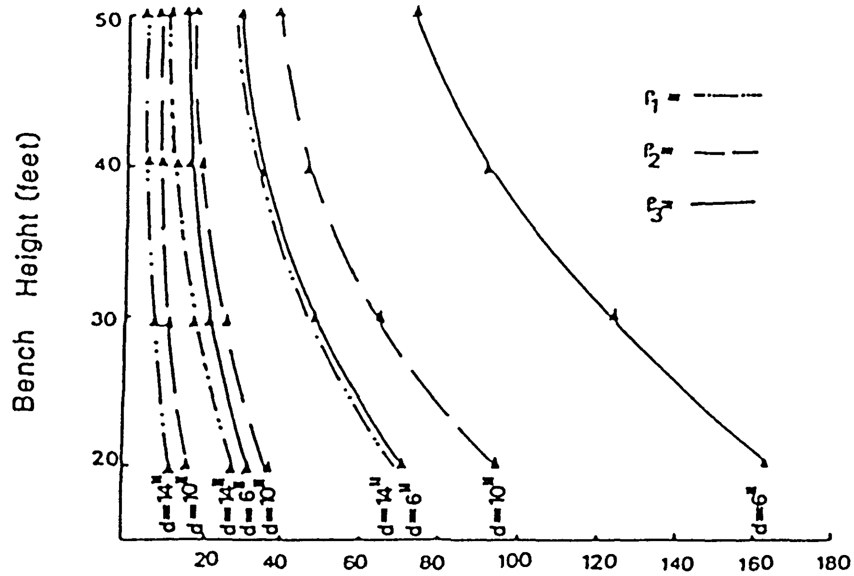


Fig 33 Combined Blasthole and Density Effects on Drill-Compressor Requirements

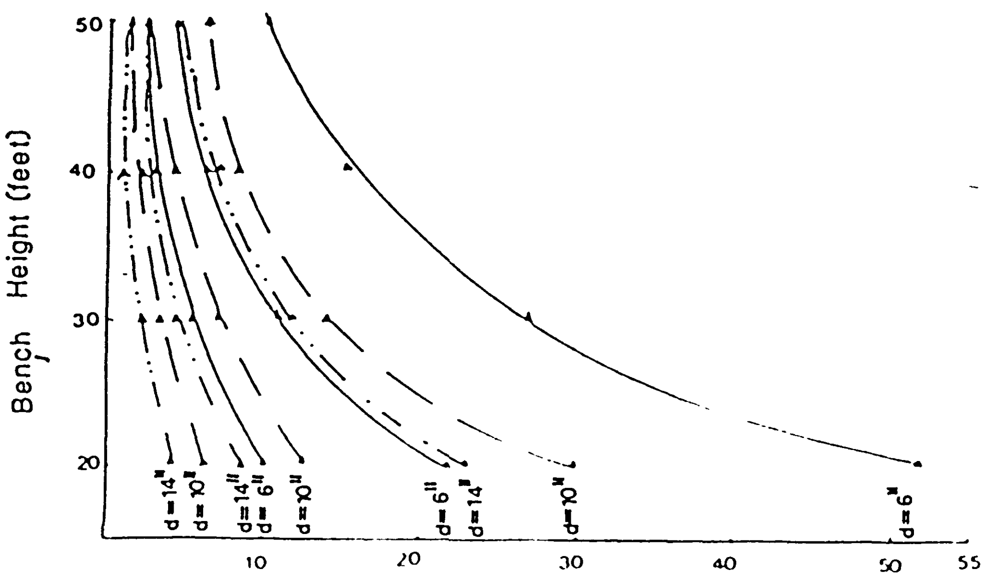


Fig 34 MINEX Verification Cash-flow Profile

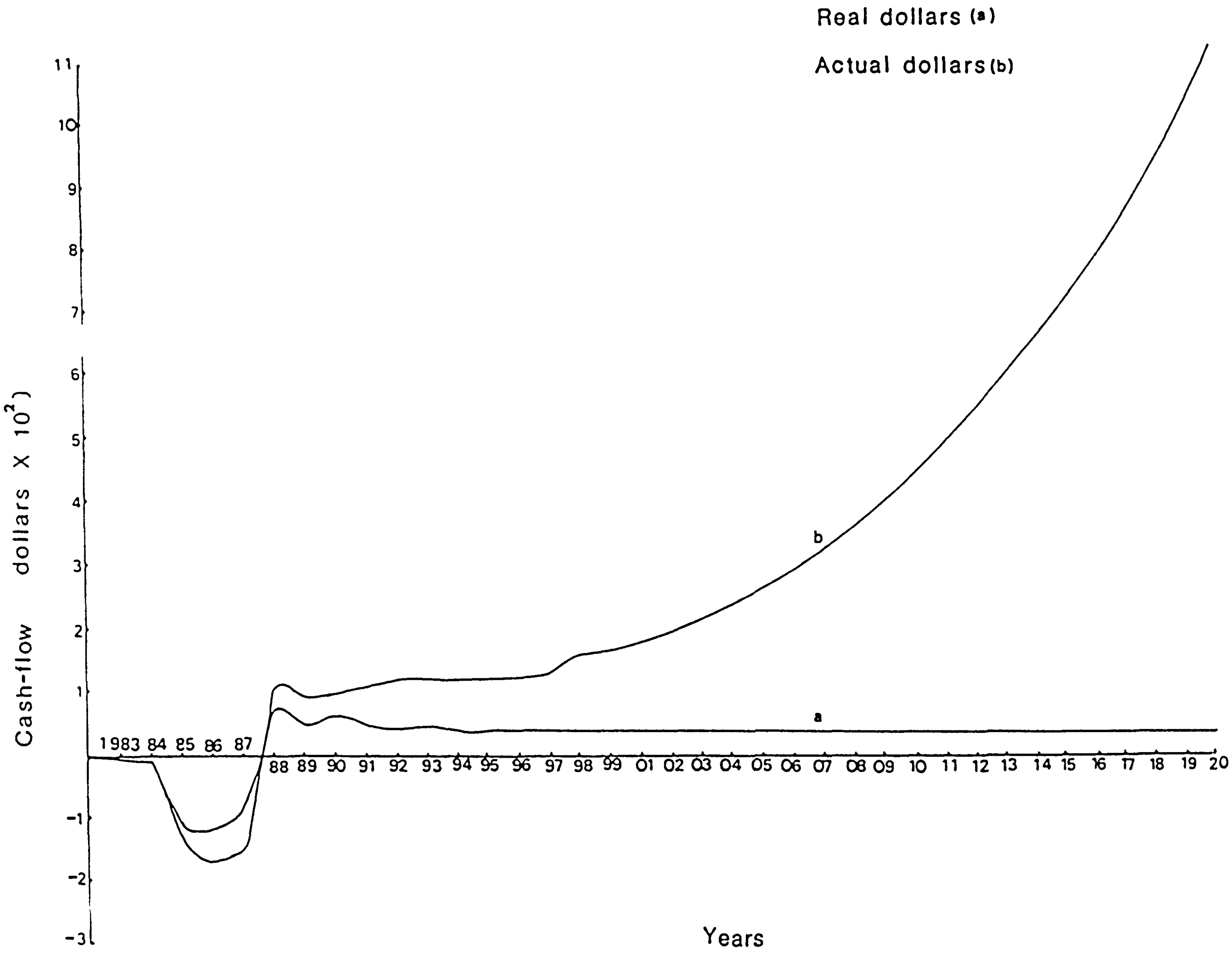


Fig 35 MINEX Verification Cumulative Cash-flow Profile

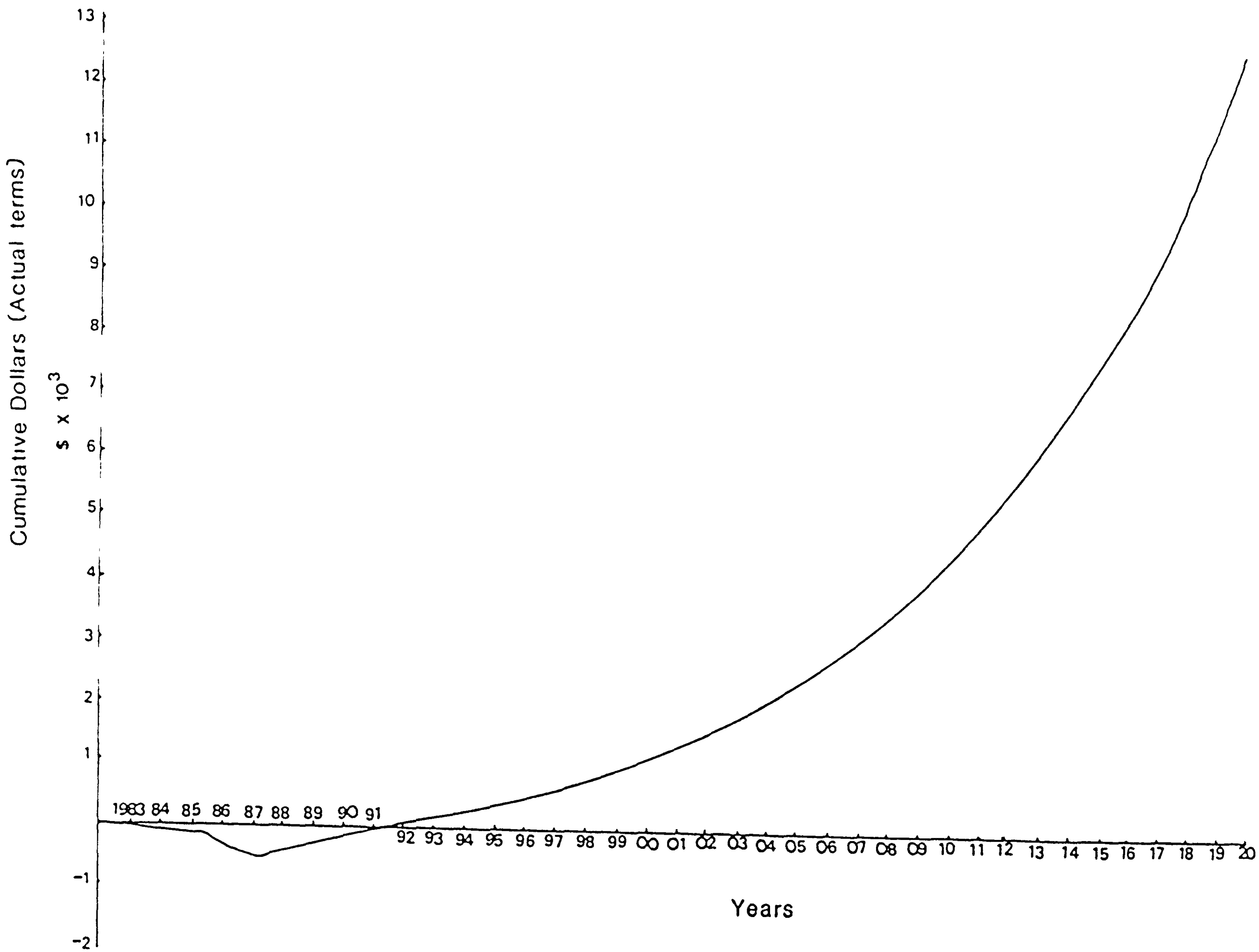
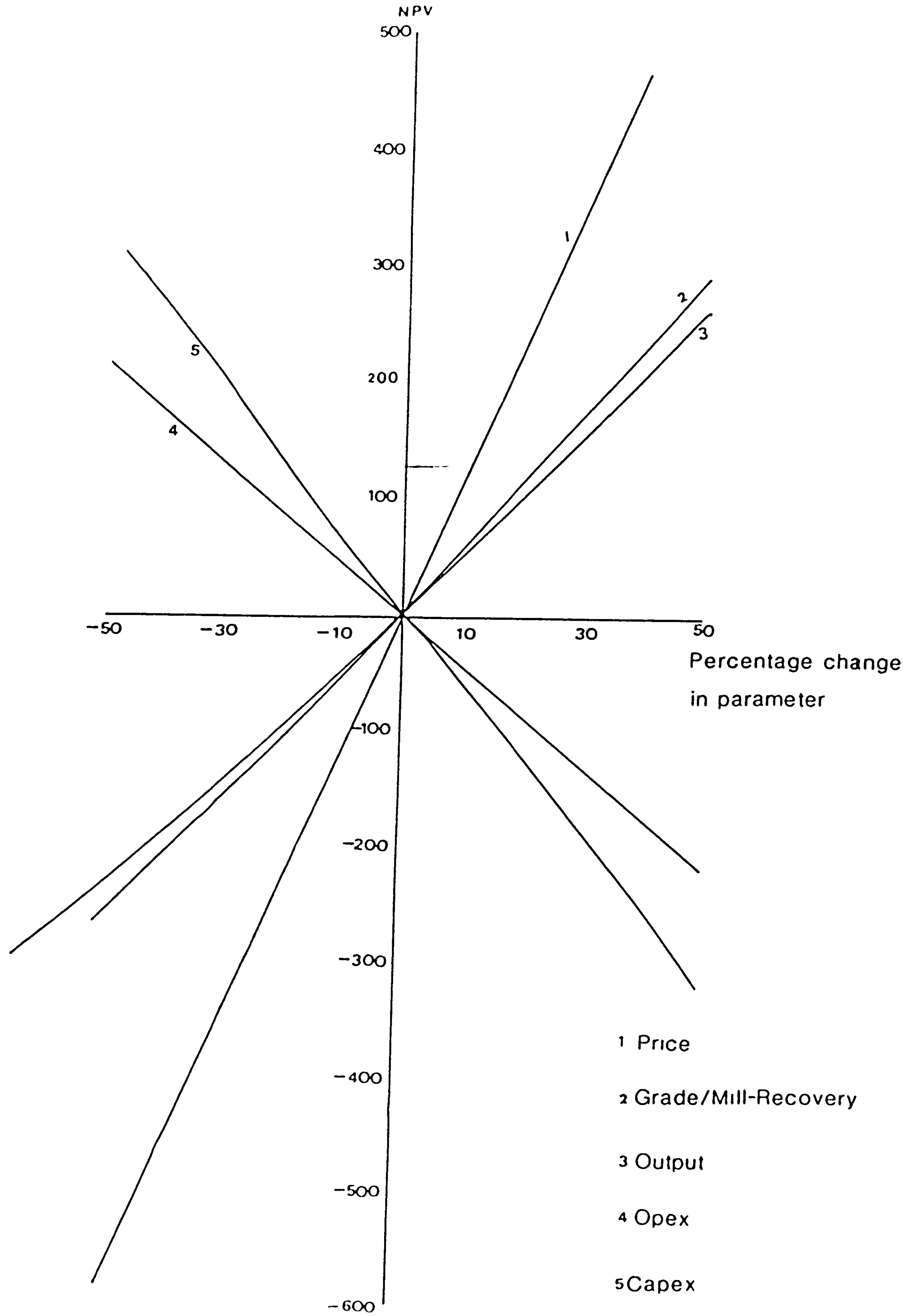


Fig 36 Minex Base-case Sensitivity of NPV to changes in Select geoeconomic parameters



CHAPTER 8 Mineral Exploitation Case-Study Models

8.1 Geo-economic models

Exploitation models of the four mineral base-cases have been presented and briefly discussed in this chapter. They comprise a summary of geo-technical, equipment and job parameters including capital and operating cost summaries with their corresponding cash-flow profiles. Other details of these models (miscellaneous parameters) have been presented and discussed in part two of the thesis (Chapter 4,5,6 and 7).

8.1.1 Geo-technical equipment and job parameters of base-cases.

8.1.1-1 Labour

Tables 11 , 12 , 15 , 16 , 19 , 20 , 23 , and 24 , are summaries of geo-technical equipment and job parameters of the mineral base-cases being studied. The bauxite-alumina case has a labour force of 450 men, 90 expatriates and 360 local employees. In the Kribi iron-ore model, it is envisaged that 300 men will be needed, 240 of them locals and 60 expatriates. Two hundred men (200 men) are estimated for the rutile case-study 160 locals and 40 expatriates. The Mayo-Darle project with a high profitability is expected to absorb about 550 men, 110 expatriates and 440 locals. Salary scales used in these models are quite conservative. (See section 6.3.1).

8.1.1-2 Geotechnical and equipment parameters

The basic shovel, truck and drill requirements derived from the MINEX model have been presented under each case-study heading. Explosive truck, drills and compressors have been omitted in the rutile heavy sands case-study, because excavation will not require any rock fragmentation.

Blasthole requirements which would completely satisfy the stipulated production schedule are presented in the tables. 8 blastholes per day are required in the Fongu-Tungong model, 10 in Kribi and 5 in Mayo-Darle. These blastholes will be put down by 1,2 and 1 tandem drill-compressor sets respectively (8 inch diameter drills).

Concentrate output per day is envisaged to be about 159 tons (96% TiO_2), 2307 tons (99.5% Al_2O_3), 6090 tons (67% Fe) and 102 tons (>50% Sn) for 300 days of the work year in the mills. Other important model parameter asumptions have been included within tables or in section 5.2 and 7.2. or in tables 9 and 10

8.1.2 Case-study costs

Capital and operating cost summaries have been presented in tables 13 14, 17 18. Cost totals are estimated to include a 10% contingency, within a 10% annual inflation scenario over project life.

Capital costs for the rutile case (see table 17), are approximately $79.1 million dollars with equipment replacement over project life adding up to $36.59 million dollars. Total Annual Operating costs are about $3.02 million dollars with salaries and wages making up to 89% of overall operating costs.

Capital costs per annual ton of rutile concentrate (96% TiO_2) is about $1658. Operating costs per annual ton of rutile heavy sands is about $21 or $63 per annual ton of rutile concentrate.

In the Mayo-Darle model, (Table 25) capital costs total about \$92.89 million dollars. Operating cost are estimated at \$13.17 million dollars 56% of which is due to salaries and wages, 25.7% milling and inland transportation while the rest is due to mining and overseas transportation. Capital cost per annual ton (>50% Sn) of tin concentrate is estimated at \$3035; while operating cost per annual ton cassiterite ore 1% Sn, is about \$8.

Mill utilities for acquisition and erection of an alumina plant (21) make up about 60% of capital costs (see table 21). Annual operating costs are quite high \$32.3 million dollars. They are due mainly to long transportation distances inland plus the high bulk nature of the concentrate (2307 tons per day); hence milling and transportation are estimated to contribute up to 74.5% of total operating costs.

Operating cost per annual ton of ore (45% Al_2O_3) is high (\$19); or \$46.5 per annual ton of alumina (99.5% Al_2O_3); capital cost per annual ton (\$389.0) of 99.5% Al_2O_3 is very low compared to the cassiterite case (\$3,035) and the rutile case (\$1658).

Kribi capital costs are about \$120 million dollars, 43% of which is due to mill and utilities only. Annual Operating costs are \$21.49 million dollars, 37.4% of which are estimated to be overseas transportation costs. The capital cost per tonne of iron concentrate 67% Fe is estimated to be \$66 while operating cost per annual ton are about \$6.

All capital costs seem to be within the open-pit mining and beneficiation levels (\$1500 - \$3000) and come out to be predictably lower than underground mining cost averages (≥\$3000) (49) (102)

It should be mentioned however that individual case-study model peculiarities such as mine-mill location, concentrate characteristics and market location do influence the cost structure for individual case-studies. The labour force and waste-to-ore ratios do influence the cost structure too.

O'Hara (106) has also cautioned against simple generalized comparisons of project costs because the uniqueness of each mineral case occasions "variability of costs and quantities" in each of their individual cost centres.

Table 11

Kribi Iron-ore geo-technical parameters

Ore-grade	= 40% Fe
Daily ore output	= 10,000 TPD
Waste to ore ratio	= 2
Ore-waste swell factor	= 1.2
Mean ore-waste density	= 8560.0 lb/cu yd
Mill recovery ratio	= 85%
Mill concentrate grade	= 67% Fe
Concentrate output	= 6,090 TPD
Drillability factor	= 1.2
Abrasiveness factor	= 1.2
Blasthole depth	= 30 feet
Required number of blastholes per day	= 10

Table 12

Kribi Iron-ore Equipment and job parameters

Shovel struck capacity	= 6 cu. yd.
Truck struck capacity	= 35 cu. yd.
Drill diameter	= 8 inches
Shift schedule	= 8 hours/day
Mine work schedule	= 340 days/yr
Mill work schedule	= 300 days/yr
Work efficiency	= 50 min/hour
Overall break time	= 1.5 hours/day
Total work force	= 300 men
Shovel fleet requred	= 2
Truck fleet required	= 6
Tandem drill - compressor set	= 2

Table 13

Kribi Iron-ore Capital Cost Summary

(Startup 1983, U.S. $\$ \times 10^6$)

Exploration Capital	= $ 3.02
Plants and Site development	= $ 1.92
Road and Rail capital costs	= $ 15.63
Feasibility and Engineering costs	= $ 9.45
Mill utilities cost	= $ 51.44
Mine Equipment costs	= $ 13.81
Working capital	= $ 1.59
Equipment replacement	= $ 6.5
Agro-restoration	= $ 0.019

Total capital plus 10% contingency	$ 120.7
	=======

Table 14

Kribi Iron-ore annual operating cost summary

(1983 U.S. $\$ \times 10^6$, at full production).

Mining operating costs	= $ 4.6
Milling and transportation	= $ 4.82
Salaries and wages	= $ 4.03
Overseas transportation	= $ 8.04

Total annual capital plus 10% contingency	$ 21.49
	=====

Table 15

Rutile Geo-technical Parameters

Ore-grade	= 3% heavy sands
Daily ore output	= 5000 TPD
Waste to ore ratio	= 2
Ore-waste swell factor	= 1.2
Mean ore-waste density	= 6000 lbs/cu yd
Mill recovery ratio	= 85%
Mill concentrate grade	= 95% TiO_2
Concentrate output	= 159 TPD

Table 16

Equipment and Job Parameters

Shovel struck capacity	= 6 cu yd
Truck struck capacity	= 35 cu yd
Drill diameter	= 8 inches
Shift schedule	= 8 hr/day
Mine work schedule	= 340 days/yr
Mill work schedule	= 300 days/yr
Work efficiency	= 50 min/hour
Mechanical efficiency	= 85%
Overall breaktime	= 1.5 hours
Total work-force	= 200 men
Shovel fleet required	= 2
Truck fleet required	= 6

Table 17

Rutile Capital Cost Summary
(Startup 1983, U.S. \$ X 10^6)

Exploration Capital	= \$ 2.67
Plants and site development	= \$ 1.29
Road Construction capital	= \$15.63
Feasibility and Engineering	= \$ 1.9
Mill utilities cost	= \$18.97
Mine equipment	= \$ 1.67
Working capital	= \$ 0.37
Equipment replacement	= \$36.59

Total capital costs plus 10% contingency	\$ 79.1
	=====

Table 18

Rutile Annual Operating Cost Summary
(1983 U.S. \$ X 10^3 at full production).

Milling and inland transportation	= \$ 24.37
Mining Operating Costs	= \$ 6.12
Salaries and Wages	= \$2,684
Overseas transportation	= \$ 306.8

Total annual operating costs plus 10% contingency	\$ 3,021.3
	=======

Table 19

Fongu-tungong Geo-technical parameters

Ore-grade	= 45% Al_2O_3
Daily ore output	= 5,000 TPD
Waste to ore ratio	= 2
Ore-waste swell factor	= 1.2
Mean ore-waste density	= 4420 lb/cu yd
Mill recovery ratio	= 85%
Mill concentrate grade	= 99.5% Al_2O_3
Concentrate output	= 2,307 TPD
Drillability factor	= 2
Abrasiveness factor	= 1.5
Blasthole depth	= 40 feet
Required number of blastholes per day	= 8

Table 20

Fongu-tungong Equipment and job parameters

Shovel struck capacity	= 6 cu. yd.
Trudkc struck capacity	= 35 cu. yd.
Drill diameter	= 8 inches
Shift schedule	= 8 hours/day
Mine work schedule	= 340 days/yr
Mill work schedule	= 300 days/yr
Work efficiency	= 50 min/hour
Mechanical efficiency	= 85%
Overall break time	= 1.5 hours/day
Total work force	= 450 men
Shovel fleet requred	= 2
Truck fleet required	= 7
Tandem drill - compressor set	= 1

Table 21

Fongu-Tungong Capital Cost Summary
(Startup 1983 U.S. $ X 10^6)

Exploration Capital	= $ 2.31
Plants and Site development	= $ 3.87
Road and Rail capital costs	= $ 28.06
Feasibility and Engineering costs	= $ 23.47
Mill utilities cost	= $163.5
Mine Equipment costs	= $ 10.17
Working capital	= $ 0.358
Equipment replacement	= $ 37.4
Agro-restoration	= $ 0.053

Total capital plus 10% contingency	$ 269.2
	=======

Table 22

Fongu-Tungong annual operating cost summary
(1983 U.S. $ X 10^6 at full production).

Mining operating costs	= $ 2.17
Milling and transportation	= $ 24.02
Salaries and wages	= $ 6.04

Total annual capital plus 10% contingency	$ 32.23
	======

Table 23

Mayo-Darle Geo-technical Parameters

Ore-grade	= 1% Sn
Daily ore output	= 5000
Waste to ore ratio	= 2
Ore-waste swell factor	= 1.2
Mean ore-waste density	= 9000 lbs/cu yd
Mill recovery ratio	= 85%
Mill concentrate grade	= > 50% Sn
Concentrate output	= 102 TPD
Drillability factor	= 1.2
Abrassiveness factor	= 1.2
Blasthole depth	= 30 feet
Required number of blastholes per day	= 5

Table 24

Mayo-Darle Equipment and Job Parameters

Shovel struck capacity	= 6 cu yd
Truck struck capacity	= 35 cu yd
Drill diameter	= 8 inches
Shift schedule	= 8 hr/day
Mine work schedule	= 340 days/yr
Work efficiency	= 50 min/hour
Mechanical efficiency	= 85%
Overall breaktime	= 1.5 hours
Total work-force	= 550 men
Shovel fleet required	= 1
Truck fleet required	= 4
Tandem drill-compressor set	= 1

Table 25

Mayo-Darle Capital Cost Summary
(Startup 1983 U.S. $ X 10^6)

Exploration Capital cost	= $ 1.04
Plants and site development cost	= $ 5.59
Mine equipment capital cost	= $ 9.39
Road and Rail capital cost	= $24.87
Feasibility and Engineering cost	= $ 3.18
Mill utilities cost	= $19.23
Working capital	= $ 0.939
Replacement capital	= $28.62
Agro-restoration capital	= $ 0.019

Total capital costs plus 10% contingency	$ 92.89
	======

Table 26

Mayo-Darle Annual Operating Cost Summary
(1983 U.S. $ X 10^6 at full production).

Mining operating cost	= $ 2.18
Milling and inland transportation	= $ 3.39
Overseas transporation	= $ 0.20
Salaries and wages	= $ 7.40

Total annual operating costs plus 10% contingency	$ 13.17
	=====

Fig 37 Mayo-Darle Base-Case Cash-flow Profiles

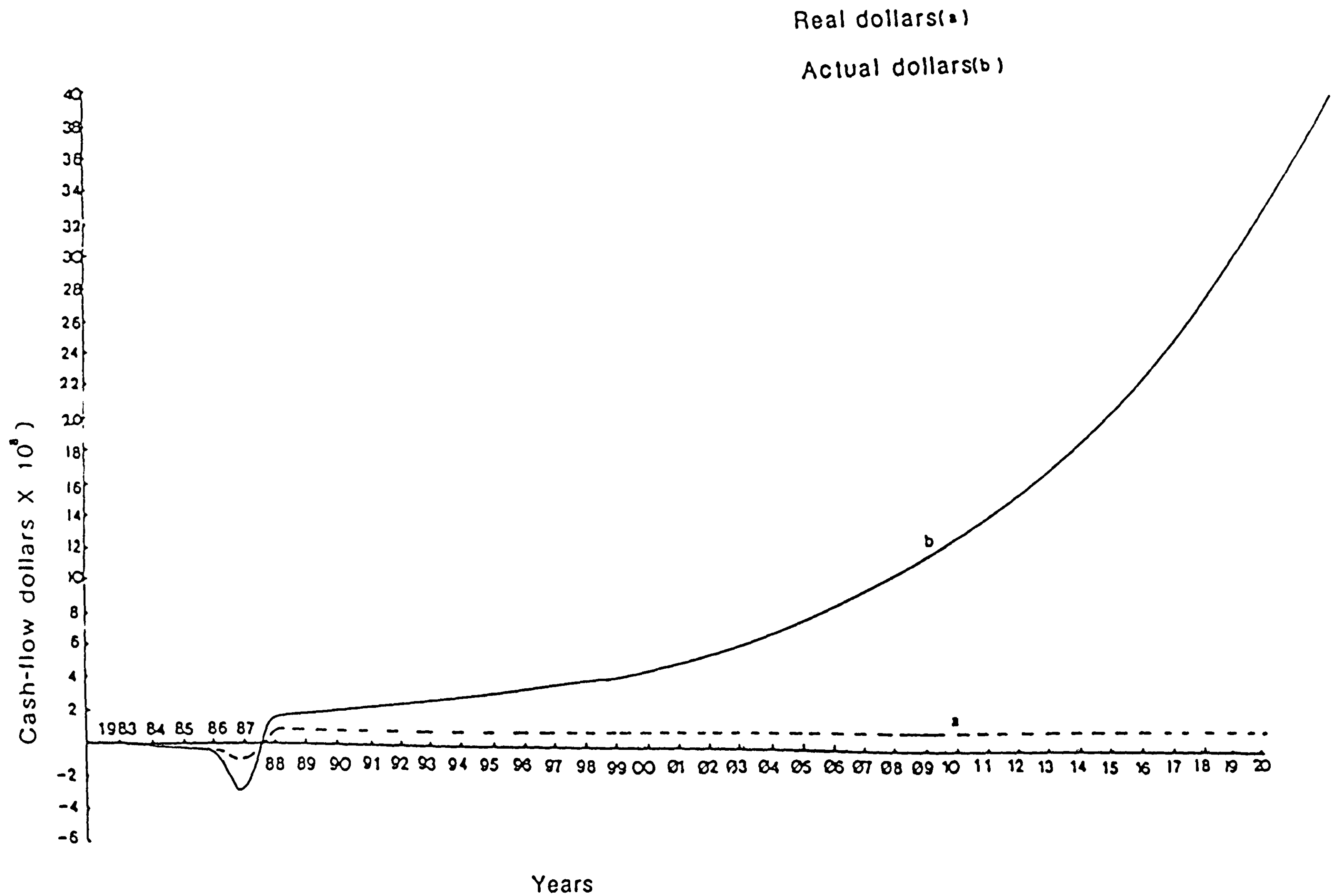
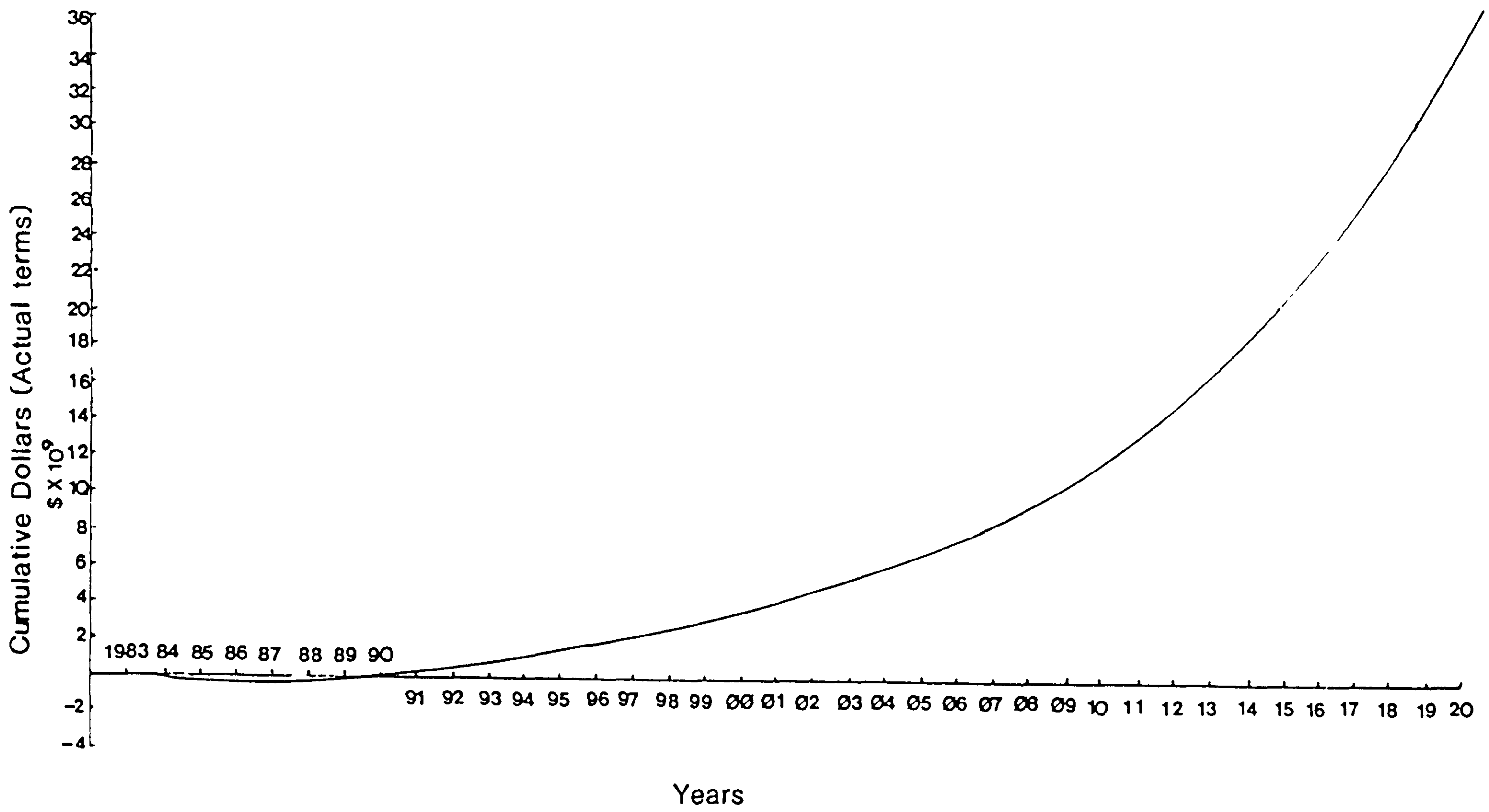


Fig 38 Mayo- Darle Base-Case Cumulative Cash-flow Profile



161

Fig 39 Fongu-Tungong Base-Case

Cash-flow Profiles

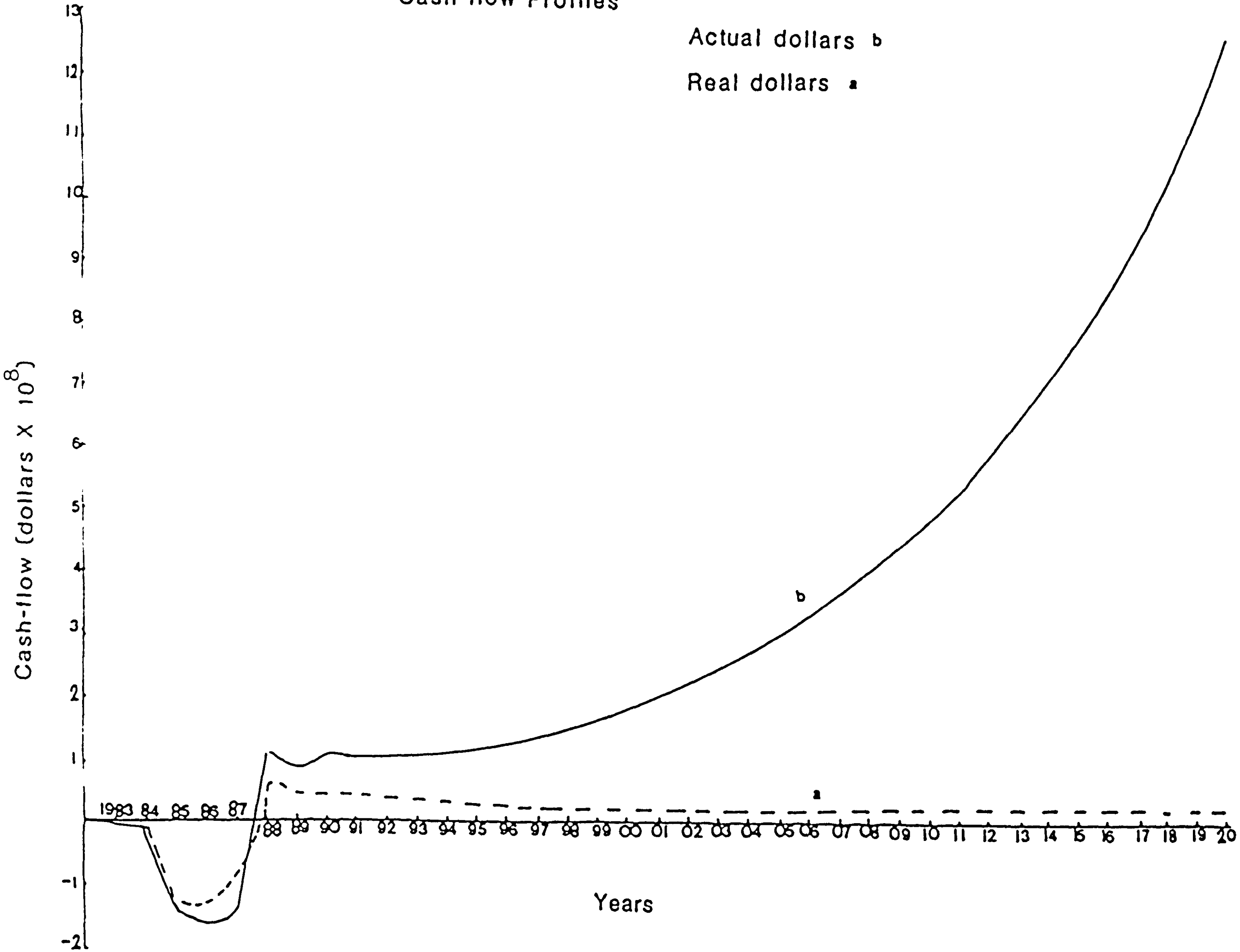


Fig 40 Fongu-Tungong, Base-Case

Cumulative Cash-flow Profile

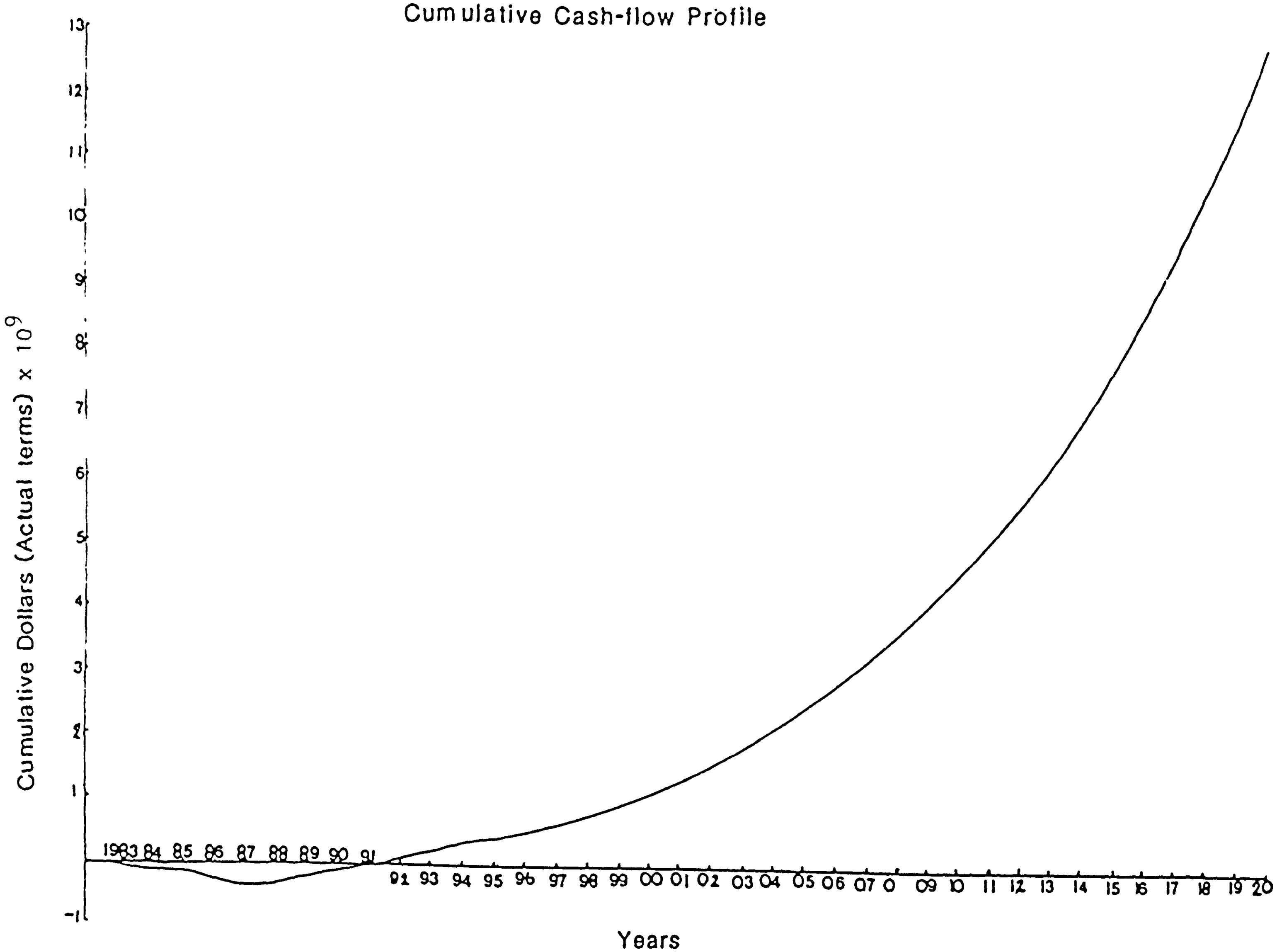


Fig 41 Douala Heavy Sands Base-Case

Cash-flow Profiles

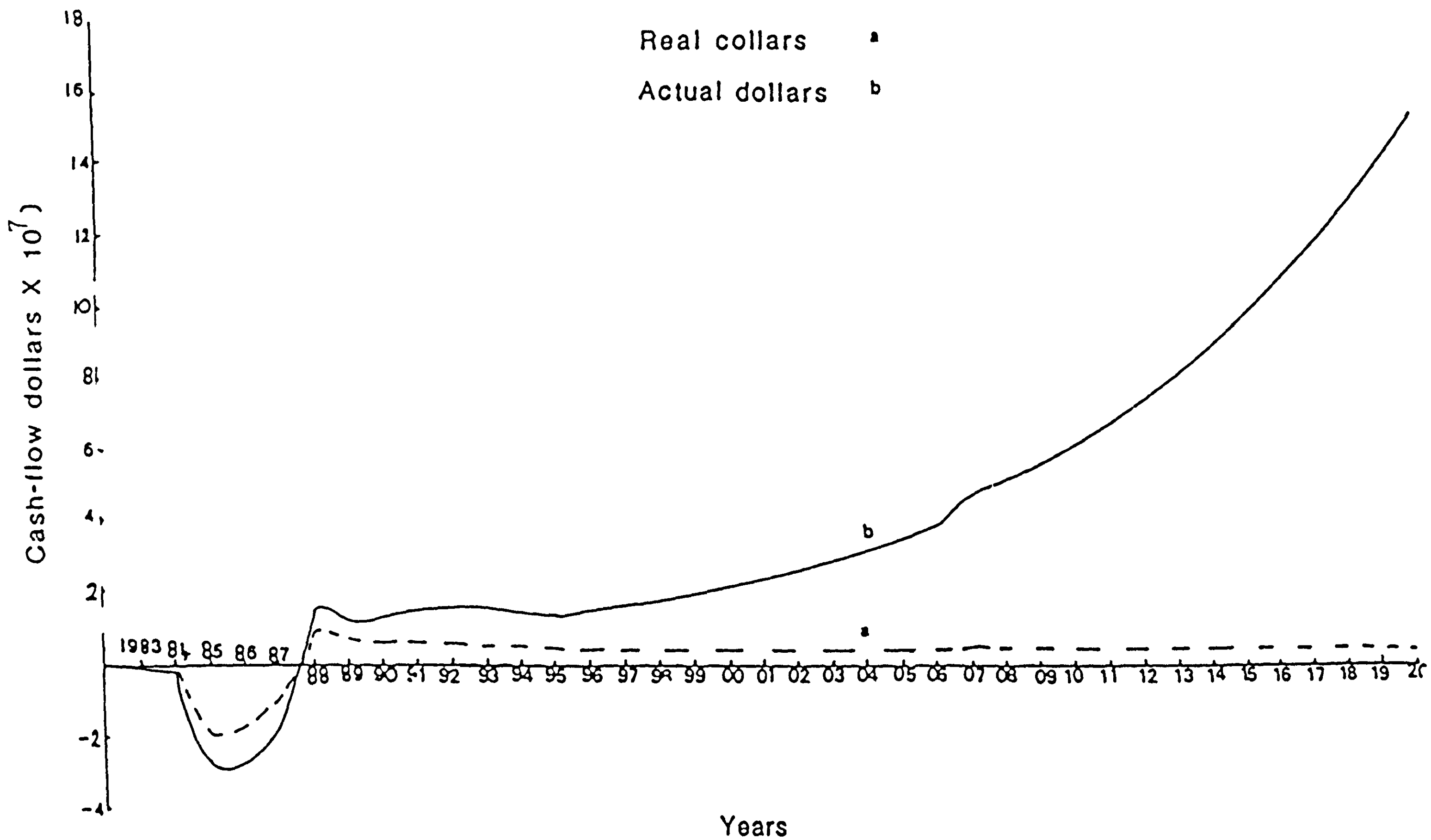


Fig 42 Douala Heavy Sands Base-Case

Cumulative Cash-flow profile

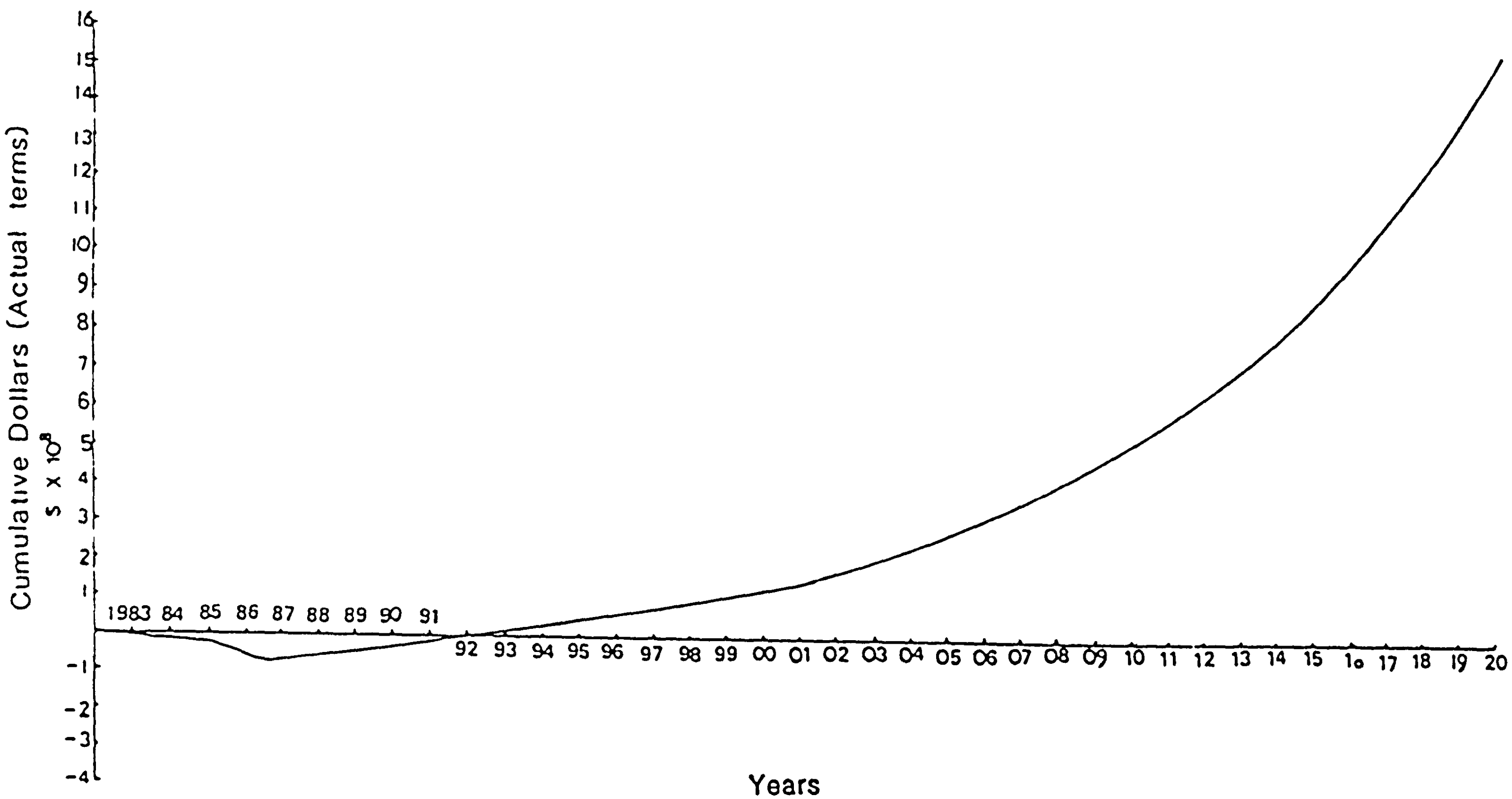


Fig 43 Kribi Base-Case Cash-flow Profiles

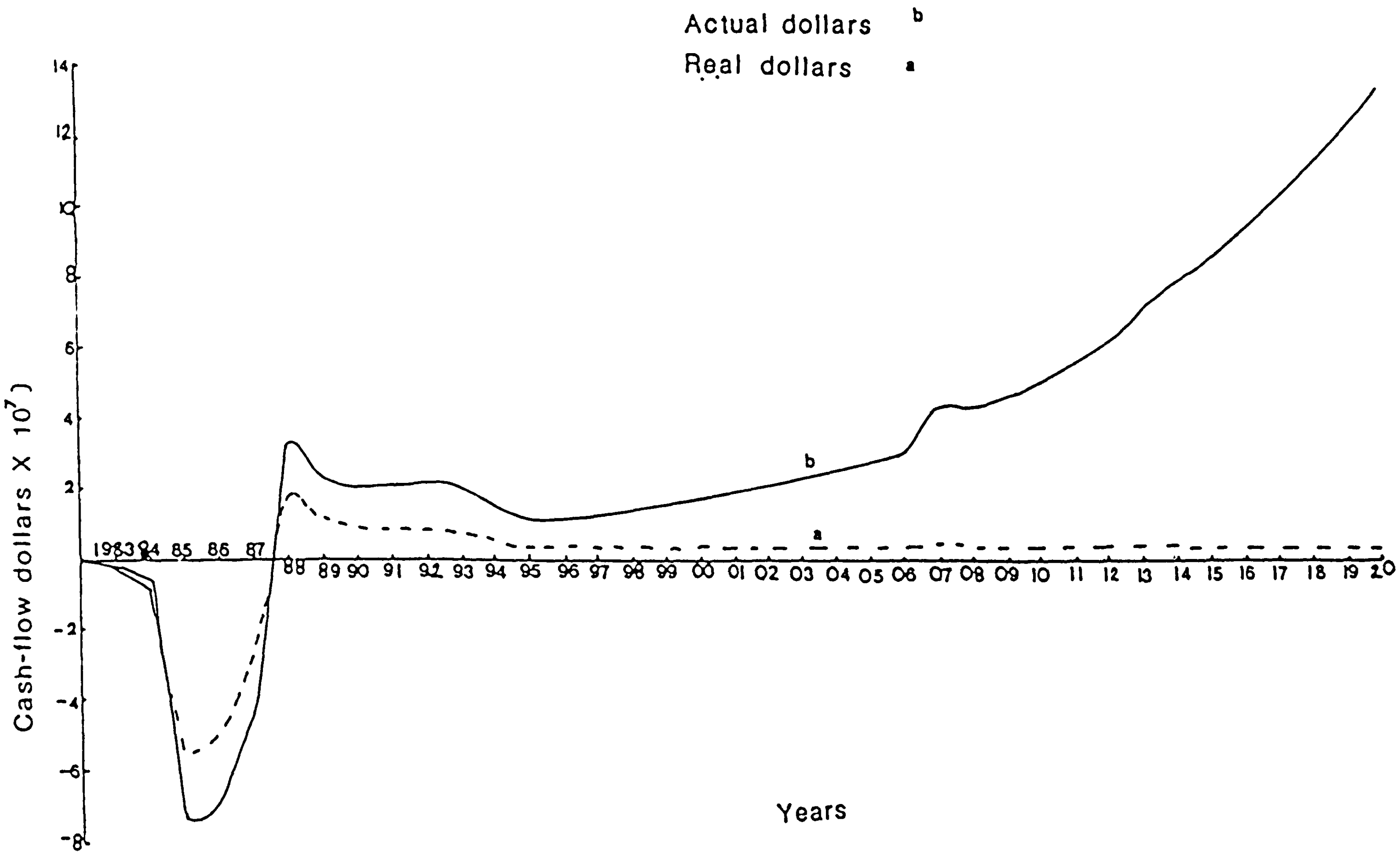
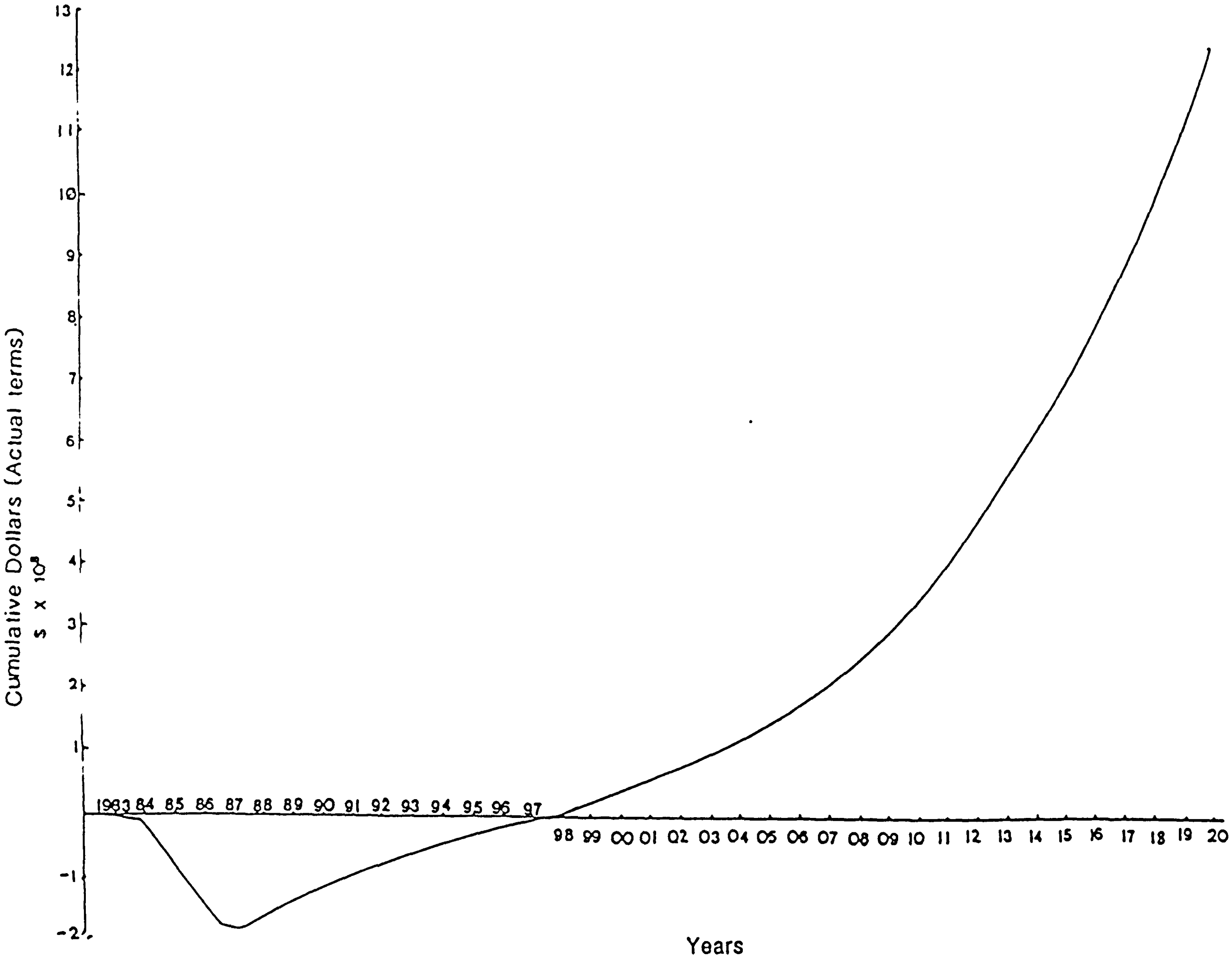


Fig 44 Kribi Base-Case Cumulative Cash-flow Profil



CHAPTER 9: Economic analysis of mineral case-studies

9.1 Generalities

Profitability outcomes of the four mineral cases have been summarised in table 27 . Geo-economic select parameters of each case-study have been analysed in the general MINEX model (fig 36) and within each base-case study model to see the impact of their variation on overall profitability (figs 45, 46 , 47 , 48 ,).

An estimation of break even economic analysis (see appendix III for methodology) of the select model parameter values has also been done, together with an estimation of the impact of transportation costs, (inland, infrastructure and overseas) and fiscal charges on profitability.

Investor and government rewards have been analysed for each case-study and the effects of select geoeconomic parameter variations measured on the proportions of investor and government rewards;. (figs 50 , 51 , 52 , 53 , and 54).

The analysis of foreign exchange gains have also been made for the four case-studies. These analysis have been used as a basis for ranking the four mineral case-studies.

9.1.1 Profitability outcome of mineral base-case studies.

Profitability outcome of the four mineral case-studies have been summarized according to Net Present Value (NPV) in millions of dollars, Internal Rate of Return (IRR) and Payback in Years (see table 27 Government take (Government tax income plus Government equity income (GVS), investor-to-Government rewards ratio (NPV/GVS) and foreign exchange gains have all been presented for the four mineral case-studies.

The cassiterite base-case study has the highest NPV ($368·8 million dollars), NPV/GVS(0.35), an IRR of (30.6)% and the shortest payback (3.4 years). The bauxite-alumina model ranks second, the rutile heavy-sands third and the iron-ore model fourth (see table 27). The bauxite-alumina and rutile heavy-sands models, with IRRs of 11.30% and 8.06% respectively are marginal case-studies in terms of the 10% bench-mark discount rate used in their analysis. The Kribi iron-ore case is clearly submarginal with an IRR of 1.86%.

9.1.1-1 Impact of geo-economic parameter changes on Net Present Value

Figs 45 , 46 , 47 , 48 present the sensitivities of select geoeconomic parameters (price, ore-grade, recovery-ratio, operating cost and capital cost) on their base-case profitabilities (NPVs). The sensitivity of NPV to changes in base parameters is more marked in the marginal cases (bauxite and rutile case studies) than in either the very profitable cassiterite case or the submarginal iron-ore case study.

Price appears to be the most important parameter in the model because variations in its base value produces the highest NPV sensitivity responses, throughout the four mineral exploitation models; causing as much as +59% (cassiterite), +402% (bauxite), +40% (iron-ore) and +259% (rutile) changes in the base-case NPVs for +50%

changes in their base prices. The cassiterite case-study is robust enough to withstand -50% price fluctuations from the $14,000/ton base value, without attaining zero NPV. This leaves an important safety margin of greater than >50% of the base price about which cassiterite price can fluctuate without producing negative NPV results. As a result the risk to the cassiterite case-study of a sudden price fall is greatly reduced. In the case of the alumina price, there is only a 10% margin below which prices can fluctuate from their base value of $300/ton, before an NPV of zero is registered. Both the rutile heavy sands and the Kribi iron-ore cases have to be raised respectively above the base-case values by about 26% for the rutile case ($628/ton) and by greater than >50% in order to sell at a profit.

The revenue lives of these mineral exploitation models have also been shown to produce negative profitability (see fig49) when they are shortened. The implication is that any factors which tend to disrupt the revenue life either through periodic closures of operations, as a result of industrial action, force majeur or political unrest would have a serious impact on the project profitability.

A cut-off grade of 40% Al_2O_3 was estimated for the bauxite alumina case-study; this implies that exploration activities should seek to delineate ore above the >40% Al_2O_3 cut-off grade. The Rutile case would need to delineate heavy-sands at a grade of >4% heavy sands. The cassiterite model would utilize ore with as low as 0.5% Sn (cut-off). The iron-ore case-study need ore greater than the concentrate grade 67% Fe to make operations marginal.

The effects of grade and mill recovery fluctuations on the NPV of the mineral cases are identical. Since mill recovery usually remains below the <90% level and is less amenable to improvement beyond this level than grade increases are than grade; efforts should be made to increase cut-off grade above which exploitation should be undertaken for marginal and submarginal case-studies while at the same finding less expensive high recovery schemes of mineral beneficiation through pilot plant and metallurgical experimentation.

High output rates should be maintained for marginal cases >4,500 tons per day (bauxite-alumina) and >7,200 tons per day rutile heavy-sands.

The analysis indicate lower output rates for the submarginal iron-ore case-study. This seems to be so because the bulky nature of the ore increases cost of transportation, and affects the case to which it is tied price of ore is low. High output rates also favour the cassiterite model.

Capital and operating costs are the least sensitive of the select geoeconomic parameters (figs 45 , 46 and 48) except for the iron-ore model, (fig 47) in which sensitive of +42% (operating cost) and +52% (capital cost) are registered for a 50% change in their base values. Generally capital costs are more sensitive than operating costs. In the cassiterite, bauxite alumina and rutile case-studies, operating cost margin is >50%. The Capital cost margin is >50% in the cassiterite model, 30% in the bauxite-alumina model and 26% in the rutile model. Both capital and operating costs need to be streamed by >50% of the base values in order to render the iron case profitable.

9.1.1-2 Impact of geo-economic parameter changes on (NPV/GVS) investor and Government ratio of rewards:

The effect of parameter variation on the ratio of investor-government rewards (NPV/GVS) has been depicted in figs 50 , 51 , 52 , 53 . 54

The NPV/GVS ratio is almost invariant for +50% changes in parameter values for high profitability models like the Mayo-Darle cassiterite model (>0.3). There is a significant variation in the NPV/GVS ratio (-1 to +1) as these select parameters are varied from +10%, +30% to +50% of their base values (rutile and bauxite marginal case-studies). The submarginal case-study produces very significant NPV/GVS variations (see fig 50 51 52 53 54

This happens to be the case because the revenues that accrue to investors from a project will diminish as the level of taxation and government participation increase. In order to improve case-study attractiveness, government rewards have to be reduced in the form cutting its share of equity participation. If taxation (corporate, withholding and royalty) levels are high they should be lowered. In the present study the Cameroonian taxation parameters were assumed to be quite fair and were left at their base values. The GVS was studied (see section 9.1.1-3). High GVS in the equity of a project decreases the profitability and hence increases risk to investor capital by prolonging the payback period. As a result, the GVS should be reduced as recommended in 10.1·1

Investor-government rewards ratio increase as price increases, grade and mill-recovery increase and also as output increases.

Below their break-even values,select parameter NPV/GVS ratios are negative. Decrease in operating and capital costs increase the NPV/GVS ratio. Generally all changes which tend to improve the NPV without significantly deteriorating the GVS, are conducive to Investor-Government agreements.

9.1.1-3 Impact of government participation on base-case profitability

The impact of government equity participation on project profitability has been depicted in figs 55 , 56 , 57 , 58 , and 59 . Base-case profitability outcomes due to variations in government equity participation (30%, 50% and 70%) were monitored using the Net Present Value (NPV), Internal rate of return (IRR) and Payback (years) criteria. Government profitability measures were monitored as Government tax income - income derived from corporate tax (30%), withholding tax 27.5% and royalty 5% (advalorem); Government equity participation (GVS) and Government Take (Government tax plus GVS).
Government share (GVS) was made to increase from 30%, 50% to 70% of total equity. The increase in government share income is greater in more profitable operations (Mayo-Darle tin model) than for less profitable operations.

Government tax income decreases as the government equity participation increases (56), this appears to be caused by diminishing taxable cash reserves (as government share increases) when other fiscal factors are kept constant within the model. The same observation is made in the resulting net present values as variations in government participation are effected (55).

There is a steep decline in NPV as the government participation is increased (for profitable cases). The decline in NPV becomes moderate and even insignificant for low profitability and marginal operations (Kribi-iron ore model).

The internal rate of return (IRR), the investor-government reward ratio (NPV/GVS) vary negatively against government participation (fig 59 , fig 61 , fig 60). Payback varies exponentially with increasing government equity participation.

It should be mentioned that government participation is a very crucial factor in determining project profitability, infact some mineral economists now consider governement participation as a tax.

9.1.1-4 Transportation Impact

With the location of new mines in remote and inhospitable country-sides, it is becoming increasingly common to lay much emphasis on the transportation of both raw and beneficiated mineral produce from these remote sites to their ultimate market destinations.

In these mineral case-models, three aspects of mineral transportation appear to be most important:-

Infrastructure capital costs,
Inland transportation costs,
Overseas transportation costs.

A discussion of transportation infrastures has already been made in section 6.4.1. Capital cost estimates for erecting an infrastructure of rail and roads have also been presented in table 21 , 25 , 17 , and 13 .

Overseas costs of concentrate delivery to market (C.I.F. Cameroon - W. Europe) have been presented side-by-side the other operating costs. Generally sea freight is low, between $10 and $5 per ton from West-African ports to ports in Western Europe or the E.E.C. . Inland freight (20 cents U.S. per ton kilometer) is about 40 times the sea freight per unit weight of ore transported from Cameroon. The disadvantages of inland transportation are quite apparent especially when the distance to be covered is important. This disadvantage is even accentuated when the material being transported is bulk (iron-ore concentrate or alumina). In the iron-ore model, mill and inland transportation costs were 22.4% of total operating costs, overseas transportation about 37.4% of total operating costs; in the alumina-bauxite model, milling and transportation was 74.5% of total operating costs. In the low bulk rutile model which is only 30 kilometers from the Douala port, milling and transportation came up to be about 11% of total operating costs.

The economic advantages of reducing transportation costs by streamlining the transportation sector are obvious, but savings resulting from this measure often occasion new capital and operating cost centers for further domestic beneficiation or processing of the end-mineral. In this way not only are transport costs saved but the mineral produce price is increased by further processing, thereby improving on the profitability.

9.1.1-5 Foreign Exchange gains

It was assumed that investor's capital will originate overseas from where it will be spent on the purchase of equipment plants and all machinery needed for the operations.

This requires about 80% - 90% of the capital expenditures to be spent abroad. About 50% of the operating costs is assumed to be spent within the country in the form of expatriate and local salaries fuel and local input purchases. (See section 10.)

Foreign exchange gains have been presented in table (27). These gains can be increased by creating linkages between mineral case-studies and the domestic industries and businesses. $720.8 million dollars will accrue from the bauxite base-case, $1.8 billion dollars from the highly profitable cassiterite case-study, $68.5 million from the rutile case and $80.0 million from the iron-ore model.

As government revenues from taxation and government participation increase, the net foreign exchange gains improve; as such a balance must be struck between how much foreign exchange gains government is prepared to make and how much investor gains to be reaped from the mineral venture.

Fig 45 Mayo-Darle Base-Case

Sensitivity of NPV to changes in select geo-economic parameters

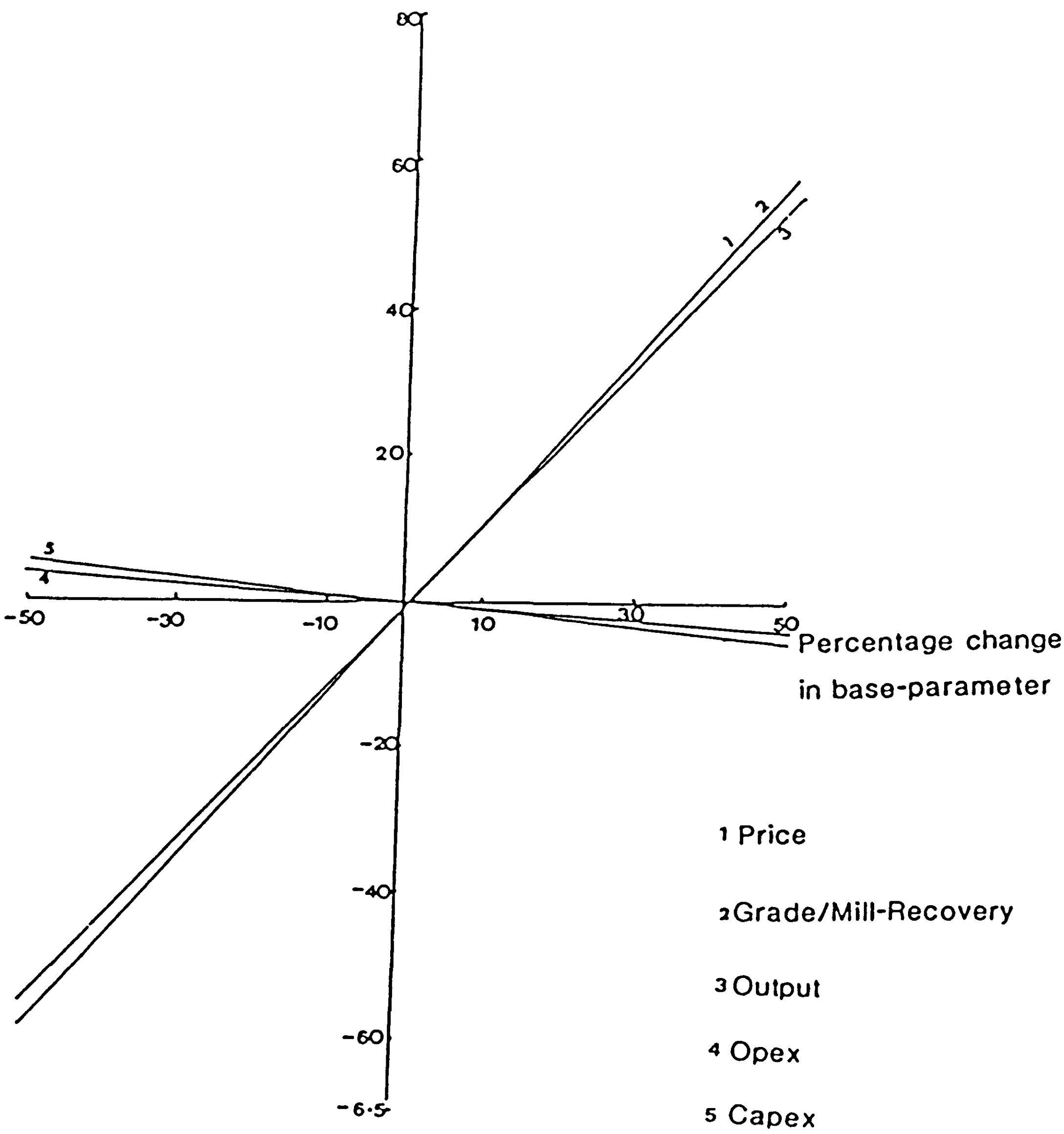


Fig 46 Fongu-Tungong Base-Case

Sensitivity of NPV to changes in select geo-economic parameters

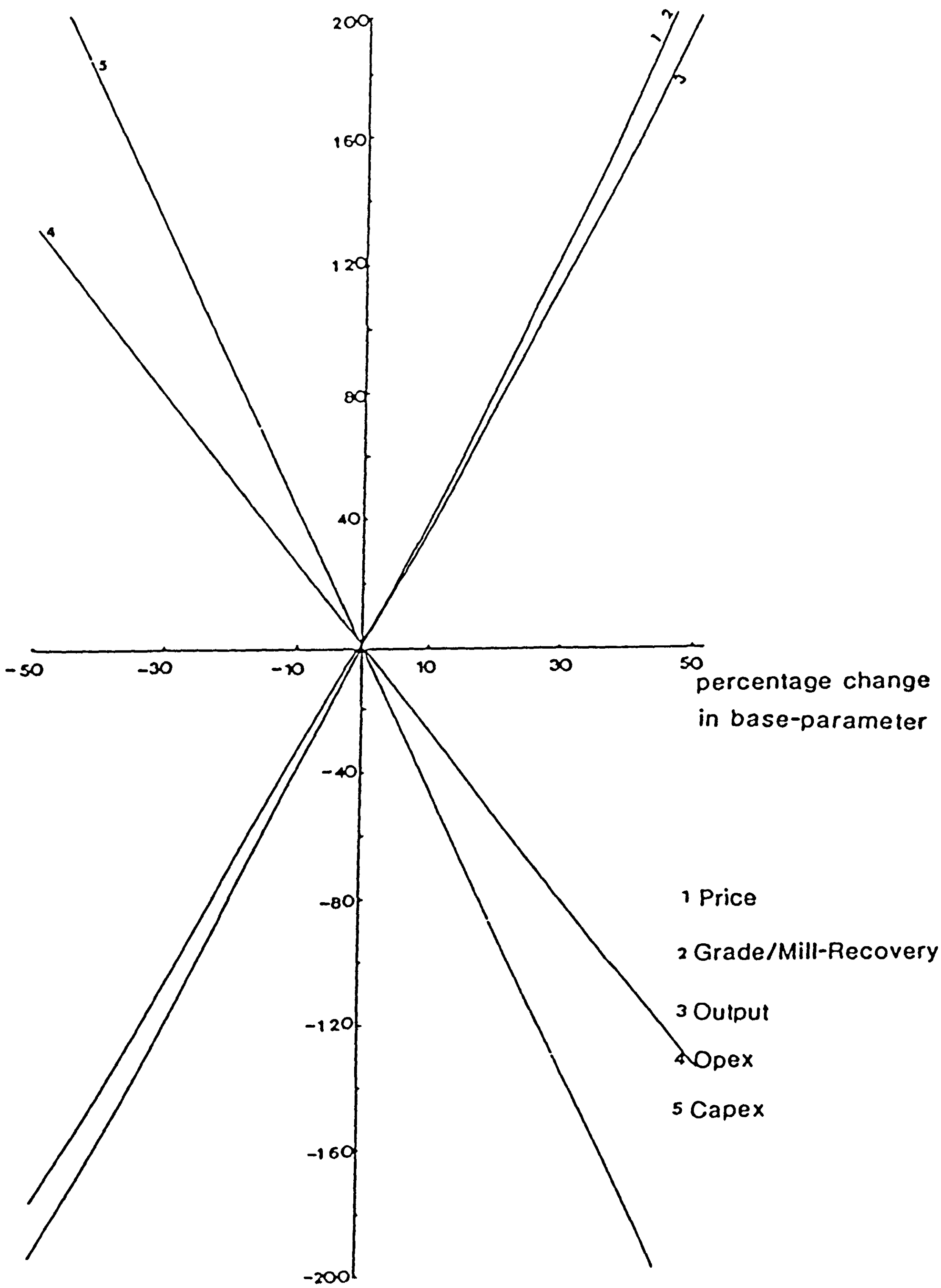


Fig 47 Kribi Base-Case

Sensitivity of NPV to changes in select geo-economic parameters

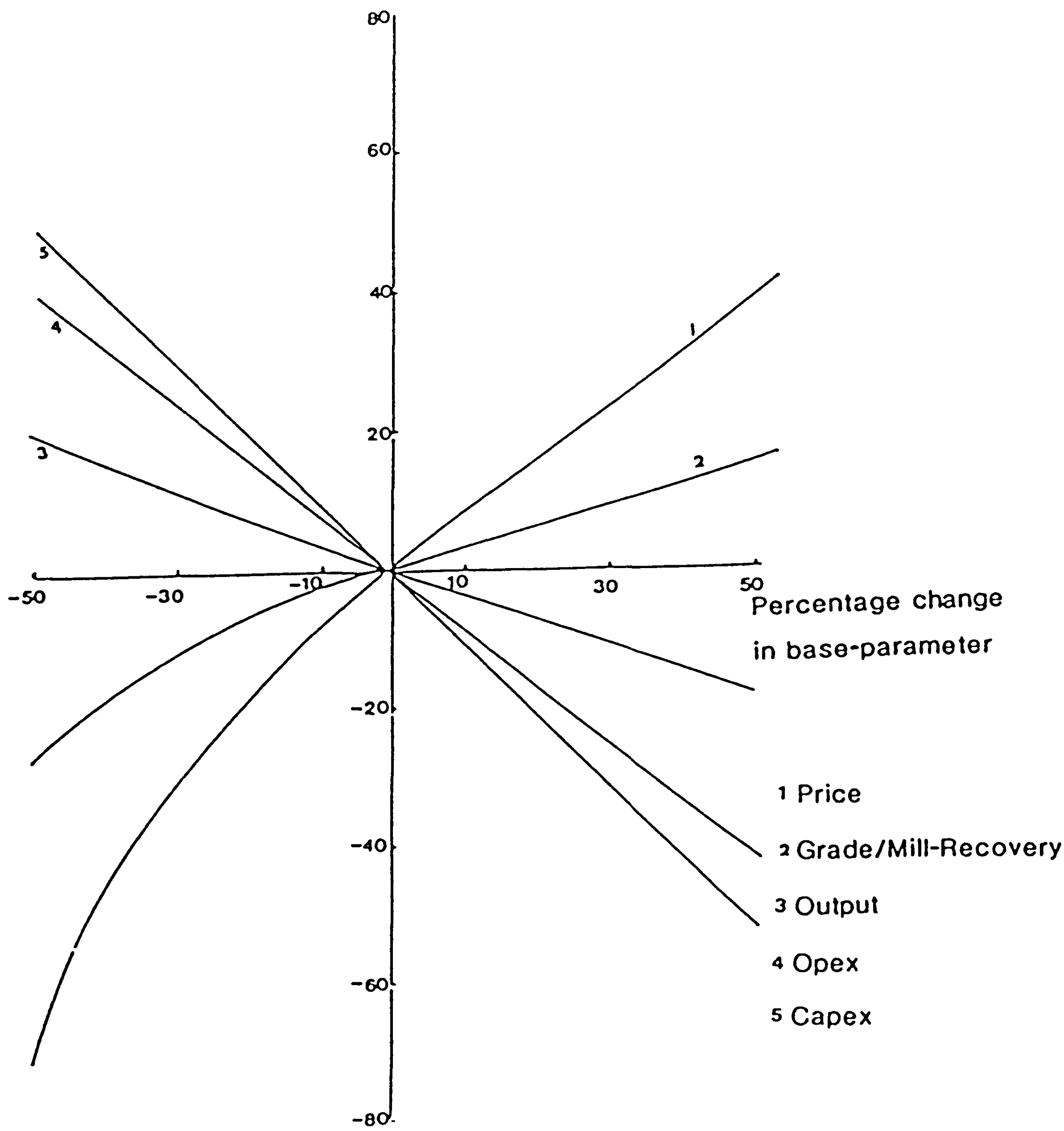


Fig 48 Douala Heavy Sands Base-Case

Sensitivity of NPV to changes in select geo-economic parameters

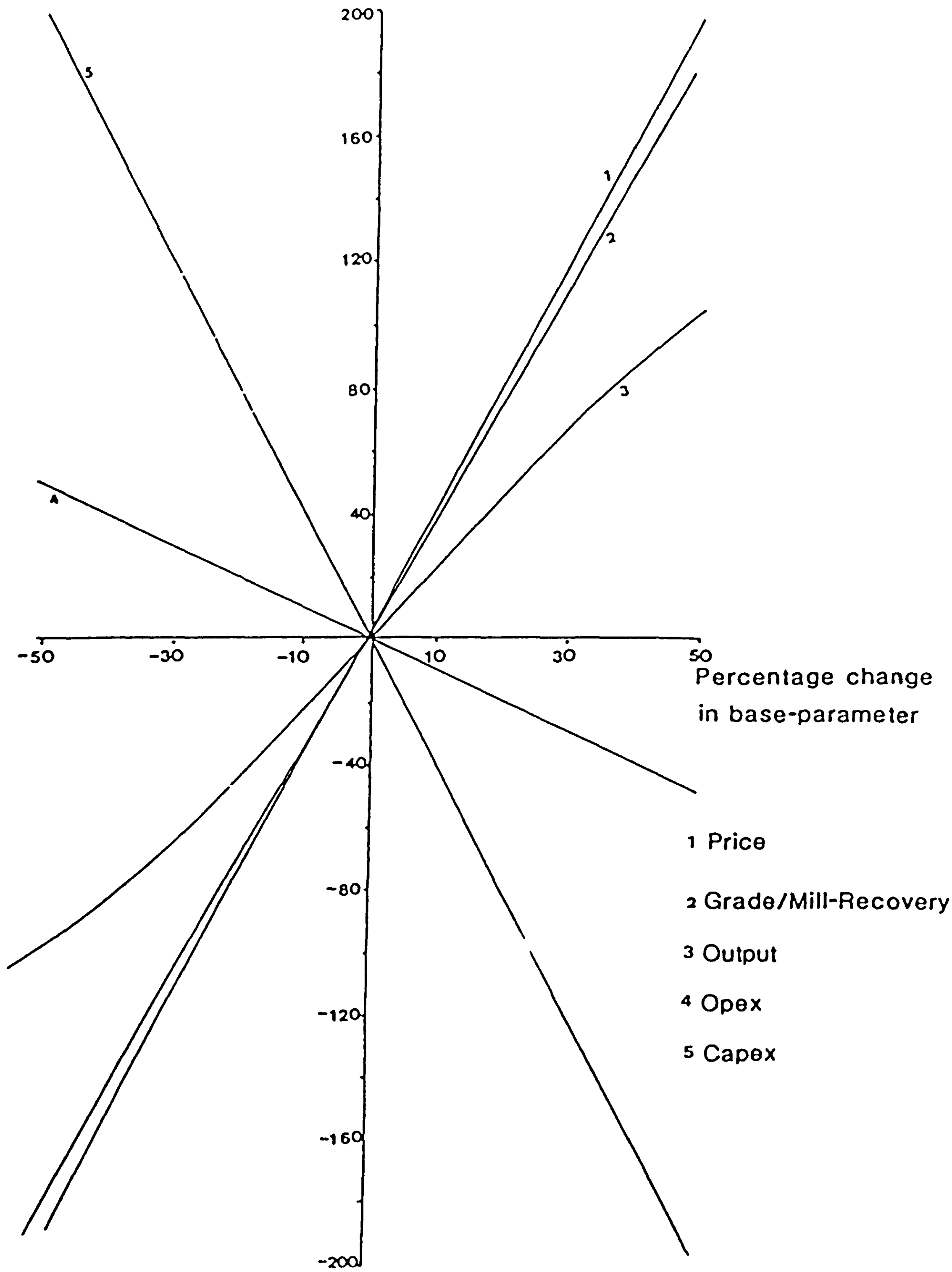
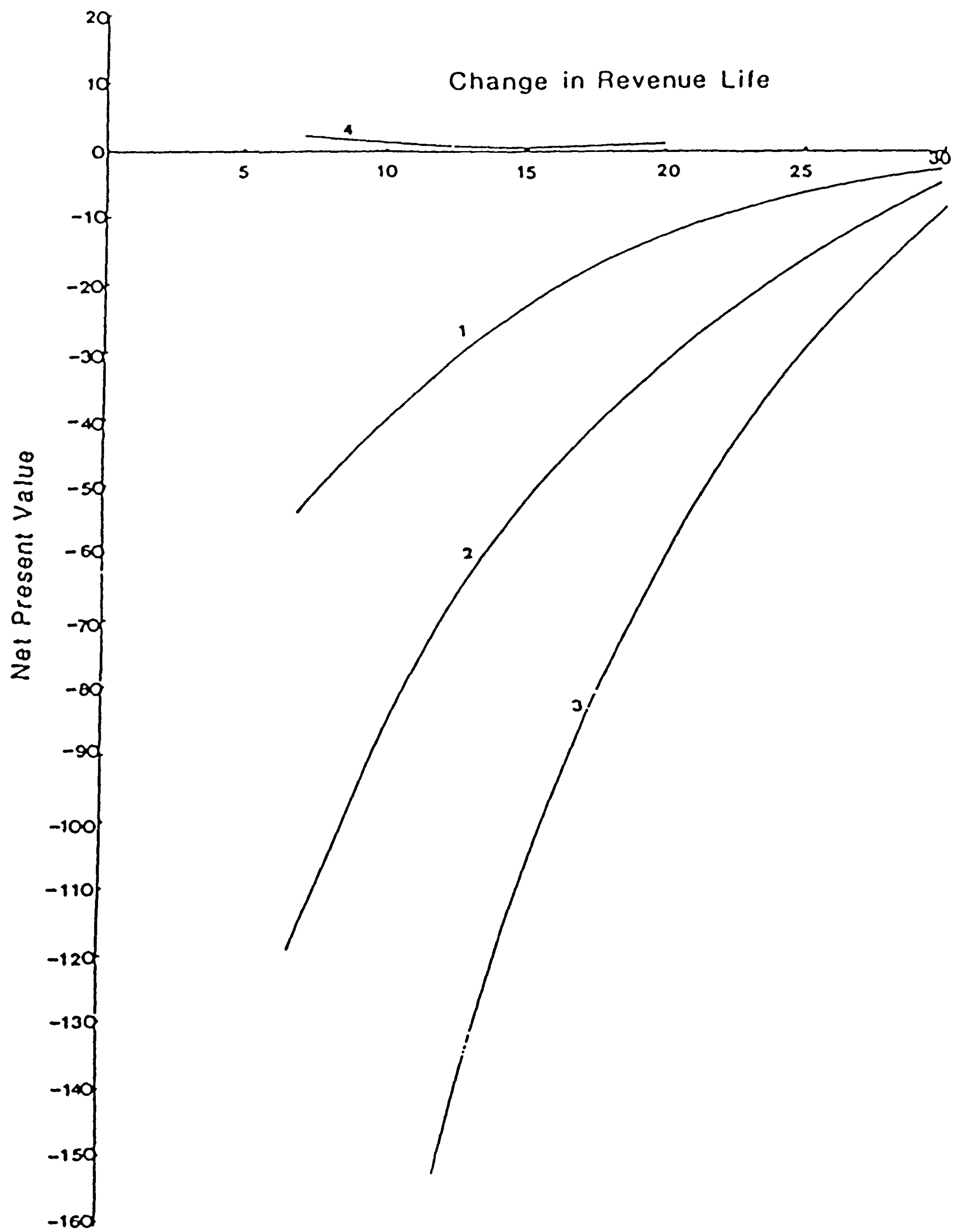


Fig 49 Sensitivity of NPV to changes in Revenue Life



1 Mayo-Darle cassiterite case

2 Fongu-Tungong bauxite-alumina case

3 Douala rutile heavy sands case

4 Kribi iron-ore case

Fig 50 Impact of Price Fluctuations on Investor-Government rewards

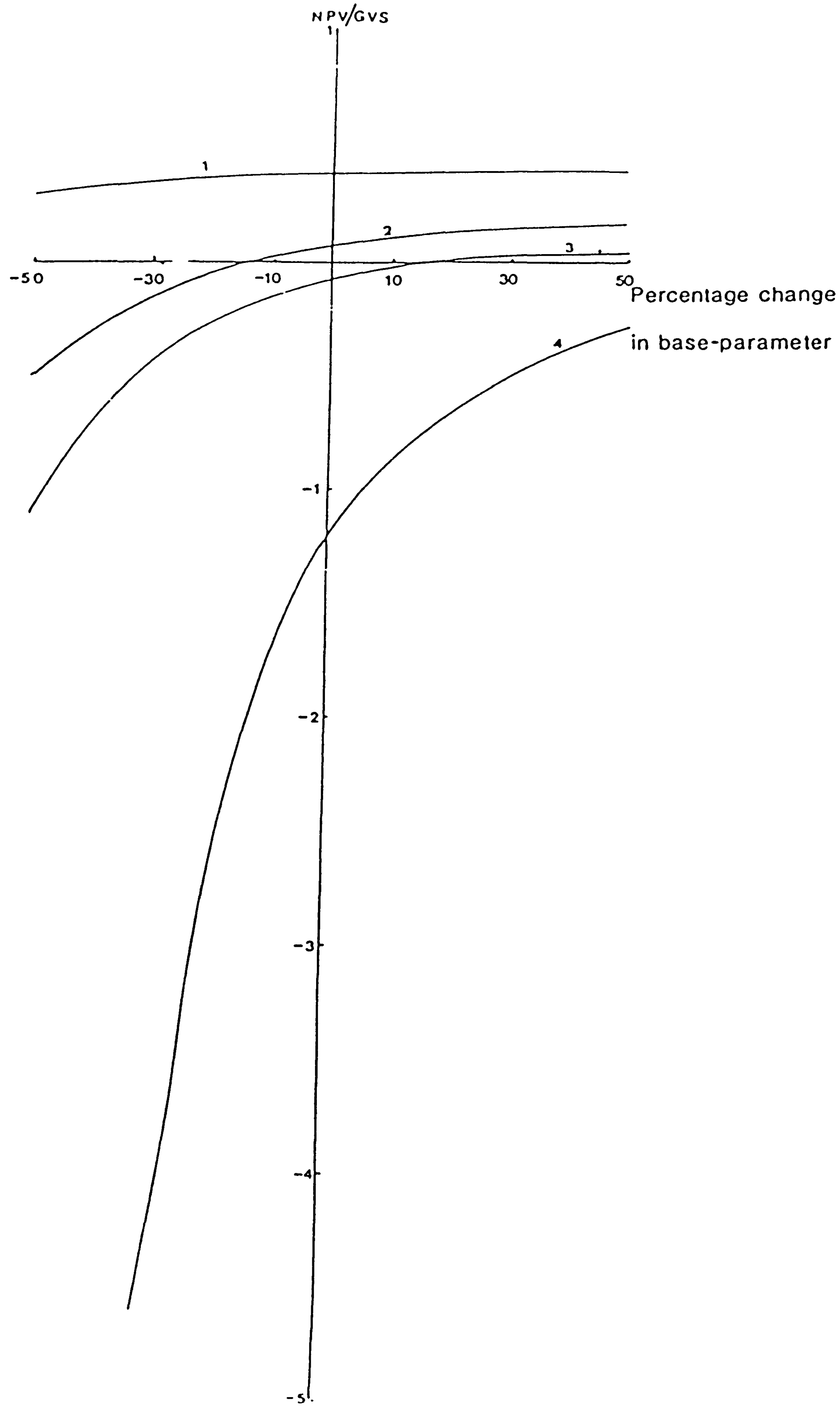


Fig 5.1

Impact of Changes in Grade and Mill-Recovery on Investor-Government Rewards

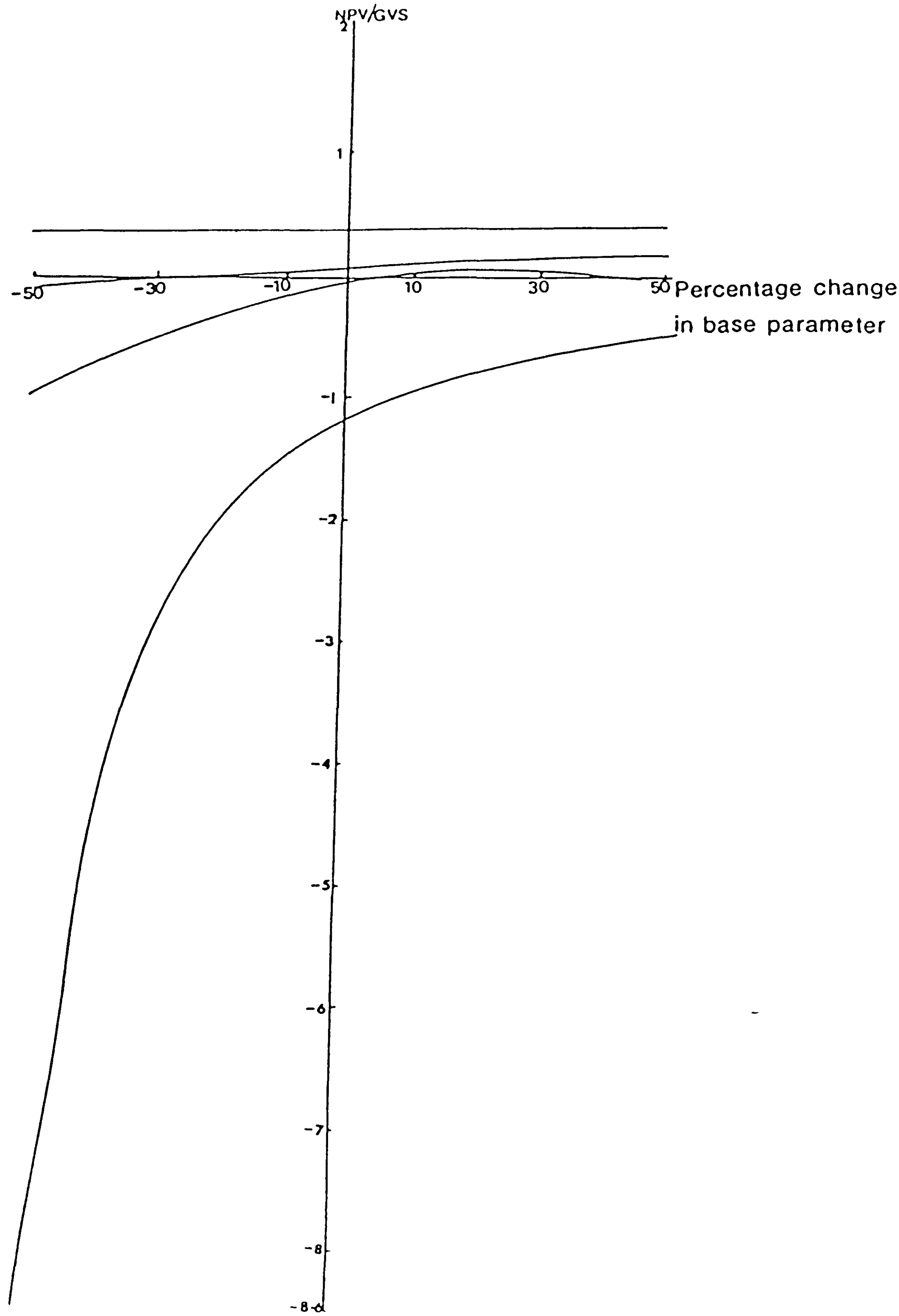


Fig 57
Impact of Changes in Output Rate on Investor-Government Rewards

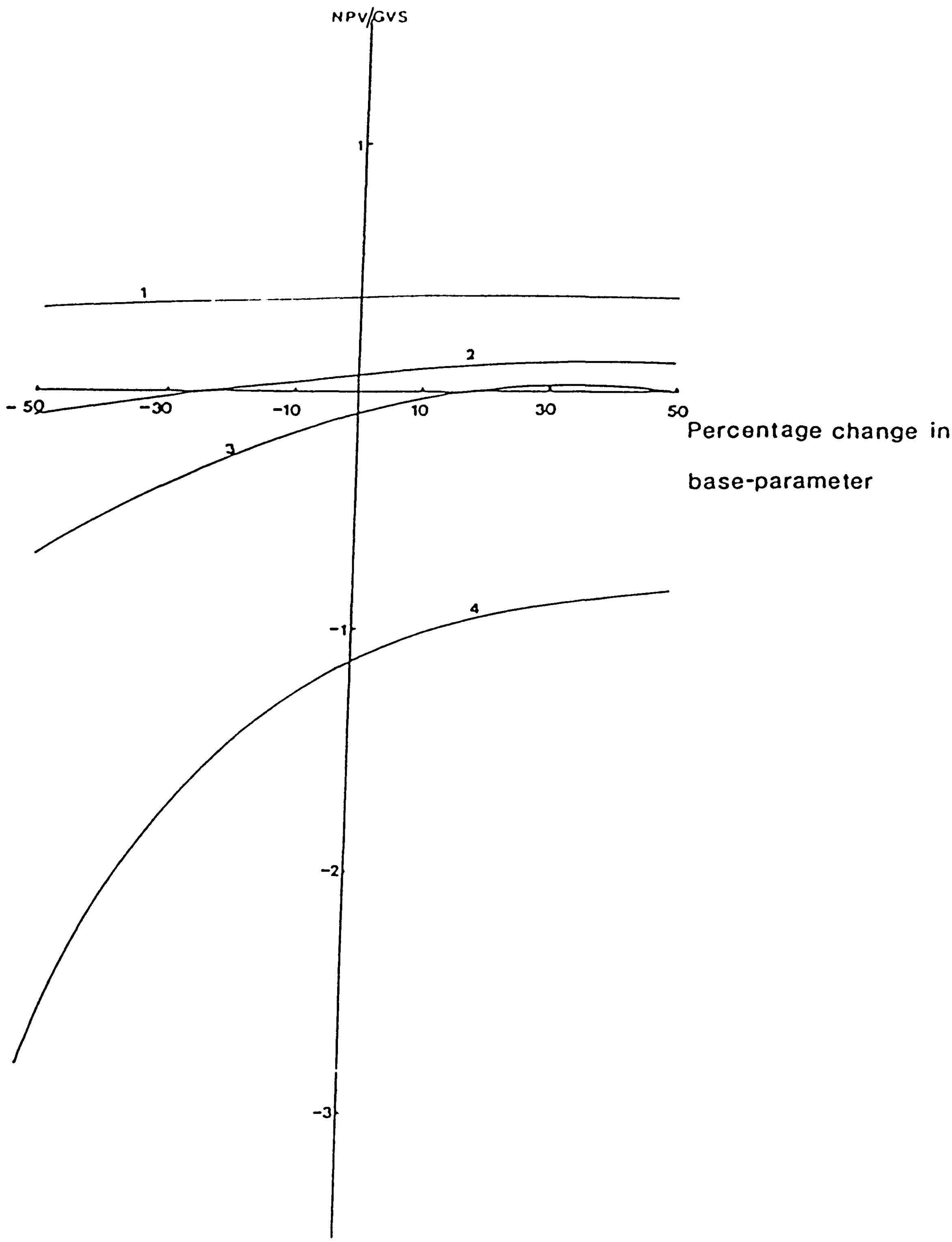


Fig 53

Impact of Changes in Capital Costs on Investor-Government Rewards

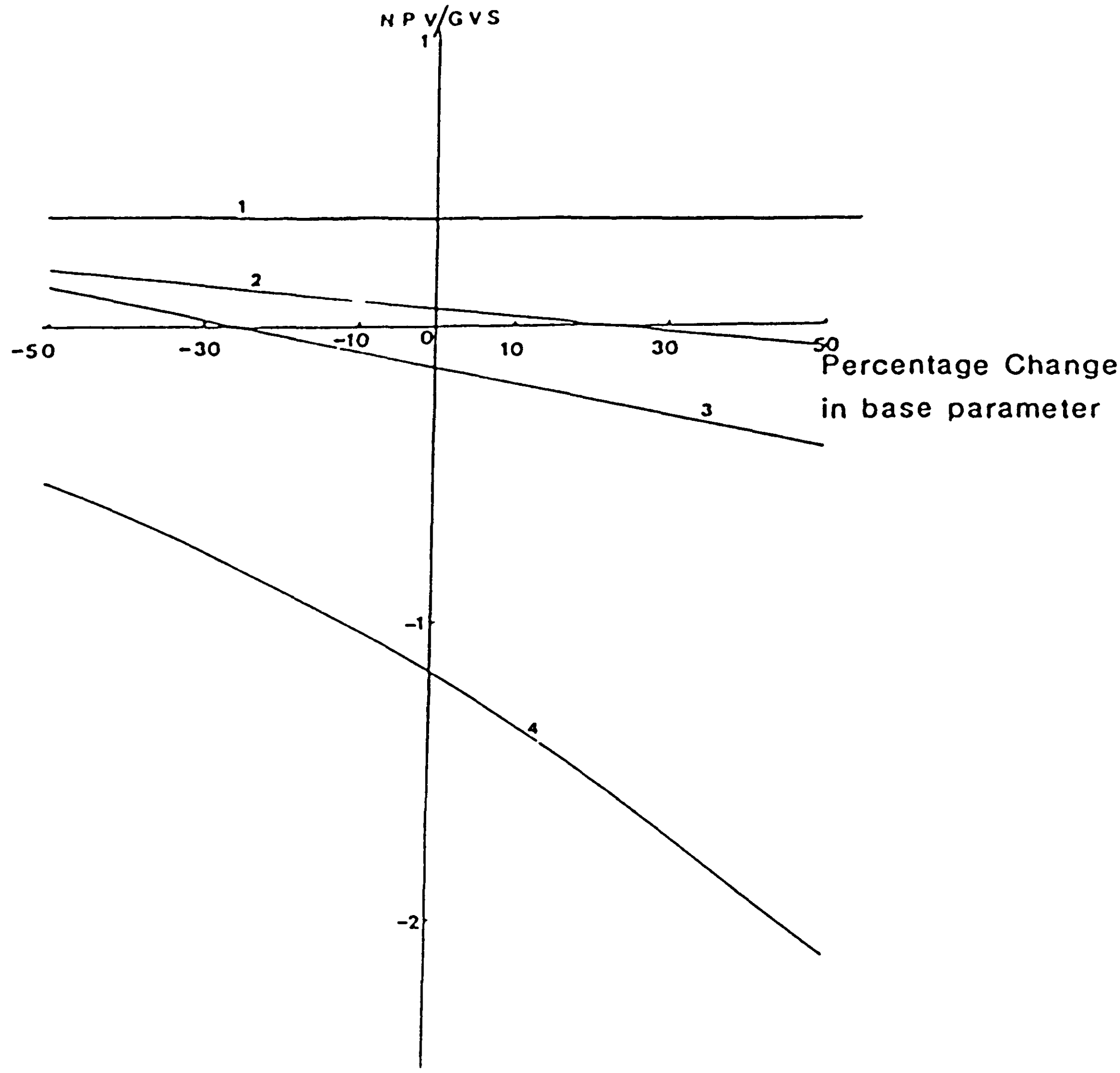


Fig 54

Impact of Changes in Operating Costs on Investor-Government Rewards

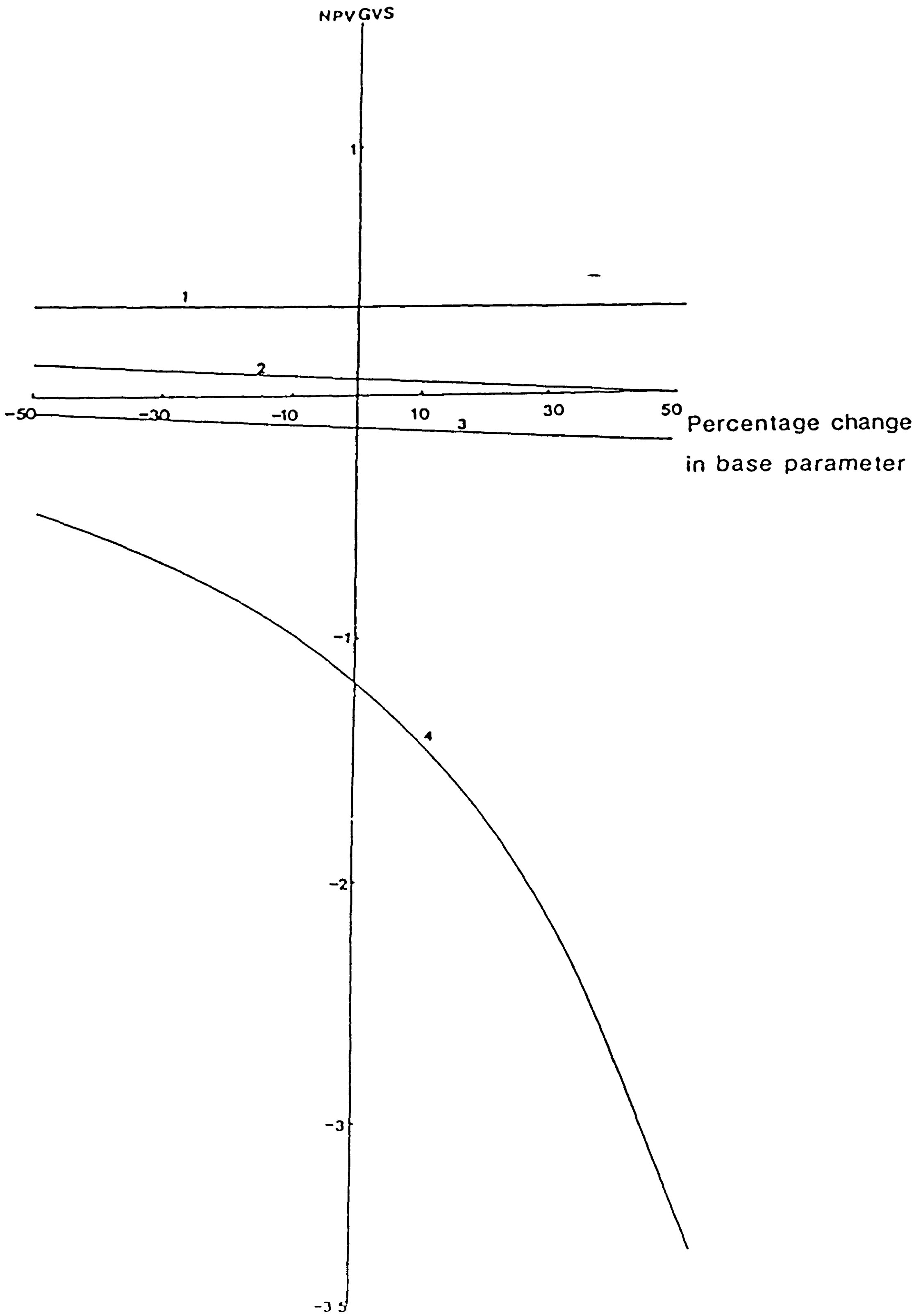


Fig 55 Impact of Government participation on Government Equity income

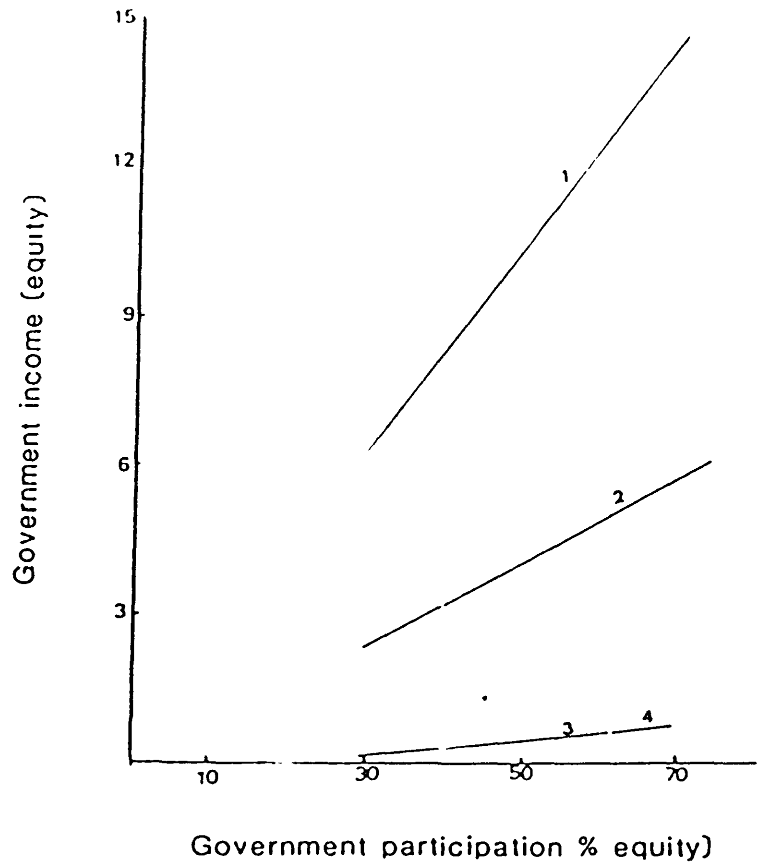


Fig 56 Impact of Government participation on Government Fiscal Income

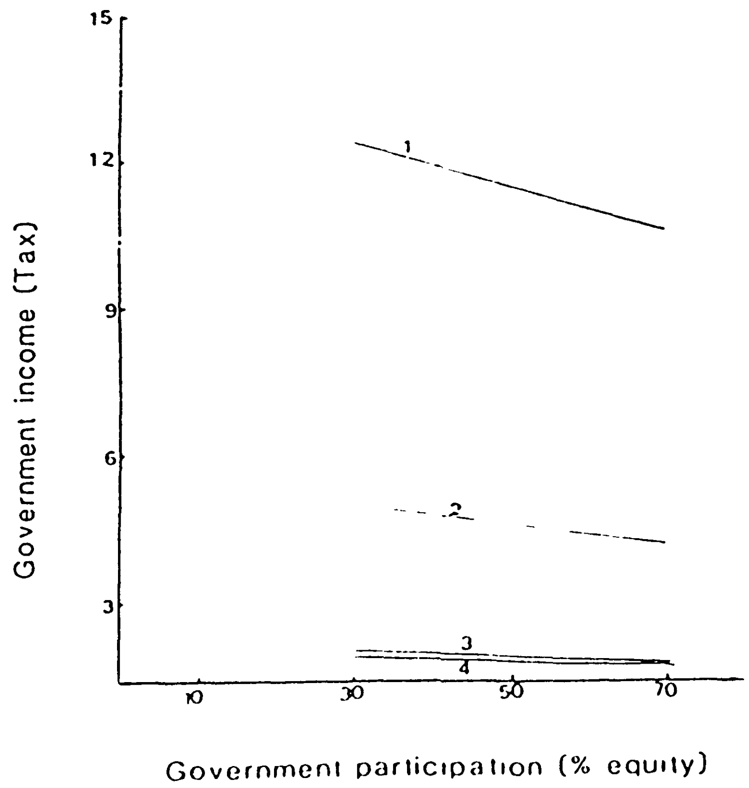


Fig 57 Impact of Government participation on Total Government Take

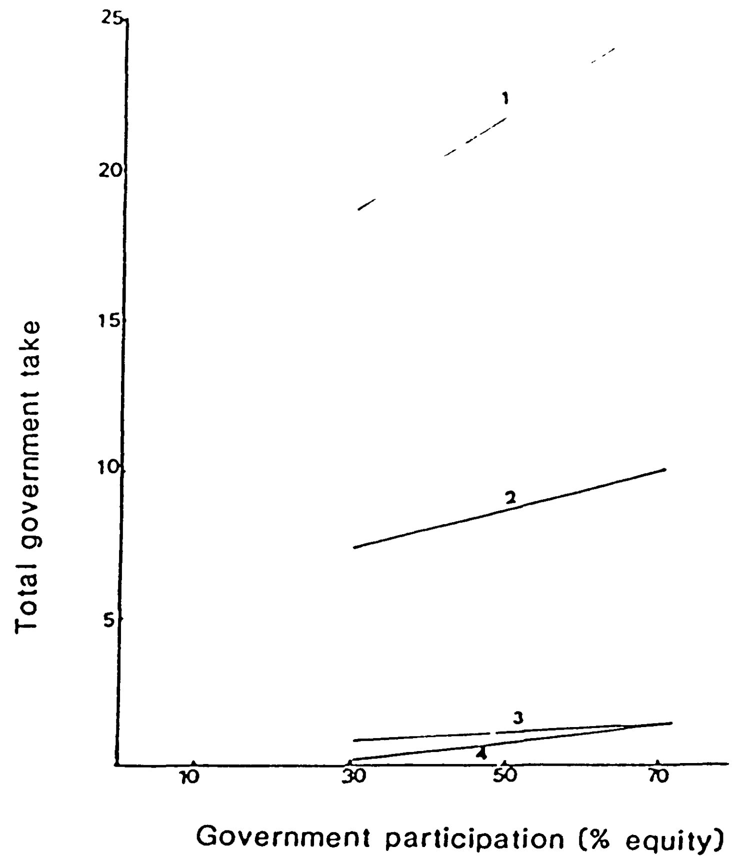


Fig 58 Impact of Government participation on Net Present Value

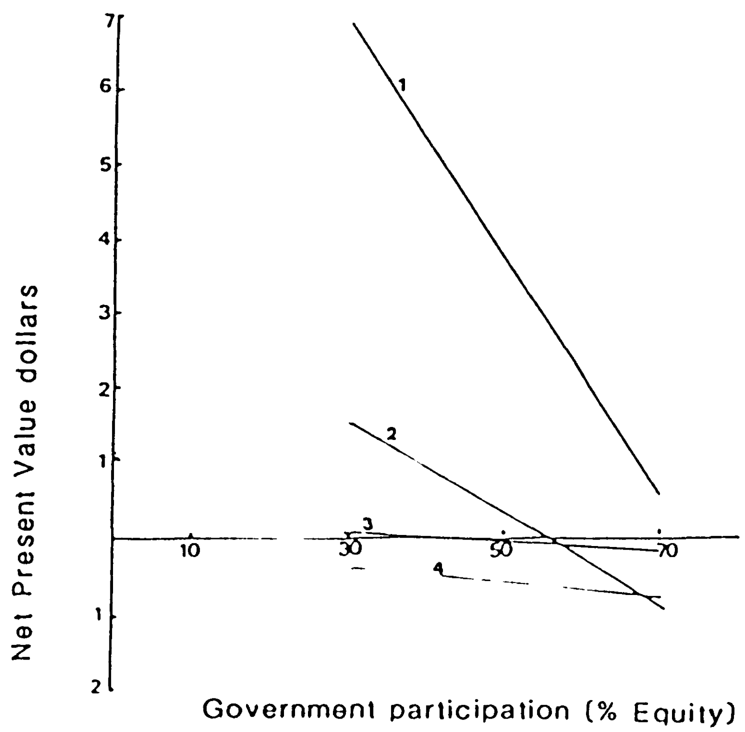


Fig 59

Impact of Government Participation on Internal rate of return

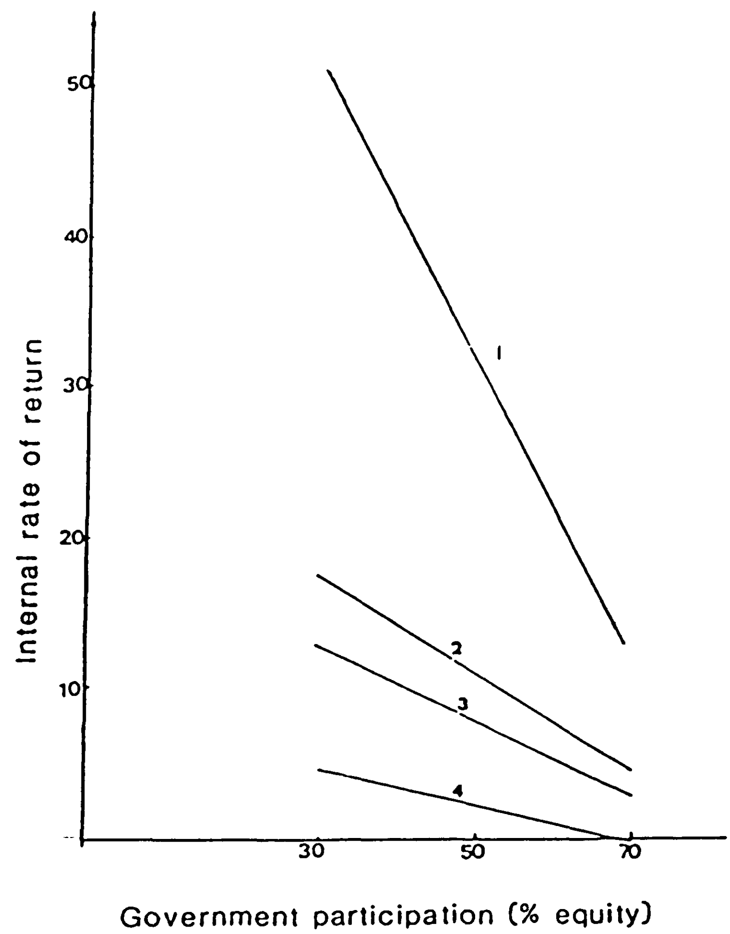


Fig 60

Impact of Government participation on Payback(Years)

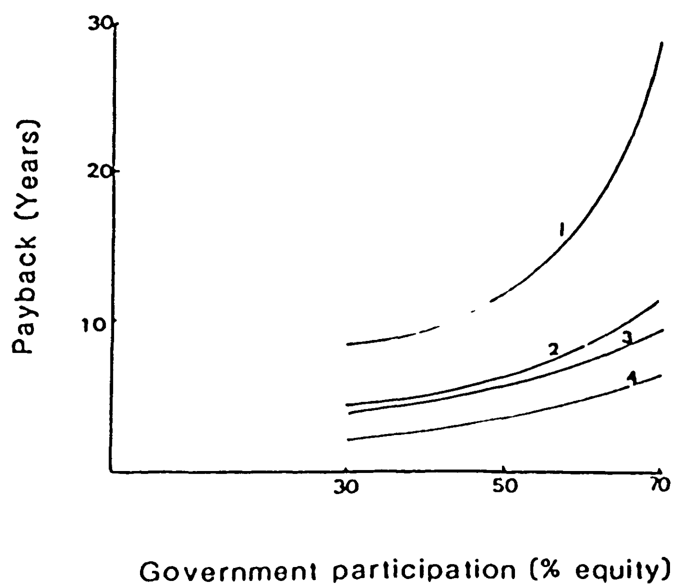


Fig 6.1 Impact of Government Participation on Investor-Government Rewards

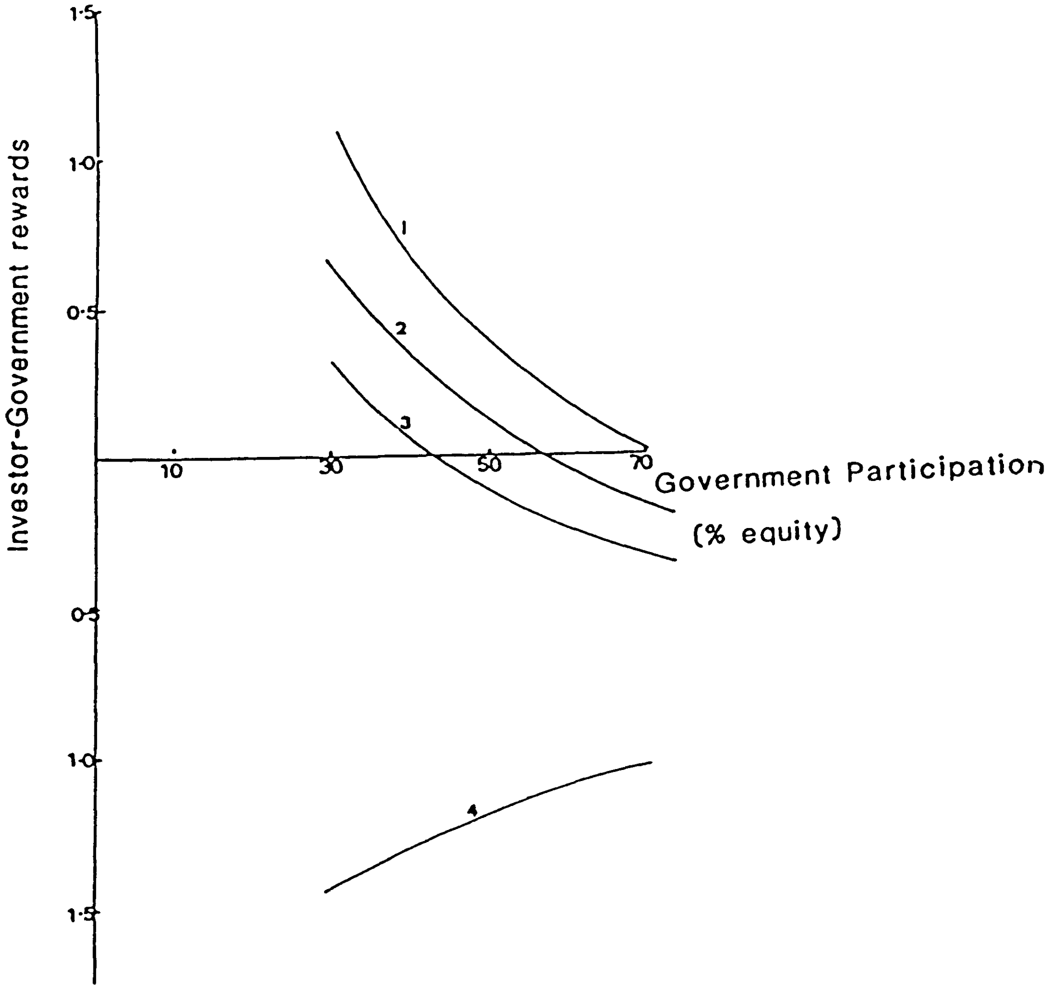


Table 27 Case-study profitability outcome and ranking

CRITERION	BAUXITE	RUTILE	CASSITERITE	IRON-ORE
Net Present Value (NPV) (U.S.x10^6$)	25.52	-6.23	368.8	-57.1
Government take (U.S.x10^6$)	866.9	100.25	2,175	109.5
Investor-Government rewards (NPV/GVS)	0.06	-0.13	0.35	- 0.52
Payback (years)	5.5	6.29	3.29	12.0
Internal rate of return (IRR) %	11.30	8.06	30.6	1.86
Foreign Exchange Gains (F.E.G.) $ x 10^6	720.8	68.46	1,810.0	80.0
NPV as a percentage of F.E.G.	3.5	-9.1	20.4	-71.4
Ranking	2	3	1	4

CHAPTER 10 Policy Recommendations and Future work.

10.1 Generalities

The original research proposition - the quantitative appraisal of 4 mineral case-studies in Cameroon and analysis of their economic rewards to investors and governments - has largely been accomplished through a two stage modelling and analysis exercise involving the development of a total mineral exploitation model (MINEX) and its application in the economic appraisal of the mineral cases in question (Chapter 4,5,6,7,8 and 9).

These mineral exploitation models have been used to demonstrate the effects of changes in geoeconomic model parameters on overall case-study effectiveness (Chapter 7 and 9) and can also be used to verify effects of actual project parameter changes on overall profitability of real projects.

The MINEX model is being proposed as a suitable scheme for quantitative appraisal of mineral potentials; it can be used in fulfilling objective (2) of the minerals section of the fifth five year economic plan (138), by increasing the scope of the study. The policy proposals have been made below, in the light of results obtained from a study of 4 selected mineral cases, and as such should be regarded as policy-formula or guidelines according to which a more comprehensive national minerals policy can be formulated.

Mineral exploitation projects are investment projects par excellence and have to be regarded as such. Capital expenditures and operating costs have to be balanced against revenues over the anticipated project life to make investment profitable. In the four case-studies, capital costs are more sensitive than operating costs in effecting changes on profitability (see figs 45 , 46 , 47 , 48 and 49); though concentrate price turns out to be the most crucial economic parameter in the models.

Fiscal costs (corporate tax, royalty payments and withholding tax) should be treated as major operating expenditures which do not have a productive effect on project profitability. They should be tailored to allow a commensurate and attractive profit margin to potential investors. Government objectives should aim at contributing to project effectiveness while assuring some kind of social benefits in the long-run i.e construction of mines infrastructures (rail lines and roads) townsite and restored mined land for agriculture.

In this way, both government and investor anticipated rewards and objectives can be resolved and realized during the exploitation of the mineral case in question.

10.1.1 <u>Government Equity participation and tax income:</u>

Governments consider the holding of shares in mineral investment projects as another means of increasing their total revenue from these and similar undertakings. Figs 57 , 58 , 59 and 60 show the effects of different levels of government equity participation on case-study profitability. Base-case studies were carried out at 50% government equity participation, 30% corporate tax rate, 27.5% withholding tax rate and 5% royalty (Ad valorem). It was observed that increases in the government equity share (58) drastically reduced case-study investor profitability while retaining the aggregate tax parameters constant (Cameroon tax structure is fair) . The profitability increases at lower government shares (58).

It has already been recommended in section 9.1.1-2 that government participation be regarded as a crucial determinant in project profitability and as such should be negotiated generally below the <50% mark. Government should also consider non-participation in sub-marginal or marginal case-studies, and should be content with its fiscal revenues. At 30% government equity share, the Kribi iron-ore and Douala rutile heavy-sands models showed marked profitability improvements (see figs , 58 , 59 , 60); internal rates of return of 1.86 to 4.54 and 8.06 to 12.42 were also respectively registered through the change. By decreasing the level of government equity participation from 50% to 30%, the Fongu-Tungong profitability (NPV) improved by three fold from $25 million dollars to $151.6 million dollars, while that of Mayo-Darle doubled from $368.8 million to $690.2 million dollars. The improved NPVs accompanied by shortened payback periods make these mineral cases automatically more attractive to potential investors because of increased profit potential and reduced risk. In order not to foresake government revenues from these mineral operations through over generous offers to investors, it is suggested that government equity participation be negotiated around the 40% mark for the Fongu-Tungong case-study and zero for the Kribi iron-ore case-study. The Mayo Darle case can be maintained at 50% of government equity participation.

The anticipated income from government participation in the rutile case will be $28.24 million dollars with $57.46 million dollars in tax receipts. The Fongu-Tungong case-study will yield about $326.4 million dollars in equity income and $477.45 million dollars in tax income.

10.1.2 Increase in employment, labour income and training.

A labour force of 1500 workers is envisaged in the four mineral cases, 1200 of whom are Cameroonians and 300 expatriate. Cameroonian workers will earn very attractive salaries (between $4000 dollars per annum or 100,000 CFA Francs per month or $24,000 dollars per annum or 600,000 CFA francs per month) by local standards. Cameroonians will earn as much as $3.87 million dollars per annum in the Mayo-Darle case, $2.11 million in the Kribi case $3.17 million in the Fongu-Tungong case and $2.11 million in the Kribi case $1.4 million dollars in the Kribi case. This excludes expatriate salaries a substantial fraction of which is likely to be spent within the country. In addition, workers will be able to take up residence in modern living conditions (mine town sites).

Provision for in-house training and refresher courses will enable workers to gain new knowledge in addition to the skills they will cultivate at their job sites; skills and expertise which can easily find other applications in industry.

10.1.3 Infrastructure development:

Road and rail construction have been modelled in the four case studies (section 6.2.4-1). In all, 100 kilometers of rail lines will be laid between Nkongsamba and Bafousam so that the latter could become a rail terminus serving the delivery of supplies from Douala and transportation of concentrate to the Edea smelter and the Douala port. Such a rail line will be very desirable because it would facilitate communication between the North Western Province the Bamilake (heartland of cash-crops) plateau, the economic center of Douala (the littoral province), the Central South province and the Adamawa province; thereby greatly stimulating economic transactions along this important economic axis of the country.

Fifty kilometers of trunk road - between Dschang and Fongu-Tungong, Foumban and Nkongsamba will also improve motorable network of roads in this area. Fifty kilometers of trunk road are envisaged in the Douala heavy sands area, 30 kilometers between the iron-ores of Kribi and the maritime port. In all nearly 180 kilometers of new trunk roads will be developed in 4 provinces; which itself is a means of ensuring equitable regional development in the country. $15.63 million dollars will go into road development in Kribi, $15.63 in Douala region, $28.06 million in the Fongu-Tungong Dschang area and $24.87 in the Bafousam - Mayo Darle stretch. These investments in roads and rail infrastructures could be contracted to local entrepreneurs so that more jobs are created and local inputs utilized (timber and road metal).

Three mineral cases envisage the erection of town-sites for their workers. $3.3 million dollars for a mine town-site in Mayo-Darle, $1.8 million dollars for one in Kribi and $2.7 million dollars for one in Fongu-Tungong. Some $7.8 million dollars will go into construction of these social housing facilities.

In the construction of these facilities, local raw materials, timber and woodwork and aluminium corrugated sheets from the sheet manufacturers in Douala or Edea. It is also recommended that port facilities at Kribi be improved in order to facilitate the transportation of iron-ore by large freighters.

10.1.4 Other Government Objectives

The creation of an industrial basis through establishment of domestic processing facilities has been not only a popular view but a development strategy.

Further beneficiation of minerals results in higher valued mineral produce and increases the revenue potential of mineral exploitation concentrate transportation costs are usually minimized by transacting in high graded commodities.

The Fongu-Tungong case-study is recommended to be an integrated bauxite-alumina plant so that alumina can be used as an input into the aluminium smelter at Edea. This will save imported costs and complete the integration of a bauxite-alumina aluminium industry in Cameroon.

Rutile and cassiterite are recommended to be sold in their concentrate forms to smelters and buyers in Western Europe or elsewhere. These are low bulk mineral concentrate with low overseas transportation costs (see tables 18 , 22 , 26 , 14). The iron-ore concentrate needs to be beneficiated to higher quality, preferably steel, before sale; this would increase the selling price drastically (about ten times) and will slash down the dead weight of end produce to be sold abroad. Total overseas transportation costs will drop, while revenues will increase by about ten times. Local movement of concentrate can be contracted to local entrepreneurs while part of the overseas freight can be handled by the Cameroon Shipping Lines. This move will improve the foreign exchange position and also provide more jobs for Cameroonians.

The availability of abundant fuels (gas and oil) in the coastal basin in Douala and Victoria make the study of an iron and steel complex reasonable. It should also be noted that availability of steel is an important pre-requisite and probably the backbone of industrialization.

The greatest foreign exchange gains were produced by the Mayo-Darle case ($1.81 billion dollars); the Fongu-Tungong case produced $720.8 million dollars (see table 27). Implementation of either or both of these projects will generate much foreign exchange receipts to permit the procurement of equipment and plants for the remaining two cases (Kribi and Douala).

Agricultural restoration of about 10km of mined out land will be accomplished. This would create a useful basis for mechanized or semi-mechanized agricultural enterprises, which can be sold out to investors.

10.1.5 <u>Mineral Exploitation Schedule</u>

The Fongu-Tungong bauxite-alumina case-study should be reinvestigated, with the studies aimed at establishing as much as 200 million tons of high-grade bauxite ore (.40% Al_2O_3), having a low silica content 3% $Si\,O_2$. These studies should verify beneficiation chracteristics of the bauxite ore through laboratory and/or pilot plant metallurgical tests. Market studies should aim at confirming the existence of reliable markets for the sale of both alumina and aluminium metal in the short and long term. In the first instance, the Edea alumina smelter will completely absorb alumina produce from the bauxite-alumina complex at Fongu-Tungong while further capacity expansion will result in direct shipment of alumina abroad. This project will, if commenced 4 years after pre-production development investment activities, payback in 5.5 years after the first revenue year (see fig 60 , table 27). If government participation is reduced as recommended in

10.1.2 to 40%, the payback period will fall to less than 4 years, and profitability will improve significantly (see fig 60).

The Mayo Darle case-study is recommended to be the second for implementation after the remaining 50 kilometers of rail line to the anticipated Bafousam rail terminu are laid. It will require an exploration campaign costing about $1 million dollars to delineate more than 80 million tons of (0.7% - 1.0% Sn) low grade porphyry tin ore . Allowing for a pre-production development interlude of 4-3 years, the project will breakeven 3 years after its first revenue accrual. Much effort should be devoted to intensive exploration. This case-study turned out to be the most attractive of the four (see table 27).

Once the first two case-studies are brought into operation, the other two cases (Kribi and Douala) can come on stream simultaneously, thereafter.

It is worth reiterating the sub-marginal nature of the Kribi iron-ore case-study, which needs more study to investigate the suitability of erecting an integrated iron and steel complex and the location of a reliable market for potential steel product. Overseas transportation costs are important in this case-study (37.4% of total operating costs; this is caused by the high-bulk nature of iron concentrates, especially when they have to be traded overseas. Improvement of the Kribi port facilities would be necessary to allow access to large ocean freighters which lower the bulk transportation charges considerably. Another unfavourable factor which might affect this case-study negatively is the poor market conditions that plague the iron and steel industry (96). It is advisable for government to reduce its equity participation to a minimum (<30% equity), even in the alternative case of an iron-steel complex. This would greatly enhance the profitability of the Kribi case-study, which is a vital basis for the attainment of objective 1) an industrial basis.

10.1.6 The question of technocrats

The question of cultivating relevant manpower capabilities for use in operating a healthy national minerals sector of the economy is stated in objective (3); it can partially be achieved through inhouse company training of employees within the country.

Availability of such a varied and large minerals team will require assistance from friendly countries of the developed world, especially for training these manpower needs and supplying expatriate assistance. It would be wise to rely however on indigenous capabilities by establishing home institutions for training these highly specialized manpower requirements. This necessarily means that existing academic institutions such as the Yaounde University and the planned University of Technology at Ngaoundere should be styled to train some of these technocrats by formulating curricula whose contents and quality reflect the scientific and applied scientific needs of such manpower requirements.

10.1.7 Suggestions for future work

There is need for more research studies into the economics of mineral beneficiation. These studies will seek to investigate criteria and beneficiation procedure for concentrating minerals from their ores at early conception phases when information on their occurrence is not plentiful.

More studies are also needed in the investigation of concentrate and metal pricing factors such as the costing of their transportation, storage, smelter charges, insurance and other tangible market factors which influence the pricing of the ultimate mineral produce.

Studies such as these would be helpful in understanding the detailed economics of mineral exploitation for use in mineral management and policy formulating.

In the Cameroonian context, future work should be directed at the collection and preparation of data for the creation of a national data base; upon which an appraisal of exploitability of comprehensive national minerals potential can be carried out.

References cited in text

1. ABECOR (1981)
   Cameroon. A country report prepared by Banque Brusselles Lambert, S.A. December, Brussels.

2. AGARWALA and SINGH (1958)
   The Economics of Under-development (Editors). Oxford University Press.

3. ALUTERV - FKI (1977)
   Report on the Feasibility of the Kibi Bauxite and Alumina Project, April, Būdapest.

4. ARMSTRONG F.W. (1932)
   Models of Mines and orebodies. Transactions of the Institute of Mines and Metallurgy; Volume Number 41, pages 106-131, London.

5. ATKINSON T. (1969)
   Mechanical and Electrical Aspects of Open-cast Mining. Mining Technology. Volume, Number Page 19-40, October.

6. BAILLY P.A. (1976)
   The Problems of Converting Resources to Reserves. Journal of the Institution of Mining Engineers, Volume 27, Number 1 pages 27-37, January, London.

7. BAMBRICK S and MISKELLY, N (1978)
   Mineral Development Decisions and Government Policy, Part I and II, The Australian Mineral Foundation, Adelaide.

8. BAYER P.A. (1981)
Equality, the Third World and Economic Delusion. Weidenfeld and Nicolson, London.

9. BEASLEY C.A. and PFLEIDER E.P. (1972)
Profitability Sensitivity analysis of a mining venture. Tenth International Symposium on Computer Applications in the Mineral Industry. Pages 109 - 114

10. BECHTEL PACIFIC CORPORATION LTD., (1977)
Material Handling in the Mineral Industry. The Australian Mineral Foundation. June 7th-19th Adelaide

11. BENDER F. (1979)
The Mineral Resources Potential of the Earth; Proceedings of the Second International Symposium, April 18-20 Hannover, Federal Republic of Germany.

12. BENNETT H.J. ET AL (1970)
Financial Evaluation of Mineral Deposits using Sensitivity and Probabilistic Analysis Methods; United States Bureau of Mines Information Circular. IC 8495

13. BERTOLDI MICHAEL (1977)
Preliminary Economics of Mining a Thick Coal Seam by Dragline, Shovel-Truck and Scraper Mining Systems. United States Bureau of Mines Information Circular IC8761.

14. BERTRAND RUSSELL (1968)
The Impact of Science on Society. Unwin Books, London.

15. BREALEY S.C. and ATKINSON T. (1968)
Open Cast Mining. Journal of the Institution of Mining Engineers, Volume 128 number 99, pages 147-163, December.

16. BROOKS D.B. (1976)
Mineral Supply as a Stock. "Economics of the Minerals Industries", The American Institute of Mining, Metallurgical, and Petroleum Engineers Inc. (Seeley Mudd Series) pages 127-207, New York.

17. BROWN, ROWLAND and FABER MIKE (1969)
Some Policy and Legal Issues Affecting Mining Legislation and Agreements in African Commonwealth Countries. Published by the Commonwealth Secretariat, London.

18. BRYAN CARSBERG (1964)
Analysis for Investment Decisions. Accounting Age, London.

19. BUTT N.A. and ATKINSON T. (1982)
Shortfalls in Minerals Investment Resources Policy, Volume 8, Number 4; pages 261-276 December.

20. CARMAN J.S. (1979)
Obstacles to Mineral Development; A Pragmatic View. Edited by Benison Varon. Oxford Pergamon Press.

21. CHAMBERLAIN J.A. (1958)
Uses, Design and Construction of Mine Models. The Canadian Mining and Metallurgical Bulletin, Volume 61, Number (552) Pages 219-222, April, Montreal.

22. CHRISTIE P. and DERRY R. (1976)
Aluminium from Indigenous United Kingdom Sources, a review of possibilities; London.

23. CHRONIQUE DE LA RECHERCHE MINIERE (1961)
Nouvelle Legislation Miniere du Cameroon. Volume Number Pages 29-32, May Paris.

24. CHRONIQUE DE LA RECHERCHE MINIERE (1970)
L'etain Exploitation au cameroon volume 38 number 389 398 pages 24 25

25. CHURCH H.C. (1981)
Excavation Handbook. McGraw-Hill Book Company. New York.

26. COATES D.F. (1970)
Rock Mechanics Principles. Monograph 87, Mines Branch of the Canadian Institution of Mining; Ottawa.

27. COBBE JAMES, H. (1979)
Governments and Mining Companies in Developing Countries. Westview Special Studies in Social, Political and Economic Developments; Bolder, Colorado.

28. COHEN E. HENRY (1976)
The Role of Mineral Engineering in Resources Development in Developing Countries - Economics of Mineral Engineering in Developing Countries, Mining Journal Books Limited, Pages London.

29. COLLET FRANCOIS (1950)
Notes on the Rutile Market in the Cameroons Part I and II. Echo. Min. Metall; Volume pages 54-62, 107-109. March, Paris.

30. COURT C.M.W. (1970)
International Significance of Mining Development and Overseas Development. Mining Congress Journal volume 56, Number July.

31. CROWSON PHILIP (1982)
Investment and Future Mineral Production. Resources Policy Volume 8, No 1 pages March, Surrey, U.K.

32. (1982)
Mineral Developments, Financing and Cooperation. New forms of International Participation. Resources Policy Volume 8; Number 2 pages June, Surrey, U.K.

33. DEGARMO E. PAUL, CANADA JOHN R. and SULLIVAN WILLIAM G. (1979)
Engineering Economy. Macmillan Publishing Co. Inc. New York.

34. DIXON C.J. (1979)
Atlas of Economic Mineral Deposits
Chapman and Hall, London.

35. DRAVO CORPORATION (1974)
Analysis of Large-Scale Non-Coal
Underground Mining Methods. United
States Bureau of Mines. National
Technical Information Service Number,
PB 234555

36. ELY NORTHCUTT (1974)
Summary of Mining and Petroleum Laws
of the World (in five parts). United
States Bureau of Mines Information
Circular 8482

37. ENGINEERING AND MINING JOURNAL (1978) E/MJ
Operating Handbook of Mineral Surface
Mining and Exploration, McGraw-Hill.

38. ENGINEERING AND MINING JOURNAL (1967)
Mining's Green Thumb. Simultaneous
Mining-Reclaiming,, Mine Plan for
Total Resources Management Volume 68
Number 7 Pages 77 82 July New York

39. EVAN JUST (1976)
The Production Process. Functions
and Structure of the Mineral
Industry. "Economics of the Mineral
Industries"; The American Institute
of Mining, Metallurgical and
Petroleum Engineers Inc. (Seeley
Mudd Series) Pages 379-402.

40. EVANS A.M. (1980)
Geoscience texts. An introduction to ore geology. Volume 2. Oxford Blackwell Scientific.

41. FRANK A. PETERS, PAUL W. JOHNSON and RALPH C. KIRKBY (1965)
A Cost Estimate of the Bayer Process for Producing Alumina. United States Bureau of Mines, Report of Investigation, RI6730.

42. FREIDENFELDS JOHN AND MICHAEL KENNEDY (1979)
Price, Inflation and Long-term Present Worth Studies. The Engineering Economist Volume 24, Number 3 pages 19-143, University of Florida.

43. FREYBERGER C. AND PFLEIDER E.P. (1971)
Integrated Investment Model to Evaluate Large-scale Mineral Projects. "Proceedings, Council of Economics, American Institute of Mining, Metallurgical and Petroleum Engineers.pages 283 313, march, New York.

44. FRIETSCH H. PAPUNEN, AND F.M. VOKES (1979)
The Ore Deposits of Finland, Norway and Sweden. Economic Geology Volume 74, Number 5 pages 975.

45. GOLDER MOFFITT AND ASSOCIATES LTD (1980)
Confidential Communications on Prefeasibility Project Evaluation. London.

46. GILLIS MALCOLM (1978)
Taxation and Mining. Non-fuel Minerals in Bolivia and other Countries. Ballinger Publishing Company. Cambridge Massachusettes.

47. GILLIS MALCOLM AND RALPH BEALS (1980).
Tax and Investment Policies for Hard Minerals. Public and Multinational Enterprises in Indonesia. Ballinger Publishing Company, Massachusettes.

48. GRJOTHEIM K. AND J.B. SEE (1979)
The Hall-Heroult Process and Alternative Processes for the Manufacture of Metallic Aluminium. Mineral Science Engineering, Volume 11, Number 2, April.

49. HAMILTON DAVID JOHN (1981)
Patterns and Trends in the sources and methods of Mine Finance. M.Sc. Dissertation, Royal School of Mines, Imperial College, London.

50. HAROLD J. BENNETT (1973)
Mineral Policy of Selected Foreign Nations Considering Direct Foreign Investment in the Mineral Industry. Quarterly Journal of the Colorado School of Mines. Volume 68, Number 4 pages 213. October, Colorado.

51. HAWKINS C.J. AND PEARCE D.W. (1971)
Capital Investment Appraisal Macmillan Studies in Economics London.

52. HENRIQUEZ L.N., MACKENZIE B.W. (1977)
Cost Benefit Analysis of a Custom Mill for Small Copper Mines in Northern Chile. Fourth International Symposium on Computer Applications in the Mineral Industry; American Institution of Mining Engineers, pages 1035 - 1057; New York.

53. HINDLEY B. (1979)
Third World Policy towards the Mining Industry. A new International Commodity Regime. Croom Helm Ltd; London.

54. HOEK E. AND BRAY J.W. (1981)
Rock Slope Engineering Institution of Mining and Metallurgy, London.

55. HOLLAND F.A., WATSON F.A. (1979)
Putting Inflation into Profitability Studies; Chapter in "Modern Cost Engineering Methods and Data." Chemical Engineering Magazine, London.

56. HOLLAND F.A., WATSON F.A., J.K. WILKINSON (1979)
Sensitivity Analysis of Project Profitability, "Chemical Engineering Magazine", London.

57. HOOGVELD ANKIE M.M. (1976)
The Sociology of Developing Societies. MacMillan Press Ltd; London.

58. HORNGREN CHARLES T. (1982)
Cost Accounting; a Managerial Emphasis, 5th Edition, Prentice Hall, London.

59. HOSKING K.F.G. (1974)
The Search for Tin-deposits; Fourth World Conference on Tin, 30th Oct. - 5th Nov. International Tin Council, London.

60. (1969)
Aspects of the Geology of the Tin Fields of South-East Asia. Proceedings of a 2nd Technical Conference on Tin - Bankok.

61. HOSKINS J.R, GREEN W.R. (1976)
Mining Industry Costs; North Western Mining Association, Spokane, Western Australia.

62. (1978)
Mineral Industry Costs. North Western Mining Association, Spokane, Western Australia.

63. HUGO FREUND (1966)
Applied Ore Microscopy. Theory and Technique. McMillan Company, New York.

64. IGOR' DMITRIEVICH SAVINSKII (1965)
Probability Tables for Locating Elliptical Underground Masses with a Rectangular Grid. Consultants Bureau, New York.

65. JANSON ERNEST C. (1979)
Financial Reporting and Tax Practices in Non-ferrous Mining. 8th Edition Cooper and Lybrand.

66. JENSEN M.L. and BATEMAN ALAN (1981)
Economic Mineral Deposits Third Edition, John Wiley and Sons. New York

67. JEUNE AFRIQUE (1979)
Atlast de la Republique Unie du Cameroon. paris

68. KATELL S. AND E.L. HEMINGWAY (1974)
Basic Estimated Capital Investment and Operating Costs for Coal Strip Mines. United States Bureau of Mines, Information Circular 8661.

69. KEITH BUCK (1963)
Mineral Development Policy. Mineral Resources Branch, Department of Energy, Mines and Resources, Mineral Information Bulletin MR64, Ottawa, Canada.

70. (1972)
Mineral Economics. Its Definition and Application. Mineral Resources Branch Department of Energy, Mines and Resources, Ottawa, Canada.

71. KEITH BUCK AND ELVER R.B. (1970)
An Approach to Mineral Policy Formulation. Information Bulletin MR108, Mineral Resources Branch, Department of Energy, Mines and Resources, Ottawa, Canada.

72. KIRCHNER AND SIX OTHERS (1977)
Mining Ventures in Developing Countries. Interests, Bargaining Process and Legal Concepts. Studies in Transnational Law of Natural Resources. Mahoney W.J. Vol. 1.

73. KRIGE D.G. (1971)
The Impact of Taxation Systems on Mine Economics. Decision-making in the Mineral Industry, CIM Special volume 12, pages 283-288 , Montreal.

74. KURSTEN MARIAN O.C. (1983)
The Role of Metallic Mineral Resources for Countries of the Third World. Natural Resources Forum Volume Number Pages

75. KUZNETS SIMON (1966)
Modern Economic Growth Rate, Structure and Spread. Yale University Press.

76. LAPLAINE L. (1969)
Mineral Occurrences and Resources of Cameroon. Mines Department of the Ministry of Mines and Power, Yaounde - Cameroon.

77. LANE F. (1964)
Choosing the Optimum Cut-off grade. Quarterly of the Colorado School of Mines. Volume 59 Number 4 Pages 811-829 October, Colorado.

78. LEE CALAWAY AND W.C.J. VAN RENSBURG (1982)
U.S. Strategic Minerals Policy Options Resources Policy Volume 8, Number 2, June, Surrey, U.K.

79. LEGOUX PIERRE (1978)
Mining Legislation of French Speaking Countries. Jurid. Polit. Indep. Coop; Volume 32, Number 2, pages 647-802, June.

80. (1981)
Legal aspects of mining development in developing countries. Natural Resources Forum Volume 5 Number 2 pages167 179 April New York

81. LEO BORASIO (1963)
How Mine Models are used to Design Mineral Processing Plants. Engineering and Mining Journal Volume 164 Number 7 Pages 73-79. July.

82. LENTZ O.H. (1960)
Mineral Economics and the Problem of Equitabl Taxation. Quarterly Journal of Colorado School of Mines. Volume Number Pages.

83. LOTHER BUCHTA (1982)
Open-pit bench blasting. How to improve results. World Mining Volume Number6 Page 64 June.35

84. MALEY TERRY S. (1979)
Handbook of Mineral Law (Second Edition Revised). MMEC Publications, Boise, Idaho.

85. (1978)
Studies in Transnation Law of Natural Resources, Mining Venture in Developing Countries, Bargaining Process, Legal Aspects.

86. MANULA C.B., RIVELL R.A. (1974)
A Master Design Simulator, Proceedings of the Twelth International Symposium on the Application of Computers in the Mineral Industry American Institution of Mining Engineers, Pages G200 ; New York.

87. MARMO V. (1965)
Natural Resources Research; Proceedings of the Symposium on Granites of West-Africa - Cameroon, Nigeria, UNESCO VIII, March.

88. MARSHALL AND SWIFT (1983)
Economic Indicators and Equipment Cost Index. Chemical Engineering, Volume Number , London.

89. McCLAY J. (1974)
A Direct Costing Model for Mining Systems. Msc. report, Pennsylvanian State University, College of Earth and Mineral Sciences; Pennsylvania.

90. McDIVITT AND W.G. JEFFERY (1976)
Mineral and the Developing Economies. Economics of the Mineral Industries; The American Institute of Mining, Metallurgical and Petroleum Engineers Inc. (Seeley Mudd Series) Pages 920-930. New York.

91. McNAMARA ROBERT (1980)
Report of President of the World Bank to the Board of Governors, September 30th, New York.

92. METAL BULLETIN (1979)
Proceedings of Metal Bulletin's First International Iron Ore Symposium - March, Amsterdam.

93. METAL STATISTICS (1971-1981)
Metallgesellsehaft, Aktiengesellschaft. Frankfurt, Germany.

94. MEIER GERALD (1976)
Leading Issues in Economic Development. Oxford University Press, New York.

95. MERCIER M.A., GUILLMIN M.F., TYMEN M.A. (1960)
Industrial Development in Cameroon. Society for Social and Economic Study S.E.DL.E.C., PARIS.

96. MINING ANNUAL REVIEW (1983)
June, London.

97. MINISTRY OF MINES AND POWER CAMEROON (1973-81)
Mineral Activities in Cameroon - Annual Bulletins of Information, Yaounde, Cameroon.

98. MITCHELL A.H.G., GARSON (1981)
Mineral Deposits and Global tectonic settings. Academic Press.

99. MURPHY JOHN (1980)
Personal Communications. United States Department of the Interior; Bureau of Mines, Pittsburg Research Center, September Pennsylvania.

100. MORAWIETZ F.H. (1968)
A Study of the Cassiterite and Wolframite, Deposit of Mayo-Darle Cameroon. Saltzgitter Industribau Geselschaft, MBH, February, Germany.

101. MULAR A.L. (1978)
Mineral Processing Equipment Costs and Preliminary Capital Cost. Estimations, CIM, Special Volume 48, Vancouver, Canada.

102. NATIONAL COAL BOARD (1983)
A Report on the Efficiency and Costs in the Development Production and Supply of Coal by the NCB. The Monopolies Commission, June, London.

103. NGOMBE NDOUMBE E. (1981)
Personal Communications. Documentation Department, in the National Chamber of Commerce, February, Douala.

104. NICKEL P.E., GILLIES I.R., HENLEY T.J., SAUNDERS J.O. (1978).
Economic Impacts and Linkages of the Canadian Mining Industry. Center for Resources Studies, Kingston Ontario.

105. O'HARA ALAN T. (1980)
Personal Communications. Hudson Bay Mining and Smelting Co. Ltd, Canadian Metals Division, April.

106. (1980)
Quick Guides to the Evaluation of Orebodies. Canadian Institute of Mining Bulletin Volume 87 Pages 87-99

107. OJONG EMMANUEL (1978)
Tin Exploitation: An Appraisal of the Mining and Processing of Tin Ores. Msc. Dissertation, University of Leicester, Leicester.

108. (1980/81)
TECHNICULTURE: The Cultivation of Technology. Essays on development science (unpublished) Department of Mineral Resources Engineering, Royal School of Mines, June, London.

109. (1981)
Data Acquisition from Cameroon. Internal report, Department of Mineral Resources Engineering, Royal School of Mines, London.

110. (1982)
Procedures for Quantitative Appraisal of Mineral Pre-Projects, Mineral Development and Policy Formulation. Internal Report, Royal School of Mines, London.

111. (1983)
Report of a Study tour of 3 Norwegian Mineral Projects. Department of Mineral Resources Engineering, Royal School of Mines, London.

112. OKEUDO OGAN EKE (1982)
The Nigerian Iron and Steel Industry. An aspect of raw materials supplies economics. Msc. dissertation, Department of Mineral Resources Engineering, Royal School of Mines, London.

113. O'LEARY J. and BUTLER (1978)
Metal Price Forecasting - a Mathematical Approach. Mining Magazine, Volume 138, Number 5 pages 469-473, May.

114. OTTLEY J. DEREK (1979)
TECHNICAL, Economic and Other Factors in the Gravity Concentration of Tin; Tungsten, Columbium and Tantalum Ores. Mineral Science Engineering, Volume 11, Number 2 April, London.

115. PRESIDENCY OF THE REPUBLIC OF CAMEROON (1961)
A plan for Mineral Investigation in Cameroon. Minutes on deliberations regarding the future of Mineral Exploitation in Cameroon. Yaounde.

116. PRESTON DAVIS R. (1980)
Personal Communications on Procedures for Mineral Investment Appraisal. Ketro. Inc. Philadelphia Operations, October Pennsylvania.

117. P.T. BROKEN HILL PROPRIETARY (1971)
Contract of Work Between the Government of the Republic of Indonesia and P.T. Broken Hill Proprietary, Indonesia. Confidential Communications.

118. PURGLIESE J.M. (1972)
Designing blast patterns using Empirical formulae; United States Bureau of Mines Information Circular IC8550.

119. RAGUIN, E. (1961)
Geologie des gites Mineraux. Masson et cie; PARIS.

120. RADETZKI MARIAN (1983)
Long-run Price Prospects for Aluminium and Copper, National Resources Forum Volume 7, Number 1, January London.

121. RICH P.J.H. (1980)
Future of Tin as a Tonnage Commodity, Institution of Mining and Metallurgy, Section A8-A17January, London.

122. ROBINS P. (1982)
Guide to Non-ferrous metals and their markets. 3rd Edition Kogan Page Ltd. London.

123. ROBERTS A. (1981)
Applied Geo-Technology. A Test for Students and Engineers on Rock Excavation and Related Topics, MacKay School of Mines, University of Nevada, Nevada.

124. ROUTHIER, P. (1963)
Les Gisements Metalliferes: Geologie et Principes des Recherce. TOME II. Masson et cie, PARIS.

125. SCHMIDT HELMUT (1982)
Regional Distribution of Mining Production and Reserves of Mineral Commodities in the World. Federal Institute of Geo-Sciences and Natural Resources. January, Hannover, Germany.

126. SCHWEYER H.E. (1964)
Analytical Models of Managerial and Engineering Economics. Chapman and Hall Limited, London.

127. SHEI EDWARD (1981)
Personal Communications, Mineral matters in Cameroon. Chief of Services at the Mines Division of the Ministry of Mines and Power, January, Yaounde.

128. SIDERI S. (1980)
Mining for Development in the Third World. Multinational Corporation, State Enterprises and the International Economy.

129. SPENCER RALPH (1979)
MECON Computer Package. Department of Mineral Resources Engineering, Royal School of Mines, London.

130. (1980)
Lignite Liquefaction Project, An Economic Evaluation for the Moonie Oil Company Limited. Confidential Communications.

131. STERMOLE F.J. (1980)
Economic Evaluation and Investment Decision Methods. 3rd Edition, Golden Colorado.

132. STEWARD R.M. and KENNEDY B.A. (1971)
The Role of Slope Stability in the Economics, Design and Operations of Open-pit Mines. Proceedings of 1st Symposium on Stability in Open-pit Mining, American Institution of Mining Engineers pages 5-2, New York.

133. STRAAM ENGINEERS (1978)
Capital and Operating Cost Estimation Handbook for Mining and Beneficiation of Metallic and Non-Metallic Minerals Except Fossil Fuels in the United States and Canada. National Technical Information Service, No.PB 277348 New York.

134. STRAHLER N. ARTHUR (1976)
Principles of Earth Science. Harper and Row Publishers, New York.

135. STUMPF K (1971)
Raw materials Quarrying as a first step in Cement Production. Paper in Zement - Kalk - Zips, Special Issue number 14, pages 1 - 7, Dusseldorf, Germany.

136. SZENTES TAMAS (1978)
The Political Economy of Underdevelopment. Akademiai Kiado, Budapest.

137. TAGGART A.M. (1951)
Elements of Ore dresing. John Wiley and Sons, New York.

138. TANKA (1981/82/83)
Personal Communications on the State of the Cameroonian Economy. Economic Mission at the Cameroon Embassy, London.

139. TOOMBS R.B. and ANDREWS P.W. (1976)
Minerals and the Modern Industrial Economies. Economics of the Mineral Industries; American Institution of Mining Engineers (Selley Mudd series) Pages 33 72 New York

140. UNITED NATIONS (1970)
Mineral Resources Development with Particular Reference to the Developing Countries, United Nations Conference on Trade and Development, New York.

141. (1978)
The Future of Small-scale Mining United Nations Institute for Training and Research (UNITAR). December 5th. New York. York.

142. (1978)
Development in the 1980s - Approach to a New Strategy. Views and Recommendations of the Committee for Development. United Nations Economic and Social Council, New York.

143. (1981)
Proceedings of the First Regional Conference on the Development and Utilization of Mineral Resources of Africa. United Nations Economic Commission for Africa, ARUSHA.

144. (1981)
Mineral Resources Development. The Role of the United Nations. United Nations Secretariat. Natural Resources Forum Volume 5, Number Pages London.

145. (1983)
Recent Demand and Supply and Price Trends for Minerals and Metals. United Nations Secretariat, Natural Resources and Energy Newsletter Volume 7, Number 1/2 March and May New York

146. UNITED STATES BUREAU OF MINES (1980)
Mineral Facts and Problems, United States Department of the Interior. Bureau of Mines Bulletin 671, Washington D.C.

147. (1958)
Iron-Ore in Cameroon. Mineral Trade Notes, Volume 47 Number July Washington D.C.

148. (1960)
Bauxite in Cameroon. Mineral Trade Notes; Volume 50 Number pages 4-5, May, Washington D.C.

149. (1983)
Minerals Commodity Summaries. United States Department of the Interior, New York.

150. (1974)
Minerals Availability System (MAS). Information Circular 8654; New York.

151. VAN HORNE JAMES (1977)
Financial Management and Policy. Prentice Hall International, London.

152. VAN RENSBURG W.C.J. and BAMBRIK (1977)
Applied commodity Economics for the Mineral Industry. Australian Mining Foundation Inc., 18th-22nd July Adelaide.

153. VAN RENSBURG W.C.J. 81982)
The Relationship between Resources and Reserves. Resources Policy Volume 8, Number 1, March, Surrey, U.K.

154. VUTUKERI V.S. and Bhandaris (1961)
Some Aspects of Open-pits. Colorado School of Mines Quarterly Volume 56 pages 55-61, Colorado.

155. WATSON JOHN (1979)
Blasting Vibrations. First Report to the National Coal Board, Open-cast Executive, Royal School of Mines, August, London.

156. WARNOCK DAVIES (1983)
Project Evaluation. The Neglected Variables. Engineering and Mining Journal Volume Number Pages 23 25 London.

157. WESTON J.F., BRIGHAM E.F. (1981)
Managerial Finance. Seventh Edition. Holt-Saunders International Editions; London.

158. WHITE C.G. (1969)
A rock Drillability Index. Quarterly of the Colorado School of Mines. Volume 64 Number 2 April, Colorado.

159. WILSON E.B. and NOTLEY K.R. (1977)
Application of Three Dimensional Mine Drawing System. Canadian Mining Journal, Volume 98, Number 6, pages 36-40, Ottawa.

160. WOLFGANG JACOBS, THOMPSON JAMES (1967)
BONG-Range Iron-Project. Engineering and Mining Journal, Volume168 Number4 Pages100-111April

161. WOODCOOK J.T. (1980)
Mining and Metallurgical Practices in Australasia. The Sir Maurice Mawby Memorial Volume, Monograph Series No. 10. The Australasian Institute of Mining and Metallurgy. Victoria, Australia.

APPENDIX I

SUMMARY GEOLOGY OF CAMEROON

(1) Geology

The geology of Cameroon is not complex, it is described below under five sub-headings.

(a) Basement complex
(b) Metamorphic formations
(c) Plutonic formations
(d) Volcanic formations
(e) Sedimentary formations

(a) Basement complex. The basement complex is identical to the great "African Shield" which constitutes most of the basement rocks of the entire continent. These rocks are mainly schists, mica-schists, gneisses, migmatites - anatexitic granites and syenites (granodiorites) which date between 2,500 million years to 800 million years of age. Almost 2/3 of the entire territory of Cameroon is covered by these basement rocks, on outcrop. The Ntem-complex is attached to these basement rocks. It is a highly metamorphosed granulite of calco-magnesian composition, with the pyroxenites and amphibolites being regarded as younger in metamorphic chronology.

(b) Metamorphic formations. They are all Precambrian or Cambrian in age and are denominated according to name of locality of occurrence. These rocks are given an age of about 600 million- 500 million years:

(5) Dja Series - tillites, quartzites, schists marble and dolerites
(4) Lom Series - schists and mica-schists
(3) Poli Series - schists, mica-schists and gabbro
(2) Mbalamayo Bengbis Series - schists and quartzites
(1) Ayos Series - schists and quartzites

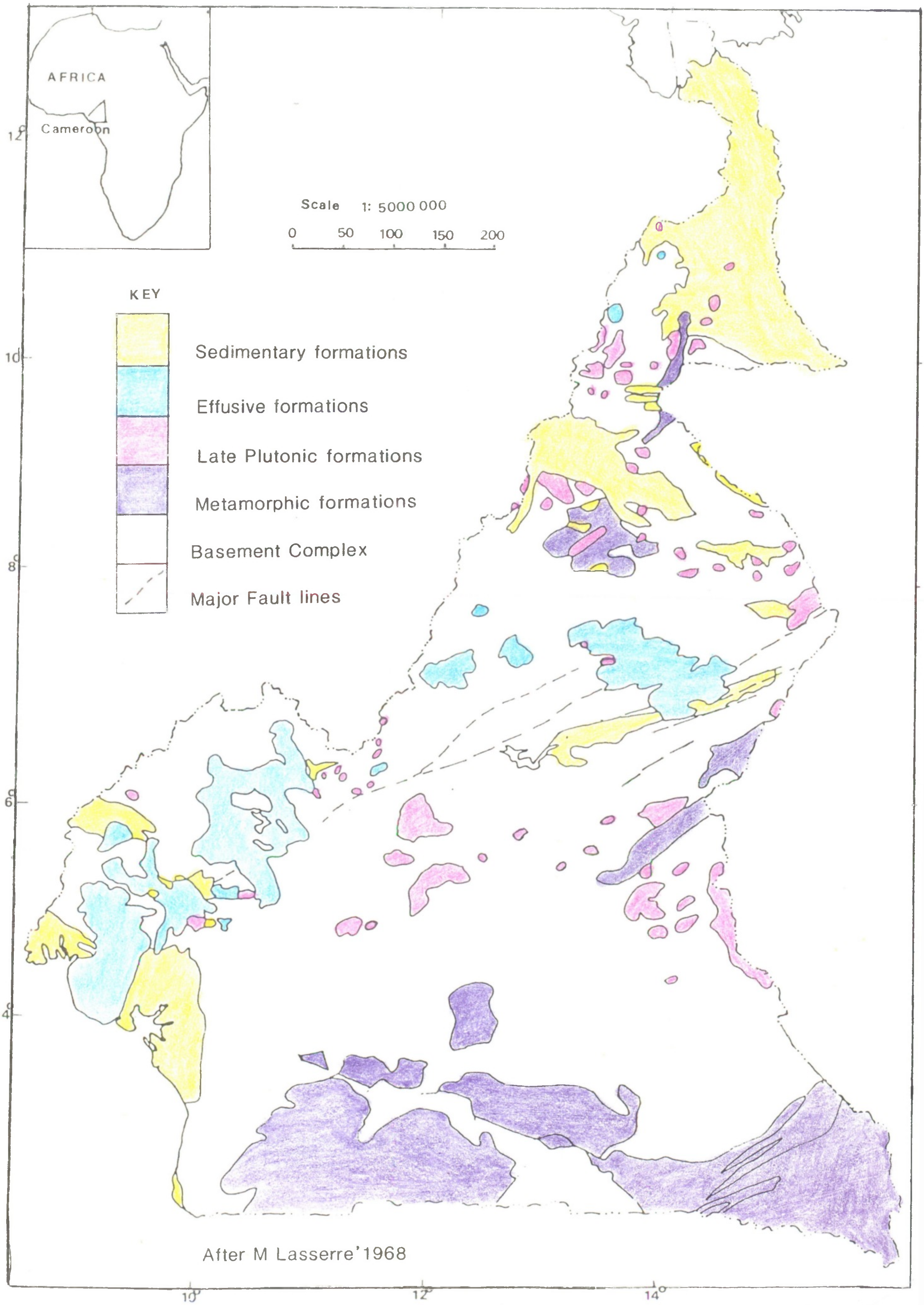
(c) Plutonic formations. Plutonic formations are widespread in the north and north-central part of the country. Classification of these plutons into the "serie ultime" and "ancien serie discordant/subconcordant syntectonique" has occasioned some confusion in the study of plutonic rocks,due probably to the granodioritization of precambrian basement rocks with anatexitization in places ,palingenesis and differentiation,giving rise to several petrographic classes. The presence of water would have played an important part in these processes especially in mobility of rocks (see proceedings of symposium on W. African granites. .). The first series is the Younger granites of the Jurassic, properly called post-tectonic granodiorites, and they are thought to be tin bearing (90-180 million years). The second and older series is a ociated with the basement complex as undifferentiated parts of the evolutive series. They are (440 million-570million years) Cambrian to Ordovician in age.

(d) Volcanic formations. There is an active volcanic apparatus which runs across Cameroon from the volcanic islands of Principe, Sao Tome, Fernando Po in the Atlantic Ocean. In Cameroon, this volcanic line is composed of Mount Buea (4070m) a giant strato-strombolian cone made up of volcanic and pyroclastic debris and basalts, the Manengouba (2396m), Koupe (2050m), Bamboutos (274m), Oku (3008m) and the Mbam (2335m).

Volcanic formations from the lava flows are grouped as the lower black series which belongs to the upper cretaceous and composed of basalts impregnated with olivine crystals. The second and middle series is called the white median series. It is made up of felsic rocks, rhyolites, trachytes, ignimbrites and phonolites. The age of the rocks is about 29 million years-8 million years (oligo-miocene). The third term is the recent basaltic flows most of which were deposited in 1922, 1954 and 1959. It is called the upper black series.

(e) Sedimentary formations. These are typically unmetamorphosed sediments generally found in coastal sedimentary basins such as in Campo, Douala, Rio-Del-Rey, Benoue, Manyu and the Chad area. Througout the coastal sedimentary basins, sediments are similar and monotonous - sandstones, clays, and basal conglomerates. The Lake-Chad basin of Cameroon is mainly made up of quaternary deposits of sands and alluvions. As for the Manyu basin, it is dominated by basal sandstones of the Albian and Aptian. The Garoua basin shows marine characteristics in many places with volcanic intercalation. It is cretaceous in age.

Fig 62 Map of a Summary Geology of Cameroon



APPENDIX II

Nomenclature of Notations

n = Number of benches and berms

b = Berm width

h = Bench-height assumed perpendicular to berm

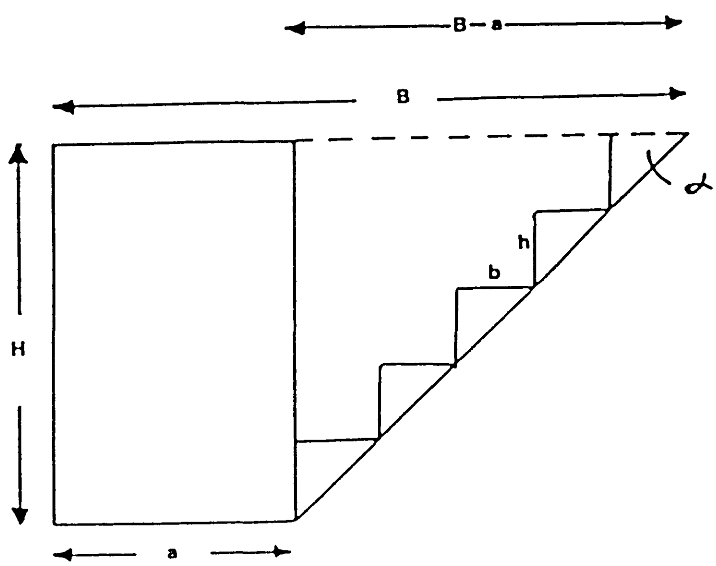
a = Radius of orebody

A = Surface area of projected right cylinder

B = Radius of pit-limit from orebody

α = Ultimate pit-angle

Fig 63 Idealized pit-model for excavating a cylindrical orebody.



The model aims at establishing relationships for ore and waste material with other open-pit parameters such as the bench height, surface area of ore outcrop (A) and the Ultimate pit angle (α).

Assumptions made in the model are that, the orebody represented by a vertical cylinder is approximate to a vertical orepipe (diamond pipe). The open-pit model will thus contain the orebody so that after the last bench is mined pit bottom approximates ore diameter. The pit itself becomes similar to an inverted/truncated cone with sides sloping at α° to the horizontal.

According to fig63 ,

$$\tan\alpha = \frac{H}{B-a}$$

$$n = \frac{B-a}{b}$$

$$n = \frac{H}{h'}$$

$$\pi = \frac{A}{a^2}$$

(a) Volume of ore and waste

The volume of the open-pit (ore/waste)

$$Vo/w = \pi a^2 \; H + \frac{1}{3} \; H \pi \frac{(B-a)^2}{H}$$

$$= \frac{1}{3} \pi H \; [3a^2 + (B-a)^2 \;]$$

$$= \pi H \; [a^2 + \left(\frac{nb}{\sqrt{3}}\right)^2]$$

Let $X = \left(\frac{nb}{\sqrt{3}}\right)$

$$Vo/w = \frac{A}{a^2} \; H \; [a^2 + \left(\frac{nb}{\sqrt{3}}\right)^2]$$

$$Vo/w = AH \; [1 + \frac{X^2}{a^2}] \qquad \ldots (1)$$

(b) Volume of waste

$$Vw = \frac{1}{3}.H \quad (H \cot \alpha)^2$$

$$= \frac{1}{3}.H.\pi \left[\frac{B-a}{H}\right]^2$$

$$= \frac{1.\pi.H}{3} (nb)^2$$

$$= \pi H \left[\frac{nb}{\sqrt{3}}\right]^2$$

$$= \frac{A}{a^2} .H.X$$

$$Vw = AH \cdot \frac{X^2}{a^2} \qquad \ldots (2)$$

(c) Volume of ore (ore/waste - waste)

$$Vo = Vo/w - Vw$$

$$= (1) - (2)$$

$$Vo = AH \qquad \ldots (3)$$

(d) Waste/ore ratio

The fundamental ratio between the total volume of waste versus ore is given as

$$Vw/Vo = \left[\frac{nb}{\sqrt{3}a}\right]^2$$

$$W.O.R. = \left[\frac{nb}{\sqrt{3}a}\right]^2$$

$$\sqrt{W.O.R.} = \text{Root of waste/ore ratio}$$

$$= \frac{nb}{\sqrt{3}a}$$

$$= \frac{H \cdot \cot \alpha \cdot \sqrt{\pi}}{\sqrt{3 \cdot A}}$$

$$= \frac{H \cdot \cot}{\sqrt{A \cdot 3/\pi}}$$

let $\sqrt{3/\pi} = 1$

$$R.W.O.R. = \frac{H}{\sqrt{A}} \operatorname{Cot} \alpha$$

These relationships were studied experimentally to see if the trend observed in experiment, accorded with empirical field observations.

Fig 64 is a presentation of trends for pit angle versus waste to ore ratio variations. The trend shows a rapid decrease in the waste to ore ratio (N) as the pit angle (α) is increased.

This trend is found to be true in most open-pit models, and Hoek and Bray (54) have demonstrated the effects of shrinking pit angles to overall excavation economics.

The effect of ore height/area versus waste to ore ratio has been presented in fig 65 .
For a given pit angle, an increase in $H/\sqrt{A}$ results in a corresponding increase in waste/ore ratio. This increase is faster for smaller slope angles than for larger slope angles.

These two experiments were carried out for both the fundamental waste to ore ratio and the root of the waste to ore ratio. Results were identical.

The preceeding ratios of waste to ore were actually inclusive of bench-rock. A second assumption is made with the bench-rock or waste being excluded from the waste rock estimates.

$$V_w^b = AH \left[\frac{X}{a}\right] - \frac{nh'\pi b^2}{3}$$

$$= AH \left[\frac{X}{a}\right] - \frac{H\ A\ b^2}{3a^2}$$

$$= AH \left[\left(\frac{x}{a}\right)^2 - \left(\frac{b^2}{3a^2}\right)\right]$$

$$= AH \left[\frac{(nb)^2}{3a^2} - \frac{b^2}{3a^2}\right]$$

$$= \frac{A \cdot H \cdot b^2}{3a^2} \cdot [n^2 - 1]$$

$$V_w^b / Vo = \frac{b^2}{3a^2} \cdot (n^2 - 1)$$

$$= \left(\frac{nb}{\sqrt{3}a}\right)^2 \cdot \left(1 - \frac{1}{n^2}\right)$$

$$V_w^b / Vo = \frac{X^2}{a^2} \cdot \left(1 - \frac{1}{n^2}\right)$$

Waste to ore ratio ex-bench rock.

In this ratio, the bench-rock proportion tends to diminish infront of the overall waste component when the number of benches (n) increase . For small number of benches, the excess bench-rock proportion tends to diminish the waste to ore ratio significantly. As such bench-rock waste should be included in waste ratios when number of benches is inferior to say 6.

Fig 64 Waste to Ore Ratio versus Pit Slope Angle

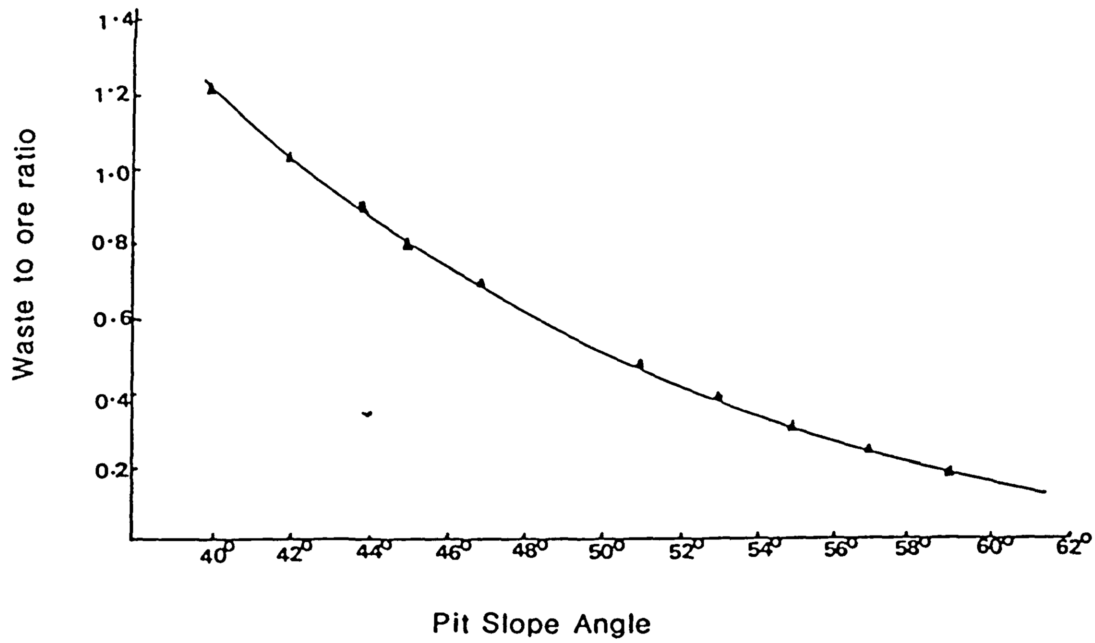
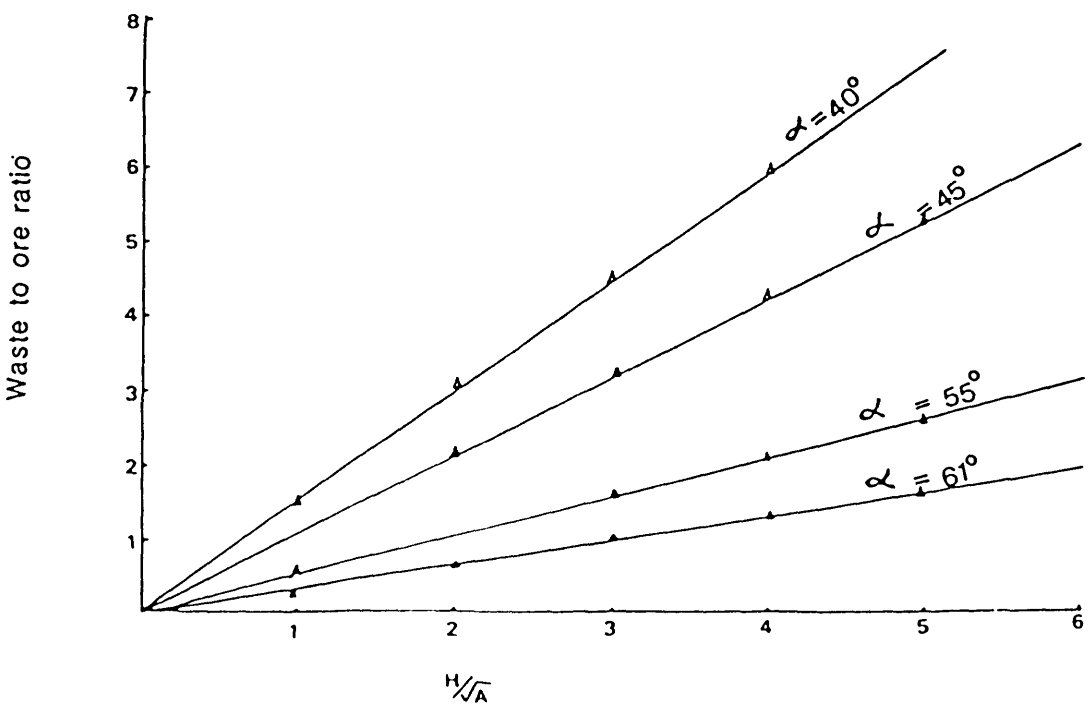


Fig 65 waste to ore ratio versus H/√A



APPENDIX III

EXTRAPOLATION OF BREAK-EVEN VALUES FROM BASE-CASE VALUES

The concept of similar triangles has been used in deriving relationships for use in the extrapolation of break-even values from their base-values.

Fig is a plot of Net Present Values (NPVs) A,C,D,B against their corresponding percentage changes of the base value in question, Δ_1 and Δ_2 The lines A C represent an extrapolation of NPVs between the two values A and C, with A being positive ($+NPV_1$) and C($-NPV_2$) negative. The break-even point is estimated by construction to pass through (X).

Δ_1 and $\Delta_2$correspond to the percentage change of base-parameter causing an NPV of value A and C respectively.

A second condition is assumed where Δ_1 and Δ_2 (percentage of parameter change) results in NPVs at D and B. D is negative and transits via X(NPV=0) to B which has a positive NPV.

The triangles $A\ X\ \Delta_1 \equiv ACD$

$B\ X\ \Delta_2 \equiv BDC$

therefore $$\frac{A\Delta_1}{AD} = \frac{\Delta_1 X}{DC}$$

and $$\frac{B\Delta_2}{BC} = \frac{\Delta_2 X}{CD}$$

But AD = BC

and DC = CD

Also, let $A\Delta_1 = NPV_1$

$AD = |NPV_1 - NPV_2|$

$DC = |\Delta_2 - \Delta_1|$

therefore $\Delta_1 X =$,

$$= \frac{A\Delta_1 \cdot DC}{AD}$$

$$= \frac{NPV_1}{|NPV_1 - NPV_2|} \cdot |\Delta_2 - \Delta_1|$$

At X, the new percentage change corresponding to an NPV = 0 is:

$$\Delta\% = \Delta_1 + \Delta_2 X$$

If the NPV changes sign from positive to negative while transiting the NPV=0 condition, then the above relationship is applicable, if on the other hand, the NPV changes sign from negative to positive as in the DB extrapolation, then:

$$\Delta\% = \Delta_2 - \Delta_2 X$$

where $\Delta_2 X$

$$= \frac{B\Delta_2 \cdot CD}{BC}$$

$$= \frac{NPV_2}{|NPV_1 - NPV_2|} \cdot |\Delta_2 - \Delta_1|$$

The break-even parameter value being extrapolated becomes equal to the relationship:

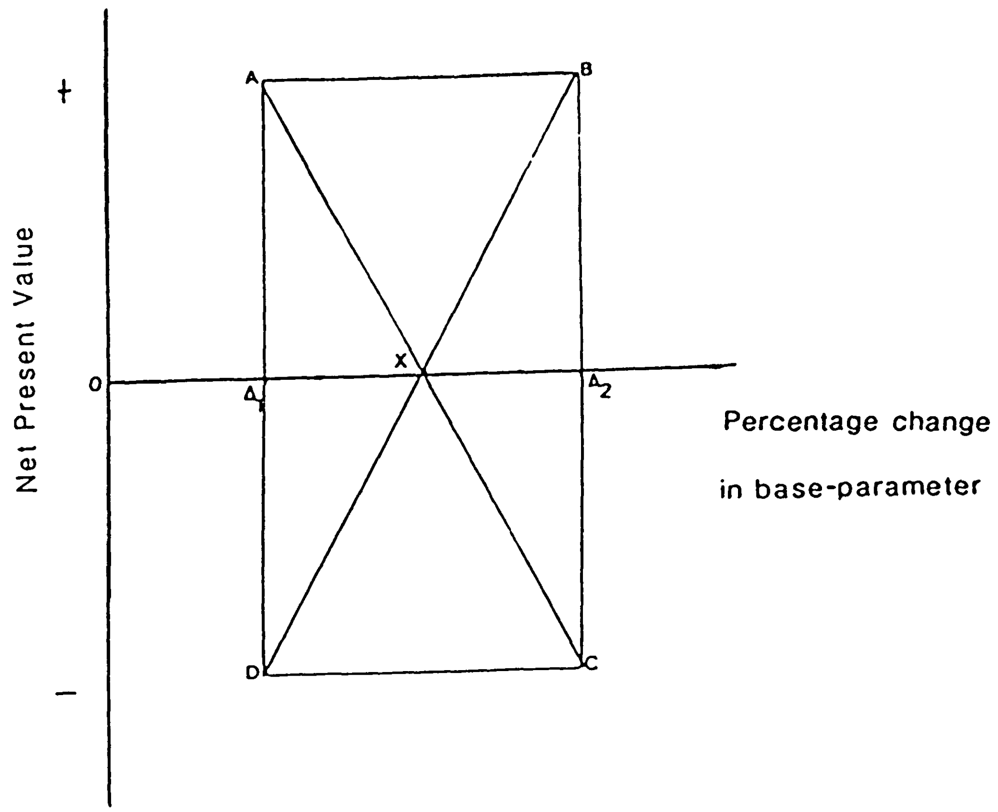
$P_{BE} = (100 - \Delta\%)\ Po$

where P_{BE} = Break-even parameter value

Po = Base-case parameter value

$\Delta\%$ = Breakeven parameter change in value

Fig 66 Extrapolating Break-even Values from Base-case Values



APPENDIX IV

	YEAR 83	YEAR 84	YEAR 85	YEAR 86	YEAR 87	YEAR 88	YEAR 89	YEAR 90
***BAUXITE GRADE AL2O3	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00
***ALUMINA PRICE(99.5)/T	0	0	0	0	.4832E+03	.5315E+03	.5846E+03	.6431E+03
***DRILL-DIAMETER INCHES	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01
***ORE-OUTPUT SCHEDULE TPD	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04
***ORE/WASTE EXC.FACTOR	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01
***ALUMINA MILL RECOVERY	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00
-C-EXPLOR AREA SQ.METERS	.4778E+07	.4778E+07	.4778E+07	.4778E+07	0	0	0	0
-C-EXPLORA DRILL/ACQ.COSTS/	.1948E+07	.1948E+07	.1948E+07	.1948E+07	0	0	0	0
-C-GEOLOGY/AERIAL PHOTO /	.7865E+05	.7865E+05	.7865E+05	.7865E+05	0	0	0	0
-C-SEISMIC,CORING,GEOCHEM /	.1798E+05	.1798E+05	.1798E+05	.1798E+05	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 1	.5820E+04	.5820E+04	.5820E+04	.5820E+04	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 2	.4887E+05	.4887E+05	.4887E+05	.4887E+05	0	0	0	0
***EXCHANGE RATE CFA//	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01
-C-TONS OF ORE/BLAST HOLE	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04
-C-REQ.NO.OF BLAST HOLES/DAY	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01
-C-REQ.NO.OF DRILLS/COMPRESS.	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00
-C-TONS ORE/FOOT DRILLED	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02
-C-SHOVEL LOADERS REQ.	.2095E+01	.2095E+01	.2095E+01	.2095E+01	0	0	0	0
-C-HAULING TRUCKS REQ.	.6983E+01	.6983E+01	.6983E+01	.6983E+01	0	0	0	0
-C-ANNUAL TONS EXP.REQ.	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04
-C-ANNUAL DRILL BITS REQ.	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02
-C-ORE/WASTE MOVED PER ANNUM	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05
-C-DAILY MILL OUTPUT	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04
-C-TOWN-SITE CAPEX/	.2709E+07	.2709E+07	.2709E+07	.2709E+07	0	0	0	0
-C-ANCILLARY.VEHICLES /CAPEX	.1968E+07	.1968E+07	.1968E+07	.1968E+07	0	0	0	0
-C-SH-TR-DR CAPEX/	.9246E+07	.9246E+07	.9246E+07	.9246E+07	0	0	0	0
-C-MINE ACCESS/STRIP CAPEX/	.3799E+07	.3799E+07	.3799E+07	.3799E+07	0	0	0	0
-C-ALUMINA/STEAM PLANT CAPEX/	.1486E+09	.1486E+09	.1486E+09	.1486E+09	0	0	0	0
-C-EXPLORATION CAPEX /	.2099E+07	.2099E+07	.2099E+07	.2099E+07	0	0	0	0
-C-DEH2O/FILT/THICK/CAPEX /	.1814E+07	.1814E+07	.1814E+07	.1814E+07	0	0	0	0
-C-TOT.M+P SUPPL.OPEX//ANN.	0	0	0	0	.5837E+06	.5837E+06	.5837E+06	.5837E+06
-C-SH-TR-DR FUEL OPEXANN/	0	0	0	0	.8322E+06	.8322E+06	.8322E+06	.8322E+06
-C-CONC/SUPP.TRANS.OPEX//ANN.	0	0	0	0	.2076E+08	.2076E+08	.2076E+08	.2076E+08
-C- DENSEMED/DRY OPEX //ANNUM	0	0	0	0	.3976E+05	.5232E+04	.3139E+04	.4185E+04
-C-EXPART.SALARIES //ANNUM	0	0	0	0	.2322E+07	.2322E+07	.2322E+07	.2322E+07
-C-CAMEROON SALARIES //ANNUM	0	0	0	0	.3168E+07	.3168E+07	.3168E+07	.3168E+07
-C-MAINT./REPAIR OPEX/ANN.	0	0	0	0	.5548E+06	.5548E+06	.5548E+06	.5548E+06
-C-CRUSHING OPEX PER ANNUM/	0	0	0	0	.3732E+06	.3732E+06	.3732E+06	.3732E+06
-C-GRINDING.OPEXS//	0	0	0	0	.7048E+06	.7048E+06	.7048E+06	.7048E+06
-C-GRAVITY SEP OPEX//ANNUM	0	0	0	0	.3367E+05	.3367E+05	.3367E+05	.3367E+05
-C-ELEC/MAG/DEH2O OPEX //ANNUM	0	0	0	0	.9450E+05	.9450E+05	.9450E+05	.9450E+05
-C-FILT/THICK OPEX //ANNUM	0	0	0	0	.4684E+05	.4684E+05	.4684E+05	.4684E+05
-C-TOTAL MILL CAPEX/	.1486E+09	.1486E+09	.1486E+09	.1486E+09	0	0	0	0
-C-SALARIES AND WAGES//ANN	0	0	0	0	.5490E+07	.5490E+07	.5490E+07	.5490E+07
-C-PLANTS/INFRST/DEV.CAPEX/	.3522E+07	.3522E+07	.3522E+07	.3522E+07	0	0	0	0
-C-TOT.MILL+TRANS OEX//ANN.	0	0	0	0	.2184E+08	.2184E+08	.2184E+08	.2184E+08
-C-MINE OPEX/ /ANN	0	0	0	0	.1971E+07	.1971E+07	.1971E+07	.1971E+07
-C-MILL UTILITIES/SHOPS CAP/	.2813E+07	.2813E+07	.2813E+07	.2813E+07	0	0	0	0
-C-START-UP,WORK CAPEX/	0	0	0	0	.3259E+06	.3259E+06	.3259E+06	.3259E+06
-C-RAIL+ROAD CAPEX/	.2551E+08	.2551E+08	.2551E+08	.2551E+08	0	0	0	0
-C-FEASITY,ENG.FEES CAPEX/	.2134E+08	.2134E+08	.2134E+08	.2134E+08	0	0	0	0
-C-OVERSEAS TRANS OPEX/	0	0	0	0	.3460E+07	.3460E+07	.3460E+07	.3460E+07
-C-EQUIP,REPLACE CAPEX/	0	0	0	0	.1000E+07	.1000E+07	.1000E+07	.1000E+07
-C-AGRO-RESTORATION CAPX/	0	0	0	0	.4778E+05	.4778E+05	.4778E+05	.4778E+05
TOTAL ANNUAL SALES REVENUE	0	0	0	0	.3789E+09	.4168E+09	.4585E+09	.5043E+09
***MINING OPSOST /	0	0	0	0	.3491E+07	.3840E+07	.4224E+07	.4647E+07
***MILLING/TRANS /	0	0	0	0	.3869E+08	.4255E+08	.4681E+08	.5149E+08
***SALARIES AND WAGES/	0	0	0	0	.9726E+07	.1070E+08	.1177E+08	.1295E+08
***OVERSEAS TRANS OPEX/	0	0	0	0	.6129E+07	.6742E+07	.7416E+07	.8158E+07
TOTAL ANNUAL OPERATING COSTS	0	0	0	0	.5803E+08	.6383E+08	.7022E+08	.7724E+08
ROYALTIES PAID PER ANNUM	0	0	0	0	.1894E+08	.2084E+08	.2292E+08	.2522E+08
***EXPLORATION CAPEX/	.1524E+07	.5589E+06	.6147E+06	0	0	0	0	0
***PLANTS/INFRST/DEV CAPX/	0	.1406E+07	.2578E+07	.1134E+07	0	0	0	0
***MINE EQUIP.CAPEX/	0	0	.6769E+07	.7446E+07	0	0	0	0
***ROAD+RAIL CONSRTUCT.CAPEX/	0	0	.1868E+08	.2054E+08	0	0	0	0
***FEASIBILITY CAPEX/	0	0	.1562E+08	.1718E+08	0	0	0	0
***EQUIP,REPLACE,CAPEX/	0	0	0	0	.1772E+07	.1949E+07	.2144E+07	.2358E+07
***AGRO-RESTORATION CAPX/	0	0	0	0	0	0	0	0
***TOT.MILL CAPEX/	0	0	.1088E+09	.1197E+09	0	0	0	0
***START-UP,WORK CAPEX/	0	0	0	0	.5773E+06	.5773E+05	.6350E+05	.6985E+05

	YEAR 83	YEAR 84	YEAR 85	YEAR 86	YEAR 87	YEAR 88	YEAR 89	YEAR 90
TOTAL ANNUAL CAPITAL COSTS	.1524E+07	.1965E+07	.1530E+09	.1660E+09	.2349E+07	.2006E+07	.2207E+07	.2428E+07
ANNUAL RECOVERED CAPITAL COSTS	0	0	0	0	0	0	0	0
UNADJUSTED CAPITAL ALLOWANCES	0	0	0	0	.1038E+09	.3954E+08	.3983E+08	.4016E+08
ANNUAL TAXATION ALLOWANCES	0	0	0	0	.1038E+09	.3954E+08	.3983E+08	.4016E+08
ANNUAL TAXATION AT WBK RATE	0	0	0	0	.5945E+08	.8777E+08	.9765E+08	.1085E+09
ANNUAL WITHOLDING TAXATION	0	0	0	0	.1907E+08	.2816E+08	.3133E+08	.3481E+08
ANNUAL GOVERNMENT SHARE VALUE	0	0	0	0	.6935E+08	.1024E+09	.1139E+09	.1266E+09
ANNUAL CASH FLOW	-.1524E+07	-.1965E+07	-.1530E+09	-.1660E+09	-.1487E+09	.1113E+09	.9158E+08	.9801E+08
CUMMULATIVE CASH FLOW	-.1524E+07	-.3489E+07	-.1565E+09	-.3225E+09	-.4712E+09	-.3599E+09	-.2683E+09	-.1703E+09
PRESENT VALUES AT SELECTED DISCOUNT RATES								
DISCOUNT RATE 0 PERCENT	-.1386E+07	-.1624E+07	-.1150E+09	-.1134E+09	-.9232E+08	.6282E+08	.4700E+08	.4572E+08
DISCOUNT RATE 5.0 PERCENT	-.1320E+07	-.1473E+07	-.9933E+08	-.9326E+08	-.7233E+08	.4688E+08	.3340E+08	.3095E+08
DISCOUNT RATE 10.0 PERCENT	-.1260E+07	-.1342E+07	-.8639E+08	-.7743E+08	-.5732E+08	.3546E+08	.2412E+08	.2133E+08
DISCOUNT RATE 15.0 PERCENT	-.1205E+07	-.1228E+07	-.7560E+08	-.6481E+08	-.4590E+08	.2716E+08	.1767E+08	.1495E+08
INTERNAL RATE OF RETURN = 10.88 PERCENT								
PAYBACK PERIOD - FROM FIRST REVENUE YEAR								
UNDISCOUNTED VALUE 5.58 YEARS								
DISCOUNT RATE 0 PERCENT EXCESS 35 YEARS								
DISCOUNT RATE 5.0 PERCENT EXCESS 35 YEARS								
DISCOUNT RATE 10.0 PERCENT EXCESS 35 YEARS								
DISCOUNT RATE 15.0 PERCENT EXCESS 35 YEARS								
GOVERNMENT TAX INCOME NET OF BONUSES ETC.								
UNDISCOUNTED INCOME	0	0	0	0	.1894E+08	.9936E+08	.1389E+09	.1542E+09
DISCOUNT RATE 0 PERCENT	0	0	0	0	.1176E+08	.5609E+08	.7126E+08	.7193E+08
DISCOUNT RATE 5.0 PERCENT	0	0	0	0	.9217E+07	.4185E+08	.5064E+08	.4869E+08
DISCOUNT RATE 10.0 PERCENT	0	0	0	0	.7304E+07	.3166E+08	.3657E+08	.3356E+08
DISCOUNT RATE 15.0 PERCENT	0	0	0	0	.5848E+07	.2425E+08	.2679E+08	.2351E+08
VALUE OF RETAINED FOREIGN EARNINGS								
UNDISCOUNTED INCOME	-.1372E+07	-.1769E+07	-.1359E+09	-.1473E+09	-.8386E+08	.2682E+09	.2941E+09	.3226E+09
DISCOUNT RATE 0 PERCENT	-.1247E+07	-.1462E+07	-.1021E+09	-.1006E+09	-.5207E+08	.1514E+09	.1509E+09	.1505E+09
DISCOUNT RATE 5.0 PERCENT	-.1188E+07	-.1326E+07	-.8818E+08	-.8278E+08	-.4080E+08	.1130E+09	.1073E+09	.1019E+09
DISCOUNT RATE 10.0 PERCENT	-.1134E+07	-.1208E+07	-.7670E+08	-.6872E+08	-.3233E+08	.8547E+08	.7745E+08	.7021E+08
DISCOUNT RATE 15.0 PERCENT	-.1084E+07	-.1105E+07	-.6712E+08	-.5753E+08	-.2589E+08	.6546E+08	.5674E+08	.4920E+08
VALUE OF GOVERNMENT PARTICIPATION								
UNDISCOUNTED INCOME	0	0	0	0	.6935E+08	.1024E+09	.1139E+09	.1266E+09
DISCOUNT RATE 0 PERCENT	0	0	0	0	.4306E+08	.5780E+08	.5846E+08	.5906E+08
DISCOUNT RATE 5.0 PERCENT	0	0	0	0	.3374E+08	.4313E+08	.4155E+08	.3997E+08
DISCOUNT RATE 10.0 PERCENT	0	0	0	0	.2674E+08	.3263E+08	.3000E+08	.2755E+08
DISCOUNT RATE 15.0 PERCENT	0	0	0	0	.2141E+08	.2499E+08	.2198E+08	.1931E+08

	YEAR 91	YEAR 92	YEAR 93	YEAR 94	YEAR 95	YEAR 96	YEAR 97	YEAR 98
***BAUXITE GRADE AL2O3	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00
***ALUMINA,PRICE(99.5)£/T	.7074E+03	.7781E+03	.8559E+03	.9415E+03	.1036E+04	.1139E+04	.1253E+04	.1378E+04
***DRILL-DIAMETER INCHES	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01
***ORE-OUTPUT SCHEDULE TPD	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04
***ORE/WASTE EXC.FACTOR	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01
***ALUMINA MILL RECOVERY	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00
-C-EXPLOR AREA SQ.METERS	0	0	0	0	0	0	0	0
-C-EXPLORA DRILL/ACQ.COST£	0	0	0	0	0	0	0	0
-C-GEOLOGY/AERIALPHOTO £	0	0	0	0	0	0	0	0
-C-SEISMIC,CORING,GEOCHEM £	0	0	0	0	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 1	0	0	0	0	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 2	0	0	0	0	0	0	0	0
***EXCHANGE RATE CFA/£	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01
-C-TONS OF ORE/BLAST HOLE	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04
-C-REQ.NO.OF BLAST HOLES/DAY	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01
-C-REQ.NO.OF DRILLS/COMPRESS.	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00
-C-TONS ORE/FOOT DRILLED	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02
-C-SHOVEL LOADERS REQ.	0	0	0	0	0	0	0	0
-C-HAULING TRUCKS REQ.	0	0	0	0	0	0	0	0
-C-ANNUAL TONS EXP.REQ.	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04
-C-ANNUAL DRILL BITS REQ.	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02
-C-ORE/WASTE MOVED PER ANNUM	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05
-C-DAILY MILL OUTPUT	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04
-C-TOWN-SITE CAPEX£	0	0	0	0	0	0	0	0
-C-ANCILLARY.VEHICLES £CAPEX	0	0	0	0	0	0	0	0
-C-SH-TR-DR CAPEX£	0	0	0	0	0	0	0	0
-C-MINE ACCESS/STRIP CAPEX£	0	0	0	0	0	0	0	0
-C-ALUMINA/STEAM PLANT CAPEX£	0	0	0	0	0	0	0	0
-C-EXPLORATION CAPEX £	0	0	0	0	0	0	0	0
-C-DEH2O/FILT/THICK£CAPEX £	0	0	0	0	0	0	0	0
-C-TOT.M+P SUPPL.OPEX£/ANN.	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06
-C-SH-TR-DR FUEL OPEXANN£	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06
-C-CONC/SUPP.TRANS.OPEX£/ANN.	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08
-C- DENSEMED/DRY OPEX £/ANNUM	.1569E+04	.1360E+04	.2727E+03	.1140E+04	.4750E+09	.1980E+04	.1016E+06	.1549E+03
-C-EXPART.SALARIES £/ANNUM	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07
-C-CAMEROON SALARIES £/ANNUM	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07
-C-MAINT./REPAIR OPEX/ANN.	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06
-C-CRUSHING OPEX PER ANNUM£	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06
-C-GRINDING.OPEX££	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06
-C-GRAVITY SEP OPEX£/ANNUM	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05
-C-ELEC/MAG/DEH2O OPEX £/ANNUM	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05
-C-FILT/THICK OPEX £/ANNUM	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05
-C-TOTAL MILL CAPEX£	0	0	0	0	0	0	0	0
-C-SALARIES AND WAGES£/ANN	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07
-C-PLANTS/INFRST/DEV.CAPEX£	0	0	0	0	0	0	0	0
-C-TOT.MILL+TRANS OPEX£/ANN.	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08
-C-MINE OPEX£ /ANN	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07
-C-MILL UTILITIES/SHOPS CAP£	0	0	0	0	0	0	0	0
-C-START-UP,WORK.CAPEX£	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06
-C-RAIL+ROAD CAPEX£	0	0	0	0	0	0	0	0
-C-FEASIBILITY,ENG.FEES CAPEX£	0	0	0	0	0	0	0	0
-C-OVERSEAS TRANS OPEX£	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07
-C-EQUIP,REPLACE CAPEX£	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07
-C-AGRO-RESTORATION CAPX£	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05
TOTAL ANNUAL SALES REVENUE	.5547E+09	.6102E+09	.6712E+09	.7384E+09	.8122E+09	.8934E+09	.9828E+09	.1081E+10
***MINING OPCOST £	.5111E+07	.5622E+07	.6185E+07	.6803E+07	.7484E+07	.8232E+07	.9055E+07	.9961E+07
***MILLING/TRANS £	.5664E+08	.6230E+08	.6853E+08	.7539E+08	.8292E+08	.9122E+08	.1003E+09	.1104E+09
***SALARIES AND WAGES£	.1424E+08	.1566E+08	.1723E+08	.1895E+08	.2085E+08	.2293E+08	.2523E+08	.2775E+08
***OVERSEAS TRANS OPEX£	.8974E+07	.9871E+07	.1086E+08	.1194E+08	.1314E+08	.1445E+08	.1590E+08	.1749E+08
TOTAL ANNUAL OPERATING COSTS	.8496E+08	.9346E+08	.1028E+09	.1131E+09	.1244E+09	.1368E+09	.1505E+09	.1656E+09
ROYALTIES PAID PER ANNUM	.2774E+08	.3051E+08	.3356E+08	.3692E+08	.4061E+08	.4467E+08	.4914E+08	.5405E+08
***EXPLORATION CAPEX£	0	0	0	0	0	0	0	0
***PLANTS/INFRST/DEV CAPX£	0	0	0	0	0	0	0	0
***MINE EQUIP.CAPEX£	0	0	0	0	0	0	0	0
***ROAD+RAIL CONSTRUCT.CAPEX£	0	0	0	0	0	0	0	0
***FEASIBILITY CAPEX£	0	0	0	0	0	0	0	0
***EQUIP,REPLACE,CAPEX£	.2594E+07	.2853E+07	.3138E+07	.3452E+07	.3797E+07	.4177E+07	.4595E+07	.5054E+07
***AGRO-RESTORATION CAPX£	.6197E+04	.6817E+04	.7498E+04	.8248E+04	.9073E+04	.9980E+04	.1098E+05	.1208E+05
***TOT.MILL CAPEX£	0	0	0	0	0	0	0	0
***START-UP,WORK CAPEX£	.7684E+05	.8452E+05	.9297E+05	.1023E+06	.1125E+06	.1237E+06	.1361E+06	.1497E+06

	YEAR 91	YEAR 92	YEAR 93	YEAR 94	YEAR 95	YEAR 96	YEAR 97	YEAR 98
TOTAL ANNUAL CAPITAL COSTS	.2677E+07	.2944E+07	.3239E+07	.3563E+07	.3919E+07	.4311E+07	.4742E+07	.5216E+07
ANNUAL RECOVERED CAPITAL COSTS	0	0	0	0	0	0	0	0
UNADJUSTED CAPITAL ALLOWANCES	.4052E+08	.4091E+08	.2838E+08	.2836E+07	.3121E+07	.3433E+07	.3777E+07	.4155E+07
ANNUAL TAXATION ALLOWANCES	.4052E+08	.4091E+08	.2838E+08	.2836E+07	.3121E+07	.3433E+07	.3777E+07	.4155E+07
ANNUAL TAXATION AT WRK RATE	.1205E+09	.1336E+09	.1519E+09	.1757E+09	.1932E+09	.2125E+09	.2338E+09	.2572E+09
ANNUAL WITHOLDING TAXATION	.3865E+08	.4286E+08	.4875E+08	.5636E+08	.6199E+08	.6819E+08	.7501E+08	.8251E+08
ANNUAL GOVERNMENT SHARE VALUE	.1405E+09	.1559E+09	.1773E+09	.2049E+09	.2254E+09	.2480E+09	.2728E+09	.3000E+09
ANNUAL CASH FLOW	.1051E+09	.1129E+09	.1169E+09	.1120E+09	.1120E+09	.1232E+09	.1355E+09	.1491E+09
CUMMULATIVE CASH FLOW	-.6521E+08	.4765E+08	.1645E+09	.2766E+09	.3886E+09	.5118E+09	.6473E+09	.7964E+09
PRESENT VALUES AT SELECTED DISCOUNT RATES								
DISCOUNT RATE 0 PERCENT	.4456E+08	.4351E+08	.4096E+08	.3570E+08	.3244E+08	.3244E+08	.3244E+08	.3244E+08
DISCOUNT RATE 5.0 PERCENT	.2873E+08	.2671E+08	.2395E+08	.1988E+08	.1720E+08	.1639E+08	.1561E+08	.1486E+08
DISCOUNT RATE 10.0 PERCENT	.1890E+08	.1678E+08	.1436E+08	.1138E+08	.9397E+07	.8543E+07	.7767E+07	.7061E+07
DISCOUNT RATE 15.0 PERCENT	.1267E+08	.1076E+08	.8805E+07	.6673E+07	.5273E+07	.4585E+07	.3987E+07	.3467E+07
INTERNAL RATE OF RETURN = 10.98 PERCENT								
PAYBACK PERIOD - FROM FIRST REVENUE YEAR								
UNDISCOUNTED VALUE	5.58 YEARS							
DISCOUNT RATE 0 PERCENT	EXCESS 35 YEARS							
DISCOUNT RATE 5.0 PERCENT	EXCESS 35 YEARS							
DISCOUNT RATE 10.0 PERCENT	EXCESS 35 YEARS							
DISCOUNT RATE 15.0 PERCENT	EXCESS 35 YEARS							
GOVERNMENT TAX INCOME NET OF BONUSES ETC.								
UNDISCOUNTED INCOME	.1711E+09	.1896E+09	.2100E+09	.2376E+09	.2726E+09	.2999E+09	.3299E+09	.3629E+09
DISCOUNT RATE 0 PERCENT	.7255E+08	.7311E+08	.7361E+08	.7571E+08	.7897E+08	.7897E+08	.7897E+08	.7897E+08
DISCOUNT RATE 5.0 PERCENT	.4676E+08	.4498E+08	.4304E+08	.4216E+08	.4168E+08	.3988E+08	.3799E+08	.3618E+08
DISCOUNT RATE 10.0 PERCENT	.3077E+08	.2819E+08	.2580E+08	.2412E+08	.2287E+08	.2080E+08	.1890E+08	.1719E+08
DISCOUNT RATE 15.0 PERCENT	.2062E+08	.1807E+08	.1582E+08	.1415E+08	.1283E+08	.1116E+08	.9705E+07	.8439E+07
VALUE OF RETAINED FOREIGN EARNINGS								
UNDISCOUNTED INCOME	.3539E+09	.3884E+09	.4230E+09	.4581E+09	.5040E+09	.5543E+09	.6098E+09	.6708E+09
DISCOUNT RATE 0 PERCENT	.1501E+09	.1499E+09	.1483E+09	.1460E+09	.1460E+09	.1460E+09	.1460E+09	.1460E+09
DISCOUNT RATE 5.0 PERCENT	.9676E+08	.9193E+08	.8669E+08	.8129E+08	.7741E+08	.7373E+08	.7022E+08	.6687E+08
DISCOUNT RATE 10.0 PERCENT	.6366E+08	.5774E+08	.5197E+08	.4651E+08	.4228E+08	.3844E+08	.3495E+08	.3177E+08
DISCOUNT RATE 15.0 PERCENT	.4267E+08	.3702E+08	.3187E+08	.2728E+08	.2373E+08	.2063E+08	.1794E+08	.1560E+08
VALUE OF GOVERNMENT PARTICIPATION								
UNDISCOUNTED INCOME	.1405E+09	.1559E+09	.1773E+09	.2049E+09	.2254E+09	.2480E+09	.2728E+09	.3000E+09
DISCOUNT RATE 0 PERCENT	.5960E+08	.6009E+08	.6213E+08	.6530E+08	.6530E+08	.6530E+08	.6530E+08	.6530E+08
DISCOUNT RATE 5.0 PERCENT	.3842E+08	.3689E+08	.3633E+08	.3636E+08	.3463E+08	.3298E+08	.3141E+08	.2991E+08
DISCOUNT RATE 10.0 PERCENT	.2528E+08	.2317E+08	.2178E+08	.2081E+08	.1891E+08	.1720E+08	.1563E+08	.1421E+08
DISCOUNT RATE 15.0 PERCENT	.1694E+08	.1485E+08	.1336E+08	.1220E+08	.1061E+08	.9228E+07	.8025E+07	.6978E+07

	YEAR 99	YEAR 00	YEAR 01	YEAR 02	YEAR 03	YEAR 04	YEAR 05	YEAR 06
***BAUXITE GRADE AL2O3	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00
***ALUMINA,PRICE(99.5)£/T	.1516E+04	.1668E+04	.1835E+04	.2018E+04	.2220E+04	.2442E+04	.2686E+04	.2955E+04
***DRILL-DIAMETER INCHES	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01
***ORE-OUTPUT SCHEDULE TPD	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04
***ORE/WASTE EXC.FACTOR	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01
***ALUMINA MILL RECOVERY	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00
-C-EXPLOR AREA SQ.METERS	0	0	0	0	0	0	0	0
-C-EXPLORA DRILL/ACQ.COST£	0	0	0	0	0	0	0	0
-C-GEOLOGY/AERIALPHOTO £	0	0	0	0	0	0	0	0
-C-SEISMIC,CORING,GEOCHEM £	0	0	0	0	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 1	0	0	0	0	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 2	0	0	0	0	0	0	0	0
***EXCHANGE RATE CFA/£	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01
-C-TONS OF ORE/BLAST HOLE	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04
-C-REQ.NO.OF BLAST HOLES/DAY	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01
-C-REQ.NO.OF DRILLS/COMPRESS.	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00
-C-TONS ORE/FOOT DRILLED	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02
-C-SHOVEL LOADERS REQ.	0	0	0	0	0	0	0	0
-C-HAULING TRUCKS REQ.	0	0	0	0	0	0	0	0
-C-ANNUAL TONS EXP.REQ.	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04
-C-ANNUAL DRILL BITS REQ.	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02
-C-ORE/WASTE MOVED PER ANNUM	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05
-C-DAILY MILL OUTPUT	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04
-C-TOWN-SITE CAPEX£	0	0	0	0	0	0	0	0
-C-ANCILLARY,VEHICLES £CAPEX	0	0	0	0	0	0	0	0
-C-SH-TR-DR CAPEX£	0	0	0	0	0	0	0	0
-C-MINE ACCESS/STRIP CAPEX£	0	0	0	0	0	0	0	0
-C-ALUMINA/STEAM PLANT CAPEX£	0	0	0	0	0	0	0	0
-C-EXPLORATION CAPEX £	0	0	0	0	0	0	0	0
-C-DEH2O/FILT/THICK£CAPEX £	0	0	0	0	0	0	0	0
-C-TOT.M+P SUPPL.OPEX£/ANN.	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06
-C-SH-TR-DR FUEL OPEX£ANN£	.8322E+05	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06
-C-CONC/SUPP.TRANS.OPEX£/ANN.	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08
-C- DENSEMED/DRY OPEX £/ANNUM	.1429E+04	.1220E+04	.1546E+04	.1990E+04	.6278E+02	.2093E+01	.2407E+03	.2218E+03
-C-EXPART.SALARIES £/ANNUM	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07	.2322E+07
-C-CAMEROON SALARIES £/ANNUM	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07
-C-MAINT./REPAIR OPEX/ANN.	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06
-C-CRUSHING OPEX PER ANNUM£	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06
-C-GRINDING.OPEXS£/	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06
-C-GRAVITY SEP OPEX£/ANNUM	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05
-C-ELEC/MAG/DEH2O OPEX £/ANNUM	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05
-C-FILT/THICK OPEX £/ANNUM	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05
-C-TOTAL MILL CAPEX£	0	0	0	0	0	0	0	0
-C-SALARIES AND WAGES£/ANN	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07	.5490E+07
-C-PLANTS/INFRST/DEV.CAPEX£	0	0	0	0	0	0	0	0
-C-TOT.MILL+TRANS OEX£/ANN.	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08
-C-MINE OPEX£ /ANN	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07
-C-MILL UTILITIES/SHOPS CAP£	0	0	0	0	0	0	0	0
-C-START-UP,WORK CAPEX£	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06
-C-RAIL+ROAD CAPEX£	0	0	0	0	0	0	0	0
-C-FEASIBILITY,ENG.FEES CAPEX£	0	0	0	0	0	0	0	0
-C-OVERSEAS TRANS OPEX£	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07
-C-EQUIP,REPLACE CAPEX£	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07
-C-AGRO-RESTORATION CAPX£	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05
TOTAL ANNUAL SALES REVENUE	.1189E+10	.1308E+10	.1439E+10	.1583E+10	.1741E+10	.1915E+10	.2107E+10	.2317E+10
***MINING OPCOST £	.1096E+08	.1205E+08	.1326E+08	.1458E+08	.1604E+08	.1765E+08	.1941E+08	.2135E+08
***MILLING/TRANS £	.1214E+09	.1336E+09	.1469E+09	.1616E+09	.1778E+09	.1955E+09	.2151E+09	.2366E+09
***SALARIES AND WAGES£	.3052E+08	.3358E+08	.3693E+08	.4063E+08	.4469E+08	.4916E+08	.5408E+08	.5948E+08
***OVERSEAS TRANS OPEX£	.1924E+08	.2116E+08	.2328E+08	.2560E+08	.2816E+08	.3098E+08	.3408E+08	.3749E+08
TOTAL ANNUAL OPERATING COSTS	.1821E+09	.2003E+09	.2204E+09	.2424E+09	.2667E+09	.2933E+09	.3226E+09	.3549E+09
ROYALTIES PAID PER ANNUM	.5946E+08	.6540E+08	.7194E+08	.7914E+08	.8705E+08	.957[illegible]E+08	.1053E+09	.1159E+09
***EXPLORATION CAPEX£	0	0	0	0	0	0	0	0
***PLANTS/INFRST/DEV CAPEX£	0	0	0	0	0	0	0	0
***MINE EQUIP,CAPEX£	0	0	0	0	0	0	0	0
***ROAD+RAIL CONSTRUCT.CAPEX£	0	0	0	0	0	0	0	0
***FEASIBILITY CAPEX£	0	0	0	0	0	0	0	0
***EQUIP,REPLACE,CAPEX£	.5560E+07	.6116E+07	.6727E+07	.7400E+07	.8140E+07	.8954E+07	.9850E+07	.1083E+08
***AGRO-RESTORATION CAPX£	.1378E+05	.1461E+05	.1607E+05	.1748E+05	.1945E+05	.2139E+05	.2353E+05	.2589E+05
***TOT.MILL CAPEX£	0	0	0	0	0	0	0	0
***START-UP,WORK CAPEX£	.1557E+06	.1812E+06	.1993E+06	.2192E+06	.2411E+06	[illegible]	.2918E+06	.3210E+06

	YEAR 99	YEAR 00	YEAR 01	YEAR 02	YEAR 03	YEAR 04	YEAR 05	YEAR 06
TOTAL ANNUAL CAPITAL COSTS	.5738E+07	.6312E+07	.6943E+07	.7637E+07	.8401E+07	.9241E+07	.1017E+08	.1118E+08
ANNUAL RECOVERED CAPITAL COSTS	0	0	0	0	0	0	0	0
UNADJUSTED CAPITAL ALLOWANCES	.4570E+07	.5027E+07	.5530E+07	.6083E+07	.6691E+07	.7361E+07	.8097E+07	.8906E+07
ANNUAL TAXATION ALLOWANCES	.4570E+07	.5027E+07	.5530E+07	.6083E+07	.6691E+07	.7361E+07	.8097E+07	.8906E+07
ANNUAL TAXATION AT VRK RATE	.2829E+09	.3112E+09	.3423E+09	.3765E+09	.4142E+09	.4556E+09	.5012E+09	.5513E+09
ANNUAL WITHOLDING TAXATION	.9076E+08	.9984E+08	.1098E+09	.1208E+09	.1329E+09	.1462E+09	.1608E+09	.1769E+09
ANNUAL GOVERNMENT SHARE VALUE	.3300E+09	.3631E+09	.3994E+09	.4393E+09	.4832E+09	.5315E+09	.5847E+09	.6432E+09
ANNUAL CASH FLOW	.1640E+09	.1804E+09	.1984E+09	.2183E+09	.2401E+09	.2641E+09	.2905E+09	.3196E+09
CUMMULATIVE CASH FLOW	.9603E+09	.1141E+10	.1339E+10	.1557E+10	.1797E+10	.2062E+10	.2352E+10	.2672E+10
PRESENT VALUES AT SELECTED DISCOUNT RATES								
DISCOUNT RATE 0 PERCENT	.3244E+08	.3244E+08	.3244E+08	.3244E+08	.3244E+08	.3244E+08	.3244E+08	.3244E+08
DISCOUNT RATE 5.0 PERCENT	.1415E+08	.1348E+08	.1284E+08	.1223E+08	.1165E+08	.1109E+08	.1056E+08	.1006E+08
DISCOUNT RATE 10.0 PERCENT	.6419E+07	.5835E+07	.5305E+07	.4822E+07	.4384E+07	.3985E+07	.3623E+07	.3294E+07
DISCOUNT RATE 15.0 PERCENT	.3015E+07	.2622E+07	.2280E+07	.1982E+07	.1724E+07	.1499E+07	.1303E+07	.1133E+07
INTERNAL RATE OF RETURN = 10.88 PERCENT								
PAYBACK PERIOD - FROM FIRST REVENUE YEAR								
UNDISCOUNTED VALUE	5.58 YEARS							
DISCOUNT RATE 0 PERCENT	EXCESS 35 YEARS							
DISCOUNT RATE 5.0 PERCENT	EXCESS 35 YEARS							
DISCOUNT RATE 10.0 PERCENT	EXCESS 35 YEARS							
DISCOUNT RATE 15.0 PERCENT	EXCESS 35 YEARS							
GOVERNMENT TAX INCOME NET OF BONUSES ETC.								
UNDISCOUNTED INCOME	.3991E+09	.4391E+09	.4830E+09	.5313E+09	.5844E+09	.6428E+09	.7071E+09	.7778E+09
DISCOUNT RATE 0 PERCENT	.7897E+08	.7897E+08	.7897E+08	.7897E+08	.7897E+08	.7897E+08	.7897E+08	.7897E+08
DISCOUNT RATE 5.0 PERCENT	.3445E+08	.3281E+08	.3125E+08	.2976E+08	.2835E+08	.2700E+08	.2571E+08	.2449E+08
DISCOUNT RATE 10.0 PERCENT	.1562E+08	.1420E+08	.1291E+08	.1174E+08	.1067E+08	.9701E+07	.8819E+07	.8017E+07
DISCOUNT RATE 15.0 PERCENT	.7338E+07	.6381E+07	.5549E+07	.4825E+07	.4196E+07	.3648E+07	.3173E+07	.2759E+07
VALUE OF RETAINED FOREIGN EARNINGS								
UNDISCOUNTED INCOME	.7378E+09	.8116E+09	.8928E+09	.9821E+09	.1080E+10	.1188E+10	.1307E+10	.1438E+10
DISCOUNT RATE 0 PERCENT	.1460E+09	.1460E+09	.1460E+09	.1460E+09	.1460E+09	.1460E+09	.1460E+09	.1460E+09
DISCOUNT RATE 5.0 PERCENT	.6369E+08	.6066E+08	.5777E+08	.5502E+08	.5240E+08	.4990E+08	.4753E+08	.4526E+08
DISCOUNT RATE 10.0 PERCENT	.2888E+08	.2626E+08	.2387E+08	.2170E+08	.1973E+08	.1793E+08	.1630E+08	.1482E+08
DISCOUNT RATE 15.0 PERCENT	.1357E+08	.1180E+08	.1026E+08	.8919E+07	.7756E+07	.6744E+07	.5865E+07	.5100E+07
VALUE OF GOVERNMENT PARTICIPATION								
UNDISCOUNTED INCOME	.3300E+09	.3631E+09	.3994E+09	.4393E+09	.4832E+09	.5315E+09	.5847E+09	.6432E+09
DISCOUNT RATE 0 PERCENT	.6530E+08	.6530E+08	.6530E+08	.6530E+08	.6530E+08	.6530E+08	.6530E+08	.6530E+08
DISCOUNT RATE 5.0 PERCENT	.2849E+08	.2713E+08	.2584E+08	.2461E+08	.2344E+08	.2232E+08	.2126E+08	.2025E+08
DISCOUNT RATE 10.0 PERCENT	.1292E+08	.1174E+08	.1068E+08	.9706E+07	.8824E+07	.8022E+07	.7292E+07	.6629E+07
DISCOUNT RATE 15.0 PERCENT	.6068E+07	.5276E+07	.4588E+07	.3990E+07	.3469E+07	.3017E+07	.2623E+07	.2281E+07

	YEAR 07	YEAR 08	YEAR 09	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14
***BAUXITE GRADE AL2O3	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00
***ALUMINA,PRICE(99.5)£/T	.3250E+04	.3575E+04	.3933E+04	.4326E+04	.4759E+04	.5235E+04	.5758E+04	.6334E+04
***DRILL-DIAMETER INCHES	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01
***ORE-OUTPUT SCHEDULE TPD	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04
***ORE/WASTE EXC.FACTOR	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01
***ALUMINA MILL RECOVERY	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00
-C-EXPLOR AREA SQ.METERS	0	0	0	0	0	0	0	0
-C-EXPLORA DRILL/ACQ.COST£	0	0	0	0	0	0	0	0
-C-GEOLOGY/AERIALPHOTO £	0	0	0	0	0	0	0	0
-C-SEISMIC,CORING,GEOCHEM £	0	0	0	0	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 1	0	0	0	0	0	0	0	0
-C-WAGES,SUPPLIES,VEHICLE 2	0	0	0	0	0	0	0	0
***EXCHANGE RATE CFA/£	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01
-C-TONS OF ORE/BLAST HOLE	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04
-C-REQ.NO.OF BLAST HOLES/DAY	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01
-C-REQ.NO.OF DRILLS/COMPRESS.	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00
-C-TONS ORE/FOOT DRILLED	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02
-C-SHOVEL LOADERS REQ.	0	0	0	0	0	0	0	0
-C-HAULING TRUCKS REQ.	0	0	0	0	0	0	0	0
-C-ANNUAL TONS EXP.REQ.	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04
-C-ANNUAL DRILL BITS REQ.	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02
-C-ORE/WASTE MOVED PER ANNUM	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05
-C-DAILY MILL OUTPUT	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04
-C-TOWN-SITE CAPEX£	0	0	0	0	0	0	0	0
-C-ANCILLARY,VEHICLES £CAPEX	0	0	0	0	0	0	0	0
-C-SH-TR-DR CAPEX£	0	0	0	0	0	0	0	0
-C-MINE ACCESS/STRIP CAPEX£	0	0	0	0	0	0	0	0
-C-ALUMINA/STEAM PLANT CAPEX£	0	0	0	0	0	0	0	0
-C-EXPLORATION CAPEX £	0	0	0	0	0	0	0	0
-C-DEH2O/FILT/THICK£CAPEX £	0	0	0	0	0	0	0	0
-C-TOT.M+P SUPPL.OPEX£/ANN.	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06
-C-SH-TR-DR FUEL OPEXANN£	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06
-C-CONC/SUPP.TRANS.OPEX£/ANN.	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08
-C- DENSEMED/DRY OPEX £/ANNUM	.1716E+03	.1567E+04	.8371E+07	.4918E+03	.2093E+07	.1256E+03	.2511E+06	.6278E+06
-C-EXPART.SALARIES £/ANNUM	0	0	0	0	0	0	0	0
-C-CAMEROON SALARIES £/ANNUM	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07
-C-MAINT./REPAIR OPEX/ANN.	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06
-C-CRUSHING OPEX PER ANNUM£	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06
-C-GRINDING,OPEXS££	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06
-C-GRAVITY SEP OPEX£/ANNUM	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05
-C-ELEC/MAG/DEH2O OPEX £/ANNUM	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05
-C-FILT/THICK OPEX £/ANNUM	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05
-C-TOTAL MILL CAPEX£	0	0	0	0	0	0	0	0
-C-SALARIES AND WAGES£/ANN	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07
-C-PLANTS/INFRST/DEV.CAPEX£	0	0	0	0	0	0	0	0
-C-TOT.MILL+TRANS OPX£/ANN.	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08
-C-MINE OPEX£ /ANN	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07
-C-MILL UTILITIES/SHOPS CAP£	0	0	0	0	0	0	0	0
-C-START-UP,WORK CAPEX£	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06
-C-RAIL+ROAD CAPEX£	0	0	0	0	0	0	0	0
-C-FEASITY,ENG.FEES CAPEX£	0	0	0	0	0	0	0	0
-C-OVERSEAS TRANS OPEX£	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07
-C-EQUIP,REPLACE CAPEX£	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07
-C-AGRO-RESTORATION CAPX£	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05
TOTAL ANNUAL SALES REVENUE	.2549E+10	.2804E+10	.3084E+10	.3393E+10	.3732E+10	.4105E+10	.4516E+10	.4967E+10
***MINING OPCOST £	.2349E+08	.2584E+08	.2842E+08	.3126E+08	.3439E+08	.3783E+08	.4161E+08	.4577E+08
***MILLING/TRANS £	.2603E+09	.2863E+09	.3149E+09	.3464E+09	.3810E+09	.4191E+09	.4611E+09	.5072E+09
***SALARIES AND WAGES£	.3776E+08	.4153E+08	.4569E+08	.5025E+08	.5528E+08	.6081E+08	.6689E+08	.7358E+08
***OVERSEAS TRANS OPEX£	.4123E+08	.4536E+08	.4989E+08	.5488E+08	.6037E+08	.6641E+08	.7305E+08	.8035E+08
TOTAL ANNUAL OPERATING COSTS	.3627E+09	.3990E+09	.4389E+09	.4828E+09	.5311E+09	.5842E+09	.6426E+09	.7069E+09
ROYALTIES PAID PER ANNUM	.1275E+09	.1402E+09	.1542E+09	.1696E+09	.1866E+09	.2053E+09	.2258E+09	.2484E+09
***EXPLORATION CAPEX£	0	0	0	0	0	0	0	0
***PLANTS/INFRST/DEV CAPX£	0	0	0	0	0	0	0	0
***MINE EQUIP.CAPEX£	0	0	0	0	0	0	0	0
***ROAD+RAIL CONSTRUCT.CAPEX£	0	0	0	0	0	0	0	0
***FEASIBILITY CAPEX£	0	0	0	0	0	0	0	0
***EQUIP,REPLACE,CAPEX£	.1192E+08	.1311E+08	.1442E+08	.1586E+08	.1745E+08	.1919E+08	.2111E+08	.2323E+08
***AGRO-RESTORATION CAPX£	.2847E+05	.3132E+05	.3445E+05	.3790E+05	0	0	0	0
***TOT.MILL CAPEX£	0	0	0	0	0	0	0	0
***START-UP,WORK CAPEX£	.3531E+06	.3944E+06	.4272E+06	.4699E+06	.5169E+06	.5686E+06	.6255E+06	.6880E+06

	YEAR 07	YEAR 08	YEAR 09	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14
TOTAL ANNUAL CAPITAL COSTS	.1230E+08	.1353E+08	.1488E+08	.1637E+08	.1797E+08	.1976E+08	.2174E+08	.2391E+08
ANNUAL RECOVERED CAPITAL COSTS	0	0	0	0	0	0	0	0
UNADJUSTED CAPITAL ALLOWANCES	.9797E+07	.1078E+08	.1185E+08	.1304E+08	.1433E+08	.1576E+08	.1733E+08	.1906E+08
ANNUAL TAXATION ALLOWANCES	.9797E+07	.1078E+08	.1185E+08	.1304E+08	.1433E+08	.1576E+08	.1733E+08	.1906E+08
ANNUAL TAXATION AT VRK RATE	.6147E+09	.6762E+09	.7438E+09	.8182E+09	.9000E+09	.9900E+09	.1089E+10	.1198E+10
ANNUAL WITHHOLDING TAXATION	.1972E+09	.2169E+09	.2386E+09	.2625E+09	.2888E+09	.3176E+09	.3494E+09	.3843E+09
ANNUAL GOVERNMENT SHARE VALUE	.7172E+09	.7889E+09	.8678E+09	.9546E+09	.1050E+10	.1155E+10	.1271E+10	.1398E+10
ANNUAL CASH FLOW	.3695E+09	.3955E+09	.4350E+09	.4785E+09	.5264E+09	.5791E+09	.6370E+09	.7007E+09
CUMMULATIVE CASH FLOW	.3041E+10	.3437E+10	.3872E+10	.4350E+10	.4877E+10	.5456E+10	.6093E+10	.6793E+10
PRESENT VALUES AT SELECTED DISCOUNT RATES								
DISCOUNT RATE 0 PERCENT	.3410E+08	.3318E+08	.3318E+08	.3318E+08	.3319E+08	.3318E+08	.3318E+08	.3318E+08
DISCOUNT RATE 5.0 PERCENT	.1007E+08	.9332E+07	.8888E+07	.8465E+07	.8062E+07	.7678E+07	.7313E+07	.6964E+07
DISCOUNT RATE 10.0 PERCENT	.3148E+07	.2784E+07	.2531E+07	.2301E+07	.2092E+07	.1902E+07	.1729E+07	.1572E+07
DISCOUNT RATE 15.0 PERCENT	.1036E+07	.8765E+06	.7622E+06	.6628E+06	.5764E+06	.5012E+06	.4358E+06	.3790E+06

INTERNAL RATE OF RETURN = 10.88 PERCENT

PAYBACK PERIOD - FROM FIRST REVENUE YEAR
UNDISCOUNTED VALUE 5.58 YEARS
DISCOUNT RATE 0 PERCENT EXCESS 35 YEARS
DISCOUNT RATE 5.0 PERCENT EXCESS 35 YEARS
DISCOUNT RATE 10.0 PERCENT EXCESS 35 YEARS
DISCOUNT RATE 15.0 PERCENT EXCESS 35 YEARS

	YEAR 07	YEAR 08	YEAR 09	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14
GOVERNMENT TAX INCOME NET OF BONUSES ETC.								
UNDISCOUNTED INCOME	.8556E+09	.9521E+09	.1047E+10	.1152E+10	.1267E+10	.1394E+10	.1533E+10	.1687E+10
DISCOUNT RATE 0 PERCENT	.7897E+08	.7989E+08	.7989E+08	.7989E+08	.7989E+08	.7989E+08	.7989E+08	.7989E+08
DISCOUNT RATE 5.0 PERCENT	.2332E+08	.2247E+08	.2140E+08	.2038E+08	.1941E+08	.1848E+08	.1760E+08	.1677E+08
DISCOUNT RATE 10.0 PERCENT	.7289E+07	.6703E+07	.6094E+07	.5540E+07	.5036E+07	.4578E+07	.4162E+07	.3784E+07
DISCOUNT RATE 15.0 PERCENT	.2399E+07	.2110E+07	.1835E+07	.1596E+07	.1388E+07	.1207E+07	.1049E+07	.9124E+06
VALUE OF RETAINED FOREIGN EARNINGS								
UNDISCOUNTED INCOME	.1588E+10	.1747E+10	.1922E+10	.2114E+10	.2326E+10	.2558E+10	.2814E+10	.3095E+10
DISCOUNT RATE 0 PERCENT	.1466E+09	.1466E+09	.1466E+09	.1466E+09	.1466E+09	.1465E+09	.1466E+09	.1466E+09
DISCOUNT RATE 5.0 PERCENT	.4329E+08	.4123E+08	.3927E+08	.3740E+08	.3562E+08	.3392E+08	.3231E+08	.3077E+08
DISCOUNT RATE 10.0 PERCENT	.1353E+08	.1230E+08	.1118E+08	.1017E+08	.9242E+07	.8402E+07	.7638E+07	.6944E+07
DISCOUNT RATE 15.0 PERCENT	.4454E+07	.3873E+07	.3368E+07	.2928E+07	.2546E+07	.2214E+07	.1925E+07	.1674E+07
VALUE OF GOVERNMENT PARTICIPATION								
UNDISCOUNTED INCOME	.7172E+09	.7889E+09	.8678E+09	.9546E+09	.1050E+10	.1155E+10	.1271E+10	.1398E+10
DISCOUNT RATE 0 PERCENT	.6619E+08	.6619E+08	.6619E+08	.6619E+08	.6619E+08	.6619E+08	.6619E+08	.6619E+08
DISCOUNT RATE 5.0 PERCENT	.1955E+08	.1862E+08	.1773E+08	.1689E+08	.1608E+08	.1532E+08	.1459E+08	.1389E+08
DISCOUNT RATE 10.0 PERCENT	.6109E+07	.5554E+07	.5049E+07	.4590E+07	.4173E+07	.3793E+07	.3449E+07	.3135E+07
DISCOUNT RATE 15.0 PERCENT	.2011E+07	.1748E+07	.1520E+07	.1322E+07	.1150E+07	.9997E+06	.8693E+06	.7559E+06

	YEAR 15	YEAR 16	YEAR 17	YEAR 18	YEAR 19	YEAR 20	YEAR ++	TOTAL
***BAUXITE GRADE AL2O3	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	.4500E+00	0	
***ALUMINA,PRICE(99.5)£/T	.6968E+04	.7464E+04	.8431E+04	.9274E+04	.1020E+05	.1122E+05	0	
***DRILL-DIAMETER INCHES	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	.8000E+01	0	
***ORE-OUTPUT SCHEDULE TPD	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	.5000E+04	0	
***ORE/WASTE EXC.FACTOR	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	.2000E+01	0	
***ALUMINA MILL RECOVERY	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	.8500E+00	0	
-C-EXPLOR AREA SQ.METERS	0	0	0	0	0	0	0	
-C-EXPLORA DRILL/ACQ.COST£	0	0	0	0	0	0	0	
-C-GEOLOGY/AERIALPHOTO £	0	0	0	0	0	0	0	
-C-SEISMIC,CORING,GEOCHEM £	0	0	0	0	0	0	0	
-C-WAGES,SUPPLIES,VEHICLE 1	0	0	0	0	0	0	0	
-C-WAGES,SUPPLIES,VEHICLE 2	0	0	0	0	0	0	0	
***EXCHANGE RATE CFA/£	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	.1000E+01	0	
-C-TONS OF ORE/BLAST HOLE	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	.2023E+04	0	
-C-REQ.NO.OF BLAST HOLES/DAY	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	.7416E+01	0	
-C-REQ.NO.OF DRILLS/COMPRESS.	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	.7558E+00	0	
-C-TONS ORE/FOOT DRILLED	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	.4045E+02	0	
-C-SHOVEL LOADERS REQ.	0	0	0	0	0	0	0	
-C-HAULING TRUCKS REQ.	0	0	0	0	0	0	0	
-C-ANNUAL TONS EXP.REQ.	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	.5304E+04	0	
-C-ANNUAL DRILL BITS REQ.	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+02	.6051E+C2	0	
-C-ORE/WASTE MOVED PER ANNUM	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	.1800E+05	0	
-C-DAILY MILL OUTPUT	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	.2307E+04	0	
-C-TOWN-SITE CAPEX£	0	0	0	0	0	0	0	
-C-ANCILLARY.VEHICLES £CAPEX	0	0	0	0	0	0	0	
-C-SH-TR-DR CAPEX£	0	0	0	0	0	0	0	
-C-MINE ACCESS/STRIP CAPEX£	0	0	0	0	0	0	0	
-C-ALUMINA/STEAM PLANT CAPEX£	0	0	0	0	0	0	0	
-C-EXPLORATION CAPEX £	0	0	0	0	0	0	C	
-C-DEH2O/FILT/THICK£CAPEX £	0	0	0	0	0	0	0	
-C-TOT.M+P SUPPL.OPEX£/ANN.	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	.5837E+06	0	
-C-SH-TR-DR FUEL OPEX£ANN£	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	.8322E+06	0	
-C-CONC/SUPP.TRANS.OPEX£/ANN.	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	.2076E+08	0	
-C- DENSEMED/DRY OPEX £/ANNUM	.2846E+06	.7324E+06	.9417E+07	.1674E+07	.1779E+08	.1046E+08	0	
-C-EXPART.SALARIES £/ANNUM	0	0	0	0	0	0	0	
-C-CAMEROON SALARIES £/ANNUM	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	0	
-C-MAINT./REPAIR OPEX/ANN.	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	.5548E+06	0	
-C-CRUSHING OPEX PER ANNUM£	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	.3732E+06	0	
-C-GRINDING.OPEX££	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	.7048E+06	0	
-C-GRAVITY SEP OPEX£/ANNUM	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	.3367E+05	0	
-C-ELEC/MAG/DEH2O OPEX £/ANNUM	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	.9450E+05	0	
-C-FILT/THICK OPEX £/ANNUM	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	.4684E+05	0	
-C-TOTAL MILL CAPEX£	0	0	0	0	0	0	0	
-C-SALARIES AND WAGES£/ANN	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	.3168E+07	0	
-C-PLANTS/INFRST/DEV.CAPEX£	0	0	0	0	0	0	0	
-C-TOT.MILL+TRANS OPEX£/ANN.	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	.2184E+08	0	
-C-MINE OPEX£ /ANN	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	.1971E+07	0	
-C-MILL UTILITIES/SHOPS CAP£	0	0	0	0	0	0	0	
-C-START-UP,WORK CAPEX£	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	.3259E+06	0	
-C-RAIL+ROAD CAPEX£	0	0	0	0	0	0	0	
-C-FEASITY,ENG.FEES CAPEX£	0	0	0	0	0	0	0	
-C-OVERSEAS TRANS OPEX£	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	.3460E+07	0	
-C-EQUIP,REPLACE CAPEX£	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	.1000E+07	0	
-C-AGRO-RESTORATION CAPX£	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	.4778E+05	0	
TOTAL ANNUAL SALES REVENUE	.5464E+10	.6011E+10	.6612E+10	.7273E+10	.8000E+10	.8800E+10	0	.9301E+11
***MINING OPCOST £	.5035E+08	.5538E+08	.6092E+08	.6701E+08	.7371E+08	.8108E+08	0	.8570E+09
***MILLING/TRANS £	.5579E+09	.6137E+09	.6750E+09	.7425E+09	.8168E+09	.8985E+09	0	.9496E+10
***SALARIES AND WAGES£	.8094E+08	.8903E+08	.9793E+08	.1077E+09	.1185E+09	.1303E+09	0	.1613E+10
***OVERSEAS TRANS OPEX£	.8839E+08	.9723E+08	.1070E+09	.1176E+09	.1294E+09	.1424E+C9	0	.1505E+10
TOTAL ANNUAL OPERATING COSTS	.7775E+09	.8553E+09	.9408E+09	.1035E+10	.1138E+10	.1252E+10	C	.1347E+11
ROYALTIES PAID PER ANNUM	.2732E+09	.3005E+09	.3306E+09	.3636E+09	.4000E+09	.4400E+09	0	.4651E+10
***EXPLORATION CAPEX£	0	0	0	0	0	0	0	.2698E+07
***PLANTS/INFRST/DEV CAPX£	0	0	0	0	0	0	0	.5119E+07
***MINE EQUIP.CAPEX£	0	0	0	0	0	0	0	.1421E+08
***ROAD+RAIL CONSRTUCT.CAPEX£	0	0	0	0	0	0	0	.3922E+08
***FEASIBILITY CAPEX£	0	0	0	0	0	0	0	.3281E+08
***EQUIP,REPLACE,CAPEX£	.2555E+08	.2810E+08	.3091E+08	.3400E+08	.3740E+08	.4114E+08	0	.4349E+09
***AGRO-RESTORATION CAPX£	0	0	0	0	0	0	0	.3549E+06
***TOT.MILL CAPEX£	0	0	0	0	0	0	0	.2284E+09
***START-UP,WORK CAPEX£	.7568E+06	.8325E+06	.9157E+06	.1007E+07	.1108E+07	.1219E+C7	0	.1341E+08

	YEAR 15	YEAR 16	YEAR 17	YEAR 18	YEAR 19	YEAR 20	YEAR ++	TOTAL
TOTAL ANNUAL CAPITAL COSTS	.2630E+08	.2893E+08	.3183E+08	.3501E+08	.3851E+08	.4236E+08	0	.7711E+09
ANNUAL RECOVERED CAPITAL COSTS	0	0	0	0	0	0	.1341E+08	.1341E+08
UNADJUSTED CAPITAL ALLOWANCES	.2096E+08	.2305E+08	.2535E+08	.2789E+08	.3067E+08	.3374E+08	0	.6763E+09
ANNUAL TAXATION ALLOWANCES	.2096E+08	.2305E+08	.2535E+08	.2789E+08	.3067E+08	.3374E+08	0	.6763E+09
ANNUAL TAXATION AT NBK RATE	.1318E+10	.1449E+10	.1594E+10	.1754E+10	.1929E+10	.2122E+10	0	.2226E+11
ANNUAL WITHOLDING TAXATION	.4228E+09	.4650E+09	.5115E+09	.5627E+09	.6190E+09	.6809E+09	0	.7143E+10
ANNUAL GOVERNMENT SHARE VALUE	.1537E+10	.1691E+10	.1860E+10	.2046E+10	.2251E+10	.2476E+10	0	.2597E+11
ANNUAL CASH FLOW	.7707E+09	.8478E+09	.9326E+09	.1026E+10	.1128E+10	.1241E+10	.6010E+10	.1875E+11
CUMMULATIVE CASH FLOW	.7564E+10	.8412E+10	.9344E+10	.1037E+11	.1150E+11	.1274E+11	.1875E+11	.1875E+11
PRESENT VALUES AT SELECTED DISCOUNT RATES								
DISCOUNT RATE 0 PERCENT	.3318E+08	.3318E+08	.3318E+08	.3318E+08	.3318E+08	.3318E+08	.1461E+09	.9975E+09
DISCOUNT RATE 5.0 PERCENT	.6633E+07	.6317E+07	.6016E+07	.5729E+07	.5457E+07	.5197E+07	.2179E+08	.2268E+09
DISCOUNT RATE 10.0 PERCENT	.1429E+07	.1299E+07	.1181E+07	.1073E+07	.9759E+06	.8872E+06	.3550E+07	.1747E+08
DISCOUNT RATE 15.0 PERCENT	.3295E+06	.2866E+06	.2492E+06	.2167E+06	.1884E+06	.1638E+06	.6271E+06	-.4991E+08

INTERNAL RATE OF RETURN = 10.88 PERCENT

PAYBACK PERIOD - FROM FIRST REVENUE YEAR
UNDISCOUNTED VALUE 5.58 YEARS
DISCOUNT RATE 0 PERCENT EXCESS 35 YEARS
DISCOUNT RATE 5.0 PERCENT EXCESS 35 YEARS
DISCOUNT RATE 10.0 PERCENT EXCESS 35 YEARS
DISCOUNT RATE 15.0 PERCENT EXCESS 35 YEARS

	YEAR 15	YEAR 16	YEAR 17	YEAR 18	YEAR 19	YEAR 20	YEAR ++	TOTAL
GOVERNMENT TAX INCOME NET OF BONUSES ETC.								
UNDISCOUNTED INCOME	.1855E+10	.2041E+10	.2245E+10	.2470E+10	.2717E+10	.2486E+10	.2803E+10	.3406E+11
DISCOUNT RATE 0 PERCENT	.7989E+08	.7989E+08	.7989E+08	.7989E+08	.7989E+08	.7989E+08	.6813E+08	.2639E+10
DISCOUNT RATE 5.0 PERCENT	.1597E+08	.1521E+08	.1448E+08	.1379E+08	.1314E+08	.1251E+08	.1016E+08	.9721E+09
DISCOUNT RATE 10.0 PERCENT	.3440E+07	.3127E+07	.2843E+07	.2584E+07	.2349E+07	.2136E+07	.1656E+07	.4507E+09
DISCOUNT RATE 15.0 PERCENT	.7233E+06	.6999E+06	.5999E+06	.5216E+06	.4536E+06	.3944E+06	.2925E+06	.2453E+09
VALUE OF RETAINED FOREIGN EARNINGS								
UNDISCOUNTED INCOME	.3405E+10	.3745E+10	.4120E+10	.4532E+10	.4985E+10	.5484E+10	.8800E+10	.6615E+11
DISCOUNT RATE 0 PERCENT	.1466E+09	.1466E+09	.1466E+09	.1466E+09	.1466E+09	.1466E+09	.2139E+09	.4808E+10
DISCOUNT RATE 5.0 PERCENT	.2930E+08	.2791E+08	.2658E+08	.2531E+08	.2411E+08	.2296E+08	.3190E+08	.1667E+10
DISCOUNT RATE 10.0 PERCENT	.6312E+07	.5739E+07	.5217E+07	.4743E+07	.4311E+07	.3920E+07	.5198E+07	.7047E+09
DISCOUNT RATE 15.0 PERCENT	.1456E+07	.1266E+07	.1101E+07	.9573E+06	.8324E+06	.7238E+06	.9182E+06	.3356E+09
VALUE OF GOVERNMENT PARTICIPATION								
UNDISCOUNTED INCOME	.1537E+10	.1691E+10	.1860E+10	.2046E+10	.2251E+10	.2476E+10	0	.2597E+11
DISCOUNT RATE 0 PERCENT	.6619E+08	.6619E+08	.6619E+08	.6619E+08	.6619E+08	.6619E+08	0	.2176E+10
DISCOUNT RATE 5.0 PERCENT	.1323E+08	.1260E+08	.1200E+08	.1143E+08	.1088E+08	.1037E+08	0	.8318E+09
DISCOUNT RATE 10.0 PERCENT	.2850E+07	.2591E+07	.2355E+07	.2141E+07	.1947E+07	.1770E+07	0	.3992E+09
DISCOUNT RATE 15.0 PERCENT	.6573E+06	.5716E+06	.4970E+06	.4322E+06	.3758E+06	.3268E+06	0	.2244E+09