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DECISION-MAKING IN THE PRELIMINARY STAGES OF MINERAL DEVELOPMENT, USING MONTE CARLO SIMULATION

by

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A thesis submitted to the Faculty and the Board of Trustees of the Colorado School of Mines in partial fulfillment of the requirements for the degree of Master of Science in Mineral Economics.

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ABSTRACT

This thesis develops a simplified model for economic decision-making in the preliminary stages of mineral deposit development. It takes into consideration exhaustibility, the time value of money, and uncertainty in the key parameters. The economic feasibility of different rates and levels of recovery, as well as optimum values for these parameters, are determined.

The model was applied to the determination of the feasibility to produce titanium sponge from a disseminated titanium deposit located in Venezuela, for which open-pit methods and selective mining practices are required.

The results of the investigation indicate a feasible project under the assumptions used. Within wide ranges of capacity and cut-off grades economic rates of return on investment are indicated.

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ACKNOWLEDGMENTS

The author wishes to express his acknowledgments to his thesis advisor, Dr. Alfred Petrick, Jr., and the other members of the thesis committee.

Thanks are also due to the Ministry of Mines and Hydrocarbons of Venezuela, for the financial assistance and information provided to make possible the satisfactory completion of this thesis.

INTRODUCTION

The purpose of this thesis is to develop and apply a simplified model for decision making in the preliminary stages of development of a mineral project. It also permits the determination of the optimum values for the cut-off grade and mine capacity that usually are not determined at this stage.

The type of analysis used in this thesis requires information from the ore deposit, the process involved, and markets. The analysis permits more rational decisions at the prefeasibility stage and provides information for more detailed studies.

It is assumed that in this stage of analysis there is no detailed engineering design and that the project is compared to similar projects to obtain an order-of-magnitude estimate of feasibility.

A Monte Carlo simulation technique is used to deal with uncertainty. The economic indicators of net present value (NPV) and discounted cash flow rate of return (DCFROR) are calculated for determination of an optimum system.

The first two chapters present a summary of the basic concepts used in developing the model and a description of the model itself.

Specific characteristics of the San Quintin titanium project in Venezuela which was evaluated for sponge production by a computer model, is discussed in the third chapter.

The final chapter presents the conclusions made from an analysis of the crucial parameters related to this project.

The appendices to the thesis present some additional aspects related to cost estimation, the listings, inputs and outputs of the computer program, and flow sheets of the processes proposed for producing titanium metal. It also includes the results of analysis of samples taken from the ore deposit.

CHAPTER 1. BASIC CONCEPTS

The information required in order to analyze a mineral project is mostly related to three fundamental sources: the ore deposit, the processing technology from extraction to refining (including transportation), and the markets. A simplified chart of procedures and calculations in mineral project evaluation is presented in Figure 1.1.

The market analysis permits determination of prices and their trends as well as the relationships between prices and output of any one producer.

The exploration of the ore deposit provides two different kinds of information:

1) It provides information on characteristics of the deposit and physical properties of the ore which affect mineability and the metallurgical process which is feasible. This information will affect the cost equations and determine parameters such as dilution and recovery.

2) It provides information on the tonnage of reserves and average ore grade. Each deposit will have a different

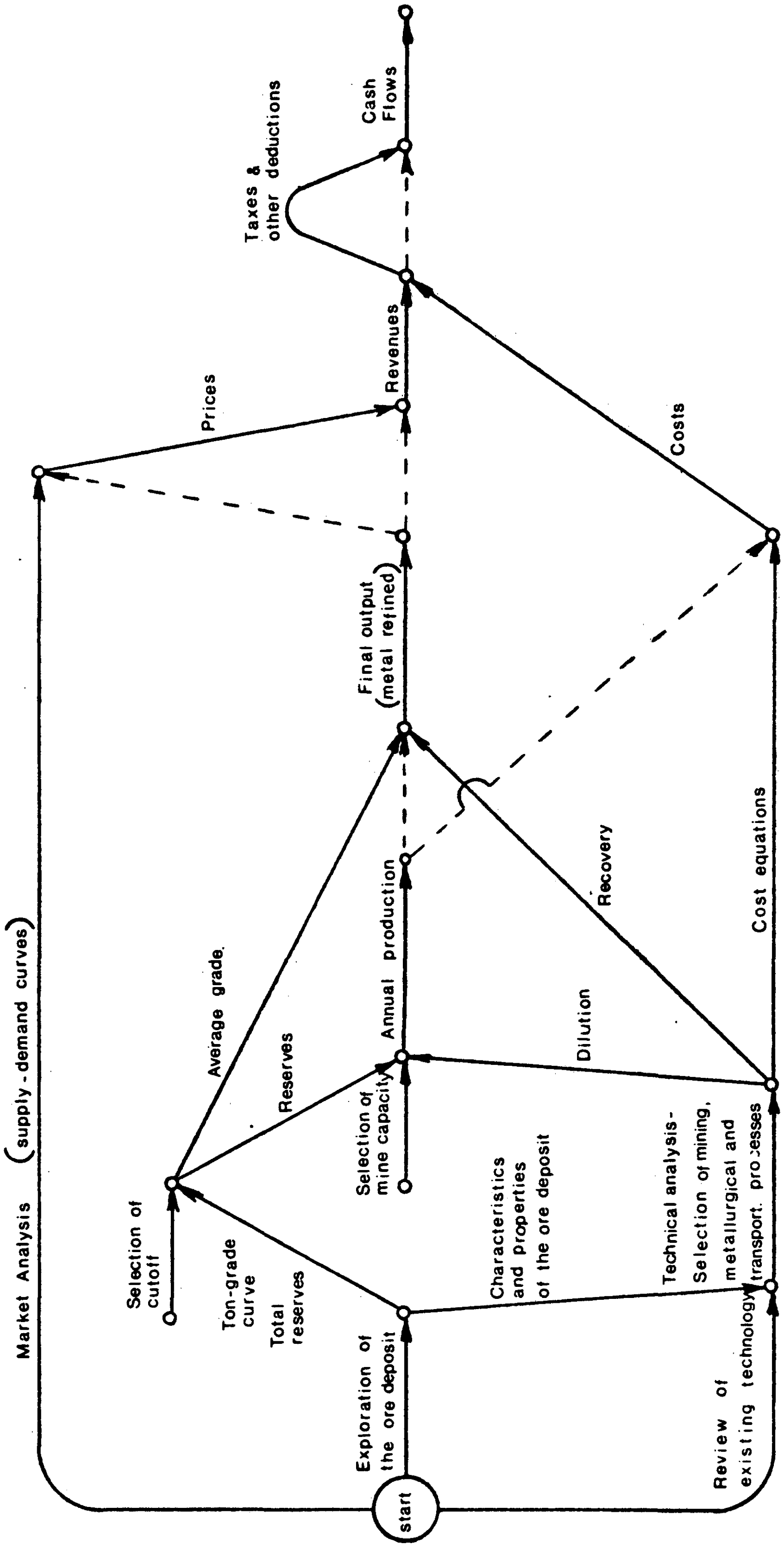


Figure 1.1 Simplified Chart of Procedures and Calculations
in Mineral Project Evaluation

tonnage-grade relationship; however, it is possible to determine generalized equations that represent approximately a type of deposit. In this thesis, the case of disseminated deposits is the only one considered.

Review of existing technology provides a third kind of information. Different mining and metallurgical processes can be analyzed and a selection made depending on the characteristics of the ore deposit. The process selected determines the cost equations and the amount of final refined metal produced. It is also necessary to consider cut-off grade and mine capacity. When analyzing a mineral project, cut-off grade and mine capacity are parameters that must be determined. They will be critical in definition of an optimum for exploitation of a mineral deposit.

The generalized equations for the ore deposit, cost behavior, and definition of an optimum are discussed in this chapter.

1.1 Ore Deposit Assumptions

There are two key factors to be determined with respect to the ore deposit.

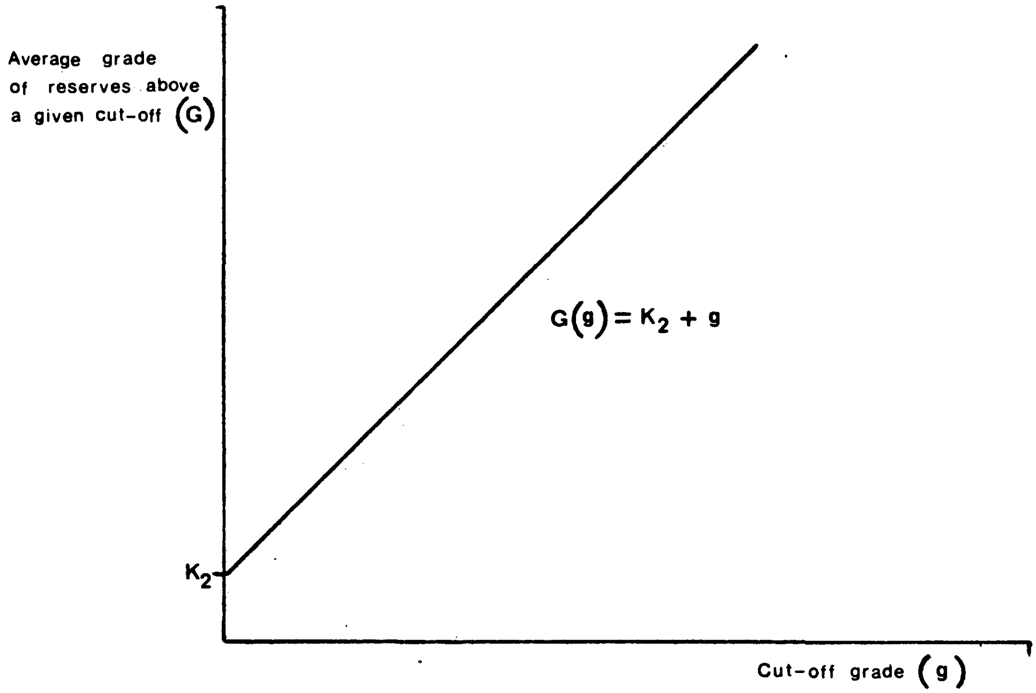
1) The physical properties and the mineralogy of the ore affect mineability and choice of metallurgical process. These in turn determine the cost behavior.

2) The tonnage-grade relationship must be determined in order to calculate the ore reserve tonnage for a specific cut-off grade. Independent samples from an ore deposit can be used to determine the average grade and volume of a particular deposit. This method will give an approximate value for reserves; however, more detailed techniques would be required for a higher degree of accuracy.

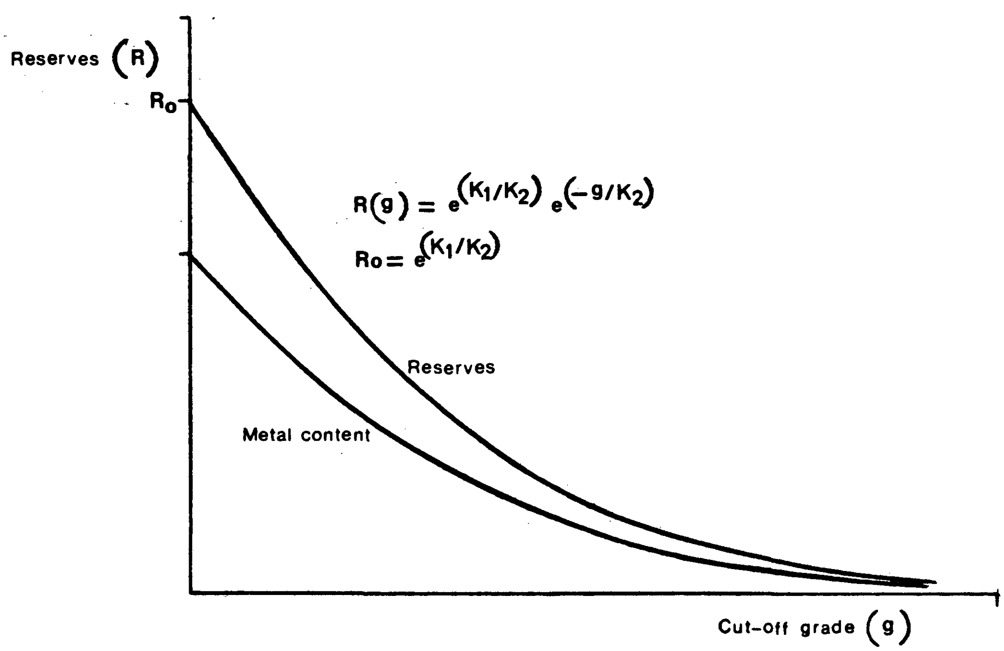
Figure 1.2 shows some conceptual relationships between the amount of ore reserves and the average grade and cut-off grade of reserves. The exponential relationships shown in Figure 1.2-b is frequently used (Koch, 1971) for deposits where reserves gradually expand as cut-off grade is lowered. This is the case of disseminated deposits which are considered in this work. The following equation was developed by Lasky (1950) upon analysis of the tonnage-grade relationships for several disseminated ore deposits.

According to Lasky tonnage and grade are related by the equation:

$$R(g) = e^{(k_1/k_2)}e^{(-g/k_2)},$$



a) Cut-off grade vs average grade of reserves



b) Cut-off grade vs reserves

Figure 1.2 Approximated Relationships Between Reserves, Cut-off Grade, and Average Grade in Disseminated Mineral Deposits where: "R(g)" represents the total amount of reserves above the cut-off grade "g",

The constants "k_1" and "k_2" have different values for different deposits, "k_2" represents the average grade of total deposit,

"e" is the base of natural logarithms, and

"g" represents the cut-off grade.

The total tonnage of the deposit is represented by R_o, whose equation is:

$$R_o = e^{(k_1/k_2)}.$$

In order to calculate the average grade of reserves above a given cut-off, also an approximate equation can be used (Mackenzie, 1974). The relationship between these two parameters is shown in Figure 1.2-a and is represented by the equation:

$$G(g) = k_2 + g,$$

where: "g" is the cut-off grade considered,

"G(g)" represents the average grade of reserves above the cut-off grade "g", and

"k_2" is the average grade of the deposit.

1.2 Cost Behavior

When some parameters as capacity are still not determined, the general cost behavior of the project is a typical case of the long-run in which all resource inputs have to be considered variable. Thus, there are not fixed costs in this case and the shape of the long-run cost curves, which differs from those in the short-run, have to be analyzed in order to determine costs at different rates of production.

Constructing large size plants results in greater efficiency up to a certain output, due to economies of large-scale production, but beyond this point, economies of large scale disappear and larger plants become progressively less efficient and ential higher unit costs.

Even though the determination of the point where economies of scale disappear is difficult, it is reasonable to assume that economies of scale diminish gradually and diseconomies of scale predominate after a specific rate of production.

A general function representing increasing returns to scale at the beginning and decreasing returns to scale at the end is illustrated in Figure 1.3. This function can be written as follows:

$$y = a + bx - cx^2 + dx^3,$$

where: "y" represents costs, and

"x" represents input capacity.

The different values of the constants a, b, c, and d, depend on the particular case and determine where economies of scale disappear and diseconomies of scale control.

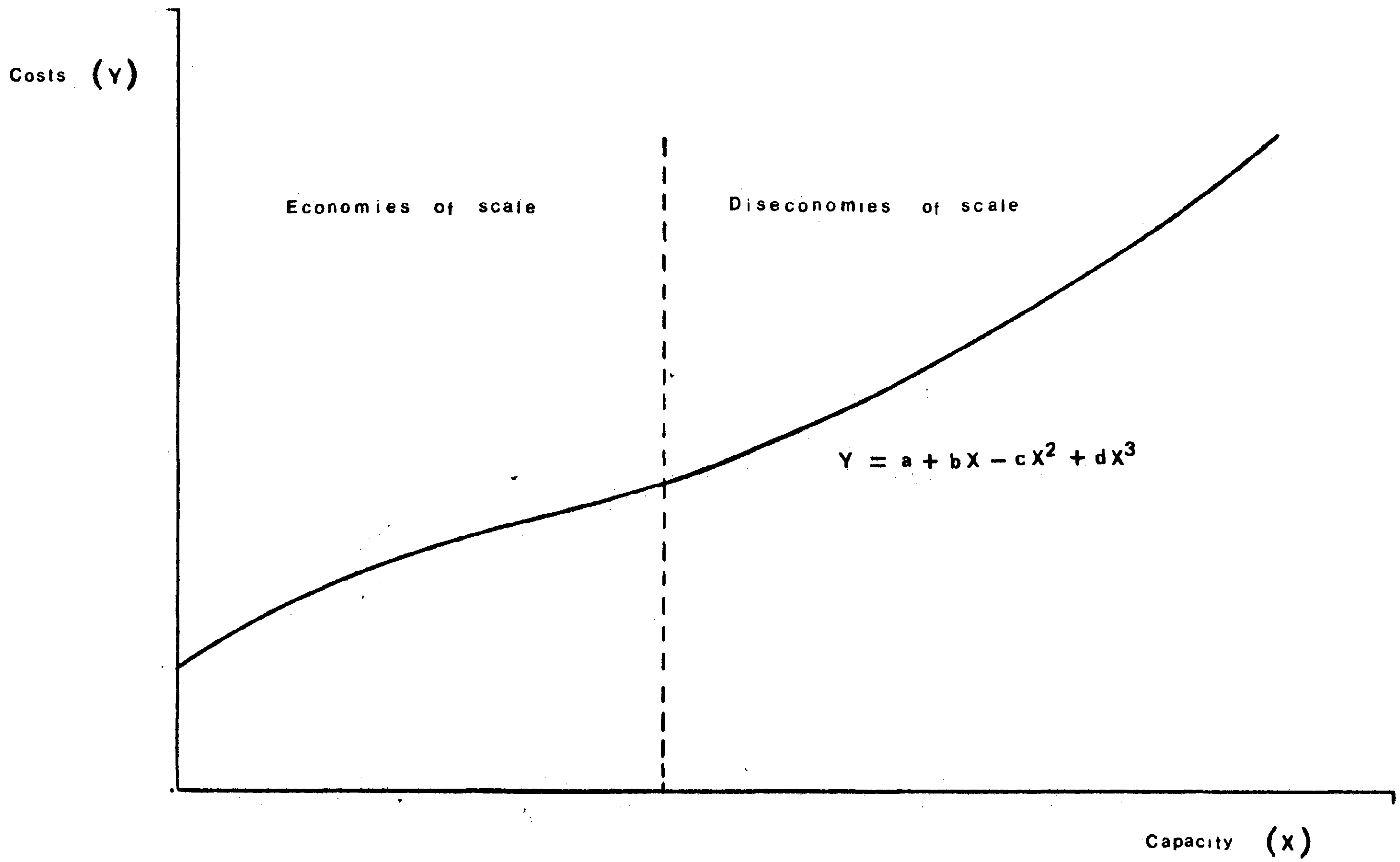


Figure 1.3 Cost Behavior with Increasing and Decreasing Returns as Capacity is Increased

1.3 The Optimum in the Mineral Industry

The maximization of profits is adopted in this thesis as the principal goal to achieve.

Usually in microeconomic analysis, the point of maximum profits in the short-run is identified at the point where marginal costs equals marginal revenue. In the case of mineral projects in a long-run analysis, factors such as time value of money and exhaustibility are involved, and it is necessary to modify the concept.

In order to achieve the point of profit maximization two basic parameters have to be determined in operating a mine: one, the rate of recovery or the total amount of mineral to be extracted in a specific period of time, and, in the case of an ore deposit in which selective mining is required, the level of recovery or cut-off grade (the minimum mineral grade to be extracted) which determines the total amount of ore in the deposit.

When analyzing a mineral project, for a given cut-off grade and where other parameters are held constant, it is possible to determine one value of the rate of production that permits maximization of NPV. Some authors agree that this point will fall somewhere in between the intersection of the average variable cost curve with the marginal cost curve and the intersection of the marginal revenue curve with the marginal cost curve (Carlisle, 1954).

What is important here is to achieve the optimization of a given function previously selected as a goal.

The net present value of the project (NPV) is adopted in this work as the objective function and optimum is defined as the values of rate of recovery and level of recovery which maximize the NPV given a specific attractive rate of interest.

When, for any reason, the attractive rate of interest is not known or not considered in the analysis, the maximization of the discounted cash flow rate of return (DCFROR) is used for determination of the optimum.

Summarizing, the objective function when assuming a specific attractive rate of interest can be written as follows:

$$\text{maximize NPV} = \sum_{j=1}^{n} C_j \frac{1}{(1+1)^j},$$

where: "n" represents the life of the project in years,

"C_j" represents the cash flow of the year "j", and

"i" is the attractive rate of interest.

When no attractive rate of interest is considered, the objective function can be written as follows:

Maximize i which satisfies the equation:

$$\sum_{j=1}^{n} C_j \frac{1}{(1+i)^j} = 0 \ ,$$

where: "n" represents the life of the project in years,

"C_j" represents the cash flow of the year "j", and

"i" represents the DCFROR.

CHAPTER 2. THE COMPUTER MODEL

2.1 Purpose

The purpose of the computer model is to determine the most likely value and lower limit for the NPV and the DCFROR of the mineral project, considering all possible combinations of cut-off grade and mine capacity and consequently the determination of the optimum.

As mentioned earlier, it assumes a minimum of available information on characteristics of the ore deposit, the metallurgical process involved, and the markets.

2.2 Model Elements

This section presents some of the most important equations used to develop the computer model.

2.2.1 Geological Parameters

As we said earlier, only the case of disseminated deposits is considered in this thesis. Also, some simplified equations are used to represent the relationships between parameters of the ore deposit. It is supposed that, for more detailed studies, these relationships have to be determined more precisely.

The relationships used in the model are defined as follows:

$$T = A/e^{(co/G)}$$

$$g = G + co$$

where: "A" represents the tonnage of total deposit,

"G" represents the average grade of total reserves,

"e" is the base of natural logarithms,

"co" is the cut-off grade, and

"g" is the average grade of reserves above the cut-off grade "c".

2.2.2 Economic Parameters

Costs and revenues in a mineral project depend on the quantity and quality of mineral extracted.

If the whole process in any metal industry is divided into subprocesses, the cost of each subprocess will depend mostly on the amount of mineral or metal which is introduced as input in each subprocess.

The total revenue depends on the amount and quality of the final products and the market prices.

Inputs and outputs for each subprocess are calculated by using the following equations:

$$X_2 = X_1 \times d,$$

$$X_n = \frac{P_{n-1}}{P_n} \times [\, r(n-1) + (AG - AGB) \times f(n-1)] \times X(n-1),$$

where: "X_1" is the first input. This value represents the total amount of material extracted or mine capacity.

"X_2" represents the mine output or total ore extracted. This amount is the input for the second subprocess,

"d" represents the dilution factor used to calculate the amount of ore,

"X_n" represents the input in the subprocess "n",

"P_n" represents the percentage of metal in the input of the subprocess "n",

"r_n" represents the recovery factor in the subprocess "n",

"AG" represents the percentage of metal in the input of the subprocess "n",

"AGB" is the percentage of metal in the input of the subprocess "n" which permits us to obtain a recovery equal to REC_n. When AG is different from AGB recovery could be different to REC_n, therefore a correction is required,

"f_{n-1}" is the factor used for correction of recovery when "AG" is different to "AGB." Its value depends on how the recovery is affected when "AG" is different to "AGB."

The final production (amount of mineral, concentrates, or metal refined to be sold in the markets) is used for calculation of revenues.

Revenues are obtained by using the following equation:

$$R_i = X_f \times P_i \ ,$$

where "R_i" represents the revenue in year "i",

"X_f" is the final production, and

"P_i" is the price in the year "i".

Both capital investment and operating costs are estimated by using the following equation:

$$C = \sum_{i=1}^{n} a_i + b_i X_i - c_i X_i^2 + d_i X_i^3 \ ,$$

where: "C" represents total investment or operating costs,

"n" is the number of subprocesses,

"a_i", "b_i", "c_i", and "d_i" are the coefficients of the cost equation for subprocess "i",

"X_i" is the capacity or input of the subprocess "i".

This value is corrected due to uncertainties by using the equation:

$$C' = S \frac{C}{E} \ ,$$

where: "C'" represents the final simulated cost,

"C" represents total cost (investment or operating cost) estimated from the cost equations,

"S" is the simulated value from the base probability distribution of costs, "E" represents the mean in the base probability distribution.

Trends in prices and costs are represented by the following function:

$$V_n = V_1[1 + G\ (D)^{n-1}\ (n-1)]\ ,$$

where: "V_n" represents the value of the variable in year "n",

"V_1" represents the value of the variable in year 1,

"G" is the annual trend in variable, and

"D" is a discount parameter which influences rate of change.

The calculation of cash flows from costs and revenues are performed by the subroutine CASHF in the computer program and are presented in the section program listings (Appendix II).

2.2.3 Uncertinaty Variables

In developing this model only four uncertainty variables were considered:

1) The average grade of the deposit,
2) the metal prices,
3) the capital costs, and
4) the operating costs.

The average grade (parameter "G" in the exponential tonnage-grade function) can be represented by any kind of probability distribution. Once "G" is determined, the tonnage-grade relationship is fixed and, for a given cut-off grade, the deposit is mined to the average grade of ore reserves over its productive life. One value is sampled from the distribution before every simulation.

Metal price values are sampled, independently, for each production year before every simulation.

The capital cost and annual operating costs in the first production year are derived from the cost equations taking into consideration capacities for every subprocess. These values are corrected using the values sampled from base probability distribution and time trends. The value obtained for total capital cost is divided by the number of pre-operational years and one operating cost is estimated for every year of the production period.

2.3 Method

The algorithm is based on the Monte Carlo Simulation techniques. Calculations are performed by generating a random number which results in selection of values of the variables consistent with their probability distribution.

A generalized flowsheet of the whole process is illustrated in Figure 2.1.

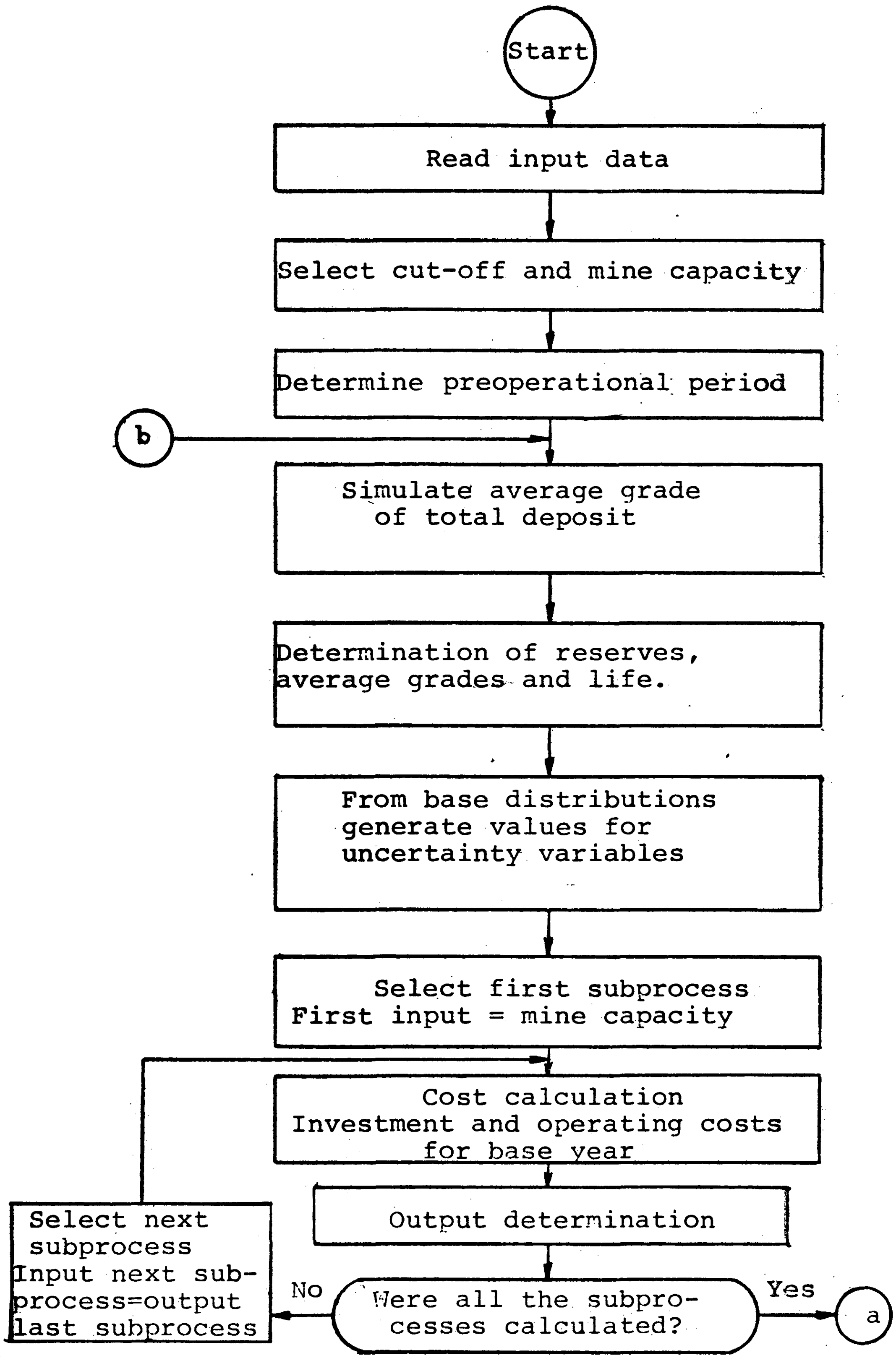


Figure 2.1 Simplified Flow-Diagram of the Computer Model

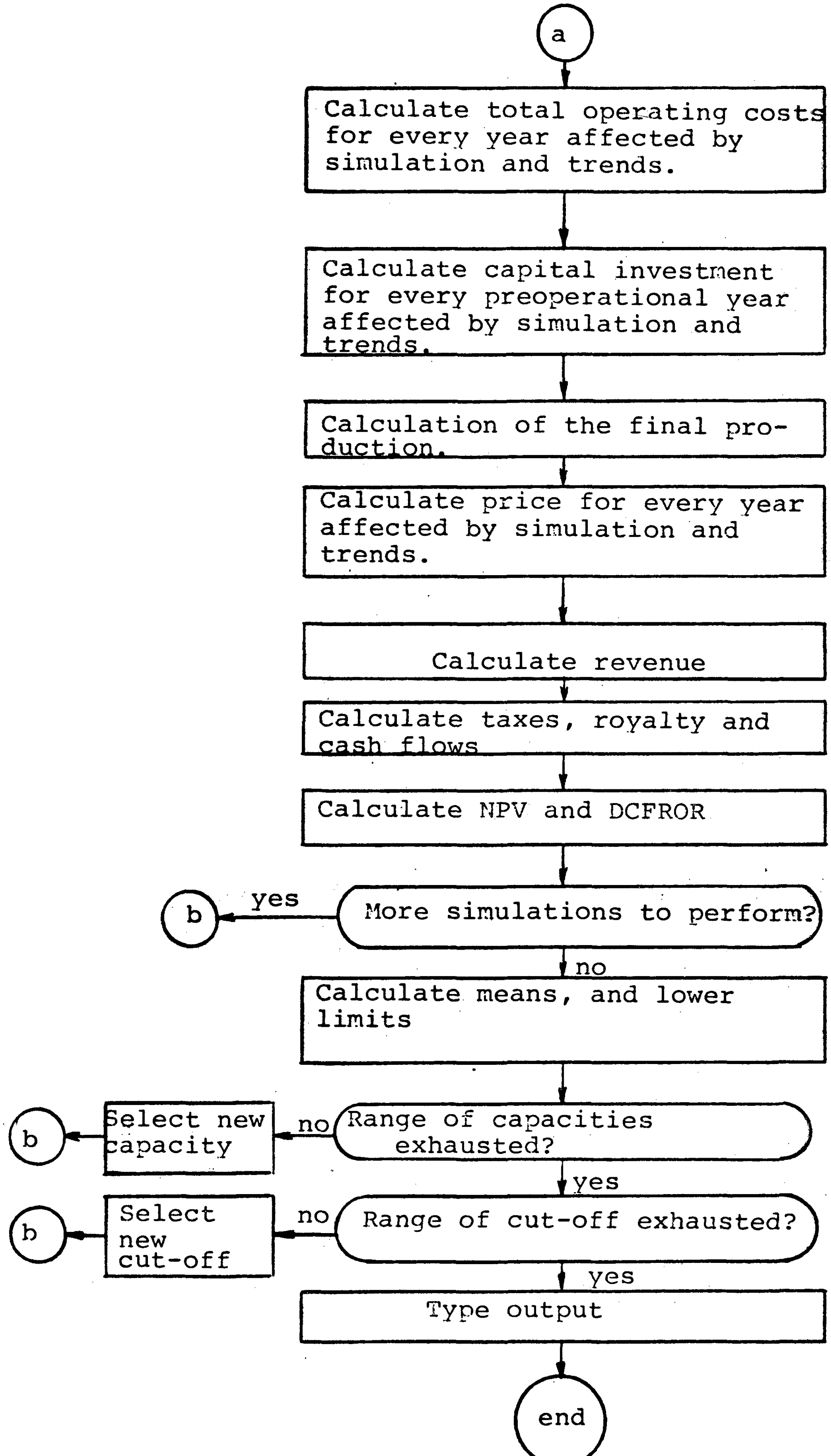


Figure 2.1 Simplified Flow-Diagram of the Computer Model (continued).

The different steps are as follows:

1) Selection of cut-off grade, mine capacity, and determination of preoperational period.

2) Simulation of average grade of total deposit using the distribution of means of the deposit.

3) Determine tonnage and average grade of reserves above the cut-off considered. Input data include tonnage of total deposit, average grade simulated, and equations which represent relationships among parameters of the ore deposit.

4) Determine the economic life of the project. Data required include total reserves, mine capacity, and dilution factors.

5) Simulate values for the remaining variables; one for each year for the life of the project. This includes investment, operating cost, and prices.

6) Calculation of inputs and outputs for each subprocess. The output of each subprocess is the input of the next subprocess. The first input is the mine capacity and the last output is the final production. These values depend on the annual production, the average grade, and recoveries in each subprocess.

7) Calculation of operating costs and working capital. The operating cost for each subprocess depends on the input to the subprocess. Total operating cost for each year is calculated based on results of the simulation.

8) Calculation of investment. The investment for each subprocess depends on the respective input calculated before. The total investment is scattered in the preoperational period affected by the simulated value for investment and trends.

9) Calculation of the final production. This value depends on the last output and the way the prices are given. When price is given per unit of metal the final product is the units of metal in the last output; if prices are given per unit of last output, the final product is the last output.

10) Calculation of prices for each year of the life of the project. Each price is obtained from the value previously simulated and affected by trends.

11) Calculation of revenues. Use corresponding values of production and prices. The revenue is estimated for each year of the life of the project.

12) Calculation of cash flows. This requires determination of depreciation rate, depletion rate, taxes, and royalties.

13) Calculation of the Net Present Value and Discounted Cash Flow Rate of Return using the cash flows.

14) Calculate the mean and lower limit of the net present values and rates of return.

15) Select new values for mine capacity and cut-off grade and repeat the process until all possible combinations or cut-off grade and mine capacity are calculated.

2.4 Program Description

2.4.1 Usage

The program was written in FORTRAN IV and it consists of a main program and seven subroutines, MAXCC, RAND, CASHF, RATE, PVALE, PVTAB, and WIRTE2. The main program reads the data, prints some results, and gives instructions to perform the principal subroutine, the MAXCC.

2.4.2 Subroutines Required

Subroutine MAXCC calculates investment, operating costs, prices, and final production for every year of the life of the project and gives instructions to execute the rest of the subroutines. Subroutine RAND is used to generate random numbers. It requires different seeds for each uncertainty variable in order to avoid correlation among random numbers generated. Subroutine CASHF determines cash flows. Before cash flows are determined, depreciation, depletion, and other deductions such as royalties are generated. In this thesis, a particular subroutine was used dealing with the specific case of exploitation of mineral deposits in Venezuela in which depletion is not considered. Subroutine RATE calculates the Discounted Cash Flow Rate of Return from the cash flows previously determined using the bisection method. Subroutine PVALUE claculates the Net Present Value on Profits given a specific attractive rate of interest. Subroutine PVTAB determines the means and lower limits for a specific confidence interval. Subroutine WRITE2 types means and lower limits obtained for all the combinations of cut-off grade and mine capacity.

2.4.3 Input Data Description

Parameters related to the characteristics of the ore deposit are:

NFTR: Kind of deposit,

A: Reserves of total deposit,

NAG: Number of class intervals in the average grade distribution,

VAG_i: Class midpoints in the average grade distribution,

$PIAG_i$: Corresponding relative frequency for each class interval in the average grade distribution,

DILU: Dilution factor relationship between the ore and the total material extracted,

Parameters related to the technology selected are:

NPRO: Number of subprocesses considered,

REC_i: Recovery factor for subprocess "i",

POR_{i+1}: Percentage of metal in the output of subprocess "i". When it is not known or equal to the last subprocess "POR_{i+1}" has to be defined zero, the correspondent value is assigned in the program,

CCJ_i, CCK_i, CCL_i, CCM_i: Independent coefficients of the capital investment functions for subprocess "i",

COJ_i, COK_i, COL_i, COM_i: Independent coefficients of the operating cost functions for subprocess "i",

PDI: Percentage of depreciable investment over total capital costs,

PWC: Percentage of working capital over operating costs,

AGB: Average grade reference used for determination of cost distributions,

CCO_i: Capacity reference of subprocess "i" used to determine cost distributions,

NCC: Number of class intervals in the capital investment cost distribution,

VCC_i: Class midpoints in the capital investment cost distribution,

$PICC_i$: Corresponding relative frequency for each class interval in the capital investment cost distribution,

NOC: Number of class intervals in the operating cost distribution,

VOC_i: Class midpoints in the operating cost distribution,

$PIOC_i$: Corresponding relative frequency for each class interval in the operating cost distribution,

$FREC_i$: Factor used to determine how the operating costs of subprocesss "i" are affected when "AGB" is modified,

IPPMI: Minimum preoperational period,

IPPMA: Maximum preoperational period,

LIFE: Maximum operational period,

RIC: Annual trend in capital investment,

ATOC: Annual trend in operating costs,

DPOC: Discount parameter which influences the operating costs rate of change,

Parameters related to the markets are:

NPR: Number of class intervals in the price distribution,

VPR(I): Class midpoints in the price distribution,

PIPR(I): Corresponding relative frequency for each class interval in the price distribution,

ATPR : Annual trend in prices,

DPPR : Discount parameters which influence rate of change,

NAFCPR: Factor used to determine the amount of final product.When "NFACPR" equals 1, price is given per unit of final product. When equal to zero, price is given per quantity of metal in the final product.

Other required parameters are:

TAX: Percentage of taxes on profits,

ROYAL: Royalties-dollars per ton of ore exploited,

IC: Confidence interval,

COMI- COMA: Minimum and maximum cut-off grades to be considered,

NCO: Number of cut-off grades,

CAPMI, CAPMA: Minimum and maximum capacities to be considered,

NCA: Number of capacities,

RINT1, RINT2: Minimum and maximum values for the attractive rate of interest,

NINCR: Number of interest rates,

IR: Number of data file,

IW: Number of output file.

CHAPTER 3. CASE STUDY

3.1 The Titanium Deposit of San Quintin

3.1.1 Location and Geology (Ministry of Mines and Hydrocarbons of Venezuela, 1975)

The disseminated titanium deposit of San Quintin is located in the north-central region of Venezuela (longitude E 68°45' and latitude North 10°40'). The deposit is located approximately 60 miles West of Puerto Cabello, an important port on the Caribbean Sea, and is shown in Figure 3.1. The area is easily accessible by road.

The San Quintin complex is a geologic unit which belongs to the orogenic system located in the northern part of the State of Yaracuy. It is surrounded by young tertiary units, specifically the Pozon formation, which is characterized by conglomerates, limestones, sandstones, marls, and shales. These rocks separate the units of the San Quintin complex from the metamorphic rocks of the Tarana and La Surda complexes located to the west and to the east.

The following lithologic associations crop out in the San Quintin Complex.

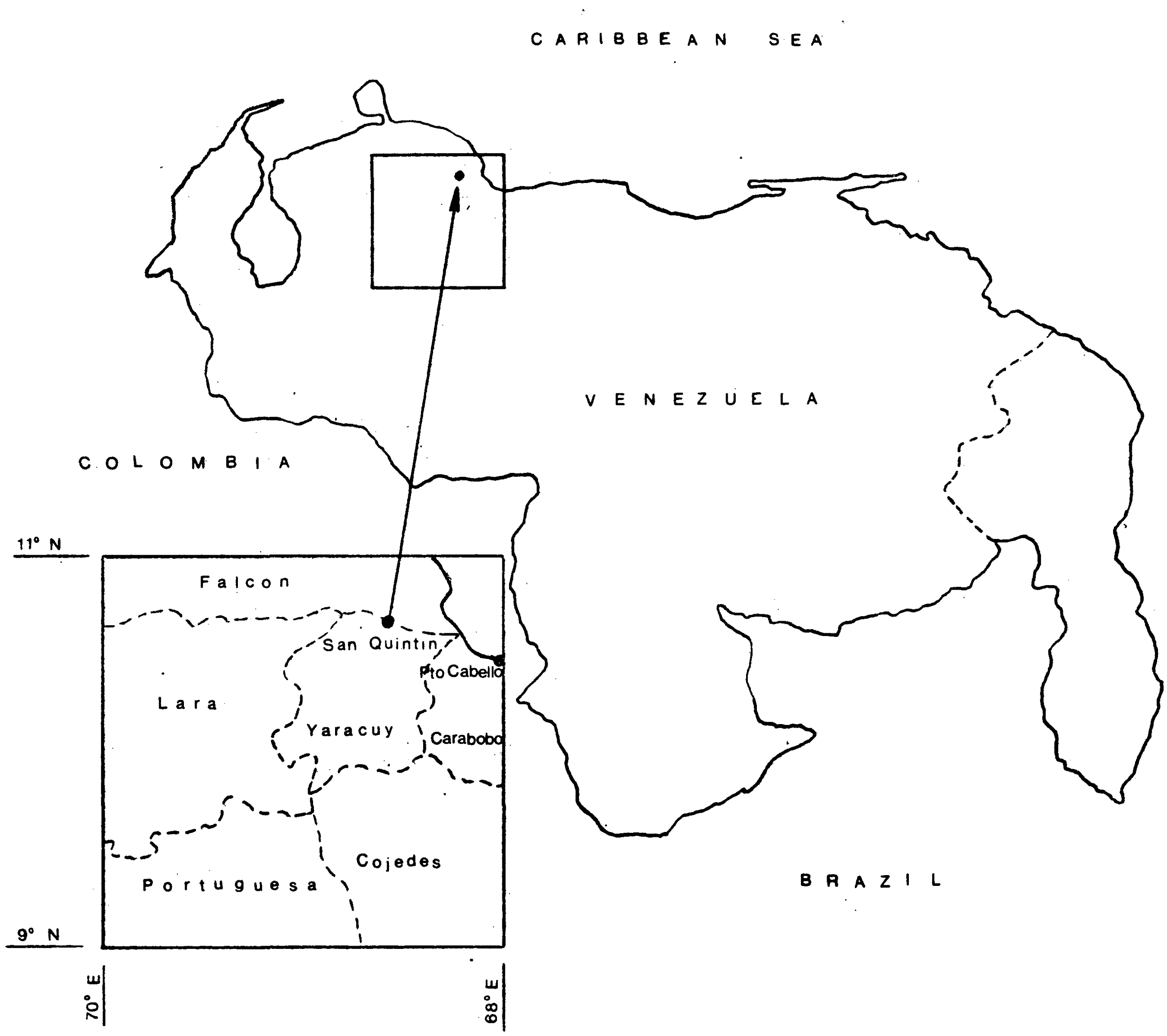


Figure 3.1 San Quintin Deposit Location Map

a) Meta - sandstone, meta-conglomerates, meta-cherts, and meta-marls - Casapal formation.

b) Andesites and basalt flows, metamorphosed and in contact with ultrabasic and basic igneous rocks - San Quintin Volcanics.

c) Gabbroic and chloritics facies, including gneisses sometimes altered. They present pyroxenitic and amphibolitic differentiations, infrequently with mineralization of titaniferous magnetite.

d) Amphibolitic pyroxenites, garnitiferous amphibolitic pyroxenites, pyroxenitic amphibolities, and hematite.

The southern part of the San Quintin Complex contains a higher enrichment of ilmenite than the northern part, and studies there have indicated a lithology characterized by remarkably banded anorthosites containing ferromagnesian facies. These rocks are in direct contact with very dense andesite-and-basalt flows that are lacking in titaniferous minerals.

The metallic minerals, especially ilmenite and hematite, in lamellar intergrowth, are very common, and commonly constitute up to 40 percent of the rock.

The anorthosites of the San Quintin Complex range in composition from monomineralic (plagioclase) to rocks of dominantly amphibole composition.

In the titaniferous ore, the ilmenite-hematite minerals occur with intimate intergrowths and are constituents in an anorthositic sequence of the igneous section.

The ore occurs as elongated lenses, massive bodies, and in disseminated form, throughout the anorthositic host rock. The percentage of ilmenite-hematite decreases in some ferromagnesian facies within the anorthositic rock. Rutile is present in the rock, but its percentage is much smaller than that of ilmenite, and occurs as irregular grains in the plagioclase crystals.

The mineralogic x-ray diffraction analyses on massive ore gives a ratio of ilmenite-hematite from 2/3 to 1/3. Plagioclase is the most important gangue mineral, along with pyrite and quartz. Magnetite constitutes approximately 2 percent of the iron oxides.

The combined iron and titanium oxides represent 98.02 percent of the ore. The specific gravity of the ore ranges from 4.1 to 4.5.

The chemical analysis of the massive ilmenite-hematite ore from the drilling program is shown in Table 3.1.

TABLE 3.1

Results of the Chemical Analysis of the Massive Ilmenite-Hematite Ore

TiO_2	32.7%
FeO	39.8%
Fe_2O_3	35.92%
MnO	traces
CrO_2	traces
CaO	nill
MgO	nill
Volatiles	.94%
HCl (insoluble)	1.04%

The above results are the basis for ore-reserve estimation. The data were obtained after initial exploration, which consisted of 52 drill holes. The location map of drill holes and a diagram summarizing results obtained from drill hole S-14 are illustrated in Figures 3.2 and 3.3, respectively.

3.1.2 Ore Reserves

A preliminary estimate of all reserves in the San Quintin deposit was made by the Ministry of Mines and Hydrocarbons of Venezuela (MMH) in 1975. This ore reserve study was based on the results obtained from 52 drill holes completed during initial exploration. The Ministry's study estimates that the

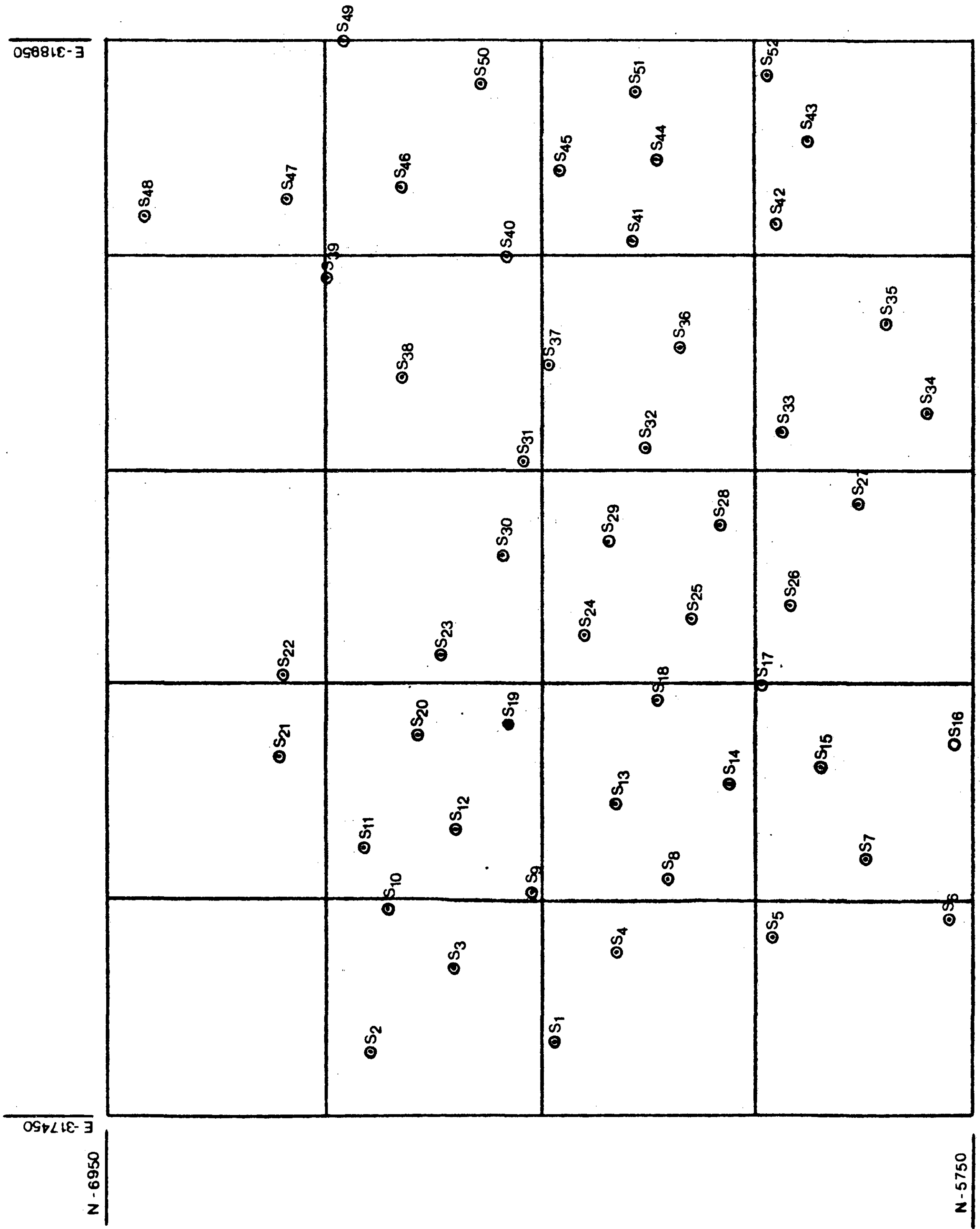


Figure 3.2 Location Map of Drill Holes

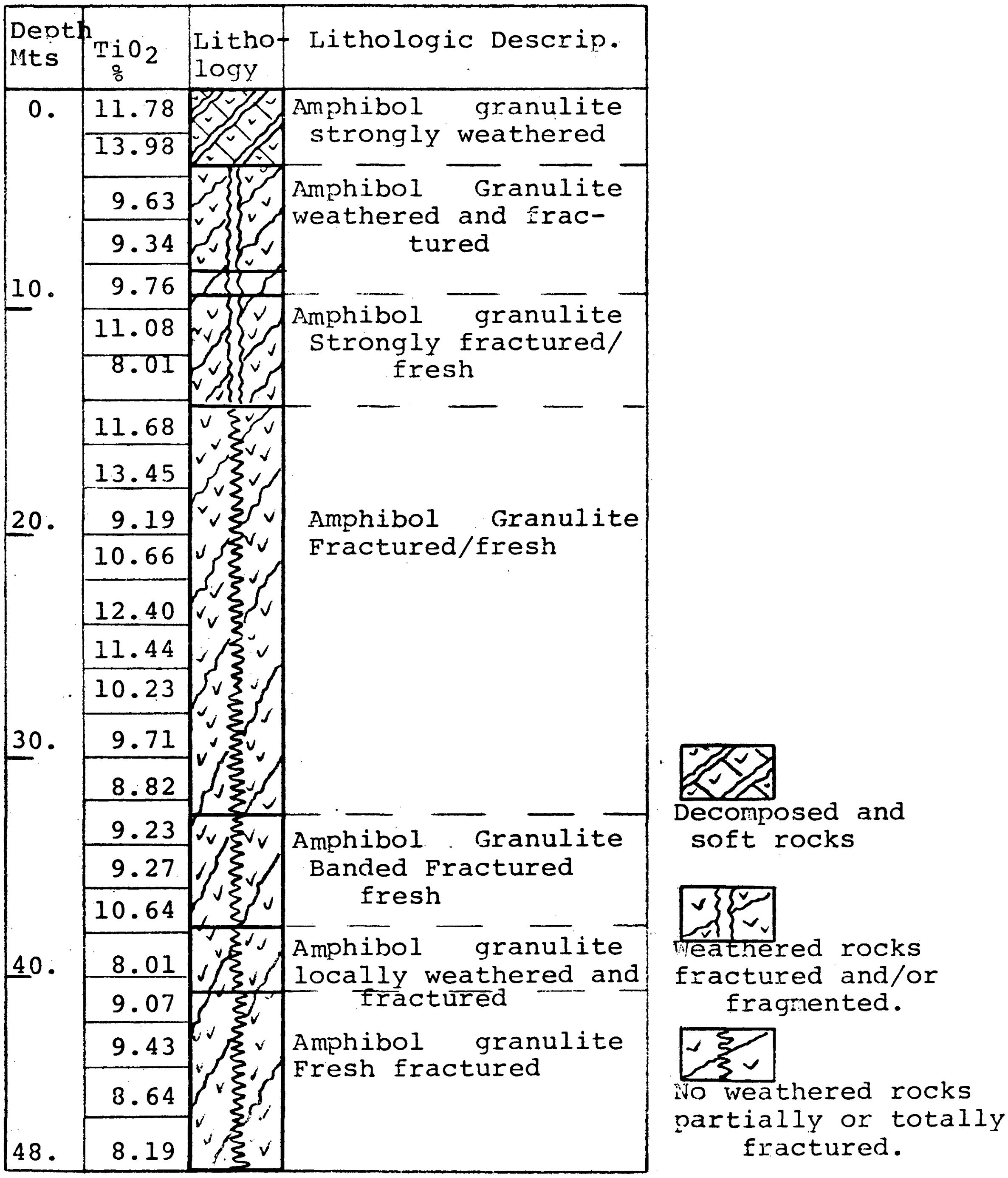


Figure 3.3 Drill Hole S-14 Showing Titanium Assays and Lithologic Units.

deposit contains about 30 million tons of ore averaging 5.98 percent titanium dioxide and 3.59 percent titanium metal. At the time of preparation of this thesis, no additional information on the deposit was available, therefore it was necessary to make assumptions about the general statistical distribution of means obtained from independent samples, which is the number of possible average grades of the total deposit, each one associated with one probability of occurrence.

3.2 Current Technology

The mining and processing of titanium ores includes the several steps which are described in this section. A general flowsheet for the process is represented in Figure 3.4.

3.2.1. Mining and Benefication (Industrial Minerals and Rocks, 1975)

The method of mining and beneficiating titanium minerals depends on whether the ore to be mined is a sand deposit or rock deposit.

Rock deposits located on or near the surface require open pit methods. Such is the case in the San Quintin deposit. Similar deposits are mined out using 45 ft. benches in an open pit, large blast-hole drills, electric shovels, and diesel-electric trucks.

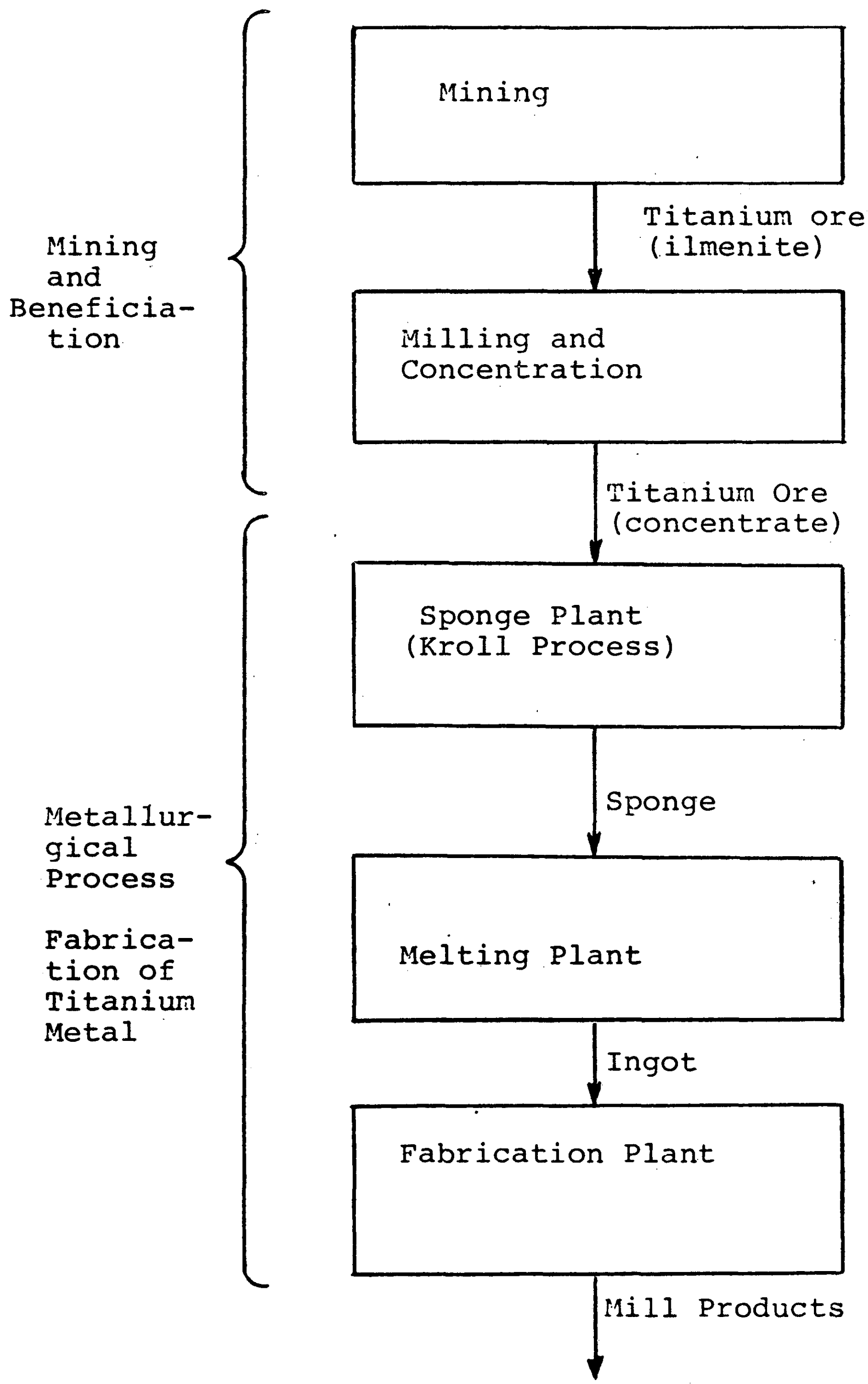


Figure 3.4 Generalized Flowsheet of the Process of Mining and Extracting Titanium Metal from Ores

The beneficiation process includes crushing, grinding, flotation, and sometimes magnetic separation.

In the case of the ilmenite deposit of MacIntyre Development, N.Y. (USA), ore from the mine is reduced in three stages of crushing to 9/16-in. size. At the 2 1/2-in. size there is a magnetic separation step which discards about 20 percent by weight as a waste. Ore is reduced in size to 65 mesh in rod mills and ball mills. Magnetic separators remove the magnetite fraction of the ore, and the non-magnetics are treated by flotation to produce ilmenite concentrate and tailing.

A flowsheet for all flotation and another for magnetic and flotation ilmenite are presented in Appendix III.

3.2.2 Commercial Processing and Fabrication of Titanium (Williams, 1965)

The commercial manufacture and fabrication of titanium involves (1) the production of sponge, (2) the conversion of sponge and alloying materials to ingots, (3) the conversion of ingots to mill shapes, and (4) the fabrication of finished products from the mill shapes.

Sponge is produced by the basic Kroll process using either magnesium or sodium to reduce the titanium tetrachloride previously obtained from the titanium ore by a process of chlorination. Magnesium is recovered from the resulting magnesium chloride for reuse.

The ingots are produced by double-melting consumable electrodes in a vacuum. The electrodes are made of sponge, home scrap, and alloying metals. The ingots are converted into mill shapes by conventional processes used in the stainless steel industry and in some cases on the identical equipment. The home scrap is recoverable and is returned to the melt-shop from which it reappears in ingots.

The mill shapes are converted to finished products in fabricating shops which generally are not an integral part of the titanium metal industry.

Shop equipment and the techniques of fabricating titanium are quite similar to those employed by fabricators of stainless steel and similar industrial metals.

3.3 Titanium Markets (Bureau of Mines, B750)

3.3.1. Industry Pattern

The titanium industry is characterized by a moderately high degree of integration from raw materials to semi-finished products. The ilmenite produced in the world comes mostly from two mines in the U.S., one mine in Canada, one in Norway, five in Australia, and an unknown number in the USSR.

Titanium dioxide pigment output in 22 countries comes from 70 separate facilities ranging in annual capacity from a few thousand tons to 125,000 tons. On the basis of annual titanium pigment capacity, the U.S. accounts for an estimated

35 percent of the world total, followed by the U.S.S.R. and other communist countries with an estimated 25 percent, the United Kingdom (8 percent), West Germany (7 percent), Japan (6 percent), and France (4 percent). The U.S. accounts for about one-half of the world productive capacity for titanium metal followed by the U.S.S.R., Japan, and the United Kingdom.

The National Lead Co., (National Lead), with mines in the United States and Norway, controls approximately 50 percent of the world reserves of ilmenite.

National Lead and the E.I. du Pont de Nemours & Co., Inc. (du Pont), own or control 35 percent of the world productive capacity for titanium pigment. The two British firms, BTP and La Porte Titanium, Ltd (La Porte), control about 20 percent. An estimated 25 percent is owned by Communist governments. The remaining 25 percent is principally owned by large chemical firms or groups such as the American Cyanamid Co., Glidden-Durkee Division of the SCM Corporation, The New Jersey Zinc Co.; Farbenfabriken Bayer A.G.; Montecatini Edison, S.P.A.; and Ishiharo Saugyo Kaisha, Ltd. About 85 percent of the productive capacity in the 22 producing countries is owned by firms that were based in the producing country or by its government. National Lead, with plants in West Germany, Canada, and Norway, owns about half of the remaining 15 percent, and du Pont, BTP, and La Porte own the remainder.

The National Lead Co. with perhaps 50 percent of its estimated worldwide sales of $1 billion stemming from its titanium mining, pigment and metal operations, is the most heavily invested firm in titanium in the world. Titanium is believed to represent a relatively small part of the overall operations of the other major firms.

The two largest titanium metal producers in the U.S. were jointly owned by large chemical companies (National Lead and National Distillers & Chemical Corp.) and steel companies (United Steel Corp. and Allegheny Ludlum Steel Corp.). Armco Steel Corp. is a joint owner of the third domestic sponge producer. In Japan, titanium metal production is associated principally with steel companies. In the United Kingdom, Imperial Metal Industries Ltd. produces titanium metal, zirconium, and refractory metals.

3.3.2. Production and Consumption

World production of titanium concentrates comes mainly from four major producing countries, Australia, Norway, Canada, and the U.S., with about 800,000 tpy each, and from Finland, India, Malaysia, and Sri Lauka each producing 80,000 to 175,000 tpy. The U.S. consumes a large share of the ilmenite and rutile produced in the world. Most of the ilmenite in Canada is converted to titanium slag which is shipped to the U.S., West Germany, and other European countries, and to two titanium pigment plants in Canada. Australian ilmenite output is shipped to the United Kingdom, France, Japan, the U.S., and other countries, and some is used in Australian titanium pigments plants. The U.S. consumes most of the ilmenite it produced.

3.3.3 Prices

The price for all forms of titanium metal showed a declining trend since the metal was first produced commercially and became relatively stable after the year 1962.

In the specific case of titanium sponge price remained relatively stable in $1.25/lb until the year 1972, with increases up to $2.85/lb in the year 1975, and a new decrease during 1976 in which prices for titanium sponge have been quoted in between $2.45/lb and $2.70/lb. The historical record of prices for titanium sponge is shown in Figure 3.5.

3.4 Exploitation Alternative

The possibility of producing titanium sponge from the titanium ore of San Quintin is considered in this thesis.

Five different subprocesses were analyzed. First, mining and crushing; second, transportation of the mineral from the mine to the metallurgical plant near the port by trucks; third, milling and concentration; fourth, fabrication of titanium sponge; and fifth, transportation of the titanium sponge from the port to the markets by ship.

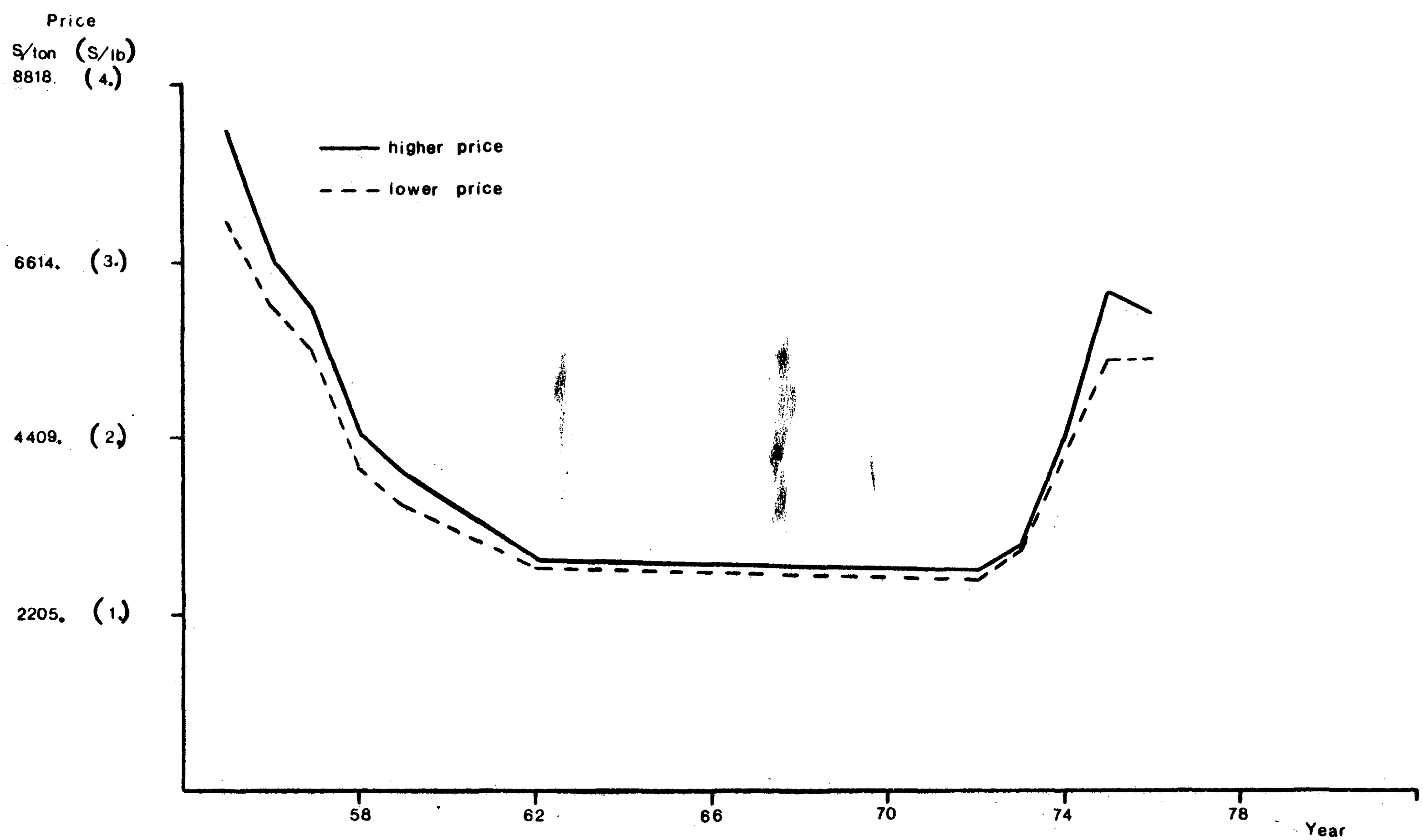


Figure 3.5 Prices for Titanium Sponge 1955-1976

Source: Engineering and Mining Journal and Mineral Industry Surveys-USBM.

A generalized flowsheet of the process is illustrated in Figure 3.6. All the calculations performed in this work are based in this alternative and are illustrated in the following section.

3.5. Parameters Estimation

The parameters required as input data in the computer model are divided into four principle groups:

1) those related to the characteristics of the ore deposit,

2) those related to the technology required for mining and processing,

3) those related to market conditions, and

4) all other parameters needed to evaluate project economics.

3.5.1 Ore Deposit Parameters

The parameters which directly depend on the ore deposit are:

1) the kind of deposit,

2) the amount of reserves of the total deposit,

3) the average grade probability distribution, and

4) the dilution factor.

The titanium deposit of San Quintin is a rock deposit located near the surface in which open pit methods and selective

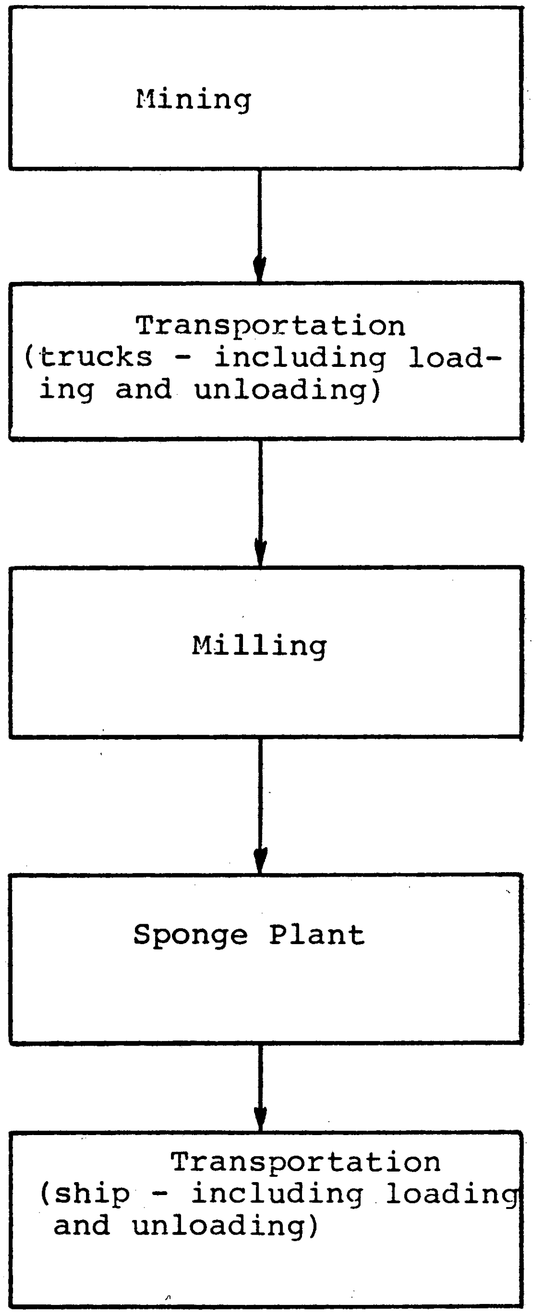


Figure 3.6 Generalized Flowsheet of the Process to Produce Titanium Sponge. San Quintin Deposit.

mining are required. This kind of deposit is identified by defining the variable NFTR equal to 1.

The amount of ore reserves estimated by the Ministry of Mines and Hydrocarbons of Venezuela is adopted in this thesis. The Ministry study estimates that the deposit contains about 30 million tons of ore.

For an analysis of this type, detailed sampling information is needed to associate probabilities with different ore grade. In this case no such information was available. Therefore, it was necessary to assume a range for the probability distribution of means based on the approximate value estimated by the MMH.

The assumed probability distribution is presented in Table 3.2.

TABLE 3.2

Average Grade Probability Distribution

Average Grade of the Deposit (percentage of Ti)	Associated Probability
3.05	.005
3.23	.045
3.41	.200
3.59	.500
3.77	.200
3.95	.045
4.13	.005

The dilution factor, which is the relationship between the ore and the total material extracted, was estimated using some statistical information for U.S. open pit ilmenite deposits (SME, 1973). From this information, a value of 0.89 for dilution was estimated.

This factor is used to determine the amount of ore extracted by multiplying it times mine capacity. It means that part of the material extracted in the mine is wasted.

3.5.2 Technology Parameters

The parameters which depend on the selected processing were grouped as follow:

1) Subprocesses, recoveries, and metal contents,
2) Cost equations, probability distributions, and trends,
3) Other parameters related to technology.

3.5.2.1 Subprocesses, Recoveries, and Metal Contents. As mentioned earlier, five different subprocesses were analyzed for production of titanium sponge from the San Quintin titanium ore. First, mining and crushing; second, transportation of the mineral from the mine to the plant near the port by trucks; third, milling and concentration; forth, fabrication of titanium sponge; and fifth, transportation of the titanium sponge from the port to the markets by ship.

Recoveries have to be considered in milling and sponge plants. In titanium milling plants recovery factors range from 81 percent to 90 percent (Bureau of Mines, 1970). A value of .85 percent was assumed in this thesis for recovery in the milling plant.

The recovery factors in sponge plants ranges around 80 percent (Williams, 1965), therefore, this value was adopted in this thesis as recovery in this subprocess. For more detailed studies, these values of recovery have to be determined more precisely by analyzing the ore of the San Quintin deposit.

The average grade of the ore depends on the cut-off selected and is determined by the computer program.

It was assumed 36 percent of titanium metal in the titanium concentrates and 99.7 percent in the titanium sponge (Williams, 1965).

<u>3.5.2.2 Cost Equations, Probability Distributions, and Trends</u>. The basic purpose of this analysis is the determination of some generalized equations which should represent as close as possible the behavior of costs of the new project for different capacities of production.

In order to update and estimate the real costs in the place where the project is located, the utilization of some escalation and location factors are required.

In order to apply correction factors, costs must be divided into major cost elements. The kind and quantity of these cost elements depends on the accuracy needed and judgement of the estimator. For preliminary estimates, only major cost elements are required. Investment can be divided into engineering, construction, and equipment categories. The construction element includes labor and material.

Operating costs will include both direct and indirect elements. The direct costs can be divided into labor, materials, and utilities categories. The materials category will include supplies and raw materials. Utilities will include fuel, electricity, and water. The indirect costs will include administration, overhead, and miscellaneous cost categories.

Sometimes it is necessary to break down costs as much as possible because of the importance and differences in trends and other factors of some particular items. But if it does not occur, it is better to use the lesser amount of cost elements as possible in order to simplify calculations.

After the determination of cost elements and correction factors the following equation will be applied in cost calculations:

$$C_{B_Y} = C_{A_X} \sum_{i=1}^{N} \left[P_{A_i} \left(I_{(Y/X)B_i} \right) \left(\frac{V_{B_i}}{V_{A_i}} \right) \right]$$

where:

A represents the place source of information,

B is the place where the new project is located,

X is the date of information,

Y is the year to update,

N is the number of cost elements,

C_{BY} is the total cost in place B and year Y,

C_{AX} is the total cost in place A and year X,

P_{A_i} is the percentage of the total cost for item i in place A,

$I_{(Y/X)B_i}$ is the escalation factor = Index year Y/ Index year X for item i in place B, and

$\frac{V_{B_i}}{V_{A_i}}$ is the location factor = value in place B/ value in place A for item i.

This section of the thesis will develop the cost equations for mining, milling, titanium sponge production, transportation by trucks from mine to port, and transportation by ship from port to markets.

In order to develop the cost equations for both capital and operating costs in mining, data from several titanium deposits in the United States and Canada were analyzed. The titanium deposit of San Quintin is a hard rock shallow deposit requiring open pit methods for extraction of the ore.

The most important cost elements to be taken into consideration for capital investment in mining are engineering, construction, and equipment.

It is convenient to break down investment costs in this way because of the availability of indices to determine escalation and location factors.

In order to determine the possible ranges for each of these cost elements, three U.S. open pit mines, an iron ore, a bauxite, and a copper mine, were analyzed (SME, 1973). These values were compared with some other kinds of industrial processes such as chemical plants in order to make some adjustments.

The final breakdown of estimated costs is presented in Table 5.3.

TABLE 3.3

Investment Costs Breakdown - Mining

Cost Element	Weight (%)
Engineering	7
Construction	26
Equipment	67

These values are used to determine the general equation which represents mining investment costs for different mining capacities.

In order to determine the general cost equation for investment, some statistical data from U.S. mines were obtained (Pfleider, 1972 and SME, 1973). Most of the information available was from the year, 1967. Therefore the utilization of indices to update these values to the year, 1976 was required. The Chemical Engineering plant cost index, for engineering and supervision, was used for engineering costs. The Engineering News Record construction cost index was used to update construction costs and the Marshall and Stevens equipment cost index for mining and milling was used to update the cost of equipment.

Various aspects influencing location factors were studied. Some aspects were taken into consideration in order to determine location factors. Freight costs were considered for equipment since most of it will be transported from foreign countries. Differences in prices for lumber, cement, steel and labor, were used to estimate location factors for construction; and, differences in cost for supervision and engineering were considered for engineering costs.

A summary of cost elements, escalation, and correction factors used to calculate a composite index to determine the generalized cost equation for investment in mining is presented in Table 3.4.

TABLE 3.4

Composite Index Estimation - Mining Investment

Cost Element	P_{A_i}	$I_{1967_{B_i}}$	$I_{1967_{B_i}}$	V_{b_i/A_i}	$P_{A_i}\left(I_{(Y/X)B_i}\right)\frac{V_{B_i}}{V_{A_i}}$
Engineering	.07	149.1	107.9	0.729	.07
Construction	.26	2326.9	1070.0	0.470	.27
Equipment	.67	469.3	263.5	1.120	1.33

Composite Index: $\sum_{i=1}^{3} P_{A_i} \left(I_{(Y/X)B_i}\right) \frac{V_{B_i}}{V_{A_i}} = 1.67$

Table 3.5 presents a summary of the mining investment costs at different production capacities (Pfleider, 1972 and SME, 1973).

TABLE 3.5

Mining Investment Costs at Different Capacities

Ton/day ore and waste	ton$\times 10^6$/year ore & waste	Investment $\$\times 10^6$(US-1967)	Correction Factor	Investment 8×10^6(Vla-1976)
29,000	9.05	15	1.67	25
38,200	11.92	19	1.67	32
40,000	12.48	20	1.67	33
45,800	14.29	23	1.67	38
56,000	17.47	28	1.67	47
74,200	23.15	37	1.67	65

These values were used to determine the general investment equation in mining. The equation obtained which is graphically represented in Fig. 3.7, is expressed as:

$$Y = (2.68)X,$$

where Y represents capital investment in mining, and

X represents mining capacity.

The breakdown of operating costs for mining into cost elements was based on the analysis of several Canadian deposits (Canadian Mining Journal, 1973); and, the percentage weights for several cost element which are derived from the 1967 Census of Mineral Industries are used to construct the indices for principal metal mining expenses (Bureau of Mines, 1973).

Both direct and indirect costs were considered for determination of percentage weights for each cost element.

It was assumed that the cost of the principal mining expenses cover approximately 80 percent of the total costs in mining. Some other costs such as administration, overhead, and other indirect costs were estimated to represent approximately 20 percent of total costs. This value was obtained as an average upon analysis of several U.S. mines (SME, 1973). Approximately the same breakdown of costs used in the Census of Mineral Industries was adopted in this thesis. But it was considered convenient to do some modifications. Explosives, steel mill shapes and forms, and all other supplies were

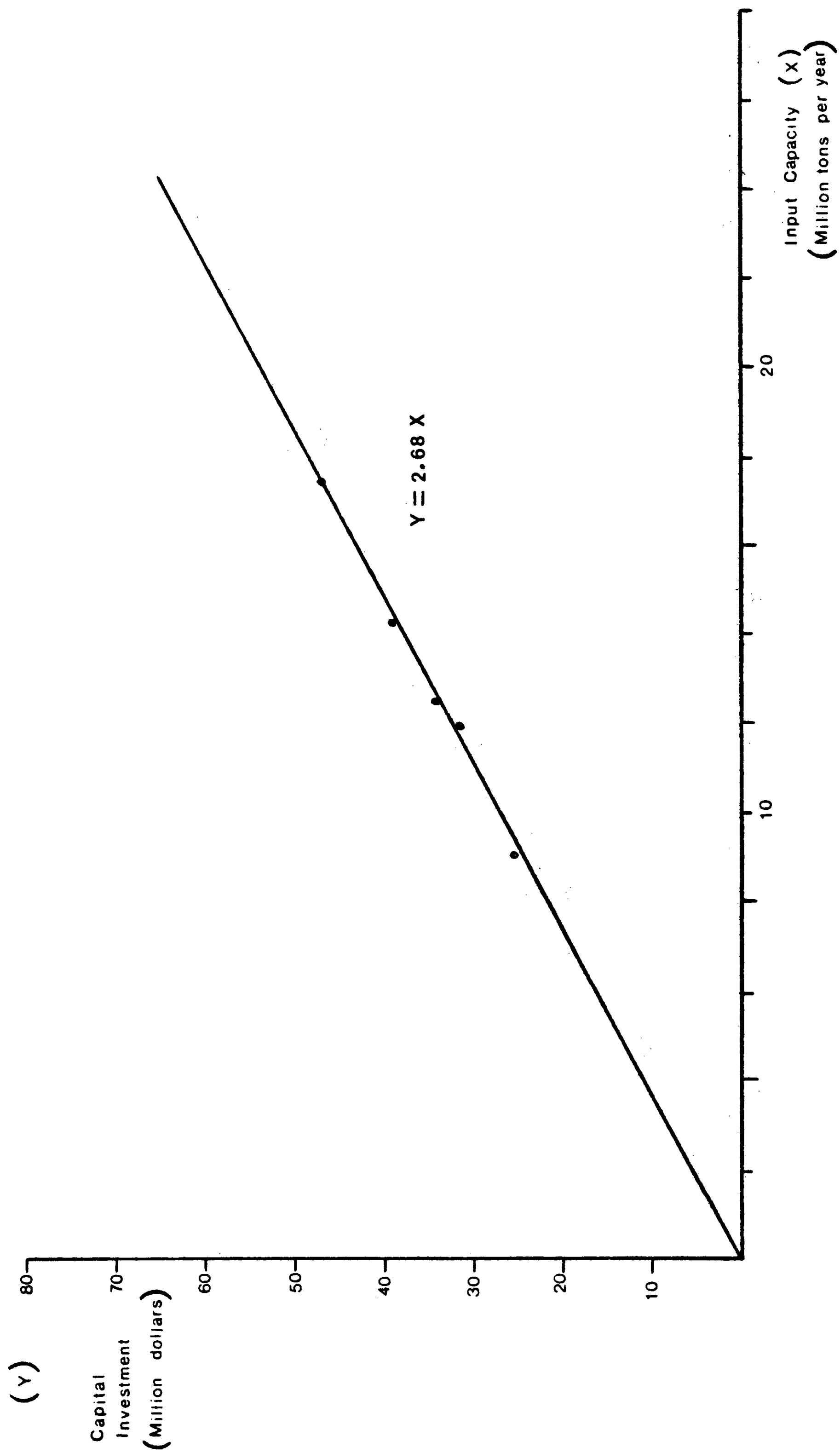


Figure 3.7 Mining Investment Costs versus Annual Capacity

grouped together as materials, and another cost element was included to consider administration, overhead, and other costs.

The final breakdown for operating costs for mining is presented in Table 3.6.

TABLE 3.6

Operating Costs Breakdown - Mining

Cost Element	Weight (%)
Labor	40
Materials	30
Fuels	5
Electric energy	5
Others (adm., overhead, etc.)	20

Most of the information used to estimate operating costs for mining was from the year, 1972. In order to determine the escalation factor, indices of Principal Mining and Milling Expenses were utilized (Bureau of Mines, 1973).

A summary of cost elements, escalation, and correction factors used to calculate a composite index is presented in Table 3.7.

TABLE 3.7

Composite Index Estimation -
Mining Operating Costs

Cost Element	P_{A_i}	$^{I}1976_{B_i}$	$^{I}1972_{B_i}$	V_{B_i}/V_{A_i}	P_{A_i} I(Y/X)Bi $\frac{V_{B_i}}{V_{A_i}}$
Labor	.400	145	120	0.3	.15
Materials	.300	145	120	1.12	.40
Fuels	.050	192	119	.50	.04
Electric Energy	.050	154	122	.66	.04
Others	.200	150	120	.725	.18
Composite Index					0.81

In order to determine the generalized cost equation for operating costs in mining, six different open pit mines were analyzed (Canadian Mining Journal, 1973). These costs for different capacities are presented in Table 3.8. The mines selected for the analysis were:

1. Asbestos Corp. British Canadian,
2. Asbestos Corp. King Beaver,
3. Bethlehem Copper Corp.,
4. Granby Mining Phoenix,
5. Granite and Copper, and
6. Norand Mines Granite Flux.

TABLE 3.8

Mining Operating Costs for Different Capacities

Mine	Ton/day (Ore+ Waste)	Tonx10^6/ year (Ore+ Waste)	Total op. cost \$/ton [direct (1.20)]	Total Cost \$x10^6/ Year	Correction Factor	Total Op. Cost Vzla-1976 \$x10^6
(1)	59,918	18.69	0.37	6.917	.81	5.603
(2)	33,000	10.30	0.42	4.326	.81	3.504
(3)	61,000	19.03	0.40	7.612	.81	6.166
(4)	14,440	4.50	0.53	2.403	.81	1.946
(5)	29,150	9.09	0.43	3.949	.81	3.198
(6)	2,845	.89	1.53	1.357	.81	1.100

These values were used to determine the general equation for operating costs in mining. The points were plotted and represented in Figure 3.8.

The equation obtained was:

$$Y = (0.89)\ 10^6 + (0.261)\ X,$$

where Y represents operating costs in mining, and

X represents mining capacity.

In order to determine milling costs, several mines from the U.S. and Canada were analyzed. Only three points with different milling capacities were estimated, but it was enough to assume a general equation for investment cost behavior.

Some statistical information from the year, 1967, for three different milling capacities was available and was used for calculation of milling investment costs (Pfleider, 1972). Three main cost elements were taken into consideration: engineering, construction, and equipment. The final breakdown of investment

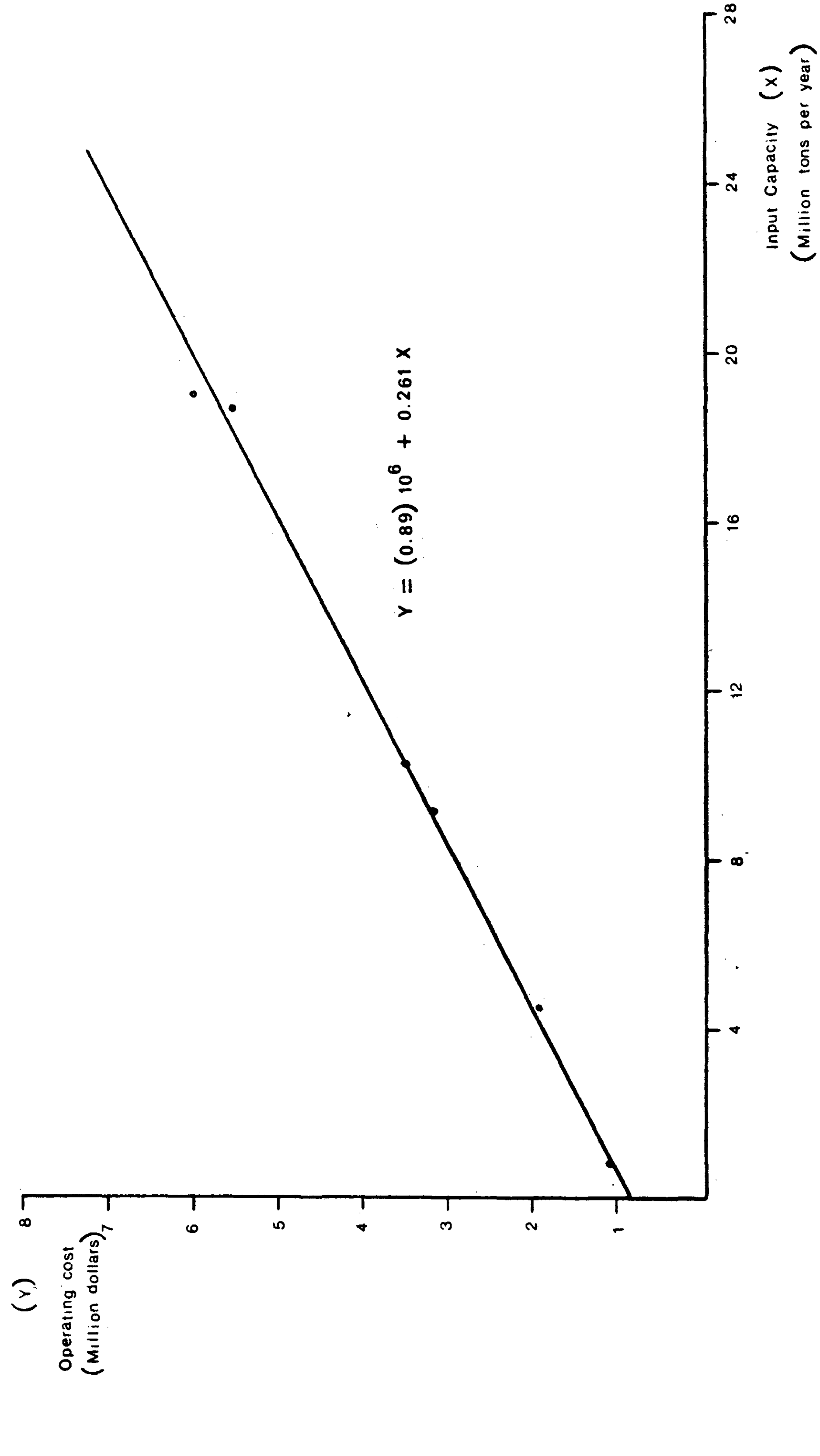


Figure 3.8 Mining Operation Costs versus Annual Capacity

costs for milling was obtained after the analysis of several different plants (Tovar, 1972). The percentage weights for the cost elements considered are presented in Table 3.9.

TABLE 3.9

Investment Costs Breakdown - Milling

	Weight (%)
Engineering	7
Construction	48
Equipment	45

The same correction factors applied for determination of mining costs were applied for milling investment costs.

A summary of cost elements, escalation, and correction factors used to calculate the composite index for milling investment are presented in Table 3.10.

TABLE 3.10

Composite Index Estimation - Milling Investment

Cost Element	P_{A_i}	I_{1976B_i}	I_{1967B_i}	V_{Bi}/V_{A_i}	$P_{A_i}\left(I(Y/X)_{B_i}\right)$	$\frac{V_{B_i}}{V_{A_i}}$
Engin.	.07	149.1	107.9	0.725	.07	
Const.	.48	2326.9	1070.0	0.470	.48	
Equip.	.45	469.3	263.5	1.120	.90	
Composite Index:					1.45	

This composite index was used to determine the milling investment costs in Venezuela for the year, 1976. The summary for the three milling capacities analyzed are presented in Table 3.11.

TABLE 3.11

Milling Investment Costs for Different Capacities

Ton/Day Ore	Tonx10^6/year Ore	Total Investment \$ x 10^6	Correction Factor	Cost Invt. Vzl-1976 \$ x 10^6
16,000	4.99	24	1.45	35
18,300	5.71	27	1.45	39
21,200	6.61	32	1.45	46

The general equation for investment in milling was determined from these values. The points were plotted and presented in Figure 3.9.

The equation obtained was:

$Y = (7.02)X,$

where Y represents total milling investment, and

X represents milling capacity.

The determination of operating costs for milling is based in the analysis of several U.S. and Canadian milling plants (SME, 1973 and Canadian Mining Journal, 1973).

In order to obtain a simplified breakdown of costs, both direct and indirect, milling costs were grouped into the same categories: labor, supplies, utilities, and others.

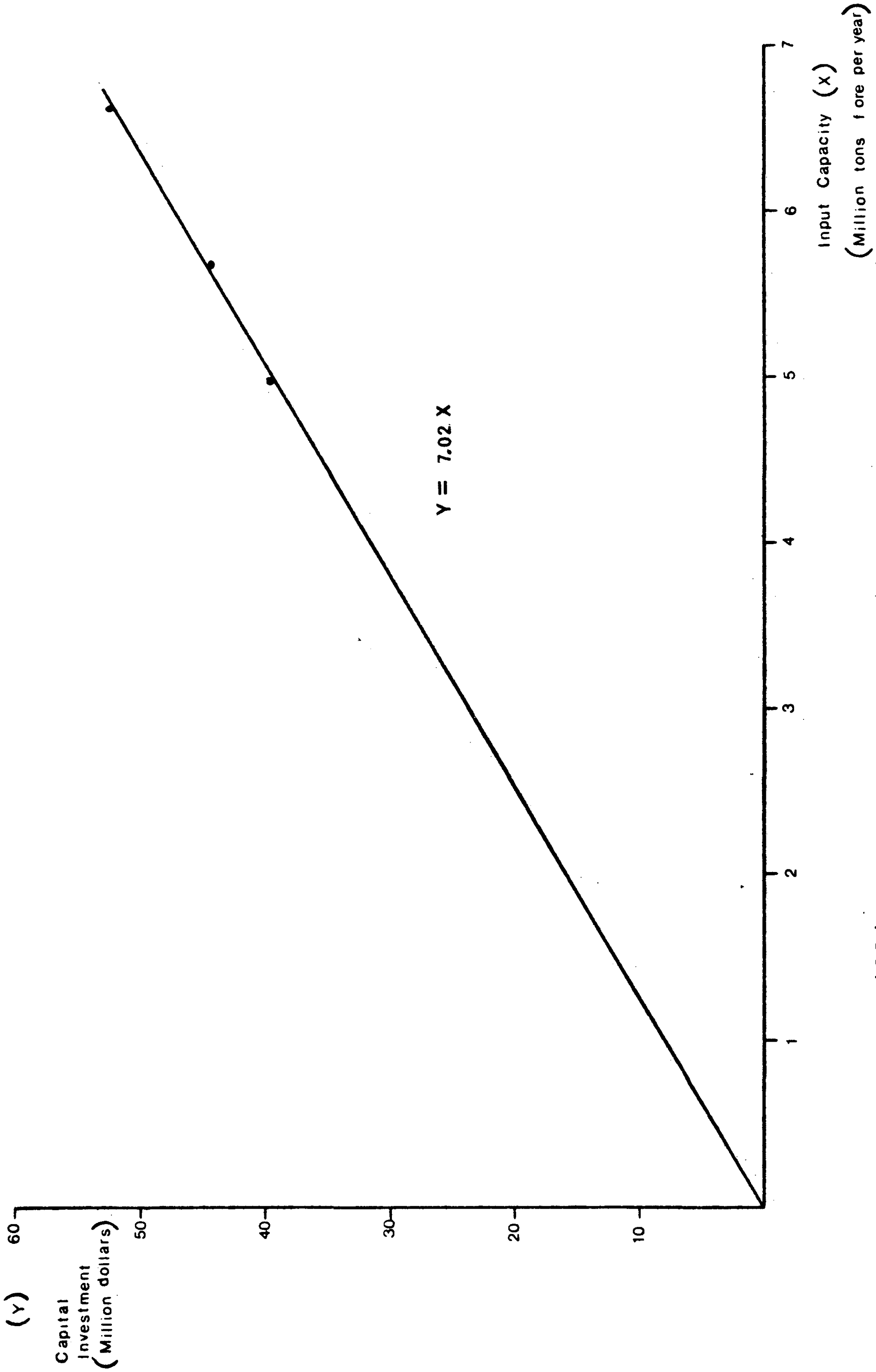


Figure 3.9. Milling Investment Costs versus Annual Capacity

The breakdown of operating costs for milling was obtained by analyzing information available from U.S. and Canadian mines. These results are presented in Table 3.12.

TABLE 3.12

Operating Costs Breakdown - Milling

	Weight (%)
Labor	39
Supplies	35
Utilities	14
Others	12

The labor category for milling plant generally ranges from 25 to 60 percent of total cost, depending on the type of operation and its size, location, and quality of labor. In this case, labor includes operating, repair, and indirect labor. The supply category is a major item of milling expense and includes reagents, grinding media, mill liners, and repair parts. Steel consumed in crushing and grinding is also a major expense item, varying with the hardness of ore, the required fineness of milling, and the characteristics of the grinding media. Consumption ranges from 1/2 lb per ton for soft ores, such as uranium ores, up to 3 1/3 lbs per ton for hard ores.

The utilities category includes power, water, and heat. Power costs for milling operations range between 10 percent and 50 percent of total costs.

The "other" category includes administration, overhead, insurance and miscellaneous costs.

Most of the information available for estimating operating costs of milling was dated 1972 or 1973. Therefore, it was necessary to use 2 different composite indices to correct the data for time and location.

The indices for Principal Mining Expenses (Bureau of Mines, 1973), were used for determination of the escalation factors.

A summary of cost elements, escalation, and location factors, and, the determination of the composite index to update information from both 1972 and 1973 are presented in tables 3.13 and 3.14, respectively.

TABLE 3.13

Cost Element	P_{A_i}	I_{1976B_i}	I_{1972}	V_{Bi}/V_{Ai}	$P_{Ai}\ I(Y/X)_{Bi}\ \frac{V_{Bi}}{V_{Ai}}$
Labor	.39	145	120	0.30	.14
Supplies	.35	145	120	1.12	.46
Util.	.14	154	122	0.66	.12
Others	.12	150	120	0.68	.11
Composite Index (1972)					0.83

TABLE 3.14

Composite Index Estimation (1973) - Milling Operating Costs

Cost Element	P_{A_i}	I_{1976B_i}	I_{1972}	V_{B_i}/V_{a_i}	$P_{A_i}\ I(Y/X)_{B_i}\ \frac{V_{B_i}}{V_{A_i}}$
Labor	.39	145	126	0.30	.13
Supplies	.35	145	128	1.12	.44
Util.	.14	154	129	0.66	.11
Others	.12	150	128	0.68	.10
Composite Index (1973)					0.78

In order to determine the generalized cost equation for operating costs in milling, several open pit Canadian mines were analyzed (Canadian Mining Journal, 1973).

The mines selected for the analysis were:

1. Bethlehem Copper Corp. (Cu),
2. Camflo Mines (Au),
3. Craigmont Mines (Cu, Fe),
4. Granby M.C. Phoenix (Cu, Ag, Au),
5. Grauisle Copper (Cu),
6. Similkameen Mining (Cu), and
7. Texada Mines (Ag, Pb, Zn).

The result of the estimation is presented in Table 3.15.

TABLE 3.15

Milling Operating Costs for Different Capacities

Mine and Year	Mill Capacity $tx10^6$/year	Milling Cost \$/ton	Total Op. Cost $\$x10^6$/year	Correction Factor	Total Op Cost Vzla 1976 \$ $x10^6$
1('72)	4.86	0.738	3.584	.83	2.975
2('73)	0.32	1.543	0.500	.78	.390
3('73)	1.68	0.462	0.779	.78	.607
4('73)	0.82	1.091	0.899	.78	.701
5('73)	4.21	0.756	3.184	.78	2.484
6('72)	4.68	0.704	3.282	.83	2.833
7('73)	1.19	1.022	1.211	.78	.945

These values were used to determine the general equation for operating costs in milling. The points were plotted and represented in Figure 3.10.

It was assumed in this case that diseconomies of scale start at any point after 2 million tons per year of milling capacity.

The equation obtained was:

$$Y = X - (2.9777)(10^{-7})\ X^2 + (4.41138)(10^{-14})X^{3*},$$

where Y represents operating costs in milling, and

X represents milling capacity.

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* An explanation of the procedure to determine the independent coefficients for this kind of equation is presented in Appendix I.

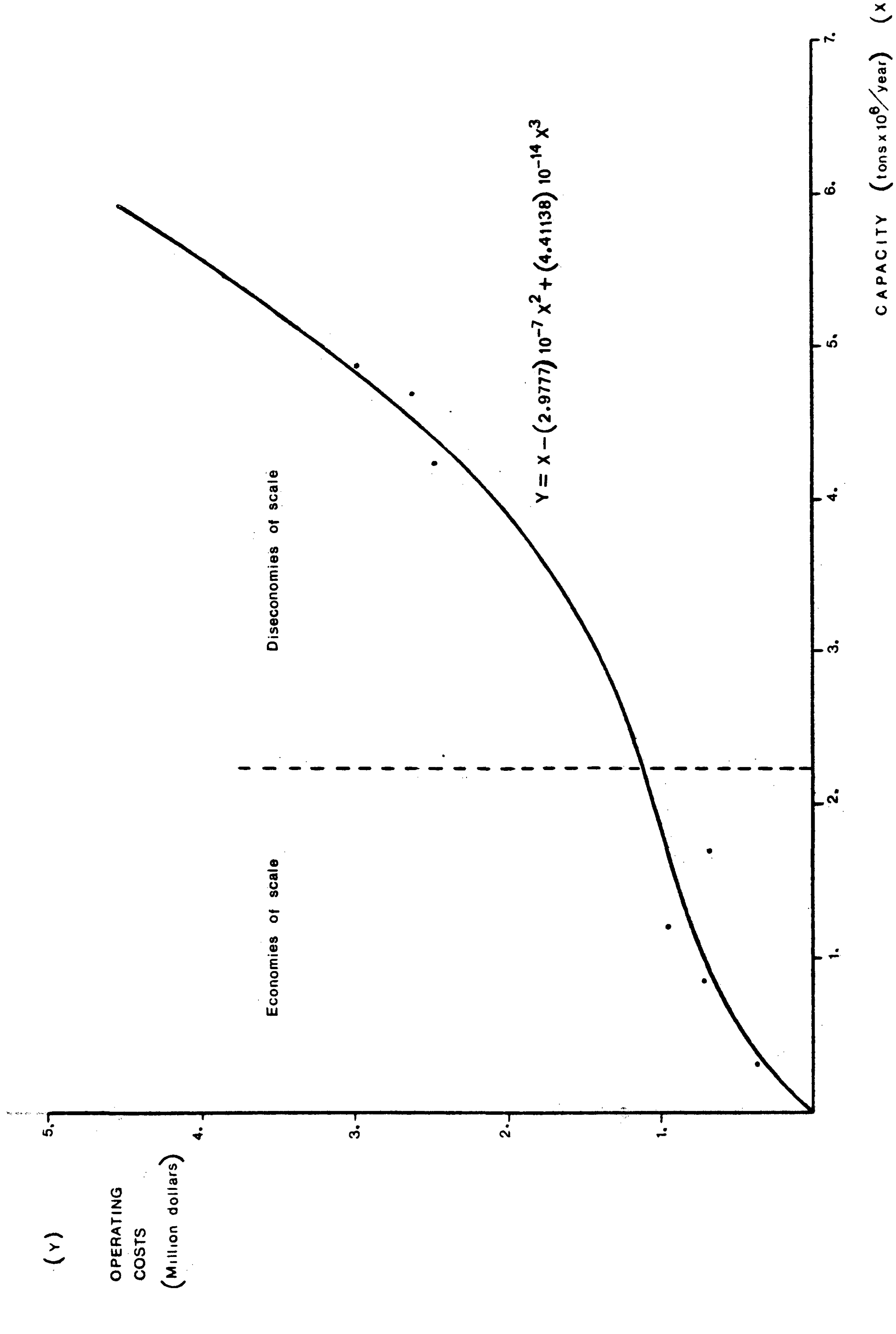

Figure 3.10 Milling Operating Costs versus Annual Capacity

Most of the information used to estimate both capital investment and operating costs for production of titanium sponge was taken from the "Report on Titanium", by Samuel C. William, in which two works published by the U.S. Bureau of Mines and two studies made by Herbert H. Kellogg are presented and used as a base for new estimations. All of these works constitute the principal information utilized concerning costs in the titanium industry.

There are three major cost categories considered for the investment in a sponge plant. These categories are engineering, construction, and equipment. Construction was further subdivided into materials and labor. In this case, the proportions used by the chemical engineering plant cost index were adopted; and the breakdown is summarized in Table 3.16.

TABLE 3.16

Investment Costs Breakdown - Sponge Plant

	Weight (%)
Engineering	10
Materials	7
Labor	22
Equipment	61

The estimates for three different sponge plant capacities taken from the Report on Titanium was used as a base to determine the general cost equation for investment in the sponge plant. These values were obtained for the year, 1965. The same sources and consideration used to estimate escalation and location factors for investment in mining were used for estimating capital investment in the sponge plant. A summary of calculations to determine the composite index for sponge plant investment is presented in Table 3.17.

TABLE 3.17

Composite Index Estimation - Sponge Plant Investment

	P_{A_i}	I_{1976B_i}	I_{1965B_i}	V_{B_i}/V_{A_i}	$I(Y/X)_{B_i}\ \frac{V_{B_i}}{V_{A_i}}$
Engineering	.10	149.1	111.83	0.725	0.10
Materials	.07	184.6	119.11	1.000	0.11
Labor	.22	173.0	120.86	0.300	0.09
Equipment	.61	200.9	116.29	1.120	1.18
Composite Index					1.48

This composite index updates the available data from 1965 to 1976 and also corrects for location in Venezuela. Results obtained for three different capacities are shown in Table 3.18.

TABLE 3.18

Sponge Plant Investment for Different Capacities

Capacity tons/tit. sponge/yr	Input tons/slag required/yr	Capital investment (US 1965) $\$ \times 10^6$	Correction factor	Capital investment (Vzla-1976) $\$ \times 10^6$
8,000	19,200	44	1.48	65
16,000	38,400	76	1.48	112
30,000	72,000	112	1.48	116

These values were used to determine the general equation for investment in the sponge plant.

The points were plotted and presented in Figure 3.11.

Even though there is no evidence of diseconomies of scale in this case, it is assumed that economies of scale are present only up to 40,000 tons of titanium sponge per year. This is an assumption which seems reasonable because the design capacities of plants have a maximum capacity of only 30,000 tons per year.

The final equation obtained was:

$$Y = (3767) - (.0243)X^2 + (5.63)(10^{-8})X^3,$$

where Y represents a capital investment in the sponge plant, and X represents input of titanium concentrates.

When considering the sponge plant's operating costs and the importance of certain raw materials, other than titanium concentrates used to produce titanium sponge, such as chlorine, magnesium, coal, argon, and leaching agents, these raw

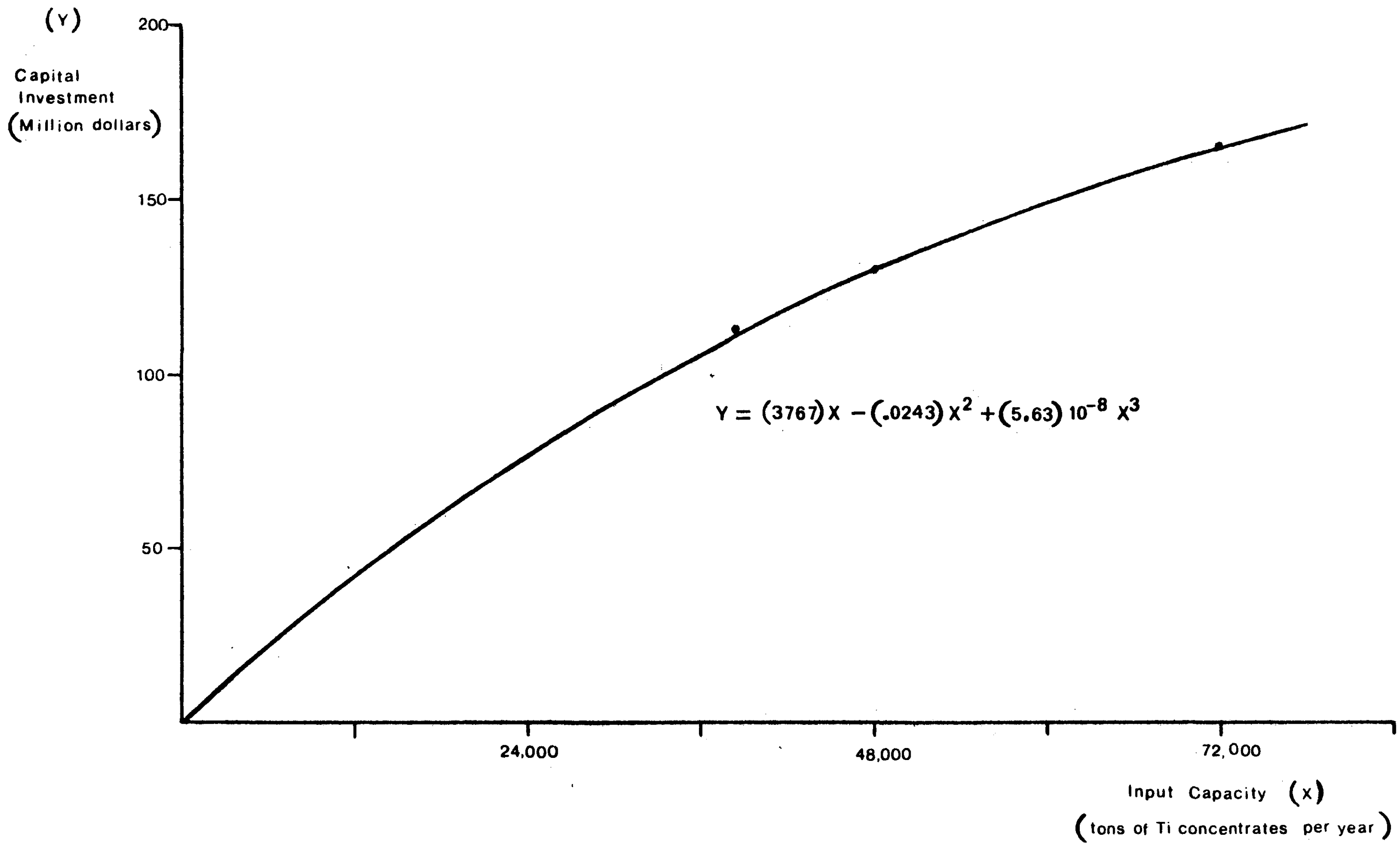

Figure 3.11 Sponge Plant Investment Costs versus Annual Capacity

materials should be considered separately. Labor, utilities (mostly electrical energy), supplies and others (administration, insurance, research and miscellaneous) were the additional cost elements considered for estimating the operating costs in the sponge plant.

The calculation of percentage weights for these cost elements were based on the assumptions used by Kellogg (1955) in two studies for estimating cost and capital investment requirements for sponge manufacture.

Two different plant capacities were analyzed. The first was designed to produce 7,800 ton/year, and the second to produce 15,600 ton/year of titanium sponge. The breakdown of costs obtained for each plant is presented in Table 3.19.

TABLE 3.19

Operating Costs Breakdown - Sponge Plant

	Weight (%)	
	7,800 tons/year	15,600 tons/year
Raw materials	22	27
Labor	45	37
Utilities	8	10
Supplies	9	9
Others	16	17

The same considerations and sources to estimate escalation and location factors for operating costs in mining were used for estimation of operating costs in the sponge plant. In addition, it was assumed that most of the raw materials other than titanium concentrates will be imported; therefore, the incrase in cost due to freight was used as the location factor, and the CE wholesale price for industrial chemicals was used as the escalation factor for this item.

A summary of calculations used to determine the composite index for each capacity with estimated updated costs for Venezuela is presented in table 3.20 and 3.21. These costs had to be updated from the year 1965, when the Kellogg report was analyzed and were updated by Samuel C. Williams.

TABLE 3.20

Composite Index Estimation - Sponge Plant Operating Costs (7,800 ton/year)

	P_{A_i}	I_{1967B_i}	U_{1965B_i}	V_{B_i}/V_{A_i}	$I(Y/X)_{B_i}\ \frac{V_{B_i}}{V_{A_i}}$
Raw material	.22	217.1	99.2	1.12	.54
Labor	.45	145.0	101.0	0.30	.19
Utilities	.08	154.0	101.0	0.66	.08
Supplies	.09	145.0	103.0	1.12	.14
Others	.16	150.0	104.0	0.73	.17
Composite Index					1.12

TABLE 3.21

Composite Index Estimation - Sponge Plant Operating Costs
(15,600 ton/year)

	P_{A_i}	I_{1976B_i}	I_{1965B_i}	V_{B_i}/V_{A_i}	$P_{A_i}\ I(Y/X)_{B_i}\ \frac{V_{B_i}}{V_{A_i}}$
Raw materials	.27	217.1	99.2	1.12	.66
Labor	.37	145.0	101.0	0.30	.16
Utilities	.10	154.0	101.0	0.66	.10
Supplies	.09	145.0	103.0	1.12	.15
Others	.17	150.0	104.0	0.73	.17
					1.24

These composite indices, updated for the year 1976, were used to estimate the operating costs in Venezuela. A summary of the calculations are presented in Table 3.22.

TABLE 3.22

Sponge Plant Operating Costs for Different Capacities

Capacity tons of titanium sponge per year	Input tons of titanium concentrate per year	Operating costs (US 1965) \$/lb	Op. Cost (US 1965) $\$x10^6$/ year	Correction factor	Op. Cost Vla-1975 $\$x10^6$/ year
7,800	18,720	.96	14.976	1.12	16.773
15,600	37,440	.79	24.554	1.24	30.447

These values were used to determine the general equation for operating costs in the sponge plant. The points were plotted and represented in Figure 3.12. Even though there is no evidence of diseconomies of scale in the range studied

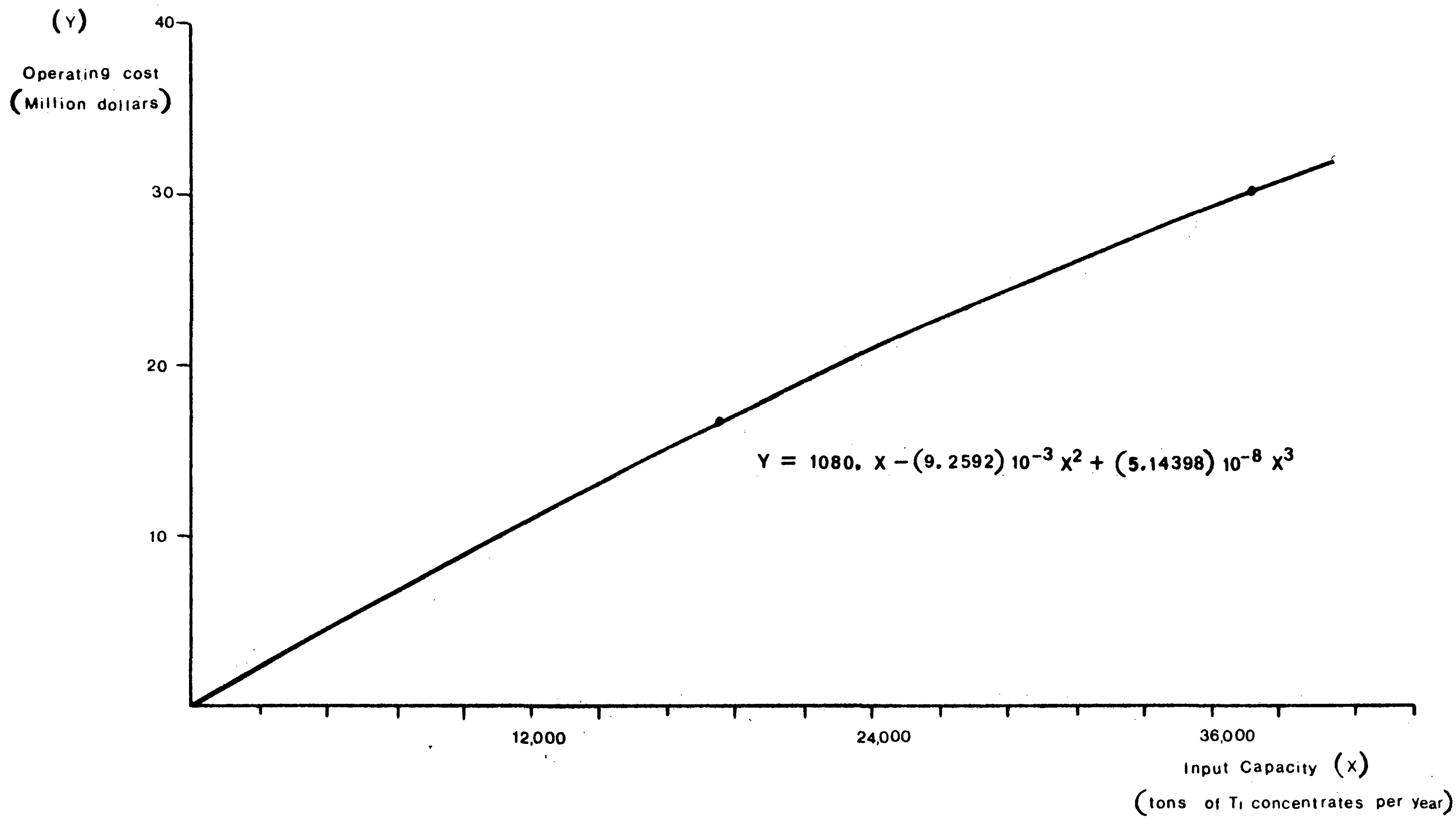

Figure 3.12 Sponge Plant Operating Costs versus Annual Capacity

it was assumed that economies of scale are present only up to an output of 25,000 tons of titanium sponge per year.

The final equation obtained was:

$$Y = (1080)X - (9.2592)(10^{-3})X^2 + (5.14398)(10^{-8})X^3,$$

where Y represents operating costs for the sponge plant, and
X represents input of titanium concentrates.

It was assumed that transportation will be conducted by contractors--including the loading and the unloading.

Two different stages of transportation were considered. First, transportation of the ore from the mine to the metallurgical plant by trucks; and secondly, transportation of the titanium sponge from the port to the markets by ship.

The mineral will be carried from the mine to the metallurgical plant approximately 60 miles. It was esimtated that costs (1976) of transportation would be $0.91 per ton when trucks are used and $2.06 per ton for loading and unloading. Thus, the equation used to represent the first stage of transportation costs is:

$$Y = (2.97)X,$$

where Y represents the total cost of transportation from mine to metallurgical plants including loading and unloading, and
X represents the tonnage of mineral.

The titanium sponge has to be carried from the metallurgical plant to the U.S. eastern coast. It was estimated that a cost (1976) of \$4.20 per ton would be required for transportation and of \$2.00 per ton for loading and unloading. Thus, the equation used to represent the second stage of transportation costs is:

$$Y = (6.20)X,$$

where Y represents the total cost of transportation from Venezuela to the U.S. eastern coast including loading and unloading, and

X represents the tonnage of titanium sponge.

Some details about the estimation of transportation costs are presented in Appendix I.

The probability distributions assumed to represent capital and operating costs are shown in table 3.23 and 3.24, respectively. The capacity reference used to determine these distributions were 500,000 tons per year for mining, transportation (1st stage) and milling, 37,440 tons per year for the sponge plant and 15,600 tons per year for transportation (2nd stage).

TABLE 3.23

Capital Investment Costs Probability Distribution

Investment (million dollars)	Associated Probability
100	.03
105	.10
110	.20
115	.35
120	.20
125	.10
130	.02

TABLE 3.24

Operating Costs Probability Distribution

Operating Costs (million dollars)	Associated Probability
22.75	.03
26.00	.06
29.25	.20
32.50	.30
35.75	.25
39.00	.12
42.25	.04

The annual increase in capital costs was assumed to be 9 percent, a little more than the increase in CE plant cost index. The CE plant cost index increased from 132.2 in 1971 to 182.4 in 1975.

The operating costs were assumed to increase linearly and similarly to the increase in the principal metal mining expenses index. The index for total mining expenses increased from 104 in 1969 to 128 in 1973 (last published value). Referring to the trend equation discussed in Chapter 2, if the discount parameter "D" is equal to 1 because of the linearity assumed, the annual trend in variable "G" will be .0577.

3.5.2.3 Other Technological Parameters

The pre-operational period was assumed to range from 2 to 4 years and the maximum operational period considered was for 20 years.

The percentage of depreciable investment over total capital costs was assumed to be 80 percent and the percentage of working capital over operating costs 25 percent.

It was also assumed that both capital and operating costs are affected only by the amount of inputs, and that the average grade of the ore will not affect these costs.

3.5.3 Market Parameters

After the analysis of historical data, the following distribution was assumed to represent prices of titanium sponge for the year, 1976.

TABLE 3.25

Price Probability Distribution

Price $/ton Ti Sponge	Associated Probability
4409	.08
4960	.20
5512	.40
6063	.20
6614	.10
7165	.02

Due to the tendency towards stabilization in titanium sponge prices during the last ten years, it was assumed that fluctuations around a mean rather than a real increase will occur; therefore, no annual trend was considered.

3.5.4 Other Parameters Required

According to Venezuelan mining law, taxes of 50 percent on profits must be payed to the government, and royalties are established depending upon the particular case. Royalties are assumed to be $1 per ton of ore extracted.

A confidence interval of 90 percent was used to estimate the lower limits. A range in mine capacity of between 100,000 and 2,100,000 tons per year, and cut-off grades of between 0 percent and 12 percent were considered when making the calculations.

CHAPTER 4. RESULTS AND CONCLUSIONS

Using the parameters developed in the text, the net present value and the discounted cash flow rate of return of the titanium project were calculated.

4.1 A Simulation Example

Before the discussion of results, a simulation example is illustrated in order to clarify the manner in which all calculations are performed. The values obtained, step by step, were as follows:

1) Selection of cut-off grade, mine capacity, and determination of pre-operational period;

a. Data required:

1. Range of cut-off grades: from 0 to 12 percent
2. Range of capacities: from 100,000 to 2,100,000 tons/year
3. Minimum pre-operational period: 2 years
4. Maximum pre-operational period: 4 years

b. Values selected:

1. Cut-off grade = 3 percent (content of titanium metal)
2. Mine capacity = 900,000 tons/year
3. Pre-operational period = 2 years

2) Simulation of average grade of total deposit;

a. Data required:

Average grade probability distribution

b. Value obtained (by simulation):

Average grade of total reserves = 3.41 percent

3) Determination of tonnage (T) and average grade of reserves (g);

a. Data required:

1. Tonnage of total deposit (A) = 30 million tons
2. Average grade of total reserves (G) = 3.41 percent
3. Cut-off grade (C) = 3 percent

b. Values obtained:

1. $T\text{-}A/e^{(c/g)}=30x10^6/2.718^{(.03/.0341)}=12,447.543$ tons
2. g=G+C=3.41+3=6.41 percent

4) Determination of the economic life of the project (L);

a. Data required:

1. Tonnage of reserves (T) = 12,447,543 tons
2. Mine capacity (C) = 900,000 tons per year
3. Dilution factor (d) = .89

b. Value obtained:

L=T/Cxd = 12,447,543/900,000x.90 = 16 years

5) Simulation of capital investment, operating costs, and prices using base distributions; these values will be used to correct those determined by use of equation.

a. Data required:

1. Capital investment probability distribution

2. Operating cost probability distribution

3. Price probability distribution

b. Values obtained (by simulation):

1. Capital investment = $100 million

2. Operating cost, first year of production = $29.25 million

3. Price, first year of production = $5512 per ton

6) Calculation of inputs and outputs of each subprocess;

a. Data required:

1. Mining capacity (X_1) = 900,000 tons per year

2. Dilution factor (d) = .89

3. Recovery milling plant (r_3) = 85%

4. Recovery sponge plant (r_4) = 80%

5. Recoveries, other processes (r_1, r_2, r_5) = 100%

6. Product metal content milling plant (P_4) = 36%

7. Product metal content sponge plant (P_5)=99.3%

8. Average grade of the ore (g) = 6.41%

It was assumed that recovery is not affected by the grade of the ore, therefore

$AG = AGB = f_i = 0.$

b. Values obtained:

1. $P_2 = g = .0641$

2. $P_3 = P_2 = .0641$

3. $P_6 = P_5 = .993$

4. X_1 = Input mining

5. X_1 = 900,000 tons per year

6. X_2 = Input transportation (first stage)

7. $X_2 = Cxd = 900{,}000 x .89 = 801{,}000$ tons per year

8. X_3 = Input milling plant

9. $X_3 = \frac{P_2}{P_3}(r_2)X_2 = \frac{.0641}{.0641}(1.) x 801{,}000 = 801{,}000$ tons/year

10. X_4 = Input sponge plant

11. $X_4 = \frac{P_3}{P_4}(r_3)X_3 = \frac{.0641}{.36}(.85) x 801{,}000 = 121{,}229$ tons/year

12. X_5 = Input transportation (second stage)

13. $X_5 = \frac{P_4}{P_5}(r_4)X_4 = \frac{.36}{.993}(.80) x 121{,}229 = 35{,}160$ tons/year

7) Calculation of operating costs and working capital; operating costs are calcuated for each year of the operational period. The process will be illustrated by calculating the first year of production. This value is used for determination of the working capital.

a. Data required:

1. Input mining (X_1)	=900,000 tons/year
2. Input transportation (1st stage) (X_2)	=801,000 tons/year
3. Input milling (X_3)	=801,000 tons/year
4. Input sponge plant (X_4)	=121,229 tons/year
5. Input transportation (2nd stage) (X_5)	=35,160 tons/year

6. Op. cost equation (mining) $I_1=(8.9)10^5X+.261X$

Op. cost equation (transportation 1st stage) $I_2=2.97X$

7. Op. cost equation (milling) $I_3=X-(2.978)10^{-7}X^2+(4.411)10^{-14}X^3$

8. Op. cost equation (transportation 2nd stage) $I_5=6.2X$

9. Annual trend in op. costs (ATOC) =.0577

10. Discount parameter (DPOC) =1

11. Reference cost (RC) $=\$33.19\text{x}10^6$

12. Simulated value (1st year of operation) (SOC) $=\$29.25\text{x}10^6$

b. Calculations:

1. Estimation of the total operating costs first year of production.

2. Total operating costs before correction (OCBC) $=I_1+I_2+I_3+I_4+I_5$

3. Operating costs (mining) $=8.9\text{x}10^5+.261(900,000)$

4. Operating costs (transportation- 1st stage) $=2.97(801,000)$

5. Operating costs (milling) $=(801,000)-2.978\text{x}10^{-7}(801,000)^2+4.4411\text{x}10^{-14}(801,000)^3$

6. Operating costs (sponge plant) $=1.08\text{x}10^3(121,229)-9.25\text{x}10^{-3}(121,229)^2+5.144\text{x}10^{-8}(121,229)^3$

7. Operating costs (transportation- 2nd stage) $=6.2(35,160)$

8. Total operating costs before correction (OCBC) $=(\$101.165)10^6$

9. Trend and simulation corrections:

$$OC=SOC(1+ATOC(DPOC)^{(i-1)}(i-1) \quad \frac{OCBC}{RC}$$

i=3 (first operational period)

$$OC = (29.25)10^6[1+.0577(1)^2(2)] \quad \frac{(101.165)10^6}{(33.19)10^6}$$

10. Operating costs (1st year of production) $=\$99.23x10^6$

11. Working capital $=(99.23)10^6(.25)-$

$(\$24.81)10^6$

8) Calculation of investment;

a. Data required:

1. Input mining =900,000 tons/year
2. Input milling =801,000 tons/year
3. Input sponge plant =121,229 tons/year
4. Cost investment equation (mining) $=I_1=2.68X$
5. Cost investment equation (milling) $=I_3=7.02X$
6. Cost investment equation (sponge plant $I_4=(3767)X-(.0243)X^2+$ $(5.6310^{-8}X^3$
7. Reference cost $=(\$114.779)10^6$
8. Simulated value $=(\$100)10^6$
9. Pre-operational period =2 years
10. Inflation factor =9%
11. Working capital $=(\$24.81)10^6$

b. Values obtained:

1. Estimate of total investment

Total investment (I) $=I_1+I_3+I_4$

$I=2.68(900{,}000)+7.02(801{,}000)+3767(121{,}229)-$

$.0243(121{,}229)^2+(5.63)10^{-8}(121{,}229)^3$

$I=(\$207.887)10^6$

2. Correction due to simulation

$$I'=(100)10^6\frac{(207.887)10^6}{(114.779)10^6} = (\$181.112)10^6$$

3. Determination of investment for each pre-operational year

a. Investment 1st year$=(181.112)10^6/2=(\$90.56)10^6$

b. Investment 2nd year$=\frac{181.112\text{x}10^6}{2}(1.09)^1 +$

$(24.81)10^6=(\$123.52)10^6$

9) Calculation of final production; price is given per unit of final output, therefore,

final production=last output=35,160 tons/year.

10) Calculation of prices;

Prices are calculated for each year of the life of the project. The process is illustrated only for the first year of production.

a. Data required:

1. Simulated value before correction (SP) =\$5512/ton

2. Annual trend in prices (ATPR) =0

3. Discount parameters (DPPR) =1

b. Calculation:

$$\text{Price} = \text{SP } (1+\text{ATPR})(\text{DPPR})^{i-1}(i-1)$$

$i = 3$ (first operational period)

Therefore:

$$\text{Price} = 5512[1+0(1)^2(2)] = \$5512/\text{ton}$$

11) Calculation of revenues;

a. Data required:

1. Final production = 35,160 ton/year
2. Price = \$5512/ton

b. Calculations:

Revenue = (final production) x (price)

Therefore:

Revenue for the first year of production =

$(35{,}160)(5512) = (\$193.80)10^6$

12) Calculation of cash flows;

The calculation of cash flows for each year of the life of the project is illustrated in Table 4.1.

13) Calculation of the net present value and discounted cash flow rate of return;

a. Data required:

1. Cash flows
2. Attractive rate of interest = 10%

b. Results obtained:

1. NPV = \$2.3 million
2. DCFROR = 11.6%.

TABLE 4.1

Determination of Cash Flows in a Simulation Example

Year	Capital Invest. (Mill. $)	Working Capital	Prices	Gross Income (Mill.$)	Operating Costs	Royalty	Depreciation	Taxes	Salvage Value	Cash Flow
1	(90.6)	-	-							(90.6)
2	(123.5)	(24.8)	-							(148.3)
3			5512	193.8	99.2	.8	8.3	42.6		51.2
4			4960	174.4	125.2	.8	8.3	19.9		28.5
5			4409	155.0	98.5	.8	8.3	23.6		32.1
6			6063	213.2	91.7	.8	8.3	56.0		64.6
7			5512	193.8	155.7	.8	8.3	14.4		22.9
8			4960	174.4	124.9	.8	8.3	20.1		28.6
9			4409	155.0	130.0	.8	8.3	7.8		16.4
10			5512	193.8	162.2	.8	8.3	11.1		19.7
11			5512	193.8	154.3	.8	8.3	15.1		23.6
12			5512	183.9	160.0	.8	8.3	12.2		20.8
13			5512	183.8	150.6	.8	8.3	16.9		25.5
14			4960	174.4	140.1	.8	8.3	12.5		21.0
15			5512	193.8	128.7	.8	8.3	27.9		36.4
16			6614	232.5	116.2	.8	8.3	53.5		62.1
17			5512	193.8	119.8	.8	8.3	32.3		40.9
18		24.8	4960	94.2	66.6	.8	8.3	9.1	34.3	76.7

4.2 Case Study Results and Determination of the Optimum

The results obtained for NPV and DCFROR are presented in contour format with the investment decision criteria portrayed as a function of cut-off grade and mine capacity.

The optimum values of mine capacity and cut-off grade or point of profit maximization depends on strategic aspects and policies previously adopted by the mining enterprise. The determination of this optimum system and the effect of policies and uncertainties are discussed in the following sections.

4.2.1 Attractive Rate of Interest Effect

Different attractive rates of interest lead to different points of profit maximization.

Figure 4.1 shows the simulated expected NPV assuming an attractive rate of interest of 10 percent. In this case the maximization of the NPV occurs above the $\$33 \times 10^6$ contour. The point of maximum expected NPV is associated with the approximated mine development characteristics presented in Table 4.2.

TABLE 4.2

Mine Development Characteristics Associated with the Point of Maximum Expected NPV When an Attractive Rate of Interest of 10 Percent is Assumed

Mine capacity	600,000 ton/year
Cut-off grade	5.5 percent
Expected NPV	$\$33^{+}$ million
Expected DCFROR	11 percent

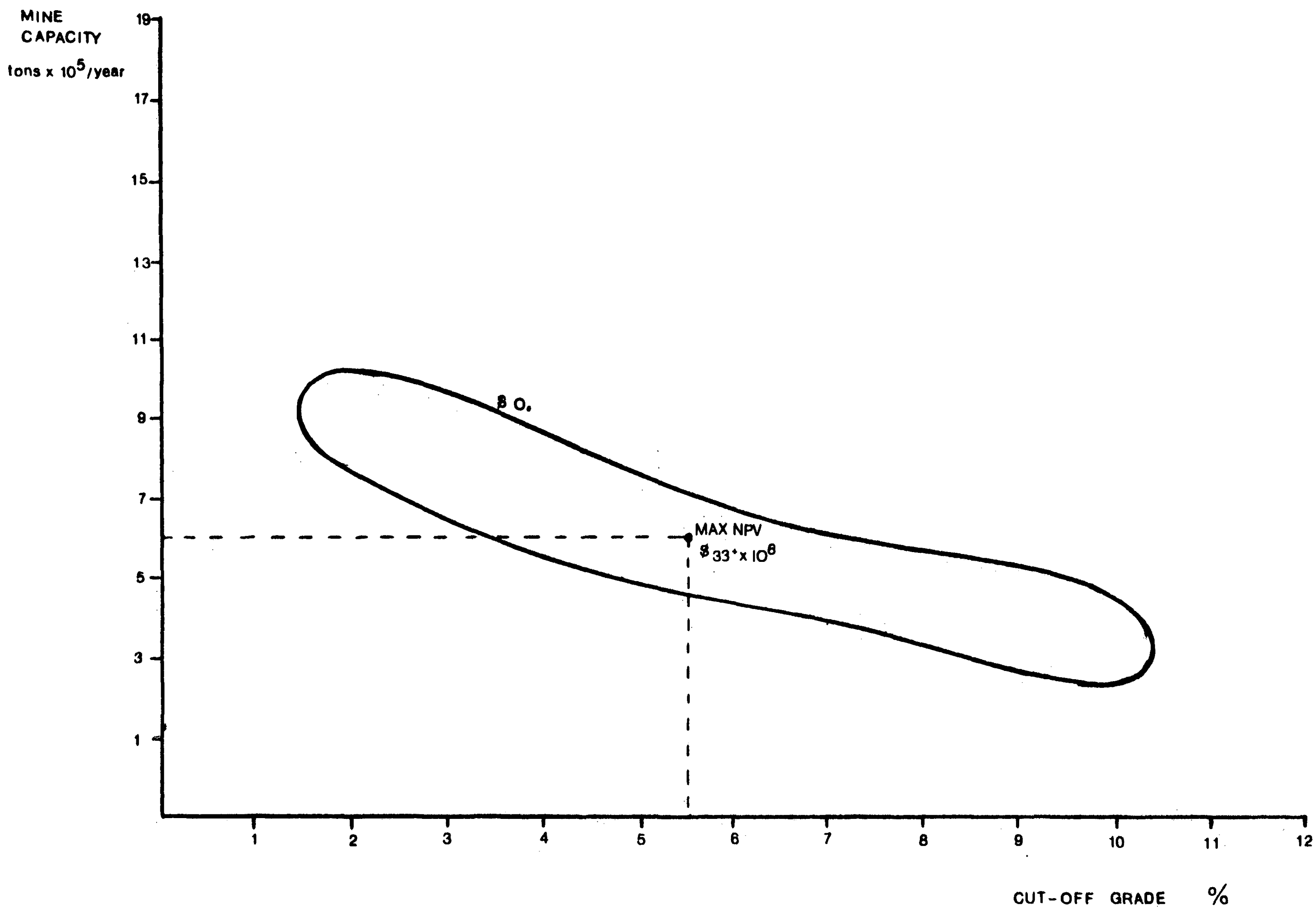

Figure 4.1 Expected Net Present Value When Assuming an Attractive Rate of Interest of 10 Percent

This point represents the maximization of expected NPV when an attractive rate of interest of 10 percent is assumed. The optimization of mine development would occur at this point when uncertainties are not taken into consideration or in the case of large mining companies which could afford large number of investment to insure averaging out to the expected value.

Figure 4.2 compares the optimum obtained when different attractive rates of interest are considered. The point of profits maximization determined when assuming an attractive rate of interest of 10 percent is moved from 600,000 tpy of mine capacity and a cut-off grade of 5.5 percent, to 700,000 tpy of mine capacity and a cut-off grade of 4 percent when an attractive rate of interest of 2 percent is considered.

When, for any reason the attractive rate of interest is not considered, the use of some other indicators such as the DCFROR is required.

Figure 4.3 shows the simulated expected DCFROR. The maximization of this value occurs above the 11 percent contour. This point of maximum expected DCFROR is associated with the following approximated mine development characteristics.

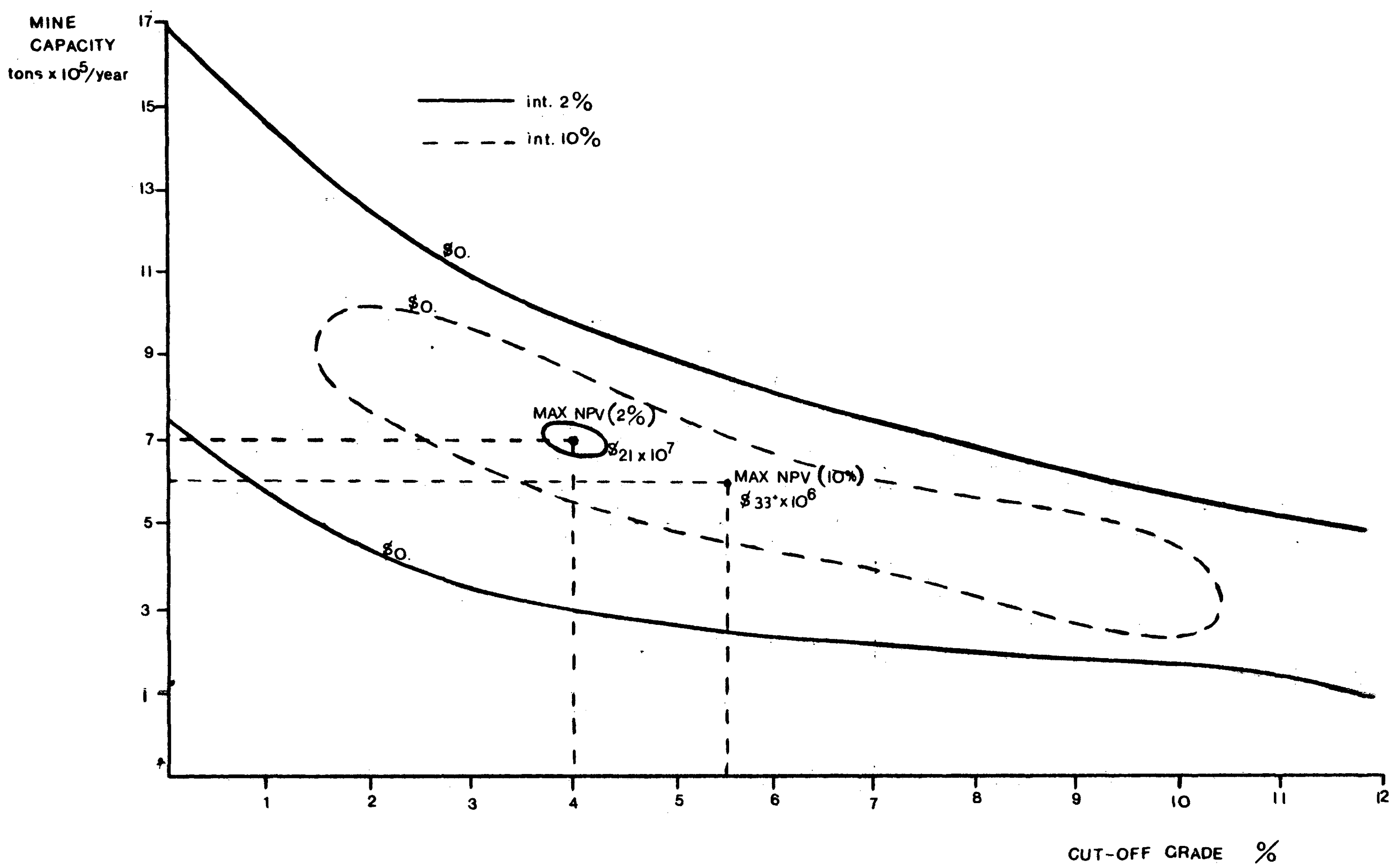

Figure 4.2 Comparison Between the Optimum Obtained for 2 percent and 10 percent Attractive Rate of Interest.

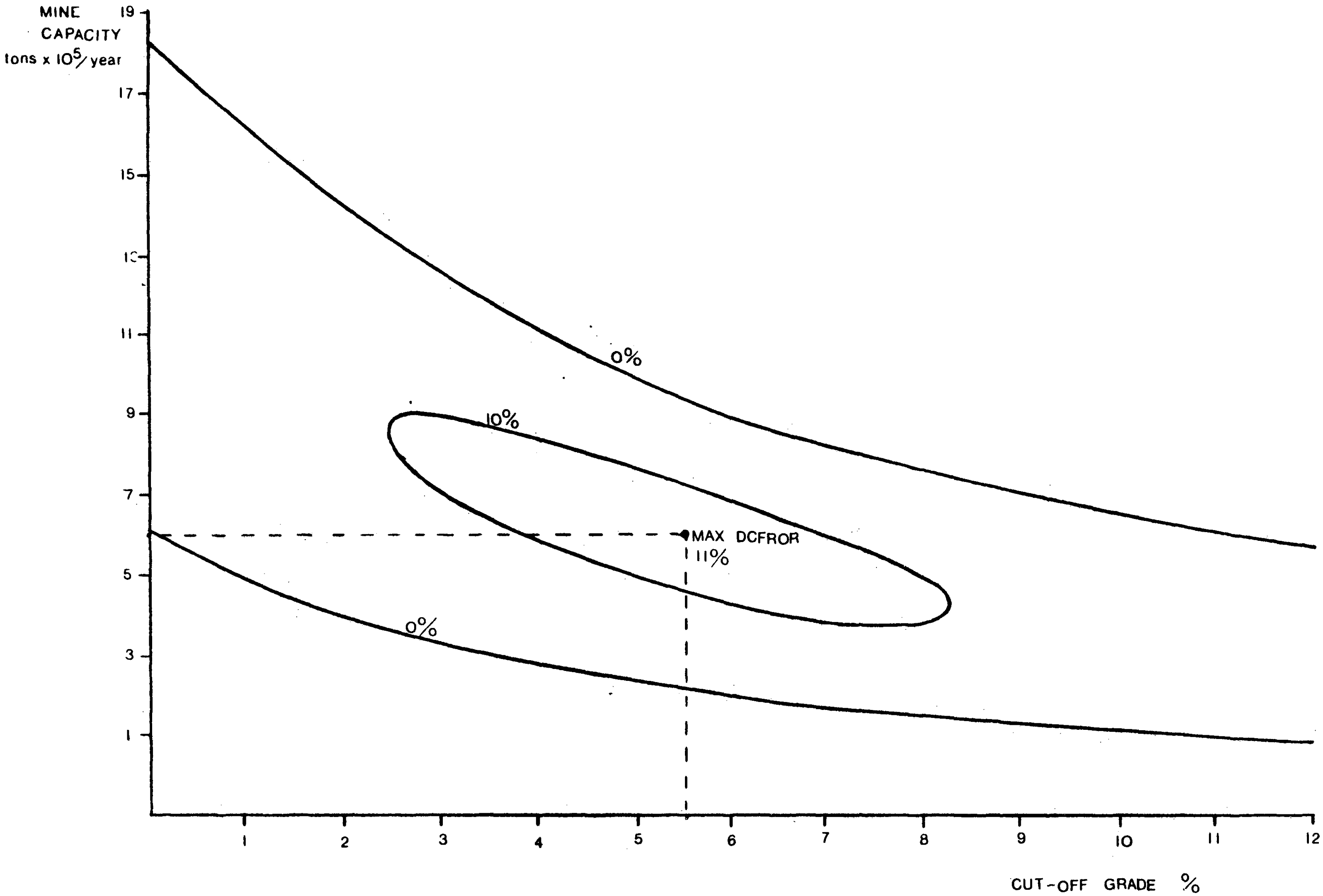

Figure 4.3 Expected Discounted Cash Flow Rate of Return

TABLE 4.3

Mine Development Characteristics Associated with the Point of Maximum Discounted Cash Flow Rate of Return

Mine Capacity	600,000 ton/year
Cut-off Grade	5.5 percent
Expected DCFROR	11 percent
90 percent lower limit DCFROR	9 percent

This point will be selected by large enterprises in which maximization of the DCFROR is selected as a goal.

4.2.2 Uncertainties Effect

The point of profit maximization can be highly affected when uncertainties are considered. The effect of economical and geological uncertainties obtained for the San Quintin project is discussed in this section.

The net present value in the 90 percent lower limit, at a rate of interest of 10 percent, is negative. Therefore, the project would be rejected by an enterprise whose survival depends on the success of this project.

Figure 4.4 defines an optimum net present value in the "90 percent lower limit" when an attractive rate of interest of 2 percent is assumed. This means that given all of the assumptions involved we would expect only 10 percent of the results to fall below this optimum present value. Hence,

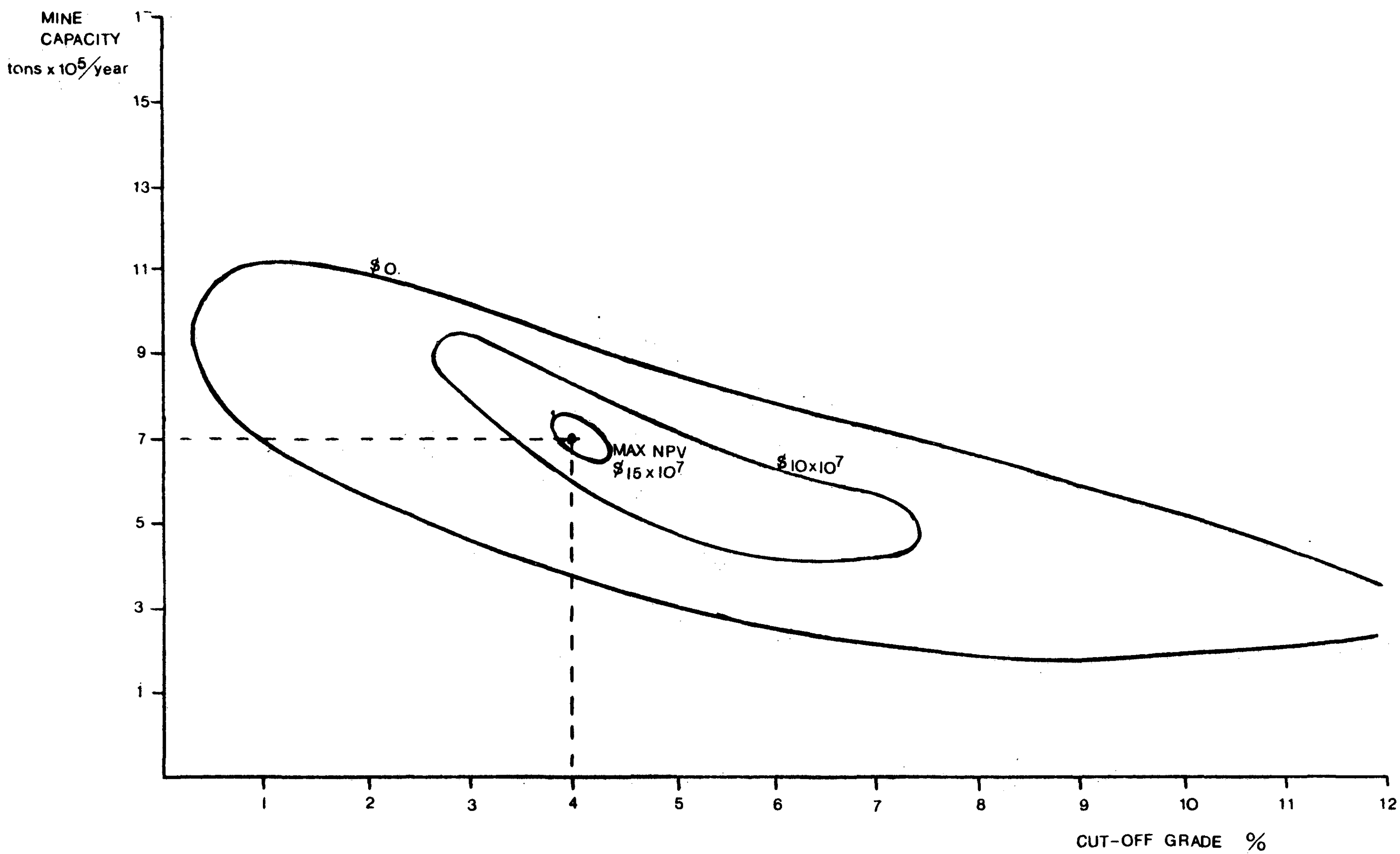

Table 4.4 Ninety Percent Lower Limit Net Present Value When Assuming an Attractive Rate of Interest of Two Percent.

this present value is the one that minimizes uncertainty about the actual value, when assuming an attractive rate of interest of 2 percent.

This point is associated with the mine development characteristics presented in Table 4.4.

TABLE 4.4

Mine Development Characteristics Associated with the Point of Maximum NPV in the "90 Percent Lower Limit"

Mine capacity	700,000 tons/year
Cut-off grade	4 percent
Expected NPV	$214 million
90 percent lower limit NPV	$155 million
Expected DCFROR	11 percent
90 percent lower limit DCFROR	9 percent

This is another point of investment preference, the minimization of uncertainty when an attractive rate of interest of 2 percent is assumed. This point will be selected by the small enterprise whose survival depends on the success of this single investment.

Figure 4.5 shows the simulated uncertainty results as reflected in 90 percent lower limit DCFROR contours. The maximization of lower limit (and, consequently, the minimization of uncertainty) occurs above the 9 percent contour. This point of minimization of uncertainty is associated with the following approximated mine development characteristics.

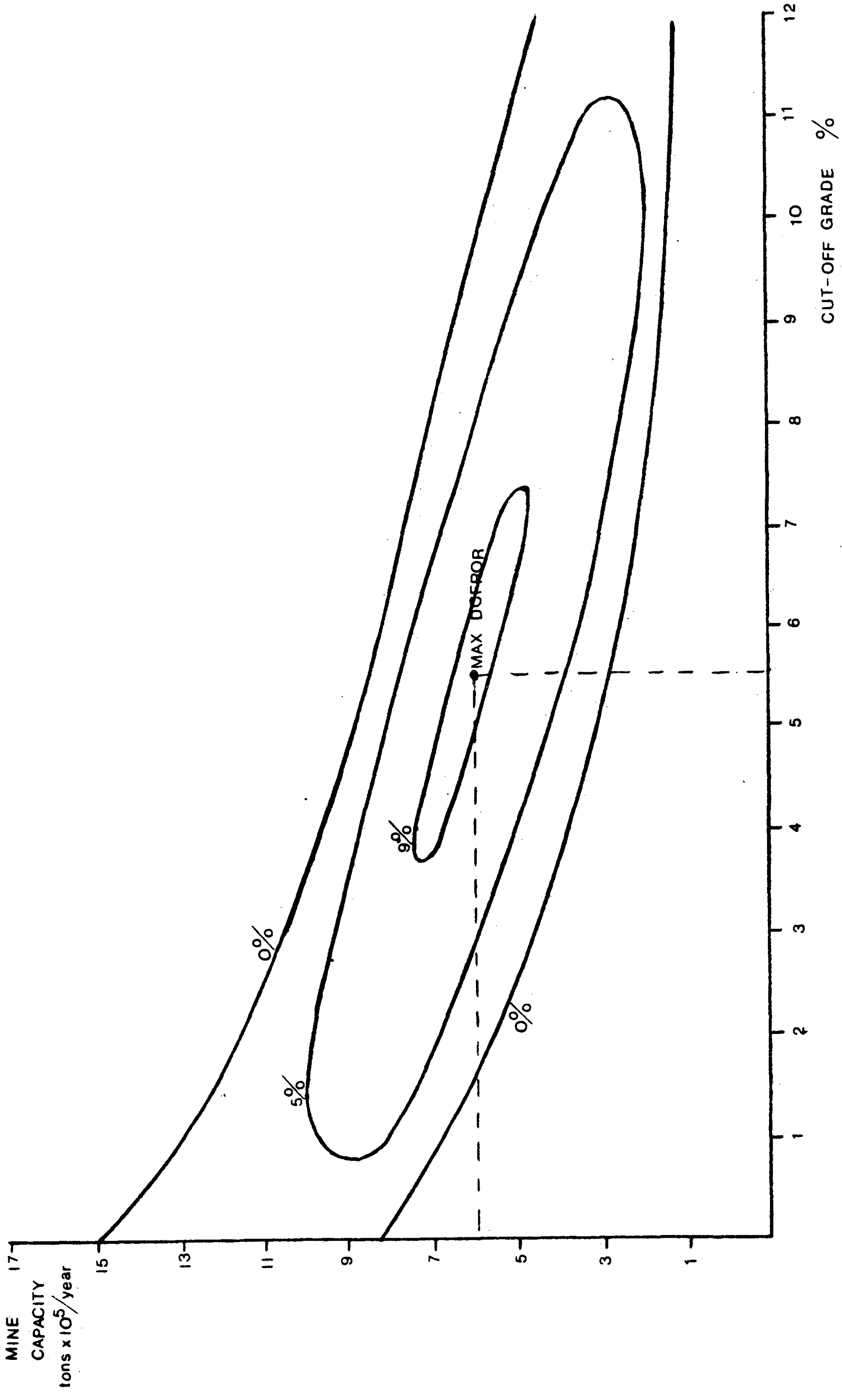

Figure 4.5 Ninety percent Lower Limit DCFROR

TABLE 4.5

Mine Development Characteristics Associated with the Point of Maximum DCFROR in the "90 percent Lower Limit"

Mine Capacity	600,000 tons/year
Cut-off grade	5.5 percent
Lower limit DCFROR	9 percent
Expected DCFROR	11 percent

This point will be selected by any enterprise which selects as goals the maximization of the DCFROR and the minimization of all uncertainties.

Following this section is a presentation of the effect of price, capital investment, operating costs, and average grade uncertainties, when assuming a 90 percent confidence interval and when the maximization of the DCFROR is adopted as a goal. The results obtained are shown in figures 4.6, 4.7, 4.8, 4.9, and Table 4.6.

TABLE 4.6

Mine Development Characteristics Associated with Points of Minimization of Uncertainties in the "90 percent Lower Limit"

	Price	Capital Investment	Operating Costs	Average Grade
Mine capacity (tpy)	400,000	600,000	400,000	700,000
Cut-off grade (%)	8.5	5.5	8.5	5.0
Lower limit DCFROR (%)	3.0	10.0	5.0	11.0

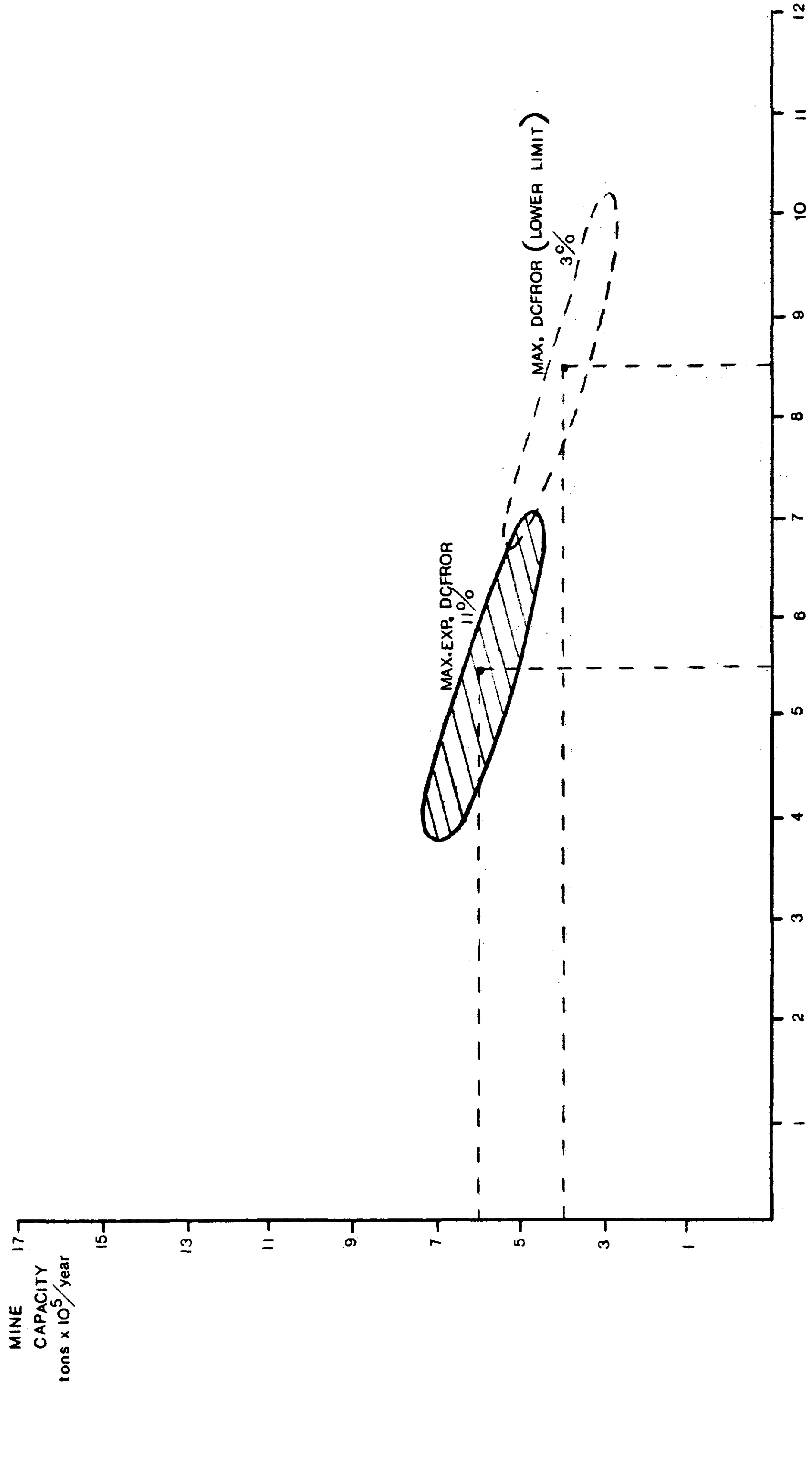

Figure 4.6 Effect of Price Uncertainties on Mine Development Results

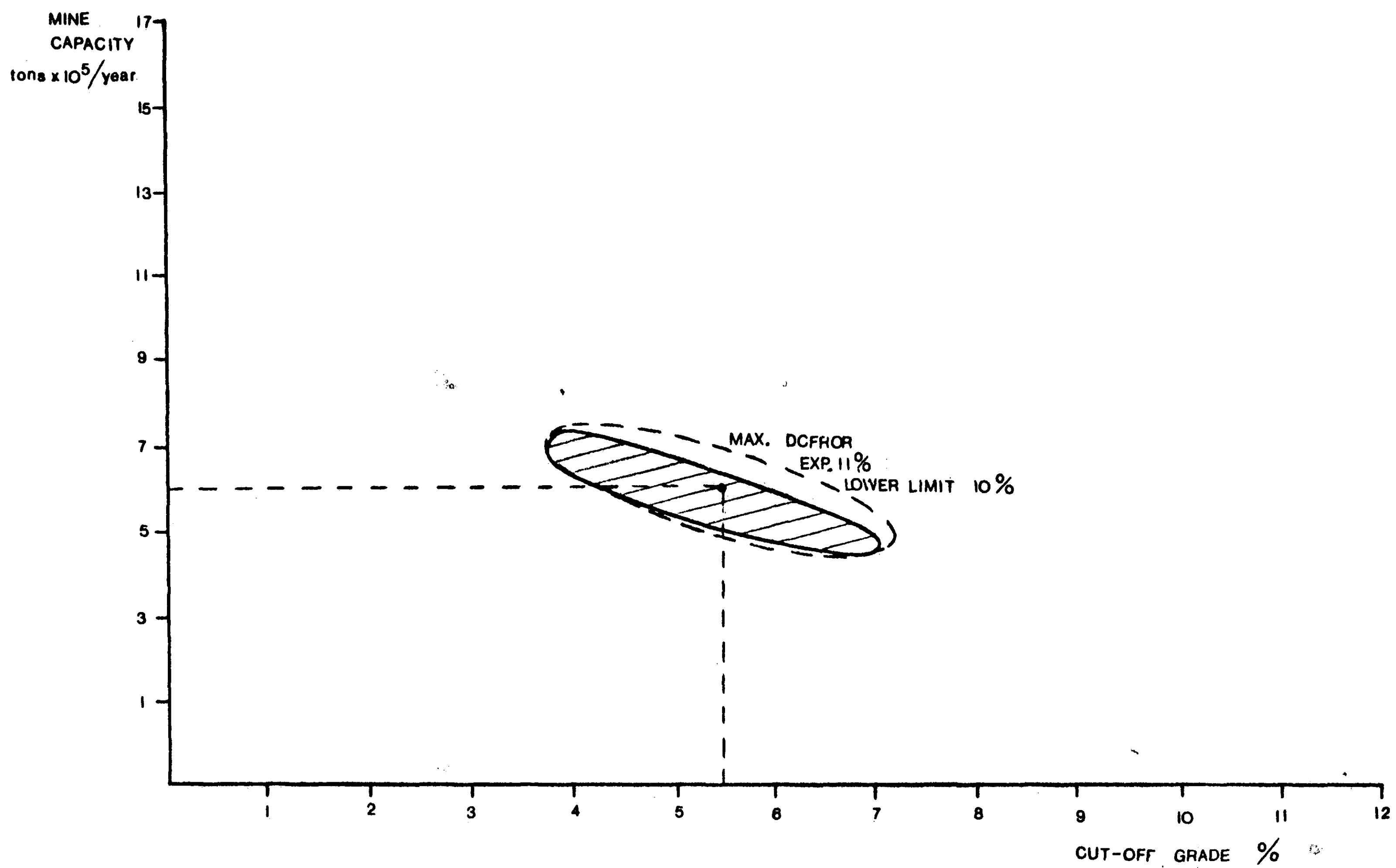

Figure 4.7 Effect of Capital Investment Uncertainties on Mine Development Results

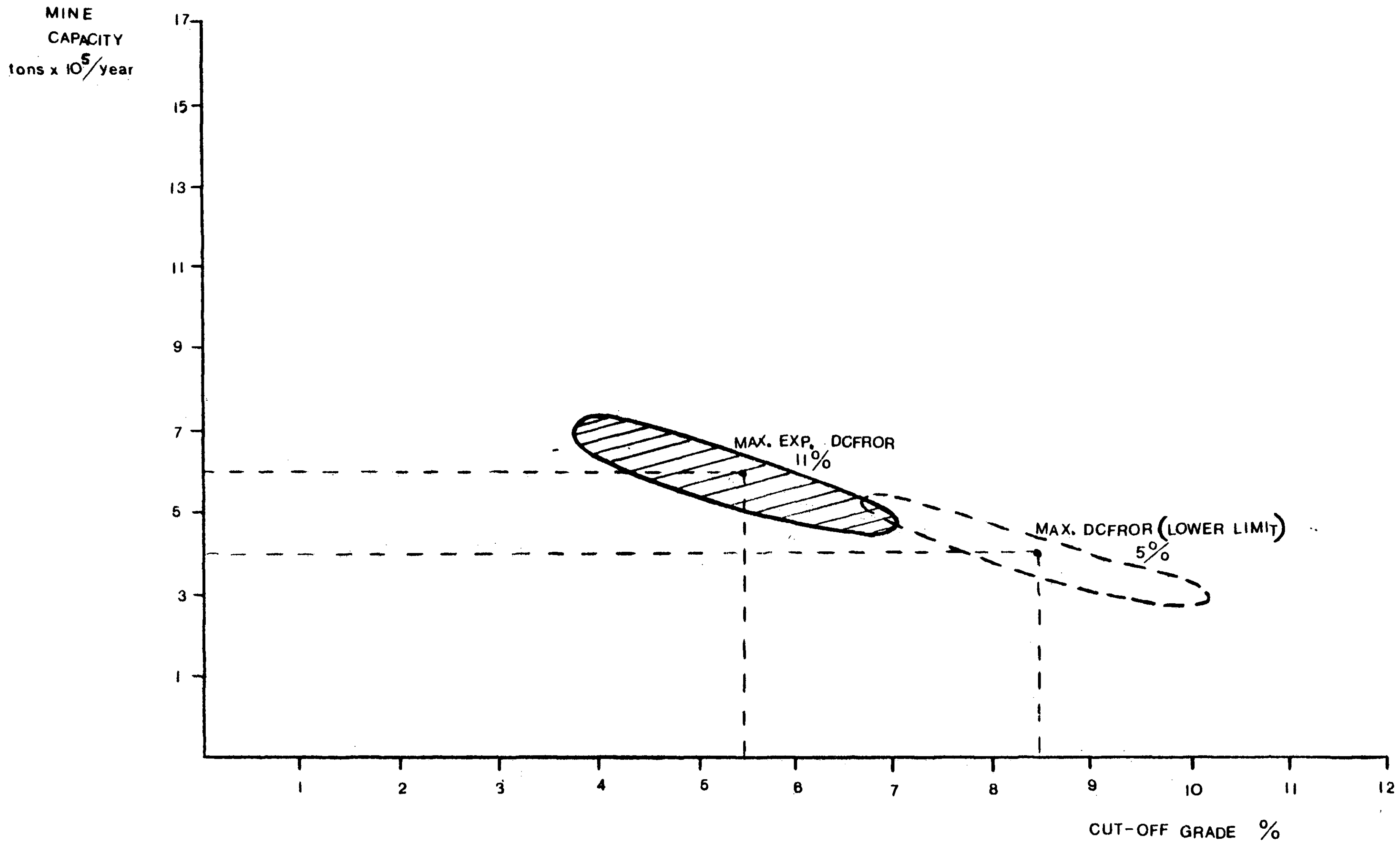

Figure 4.8 Effect of Operating Costs Uncertainties on Mine Development Results

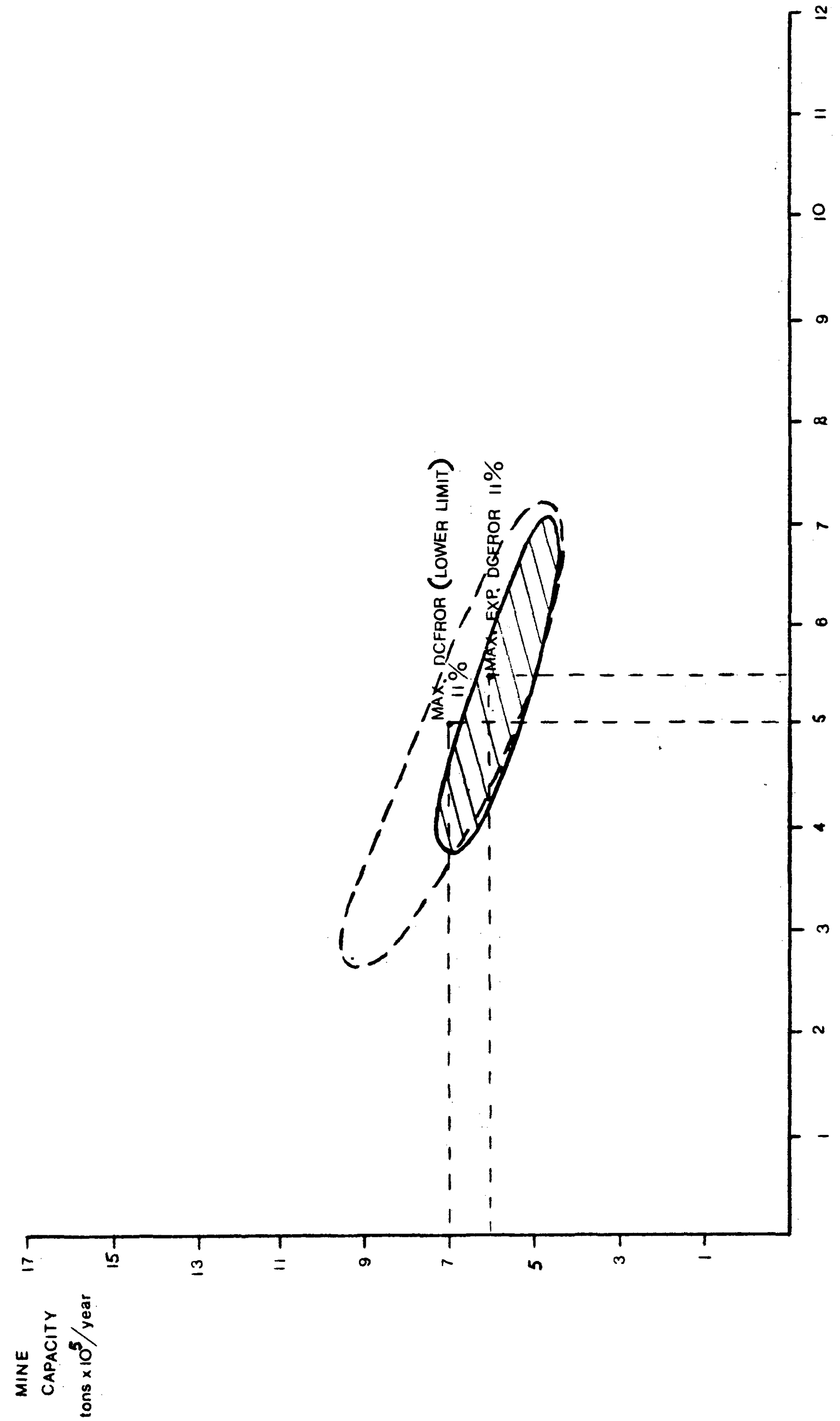

Figure 4.9 Effect of Average Grade Uncertainties on Mine Development Results

These results prove that the point of profit maximization and the profitability of this project are very sensitive to the price and operating costs uncertainties, while the capital investment and the average grade uncertainties are not highly modifying.

4.3 Conclusions

The analysis proves that the values of mine capacity and cut-off grade which maximize profits are extremely sensitive to the rate of interest required and the uncertainties involved.

Different policies adopted by mining enterprises lead to different points of profit maximization and even to contradicting results. When the uncertainties of a specific project are very important as they are in the case of small enterprise, the optimum is likely to be different from the values obtained when the uncertainties of the specific project are not crucial to the survival of the firm. Uncertainty may force rejection by a firm with limited financial resources even though the project indicates probable high expected returns.

In the specific case of the San Quintin deposit, the project to produce titanium sponge is feasible when uncertainties are not considered. When a 90 percent confidence interval and a minimum acceptable rate of return of 10 percent are assumed, the project is not considered economic. Therefore, the project would have to be rejected by a small enterprise relying on positive results over the 90 percent confidence interval.

In the case of the Venezuelan government, which will probably exploit this deposit, the project to produce titanium sponge can be considered feasible under the assumptions used. Government is able to assume a larger risk than is a small enterprise.

If we assume the goal to be the achievement of maximization of the discounted cash flow rate of return (the one which is normally adopted by the Venezuelan government), the mine development characteristics associated with the point of profit maximization can be summarized as follows.

TABLE 4.7

Approximated Mine Development Characteristics that can be Adopted by the Venezuelan Government

Ore Reserves	6,500,000 tons
Cut-off grade	5.5 percent
Mine Capacity:	
total material	600,000 tons/year
ore	534,000 tons/year
Milling Capacity	
input of ore	534,000 tons/year
output of concentrates	114,610 tons/year
Sponge Plant Capacity	
input of concentrates	114,610 tons/year
output of Ti sponge	33,240 tons/year
Life time	14 years
Pre-operational period	2 years
Approximated investment	$200 million
Expected DCFROR	11 percent

These values can be considered a guide for further detailed studies.

Special attention should also be given to prices and operating costs. This project, as was proved earlier, is very sensitive to price and operating cost uncertainties.

APPENDIX I

Some Additional Aspects Related to Cost Estimation

INDICES

EN-R index for construction

This index was used to update the capital investment in mining.

This index is published in the magazine, Engineering News-Record.

Marshall and Stevens Equipment cost index

This index is published in the magazine, Chemical Engineering, for several industries. The index for mining and milling was used to update capital investment for mining and concentration plants.

Price Index of principal metal mining expenses

This index was used to update different cost elements for operating costs in mining. It is published by the United States Bureau of Mines (USBM) in the Minerals Yearbook.

Some of the last values published are presented in Table I.1.

TABLE I.1

Price Index of Principal Metal Mining Expenses
1967=100

Year	Total	Labor	Supplies	Fuel	Elec. Energy
1969	104	104	106	101	102
1970	109	108	111	106	105
1971	114	113	116	114	114
1972	120	120	120	119	122
1973	128	126	128	146	129

CE Plant Cost Index

The CE plant cost index was used to estimate the annual increase in total capital investment and the CE plant cost index for engineering and supervision was used to update the engineering supervision and overhead costs. This index is published in the magazine, Chemical Engineering.

Location Factors

Supervision and Engineering

The following relationships was assumed between supervision and engineering costs in Venezuela and the U.S.

Cost Vla = 0.725 x Cost U.S. (Tovar, 1972).

Labor

The determination of the location factor for labor was based on the assumption that the cost of labor in the U.S. is 8.5 times the cost of labor in Venezuela (Tovar, 1972). This value is affected by differences in productivity and fringe benefits in Venezuela. The increase in costs due to differences in productivity and fringe benefits in Venezuela is assumed to be 1.75 and 1.45 respectively. Therefore the location factor for labor is: $\frac{1}{8.5} \times 1.75 \times 1.45 = 0.3$.

Imported Equipment and Materials

A factor of 1.12 was used to determine cost of equipment and materials imported from foreign countries resulting from freight costs.

Construction

In order to determine the location factor for construction, the following assumptions were made.

The cost breakdown for the Engineering News-Record construction cost index is: common labor + materials. Materials include lumber, cement, and steel. Common labor is approximately 76 percent of the total and materials is 24 percent (breakdown CE Plant Cost Index).

Presently the cost of materials in Venezuela is approximately the same as those in the U.S. Therefore, if we assume 0.3 as the correction factor for labor and 1 for materials, the correction factor for construction will be approximately

$$.76 \times .3 + .24 \times 1 = 0.47.$$

Fuel

The costs of fuel in Venezuela were assumed to be half of fuel costs in the U.S. Therefore, a factor of 0.5 is used as location factor for fuel.

Electrical energy

The average cost of electrical energy in the U.S. in 1970 was 4.64¢/Kwh. In the same year, the same cost in Venezuela was 1.15¢/Kwh. Therefore, the correction factor for electrical energy is 0.66.

Breakdown of costs used as reference

The following tables show some of the cost breakdown used as reference to determine costs in the case study.

TABLE I.2

CE Plant Cost Index Costs Breakdown

Cost element	Weight (%)
Equipment, machinery, and supports	61
Construction labor	22
Building materials and labor	7
Engineering, supervision, and manpower	10

TABLE I.3

Investment Costs Breakdown Derived from Selected U.S. Open Pit Mines (SME, 1973)

Cost Element	Weight(%) Lower	Weight(%) Upper
Equipment	57	83
Construction	7	33
Engineering	5	15

TABLE I.4

Direct Operating Costs Breakdown Derived From Selected Open Pit Canadian Mines (Canadian Mining Journal, 1973)

Cost Element	Weight(%) Lower	Weight(%) Upper
Labor	20	40
Materials	28	56
Power	5	30
Other-direct and miscellaneous	5	35

TABLE I.5

Operating Cost Breakdown Derived from the 1967 Census of Mineral Industries

Cost Element	Weight (%)
Labor	50
Explosives	3
Steel, mill shapes, and forms	7
All other supplies	27
Fuels	6
Electric energy	7

Transportation Cost Estimation

First Stage. Transportation by trucks from the mine to the metallurgical plant.

The distance between the mine and metallurgical plant is approximately 60 miles.

Transportation costs by trucks are estimated in 0.95 cents per ton per mile (Tovar, 1972). Loading is estimated as \$.735 per ton and unloading as \$.98 per ton (Link, 1974).

If we assume that the increase in transportation cost by trucks is mostly due to the increase in the cost of fuels, the cost of transportation updated to the year 1976 can be estimated as follows.

$(.0095 \times 60) \times 1.61 + (0.735 + .98) 1.2 = \$2.97$ per ton,

where the factors 1.61 and 1.21 represent the increase in fuel cost from the years 1972 and 1974, respectively.

Second Stage. Transportation by ship from the metallurgical plant to the markets.

Shipping costs from Venezuela to the U.S. East Coast are estimated as \$5.17 per ton (Link, 1974).

If we assume that the increase in shipping costs are mostly due to the increase in fuel costs, the cost of shipping updated to the year 1976 can be estimated as follows:

$5.17 \times 1.20 = \$6.20$ per ton.

Determination of the Cost Equations from Statistical Data

Cost equations are determined graphically from the statistical data available.

The calculation of the operating cost equation for milling will illustrate the way in which these equations are determined.

In Figure I.1 is illustrated the curve graphically constructed from milling operating cost data available. The general form of this equation is as follows:

$$y = a + bX - cX^2 + dX^3.$$

The coefficient, a, represents the intersection of the curve with the ordinates axis; therefore, a, is equal to zero in this case.

The value of the coefficient, b, is the derivative of the equation when X is equal to zero.

$$y' = b - 2cX + 3dX^2$$

when $X = 0$, $y' = b$.

This value is obtained graphically; and, in this case, equal to 1.

A system of two equations using the first and second derivatives permits one to obtain the values of the coefficients "c" and "d".

$$y' = \frac{2.01 - 0.35}{5} = 0.33$$

$$y'' = 0$$

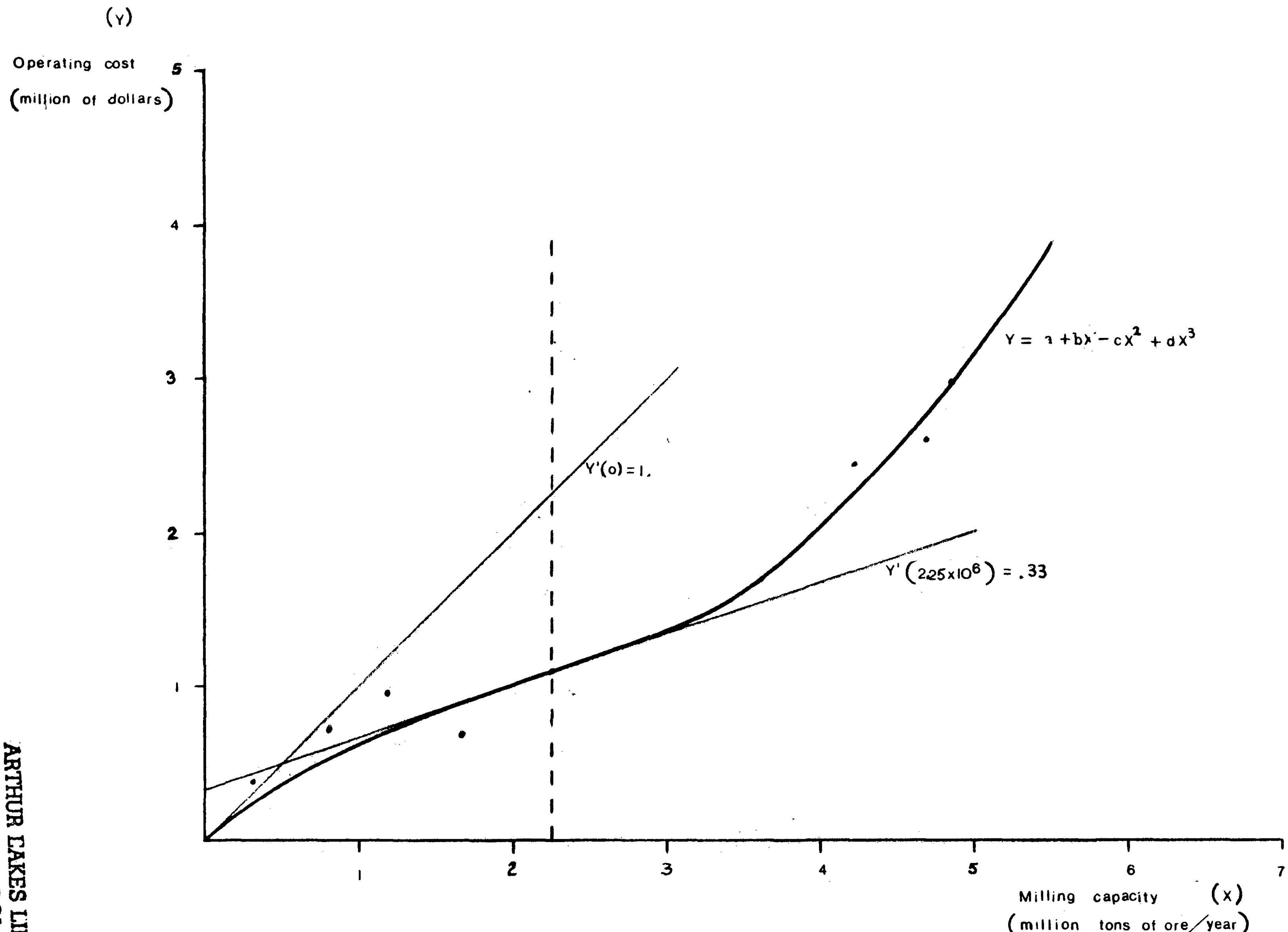

Figure I.1 Determination of Cost Equations. Milling Operating Costs versus Annual Capacity.

Therefore, when $X = 2.25 \times 10^6$, we have:

$$0.33 = 1 - 2C\,(2.25\text{x}10^6) + 3d(2.25\text{x}10^6)^2$$

$$0 = -2C + 6d(2.25\text{x}10^6), \text{ and}$$

from these two equations we obtain

$$C = 2.9777\text{x}10^{-7}$$

$$d = 4.41138\text{x}10^{-14}\,.$$

APPENDIX II

Program Listings, Inputs, and Outputs

```
C
C      OPTIMIZATION OF MINE DEVELOPMENT
C      APPLYING MONTECARLO SIMULATION
C
C      BY RAFAEL E. BORGES
C
C      MINERAL ECONOMICS DEPERTMENT
C      COLORADO SCHOOL OF MINES
C
C      DESCRIPTION OF INPUT VARIABLES
C
C      NS : NUMBER OF SIMULATIONS
C      IC : CONFIDENCE INTERVAL
C      NCO : NUMBER OF CUT-OFF GRADES TC EE CONSIDERED
C      NCA : NUMBER OF CAPACITIES
C      COMI : MINIMUM CUT-OFF
C      COMA : MAXIMUM CUT-OFF
C      CAPMI : MINIMUM CAPACITY
C      CAPMA : MAXIMUM CAPACITY
C      IPPMI : MINIMUM PREOPERATIONAL PERIOD
C      IPPMA : MAXIMUM PREOPERATIONAL PERIOC
C      A : RESERVES OF TOTAL DEPOSIT
C      DILU : DILUTION FACTOR
C      NFTR : KIND OF DEPOSIT
C      NAG : NUMBER OF VALUES IN THE AVERAGE GRADE DIST.
C      VAG(I) : VALUES OF AVERAGE GRACES
C      PIAG(I) : PERCENTAGE OF EACH VALUE IN THE AVERAGE
C                GRADE DISTRIBUTICN
C      NPR : NUMBER OF VALUES IN THE PRICE DISTRIBUTION
C      ATPR : ANNUAL TREND IN PRICES
C      DPPR : DISCOUNT PARAMETER WHICH INFLUENCE RATE OF
C             CHANGE
C      NFACPR : =1 PRICE GIVEN PER UNIT OF FINAL PRODUCT
C               =Ø PRICE GIVEN PER QUANTITY OF METAL IN
C                  THE FINAL PRODUCT
C      VPR(I) : VALUES OF PRICES
C      PIPR(I) : PERCENTAGE OF EACH VALUE IN THE PRICE
C                DISTRIBUTION
C      NPRO : NUMBER OF SUBPROCESSES
C      AGB : AVERAGE GRADE BASE
C      CCO(I) : BASIC CAPACITIES CONSIDERED TO DETERMINE
C               RANGES
C      REC(I) : RECOVERY FACTOR FOR EACH PROCESS
C      FREC(I) : FACTOR USED TO DETERMINE HOW SOME COSTS
C                ARE AFECTED WHEN AGE IS MODIFIED
C      POR (I+1) : PERCENTAGE OF METAL AT THE END OF
C                  SUBPROCESS I, WHEN IT IS NOT KNOWN
C                  OR EQUAL TO THE LAST SUBPROCESS
C                  POR (I+1) HAS TO BE DEFINED ZERO, THE
C                  CORRESPONDENT VALUE IS ASSIGNED IN THE
C                  PROGRAM
```

```
C      CCJ(I),CCK(I),CCL(I),CCM(I) : INDEPENDENT COEF.
C               OF THE COST INVESTMENT FUNCTION FOR
C               SUBPROCESS I
C      COJ(I),COK(I),COL(I),COM(I) : INDEPENDENT COEF.
C               OF THE OPERATING COST FUNCTION FOR
C               SUBPROCESS I
C      NOC : NUMBER OF VALUES IN THE OPERATING COST DIST.
C      ATOC : ANNUAL TREND IN OPERATING COSTS
C      DPOC : DISCOUNT PARAMETER WHICH INFLUENCE RATE OF
C             CHANGE
C      VOC(I) : VALUES OF OPERATING COSTS
C      PIOC(I) : PERCENTAGES OF EACH VALUE IN THE OPERATING
C                 COST DISTRIBUTION
C      NCC : NUMBER OF VALUES IN THE CAPITAL INVESTMENT
C               COST DISTRIBUTION
C      LIFE : MAXIMUM LIFE TIME CONSIDERED
C      PDI : PERCENTAGE OF DEPRECIABLE INVESTMENT
C      FWC : WORKING CAPITAL FACTOR
C      RIC : ANNUAL TREND IN CAPITAL INVESTMENT
C      VCC(I) : VALUES OF CAPITAL COSTS
C      PICC(I) : PERCENTAGES OF EACH VALUE IN THE CAPITAL
C                   COST DISTRIBUTION
C      RINT1 : MINIMUM VALUE OF INTEREST TO BE CONSIDERED
C                 TO CALCULATE PRESENT VALUES
C      RINT2 : MAXIMUM VALUE OF INTEREST
C      NINCR : NUMBER OF INCREMENT IN INTEREST
C      TAX : TAXES (% ON PROFITS BEFORE TAXES)
C      ROYAL : ROYALTIES ($/UNIT OF ORE EXPLOITED)
C
C      MAIN PROGRAM
C
       DIMENSION VAG(20),PIAG(20),VPR(20),PIPR(20),
      1   VOC(20),PIOC(20),VCC(20),PICC(20),ICOO(15),
      2   COO(15),CAA(15),IAVROR(15,15),
      3   AVROR(15,15),IALL(15,15),ALL(15,15)
      4   ,CCJ(5),CCK(5),CCL(5),CCM(5)
      5   ,COJ(5),COK(5),COL(5),COM(5)
      6   ,REC(5),POR(6),CCO(5),FREC(5)
       DOUBLE PRECISION CCJ,CCK,CCL,CCM,COJ,COK,COL,COM
       WRITE(4,2000)
2000   FORMAT(1X,'TYPE IN READ AND WRITE'/)
       READ(4,1000)IR,IW
       READ(IR,1000) NS,IC,NCO,NCA
       MNS=1
1000   FORMAT(8I)
       WRITE(IW,5000) NS,IC,NCO,NCA
5000   FORMAT('1',//,15X,'OPTIMIZATION OF MINE DEVELOPMENT',
      1   ' (DATA)',//,15X,'NS IC NCO NCA',/,15X,4I10)
       READ(IR,1001) COMI,COMA,CAPMI,CAPMA,IPPMI,IPPMA
1001   FORMAT(4F,2I)
       WRITE(IW,5001) COMI,COMA,CAPMI,CAPMA,IPPMI,IPPMA
```

```
5001  FORMAT(15X,'COMI COMA CAPMI CAPMA IPPMI IPPMA',/,
     1   15X,2F6.4,2F10.0,2I5)
      READ(IR,1009) A,DILU,NFTR
1009  FORMAT(2F,I)
1002  FORMAT(3F,I)
      WRITE(IW,5002) A,DILU,NFTR
5002  FORMAT(15X,'A DILU NFTR',/,15X,F15.0,F10.5,I10)
      READ(IR,1000) NAG
      WRITE(IW,5003) NAG
5003  FORMAT(15X,'NAG',/,15X,I10,/,15X,'VAG(I) PIAG(I)')
      DO 10 I=1,NAG
      READ(IR,1003) VAG(I),PIAG(I)
10    WRITE(IW,5004)VAG(I),PIAG(I)
5004  FORMAT(15X,2F10.5)
1003  FORMAT(2F)
      READ(IR,1004) NPR,ATPR,DPPR,NFACPR
1004  FORMAT(I,2F,I)
      WRITE(IW,5005) NPR,ATPR,DPPR,NFACPR
5005  FORMAT(15X,'NPR ATPR DPPR NFACPR',/,15X,I5,2F10.5,I5,
     1   /,15X,'VPR(I) PIPR(I)')
      DO 15 I=1,NPR
      READ(IR,1003) VPR(I),PIPR(I)
15    WRITE(IW,5006)VPR(I),PIPR(I)
5006  FORMAT(15X,F10.2,F8.5)
      READ(IR,1004) NPRO,AGB
      WRITE(IW,5007) NPRO,AGB
5007  FORMAT(15X,'NPRO AGB',/,15X,I10,F10.5,/,15X,'CCO(I)')
      READ(IR,1006) (CCO(I),I=1,NPRC)
      WRITE(IW,5008)(CCO(I),I=1,NPRC)
5008  FORMAT(15X,5F10.0)
      WRITE(IW,5017)
5017  FORMAT(15X,'REC(I) FREC(I) POR(I+1)',/,15X,
     1 'CCJ CCK CCL CCM',/,15X,'COJ COK COL COM')
      DO 16 I=1,NPRO
      READ(IR,1006) REC(I),FREC(I),POR(I+1)
      WRITE(IW,5009) REC(I),FREC(I),POR(I+1)
5009  FORMAT(15X,3F10.5)
      READ(IR,1005) CCJ(I),CCK(I),CCL(I),CCM(I)
      WRITE(IW,5010)CCJ(I),CCK(I),CCL(I),CCM(I)
5010  FORMAT(15X,4E)
      READ(IR,1005) COJ(I),COK(I),COL(I),COM(I)
16    WRITE(IW,5010)COJ(I),COK(I),COL(I),COM(I)
1005  FORMAT(8E)
1006  FORMAT(8F)
      READ(IR,1004) NOC,ATOC,DPOC
      WRITE(IW,5011)NOC,ATOC,DPOC
5011  FORMAT(15X,'NOC ATOC DPOC',/,15X,I10,2F10.5,/,
     1   15X,'VOC(I) PIOC(I)')
      DO 20 I=1,NOC
      READ(IR,1003) VOC(I),PIOC(I)
20    WRITE(IW,5012)VOC(I),PIOC(I)
```

```
5012  FORMAT(15X,F15.0,F10.5)
      READ(IR,1007)NCC,LIFE,PDI,FWC,RIC
1007  FORMAT(2I,3F)
      WRITE(IW,5013)NCC,LIFE,PDI,FWC,RIC
5013  FORMAT(15X,'NCC LIFE PDI FWC RIC',/,15X,
     1    2I10,3F10.5,/,15X,'VCC(I) PICC(I)')
      DO 25 I=1,NCC
      READ(IR,1003)VCC(I),PICC(I)
25    WRITE(IW,5014)VCC(I),PICC(I)
5014  FORMAT(15X,F15.0,F10.5)
      READ(IR,1008)RINT1,RINT2,NINCR
1008  FORMAT(2F,I)
      WRITE(IW,5015)RINT1,RINT2,NINCR
5015  FORMAT(15X,'RINT1 RINT2 NINCR',/,15X,2F10.5,I5)
      READ(IR,1003)TAX,ROYAL
      WRITE(IW,5016)TAX,ROYAL
5016  FORMAT(15X,'TAX ROYAL',/,15X,2F10.5)
      CALL MAXCC(IR,IW,COMI,COMA,CAPMA,CAPMI,IPPMI,IPP
     1 MA,IC,NS,MNS,PIAG,VAG,PICC,VCC,PICC,VOC,ATOC,
     2 DPOC,PIPR,VPR,ATPR,DPPR,CCJ,CCK,CCL,CCM,EXCC1
     3,EXCC2,EXCC3,COJ,COK,COL,COM,EXCO1,EXCO
     4    2,EXCO3,CCO,NFTR,A,DILU,LIFE,NCA,NCC,AVROR,A
     5    LL,COO,CAA,PDI,FWC,RIC,REC,ROYAL,TAX,NFACPR,
     6    NAG,FA,RINT1,RINT2,NINCR,PCR,NPRO,FREC,AGB)
      DO 30 I=1,NCO
      DO 30 J=1,NCA
      IAVROR(I,J)=AVROR(I,J)
30    IALL(I,J)=ALL(I,J)
      DO 31 I=1,NCO
31    COO(I)=COO(I)*100.
      WRITE(IW,3002)
3002  FORMAT('1',//,
     1 ' DISCOUNTED CASH FLOW RATE CF RETURN (MEANS)')
      WRITE(IW,3000) (COO(I),I=1,NCC)
3000  FORMAT(///,15X,15(F6.2))
      DO 35 J=1,NCA
35    WRITE(IW,3001) CAA(J),(IAVROR(I,J),I=1,NCO)
3001  FORMAT(//,1X,F14.0,15(4X,I2))
      WRITE(IW,3003)
3003  FORMAT('1',//,
     1 ' DISCONTED CASH FLOW RATE CF RETURN (LOW LIMIT)')
      WRITE(IW,3000) (COO(I),I=1,NCC)
      DO 40 J=1,NCA
40    WRITE(IW,3001) CAA(J),(IALL(I,J),I=1,NCO)
      STOP
      END
```

```
      SUBROUTINE MAXCC (IR,IW,COMI,COMA,CAPMA,CAPMI,IPPM
     1   I,IPPMA,IC,NS,MNS,PIAG,VAG,PICC,VCC,PIOC,VOC,
     2   ATOC,DPOC,PIPR,VPR,ATPR,DPFR,CCJ,CCK,CCL,CCM,
     3   EXCC1,EXCC2,EXCC3,COJ,COK,COL,COM,EXCO1,EXCO2
     4   ,EXCO3,CCO,NFTR,A,DILU,LIFE,NCA,NCO,AVRCR,ALL,
     5   CCO,CAA,PDI,FWC,RIC,REC,ROYAL,TAX,NFACPR,
     6   NAG,FA,RINT1,RINT2,NINCR,PCR,NPRO,FREC,AGB)
      REAL MC
      DIMENSION COO(15),CAA(15),PIAG(20),AG(100),
     1   VAG(20),PICC(20),CC(100),VCC(20),PIOC(20),A
     2   OC(50),VOC(20),PIPR(20),PR(50),VPR(20),CASH(
     3   60),ROR(100),AVROR(15,15),IROR(100),ALL
     4   (15,15),
     5   PV(2),PVV(2,100),IAVPV(2,15,15),IPVLL
     6   (2,15,15),IPVHL(2,15,15),AVPV(2),PVLL(2),
     7   PVHL(2),MAXAV(2),MAXLL(2),MAXHL(2)
     8   ,POR(6),REC(5),CCO(5),X(5,100),FREC(5)
     9   ,FPRO(100),CCJ(5),CCK(5),CCL(5),CCM(5),
     1   COJ(5),COK(5),COL(5),COM(5)
      DOUBLE PRECISION CCJ,CCK,CCL,CCM,COJ,COK,COL,COM
      N1=50
      N3=150
      N5=250
      N7=350
      DO 5 I=1,NINCR+1
      MAXAV(I)=0
      MAXLL(I)=0
5     MAXHL(I)=0
      N2=1
      N4=1
      N6=1
      N8=1
      CCCOO=0.
      CCCAA=0.
      DO 9 I=1,NPRO
      CCCAA=CCCAA+CCJ(I)+CCK(I)*CCO(I)-CCL(I)*CCO(I)**
     1   2.+CCM(I)*CCO(I)**3.
9     CCCOO=CCCOO+COJ(I)+COK(I)*CCO(I)-CCL(I)*CCO(I)**
     1   2.+COM(I)*CCO(I)**3.
      ICO=0
      ICA=0
      CO=COMA
12    ICO=ICO+1
      COO(ICO)=CO
      CA=CAPMI
15    ICA=ICA+1
      CAA(ICA)=CA
      IF(CAPMA-CAPMI)25,25,20
20    IPP=(CA-CAPMI)*(IPPMA-IPPMI)/(CAPMA-CAPMI)+IPPMI
      GO TO 35
25    IPP=IPPMI
```

```
35      DCO=(COMA-COMI)/(NCO-1)
50      DCA=(CAPMA-CAPMI)/(NCA-1)
        REIC=IC
        CI=(100.-REIC)/(2.*100.)
        DO 75 I=1,NS
        CALL RAND(N1,N2,YFL)
        PC=0.
        II=1
65      PC=PC+PIAG(II)
        IF(YFL-PC) 75,70,70
70      II=II+1
        GO TO 65
75      AG(I)=VAG(II)
        DO 90 I=1,NS
        AG22=AG(I)+CO
        X(1,I)=CA
        X(2,I)=CA*DILU
        POR(1)=AG22
        POR(2)=AG22
        IF(NPRO.EQ.2)GO TO 77
        DO 76 I2=3,NPRO
        I22=I2-1
751     IF(POR(I22).EQ.0.)GO TO 752
        GO TO 753
752     I22=I22-1
        GO TO 751
753     PI2M1=POR(I22)
        PI2=POR(I2)
        IF(PI2.LT.PI2M1)PI2=PI2M1
        X(I2,I)=PI2M1*(REC(I2-1)+(AG(I)-AGB)*
       1 FREC(I2-1))*X((I2-1),I)/PI2
76      CONTINUE
77      PORNP1=POR(NPRO+1)
        I22=NPRO+1
771     IF(POR(I22).EQ.0.)GO TO 772
        GO TO 773
772     I22=I22-1
        GO TO 771
773     PORNP1=POR(I22)
        FPRO(I)=POR(NPRO)*(REC(NPRO)+(AG22-AGB)*FREC(NPRO))
       1 *X(NPRO,I)*((NFACPR*PORNP1)+1-NFACPR)/PORNP1
        CCCA=0.
        DO 78 I2=1,NPRO
78      CCCA=CCCA+CCJ(I2)+CCK(I2)*X(I2,I)-CCL(I2)*
       1   X(I2,I)**2.+CCM(I2)*X(I2,I)**3.
        CALL RAND (N3,N4,YFL)
        PC=0.
        II=1
80      PC=PC+PICC(II)
        IF(YFL-PC) 90,85,85
85      II=II+1
```

```
      GO TO 80
90    CC(I)=VCC(II)*CCCA/CCCAA
      NSR=0
      DO 150 IS=1,NS
      GO TO (95,95,95,95,95) NFTR
95    G=CO
      AGG=AG(IS)
      T=A/2.718**(G/AGG)
110   MC=AGG*T*(1+G/AGG)
      RRL=T/(CA*DILU)
      L=RRL
      DL=RRL-L
      L=L+1
      IF(L-LIFE)111,111,113
113   L=LIFE
      DL=1.
111   NSR=NSR+1
      IOP1=IPP+1
      IOPF=IPP+L
      CCCO=0.
      DO 112 I=1,NPRO
112   CCCO=CCCO+COJ(I)+COK(I)*X(I,IS)-COL(I)*X(I,IS)**
     1  2.+COM(I)*X(I,IS)**3.
      DO 125 I=IOP1,IOPF
      CALL RAND (N5,N6,YFL)
      PC=0.
      II=1
115   PC=PC+PIOC(II)
      IF(YFL-PC)125,120,120
120   II=II+1
      GO TO 115
125   AOC(I)=VOC(II)*((1+ATOC*(DPOC)**(I-1)*(I-1)))*CCCO
     1  /CCCOO
      AOC(IOPF)=AOC(IOPF)*DL
      DO 140 I=IOP1,IOPF
      CALL RAND (N7,N8,YFL)
      PC=0.
      II=1
130   PC=PC+PIPR(II)
      IF(YFL-PC)140,135,135
135   II=II+1
      GO TO 130
140   PR(I)=VPR(II)*(1+ATPR*(DPPR)**(I-1)*(I-1))
      CCU=CC(IS)
      FPROD=FPRO(IS)
      FPRODL=FPROD*DL
      CALL CASHF (IPP,L,CCU,AOC,PR,LIFE,PDI,FWC,RIC,
     1   MC,REC,CA,DILU,ROYAL,TAX,FPROD,FPRODL,CASH)
      CALL RATE (CASH,RR,IOPF)
      ROR(NSR)=RR
      CALL PVALUE(CASH,PV,IOPF,RINT1,RINT2,NINCR)
```

```
      DO 145 I=1,NINCR+1
145   PVV(I,NSR)=PV(I)
150   CONTINUE
      CALL PVTAB(NSR,MNS,PVV,NINCR,CI,AVPV,PVLL,PVHL,
     1   KKI)
      DO 151 I=1,NINCR+1
      IAVPV(I,ICO,ICA)=AVPV(I)
      IPVLL(I,ICO,ICA)=PVLL(I)
      IPVHL(I,ICO,ICA)=PVHL(I)
      IF(IAVPV(I,ICO,ICA).GT.MAXAV(I))MAXAV(I)=IAVPV(
     1   I,ICO,ICA)
      IF(IPVLL(I,ICO,ICA).GT.MAXLL(I))MAXLL(I)=IPVLL(
     1   I,ICO,ICA)
151   IF(IPVHL(I,ICO,ICA).GT.MAXHL(I))MAXHL(I)=IPVHL(
     1   I,ICO,ICA)
152   SROR=0.
      IF(NSR-MNS)155,160,160
155   AVROR(ICO,ICA)=1000.
      ALL(ICO,ICA)=1000.
      GO TO 240
160   DO 165 I=1,NSR
165   SROR=SROR+ROR(I)
      AVROR(ICO,ICA)=SROR/NSR
      IF(NSR-1)166,166,167
166   ALL(ICO,ICA)=AVROR(ICO,ICA)
      GO TO 240
167   DO 170 I=1,NSR
170   IROR(I)=ROR(I)*10.
      MIN=IROR(1)
      MAX=MIN
      DO 195 I=2,NSR
      IF(IROR(I)-MIN)180,185,185
180   MIN=IROR(I)
      GO TO 195
185   IF(IROR(I)-MAX)195,195,190
190   MAX=IROR(I)
195   CONTINUE
      S=0.
      I=MIN-1
200   I=I+1
      NU=0
      DO 210 J=1,NSR
      IF(I-IROR(J))210,205,210
205   NU=NU+1
210   CONTINUE
      RNU=NU
      RNSR=NSR
      S=RNU/RNSR+S
      IF(S-CI)215,220,220
215   IF(I-MAX)200,220,220
220   ALL(ICO,ICA)=I/10.
```

```
240   IF(CA-CAPMA)245,250,250
245   CA=CA+DCA
      GO TO 15
250   IF(CO-COMI)270,270,260
260   CO=CO-DCO
      ICA=0
      GO TO 10
270   PQRST=0.
      CALL WRITE2(NCO,NCA,IAVPV,IPVLL,IPVHL,MAXAV,MAXLL,
     1   MAXHL,NINCR,RINT1,RINT2,COC,CAA,IW,CI)
280   RETURN
      END
```

```
      SUBROUTINE CASHF(IPP,L,CCU,AOC,PR,LIFE,PDI,FWC,
     1 RIC,MC,REC,CA,DILU,ROYAL,TAX,FPROD,FPRODL,CASH)
      IOP1=IPP+1
      IOPF=IPP+L
      REAL MC
      DIMENSION CINV(10),WCI(50),PR(50),SALER(50),
     1      AOC(50),CASH(60)
      RIPP=IPP
      CCU2=CCU/RIPP
      DO 10 I=1,IPP
10    CINV(I)=CCU2*(1+RIC)**(I-1)
      WC=AOC(IOP1)*FWC
      CINV(IPP)=CINV(IPP)+WC
      DO 15 I=IOP1,IOPF
15    WCI(I)=0.
      WCI(IOPF)=WC
      TINV=0.
      DO 20 I=1,IPP
20    TINV=TINV+CINV(I)
      DINV=TINV*PDI
      RLIFE=LIFE
      DEP=DINV/RLIFE
      SV=DINV-DEP*L
      DO 25 I=IOP1,IOPF-1
      RL=L
25    SALER(I)=FPROD*PR(I)
      SALER(IOPF)=FPRODL*PR(IOPF)
      DO 30 I=1,IPP
300   FORMAT(3F)
30    CASH(I)=-CINV(I)
      CASH(IPP)=CASH(IPP)-WC
      DO 35 I=IOP1,IOPF
      GP=SALER(I)-AOC(I)-CA*DILU*ROYAL
      BTAX=GP-DEP
      TAXES=0.
      IF(BTAX.GT.0.) TAXES=BTAX*TAX
      NETP=BTAX-TAXES
35    CASH(I)=NETP+DEP+WCI(I)
      CASH (IOPF)=CASH(IOPF)+SV
      RETURN
      END
```

```
      SUBROUTINE RAND (IR,N,Y1)
      DIMENSION IA1(30),IA2(30),IA3(30),POW(30)
      N1=29
      Y1=0.
      MAX=28+IR
      IF(N-2)10,40,40
10    DO 20 I=1,29,2
      IA2(I)=0
      IA2(I+1)=1
      POW(I)=.5**I
      POW(I+1)=.5**(I+1)
20    CONTINUE
      IA2(29)=1
30    N1=N1+29
      N2=N1-MAX
40    DO 110 I=1,29
      IF(I-2)50,60,70
50    ITEMP=IA2(28)
      GO TO 80
60    ITEMP=IA2(29)
      GO TO 80
70    ITEMP=IA3(I-2)
80    IA3(I)=ITEMP+IA2(I)
      IF(IA3(I)-2)100,90,90
90    IA3(I)=0
100   IA1(I)=IA2(I)
      IA2(I)=IA3(I)
110   CONTINUE
      IF(N-2)120,170,170
120   IF(N2)30,170,130
130   N1=29-N2
      N2=N1+1
      N=2
      IZ=0
      DO 140 I=N2,29
      IZ=IZ+1
      IA2(IZ)=IA1(I)
140   CONTINUE
      IF(N1)150,170,150
150   DO 160 I=1,N1
      IZ=IZ+1
      IA2(IZ)=IA3(I)
160   CONTINUE
170   DO 180 I=1,29
      IZ=30-I
      ITEMP=IA2(IZ)
      Y1=Y1+POW(I)*ITEMP
180   CONTINUE
      RETURN
      END
```

```
      SUBROUTINE RATE (CASH,RR,NY)
      DIMENSION CASH(6Ø)
      RR=.7ØØØ1
      HOLD=Ø.
      PREV=Ø.
1Ø    Z=Ø.
      IF(RR-1.1Ø)4Ø,4Ø,14Ø
4Ø    DO 5Ø JJ=1,NY
      Z=Z+CASH(JJ)*(EXP(RR)-1.)/(RR*EXP(RR*JJ))
5Ø    CONTINUE
      IF(Z)6Ø,14Ø,1ØØ
6Ø    IF(HOLD)7Ø,7Ø,8Ø
7Ø    RR=Ø.
      GO TO 14Ø
8Ø    PREV=HOLD
      HOLD=RR
      AVL=ABS(RR-PREV)
      IF(AVL-.ØØØ5)14Ø,14Ø,9Ø
9Ø    RR=RR-AVL/2.
      GO TO 1Ø
1ØØ   IF(PREV)11Ø,11Ø,12Ø
11Ø   HOLD=RR
      RR=RR+1
      GO TO 1Ø
12Ø   PREV=HOLD
      HOLD=RR
      AVL=ABS(RR-PREV)
      IF(AVL-.ØØØ5)14Ø,14Ø,13Ø
13Ø   RR=RR+AVL/2.
      GO TO 1Ø
14Ø   RR=RR*1ØØ.
      RETURN
      END
```

```
      SUBROUTINE PVALUE (CASH,PV,IOFF,RINT1,RINT2,NINCR)
      DIMENSION CASH(6Ø),PV(2)
      DINT=(RINT2-RINT1)/NINCR
      RINT=RINT1
      DO 15 II=1,NINCR+1
      PV(II)=CASH(1)
      DO 1Ø I=2,IOPF
1Ø    PV(II)=PV(II)+CASH(I)/(1+RINT)**(I-1)
15    RINT=RINT+DINT
      RETURN
      END
```

```
      SUBROUTINE PVTAB (NSR,MNS,PVV,NINCR,CI,AVPV,PVLL,
     1   PVHL)
      DIMENSION AVPV(2),PVLL(2),PVHL(2),PVV(2,102)
      IF(NSR-MNS)10,13,15
10    DO 12 I=1,NINCR+1
      AVPV(I)=-1000.
      PVLL(I)=-1000.
12    PVHL(I)=-1000.
      GO TO 90
13    DO 14 I=1,NINCR+1
      AVPV(I)=PVV(I,1)
      PVLL(I)=PVV(I,1)
14    PVHL(I)=PVV(I,1)
      GO TO 90
15    IX=CI*NSR
      DO 80 II=1,NINCR+1
      SPV=0.
      DO 20 I=1,NSR
20    SPV=SPV+PVV(II,I)
      AVPV(II)=SPV/NSR
      NI=0
      NF=NSR+1
30    NI=NI+1
      NF=NF-1
      MIN=PVV(II,NI)
      MAX=MIN
      NORD1=0
      NORD2=0
      DO 50 I=NI,NF
      IF(PVV(II,I)-MIN)35,40,40
35    MIN=PVV(II,I)
      IMIN=I
      NORD1=1
      GO TO 50
40    IF(PVV(II,I)-MAX)50,50,45
45    MAX=PVV(II,I)
      IMAX=I
      NORD2=1
50    CONTINUE
      IF(NORD1.EQ.0) GO TO 55
      P=PVV(II,NI)
      PVV(II,NI)=PVV(II,IMIN)
      PVV(II,IMIN)=P
55    IF(NORD2.EQ.0) GO TO 60
      P=PVV(II,NF)
      PVV(II,NF)=PVV(II,IMAX)
      PVV(II,IMAX)=P
60    IF(NI-IX-1)65,70,70
65    IF((NF-NI).GT.2) GO TO 30
70    PVLL(II)=PVV(II,NI)
      PVHL(II)=PVV(II,NF)
```

```
80      CONTINUE
90      RETURN
        END
```

1857


```
      SUBROUTINE WRITE2 (NCO,NCA,IAVPV,IPVLL,IPVHL,
     1   MAXAV,MAXLL,MAXHL,NINCR,RINT1,RINT2,COO,
     2   CAA,IW,CI)
      DIMENSION IAVPV(2,15,15),IPVLL(2,15,15),IPVHL(2,
     1   15,15),MAXAV(2),MAXLL(2),MAXHL(2),MAX(2,3),
     3   IND(2,3),RINT3(2),COO2(15),COC(15),CAA(15)
      DINT=(RINT2-RINT1)/NINCR
      RINT3(1)=RINT1*100.
      DO10 I=2,NINCR+1
10    RINT3(I)=RINT3(I-1)+DINT*100.
      DO 15 I=1,NINCR+1
      MAX(I,1)=MAXAV(I)
      MAX(I,2)=MAXLL(I)
15    MAX(I,3)=MAXHL(I)
      DO 30 I=1,NINCR+1
      DO 30 L=1,3
      ICONT=100
20    IF(MAX(I,L).LT.ICONT) GO TO 25
      ICONT=ICONT*10
      GO TO 20
25    IND(I,L)=ICONT/100
30    CONTINUE
      DO 35 I=1,NINCR+1
      DO 35 J=1,NCO
      DO 35 K=1,NCA
      IAVPV(I,J,K)=IAVPV(I,J,K)/IND(I,1)
      IPVLL(I,J,K)=IPVLL(I,J,K)/IND(I,2)
35    IPVHL(I,J,K)=IPVHL(I,J,K)/IND(I,3)
      DO 36 I=1,NCO
36    COO2(I)=COO(I)*100.
      CI2=100.-CI*200.
      DO 60 I=1,NINCR+1
      WRITE(IW,1001) RINT3(I),CI2
1001  FORMAT('1',1X,'RATE OF INTEREST= ',F5.2,' PER CENT',
     1   /,1X,'CONFIDENCE INTERVAL= ',F5.2,' PER CENT')
      WRITE(IW,1002)MAX(I,1),(COO2(INCC),INCC=1,NCO)
1002  FORMAT(//,1X,'PRESENT VALUE (MEANS)',/,1X,
     1   'MAX= ',I15,//,15X,15F6.2)
      DO 40 K=1,NCA
40    WRITE(IW,1003)CAA(K),(IAVPV(I,J,K),J=1,NCO)
1003  FORMAT(//,1X,F14.0,15(4X,I2))
      WRITE(IW,1004)MAX(I,2),(COO2(INCC),INCO=1,NCO)
1004  FORMAT('1',/////,1X,'PRESENT VALLE (LOW LIMIT)',/,1X,
     1   'MAX= ',I15,//,15X,15F6.2)
      DO 45 K=1,NCA
45    WRITE(IW,1003)CAA(K),(IPVLL(I,J,K),J=1,NCO)
      WRITE(IW,1005)MAX(I,3),(COC2(INCC),INCC=1,NCO)
1005  FORMAT('1',/////,' PRESENT VALLE (HIGH LIMIT)',/,1X,
     1   'MAX= ',I15,//,15X,15F6.2)
      DO 50 K=1,NCA
50    WRITE(IW,1003)CAA(K),(IPVHL(I,J,K),J=1,NCO)
```

```
60    CONTINUE
      RETURN
      END
```

```
OPTIMIZATION OF MINE DEVELOPMENT (DATA)

NS IC NCO NCA
        30        90        13        11
COMI COMA CAPMI CAPMA IPPMI IPPMA
 .0300 .1200  100000.  2100000.    2    4
A DILU NFTR
      30000000.   0.89000        1.
NAG
        7
VAG(I) PIAG(I)
   0.03050   0.00500
   0.03230   0.04500
   0.03410   0.20000
   0.03590   0.50000
   0.03770   0.20000
   0.03950   0.04500
   0.04130   0.00500
NPR ATPR DPPR NFACPR
    6   0.00000   1.00000    0
VPR(I) PIPR(I)
   4409.00 0.08000
   4962.00 0.20000
   5512.00 0.40000
   6063.00 0.20000
   6614.00 0.10000
   7165.00 0.02000
NPRO AGB
        5   0.00000
CCO(I)
   500000.   500000.   500000.    37440.    15600.
REC(I) FREC(I) POR(I+1)
CCJ CCK CCL CCM
COJ COK COL COM
   1.00000   0.00000   0.00000
  0.0000000E+00  0.2680000E+01  0.0000000E+00  0.0000000E+00
  0.8900000E+06  0.2610000E+00  0.0000000E+00  0.0000000E+00
   1.00000   0.00000   0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.2970000E+01  0.0000000E+00  0.0000000E+00
   0.85000   0.00000   0.36000
  0.0000000E+00  0.7020000E+01  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.1000000E+01  0.2977700E-06  0.4411380E-13
   0.80000   0.00000   0.99300
  0.0000000E+00  0.3767000E+04  0.2430000E-01  0.5630000E-07
  0.0000000E+00  0.1080000E+04  0.9259200E-02  0.5143980E-07
   1.00000   0.00000   0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.6200000E+01  0.0000000E+00  0.0000000E+00
NOC ATOC DPOC
        7   0.05770   1.00000
VOC(I) PIOC(I)
      22750000.   0.03000
      26000000.   0.06000
      29250000.   0.20000
      32500000.   0.30000
      35750000.   0.25000
      39000000.   0.12000
```

```
      42250000.     0.04000
 NCC LIFE PDI FWC RIC
        7          20    0.80000    0.25000    0.09000
 VCC(I) PICC(I)
      100000000.    0.03000
      105000000.    0.10000
      110000000.    0.20000
      115000000.    0.35000
      120000000.    0.20000
      125000000.    0.10000
      130000000.    0.02000
 RINT1 RINT2 NINCR
    0.22000    0.10000     1
 TAX ROYAL
    0.50000    1.00000
```

CONFIDENCE INTERVAL= 90.00 PER CENT

PRESENT VALUE (MEANS)
MAX= 33377374

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	**	**	**	**	**	**	**	**	**	**	**	**	**
300000.	-7	-9	3	1	-2	**	**	**	**	**	**	**	**
500000.	**	**	**	4	13	33	25	15	-6	**	**	**	**
700000.	**	**	**	**	**	**	-4	18	26	11	-9	**	**
900000.	**	**	**	**	**	**	**	**	**	11	6	-3	**
1100000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1300000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1500000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1700000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1900000.	**	**	**	**	**	**	**	**	**	**	**	**	**
2100000.	**	**	**	**	**	**	**	**	**	**	**	**	**

PRESENT VALUE (MEANS)
MAX= 214507814

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	0	-3	-6	-8	-8	-8	-9	-9	-9	-8	-8	-7	-7
300000.	6	7	10	11	12	11	6	1	-2	-3	-7	-9	**
500000.	0	3	7	10	14	19	19	19	14	8	3	-3	-7
700000.	**	**	-9	-4	2	8	13	19	21	18	14	6	-1
900000.	**	**	**	**	**	**	-8	5	14	18	17	14	4
1100000.	**	**	**	**	**	**	**	**	**	-1	7	8	5
1300000.	**	**	**	**	**	**	**	**	**	**	-4	6	9
1500000.	**	**	**	**	**	**	**	**	**	**	**	-4	7
1700000.	**	**	**	**	**	**	**	**	**	**	**	**	0
1900000.	**	**	**	**	**	**	**	**	**	**	**	**	**
2100000.	**	**	**	**	**	**	**	**	**	**	**	**	**

DISCOUNTED CASH FLOW RATE OF RETURN (MEANS)

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	1	0	0	0	0	0	0	0	0	0	0	0	0
300000.	8	8	9	9	9	8	6	3	1	0	0	0	0
500000.	3	5	7	9	10	11	11	10	9	6	4	0	0
700000.	0	0	0	0	3	7	9	10	11	10	8	5	1
900000.	0	0	0	0	0	0	0	4	8	10	9	9	4
1100000.	0	0	0	0	0	0	0	0	0	2	5	6	5
1300000.	0	0	0	0	0	0	0	0	0	0	1	4	6
1500000.	0	0	0	0	0	0	0	0	0	0	0	1	5
1700000.	0	0	0	0	0	0	0	0	0	0	0	0	2
1900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
2100000.	0	0	0	0	0	0	0	0	0	0	0	0	0

PRESENT VALUE (LOW LIMIT)
MAX= 0

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	**	**	**	**	**	**	**	**	**	**	**	**	**
300000.	**	**	**	**	**	**	**	**	**	**	**	**	**
500000.	**	**	**	**	**	**	**	**	**	**	**	**	**
700000.	**	**	**	**	**	**	**	**	**	**	**	**	**
900000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1100000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1300000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1500000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1700000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1900000.	**	**	**	**	**	**	**	**	**	**	**	**	**
2100000.	**	**	**	**	**	**	**	**	**	**	**	**	**

PRESENT VALUE (LOW LIMIT)
MAX= 155259852

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	-3	-6	**	**	**	**	**	**	**	**	**	-8	-8
300000.	1	4	6	5	9	5	0	-1	-5	-6	**	**	**
500000.	-4	-2	2	3	8	14	13	11	9	3	-3	-9	**
700000.	**	**	**	**	-4	3	5	14	15	5	9	0	-8
900000.	**	**	**	**	**	**	**	-5	3	10	11	7	-4
1100000.	**	**	**	**	**	**	**	**	**	**	-6	0	-1
1300000.	**	**	**	**	**	**	**	**	**	**	**	-7	0
1500000.	**	**	**	**	**	**	**	**	**	**	**	**	-3
1700000.	**	**	**	**	**	**	**	**	**	**	**	**	**
1900000.	**	**	**	**	**	**	**	**	**	**	**	**	**
2100000.	**	**	**	**	**	**	**	**	**	**	**	**	**

DISCONTED CASH FLOW RATE OF RETURN (LOW LIMIT)

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	0	0	0	0	0	0	0	0	0	0	0	0	0
300000.	3	5	7	6	7	5	1	0	0	0	0	0	0
500000.	0	0	4	5	6	9	8	7	6	4	0	0	0
700000.	0	0	0	0	0	3	5	7	9	6	6	1	0
900000.	0	0	0	0	0	0	0	0	3	6	7	5	0
1100000.	0	0	0	0	0	0	0	0	0	0	0	1	1
1300000.	0	0	0	0	0	0	0	0	0	0	0	0	1
1500000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1700000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
2100000.	0	0	0	0	0	0	0	0	0	0	0	0	0

```
OPTIMIZATION OF MINE DEVELOPMENT (DATA)

NS IC NCO NCA
          1          90          13          11
COMI COMA CAPMI CAPMA IPPMI IPPMA
 .0000 .1200   100000.  2100000.     2    4
A DILU NFTR
      30000000.   0.89000           1
NAG
          1
VAG(I) PIAG(I)
   0.03590   1.00000
NPR ATPR DPPR NFACPR
   1   0.00000   1.00000    0
VPR(I) PIPR(I)
   4400.00 1.00000
NPRO AGB
          5   0.00000
CCO(I)
   500000.   500000.   500000.    37440.    15600.
REC(I) FREC(I) POR(I+1)
CCJ CCK CCL CCM
COJ COK COL COM
   1.00000   0.00000   0.00000
  0.0000000E+00  0.2680000E+01  0.0000000E+00  0.0000000E+00
  0.8900000E+06  0.2610000E+00  0.0000000E+00  0.0000000E+00
   1.00000   0.00000   0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.2970000E+01  0.0000000E+00  0.0000000E+00
   0.85200   0.00000   0.36000
  0.0000000E+00  0.7020000E+01  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.1000000E+01  0.2977700E-06  0.4411380E-13
   0.80000   0.00000   0.99300
  0.0000000E+00  0.3767000E+04  0.2430000E-01  0.5630000E-07
  0.0000000E+00  0.1080000E+04  0.9259200E-02  0.5143980E-07
   1.00000   0.00000   0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.6200000E+01  0.0000000E+00  0.0000000E+00
NOC ATOC DPOC
          1   0.05770   1.00000
VOC(I) PIOC(I)
      33174863.   1.00000
NCC LIFE PDI FWC RIC
          1          20   0.80000   0.25000   0.09000
VCC(I) PICC(I)
     114850000.   1.00000
RINT1 RINT2 NINCR
   0.02000   0.10000     1
TAX ROYAL
   0.50000   1.00000
```

DISCONTED CASH FLOW RATE OF RETURN (LOW LIMIT)

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	2	0	0	0	0	0	0	0	0	0	0	0	0
300000.	2	2	3	2	1	0	0	0	0	0	0	0	0
500000.	0	0	0	2	2	3	2	0	0	0	0	0	0
700000.	0	0	0	0	0	0	0	0	0	0	0	0	0
900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1100000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1300000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1500000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1700000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
2100000.	0	0	0	0	0	0	0	0	0	0	0	0	0

```
OPTIMIZATION OF MINE DEVELOPMENT (DATA)

NS IC NCO NCA
         1          90          13          11
COMI COMA CAPMI CAPMA IPPMI IPPMA
 .0000 .1200   100000.  2100000.     2     4
A DILU NFTR
      30000000.    0.89000              1
NAG
         1
VAG(I) PIAG(I)
   0.03590   1.00000
NPR ATPR DPPR NFACPR
    1   0.00000   1.00000     0
VPR(I) PIPR(I)
   5567.20 1.00000
NPRO AGB
         5    0.00000
CCO(I)
   500000.    500000.    500000.  .  37440.    15600.
REC(I) FREC(I) POR(I+1)
CCJ CCK CCL CCM
COJ COK COL COM
   1.00000    0.00000    0.00000
  0.0000000E+00  0.2680000E+01  0.0000000E+00  0.0000000E+00
  0.8900000E+06  0.2610000E+00  0.0000000E+00  0.0000000E+00
   1.00000    0.00000    0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.2970000E+01  0.0000000E+00  0.0000000E+00
   0.85000    0.00000    0.36000
  0.0000000E+00  0.7020000E+01  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.1000000E+01  0.2977700E-06  0.4411380E-13
   0.80000    0.00000    0.99300
  0.0000000E+00  0.3767000E+04  0.2430000E-01  0.5630000E-07
  0.0000000E+00  0.1080000E+04  0.9259200E-02  0.5143980E-07
   1.00000    0.00000    0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.6200000E+01  0.0000000E+00  0.0000000E+00
NOC ATOC DPOC
         1    0.05770    1.00000
VOC(I) PIOC(I)
      33174863.    1.00000
NCC LIFE PDI FWC RIC
         1          20    0.80000    0.25000    0.09000
VCC(I) PICC(I)
     129000000.    1.00000
RINT1 RINT2 NINCR
   0.02000    0.10000     1
TAX ROYAL
   0.50000    1.00000
```

DISCONTED CASH FLOW RATE OF RETURN (LOW LIMIT)

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	1	0	0	0	0	0	0	0	0	0	0	0	0
300000.	7	7	8	8	7	6	5	3	0	0	0	0	0
500000.	1	4	6	8	9	10	10	9	8	6	3	0	0
700000.	0	0	0	0	2	6	8	10	10	9	7	5	0
900000.	0	0	0	0	0	0	0	4	7	9	9	8	4
1100000.	0	0	0	0	0	0	0	0	0	2	5	6	5
1300000.	0	0	0	0	0	0	0	0	0	0	0	5	6
1500000.	0	0	0	0	0	0	0	0	0	0	0	0	5
1700000.	0	0	0	0	0	0	0	0	0	0	0	0	3
1900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
2100000.	0	0	0	0	0	0	0	0	0	0	0	0	0

```
OPTIMIZATION OF MINE DEVELOPMENT (DATA)

NS IC NCO NCA
         1        90        13        11
COMI COMA CAPMI CAPMA IPPMI IPPMA
 .0000 .1200   100000.  2100000.     2    4
A DILU NFTR
      30000000.   0.89000         1
NAG
         1
VAG(I) PIAG(I)
   0.03590   1.00000
NPR ATPR DPPR NFACPR
    1   0.00000   1.00000    0
VPR(I) PIPR(I)
   5567.00 1.00000
NPRO AGB
         5   0.00000
CCO(I)
   500000.   520000.   500000.    37440.    15600.
REC(I) FREC(I) POR(I+1)
CCJ CCK CCL CCM
COJ COK COL COM
   1.20000   0.00000   0.00000
  0.0000000E+00  0.2680000E+01  0.0000000E+00  0.0000000E+00
  0.8900000E+06  0.2610000E+00  0.0000000E+00  0.0000000E+00
   1.00000   0.00000   0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.2970000E+01  0.0000000E+00  0.0000000E+00
   0.85000   0.00000   0.36000
  0.0000000E+00  0.7020000E+01  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.1000000E+01  0.2977700E-06  0.4411380E-13
   0.80000   0.00000   0.99300
  0.0000000E+00  0.3767000E+04  0.2430000E-01  0.5630000E-07
  0.0000000E+00  0.1080000E+04  0.9259200E-02  0.5143980E-07
   1.00000   0.00000   0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.6200000E+01  0.0000000E+00  0.0000000E+00
NOC ATOC DPOC
         1   0.05770   1.00000
VOC(I) PIOC(I)
      42200000.   1.00000
NCC LIFE PDI FWC RIC
         1       20   0.80000   0.25000   0.09000
VCC(I) PICC(I)
     114850000.   1.00000
RINT1 RINT2 NINCR
   0.02000   0.10000    1
TAX ROYAL
   0.50000   1.00000
```

DISCONTED CASH FLOW RATE OF RETURN (LOW LIMIT)

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	0	0	0	0	0	0	0	0	0	0	0	0	0
300000.	4	4	5	3	2	0	0	0	0	0	0	0	0
500000.	0	0	1	3	4	5	4	0	0	0	0	0	0
700000.	0	0	0	0	0	0	0	1	0	0	0	0	0
900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1100000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1300000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1500000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1700000.	0	0	0	0	0	0	0	0	0	0	0	0	0
1900000.	0	0	0	0	0	0	0	0	0	0	0	0	0
2100000.	0	0	0	0	0	0	0	0	0	0	0	0	0

```
OPTIMIZATION OF MINE DEVELOPMENT (DATA)

NS IC NCO NCA
          1          90          13          11
COMI COMA CAPMI CAPMA IPPMI IPPMA
 .0000 .1200   100000.  2100000.    2    4
A DILU NFTR
      30000000.    0.89000          1
NAG
          1
VAG(I) PIAG(I)
   0.03250   1.00000
NPR ATPR DPPR NFACPR
    1   0.00000   1.00000    0
VPR(I) PIPR(I)
   5567.00 1.00000
NPRO AGB
          5   0.00000
CCO(I)
   500000.   500000.   500000.    37440.    15600.
REC(I) FREC(I) POR(I+1)
CCJ CCK CCL CCM
COJ COK COL COM
   1.00000    0.00000    0.00000
  0.0000000E+00  0.2680000E+01  0.0000000E+00  0.0000000E+00
  0.8900000E+06  0.2610000E+00  0.0000000E+00  0.0000000E+00
   1.00000    0.00000    0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.2970000E+01  0.0000000E+00  0.0000000E+00
   0.85000    0.00000    0.36000
  0.0000000E+00  0.7020000E+01  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.1000000E+01  0.2977700E-06  0.4411380E-13
   0.80000    0.00000    0.99300
  0.0000000E+00  0.3767000E+04  0.2430000E-01  0.5632000E-07
  0.0000000E+00  0.1080000E+04  0.9259200E-02  0.5143980E-07
   1.00000    0.00000    0.00000
  0.0000000E+00  0.0000000E+00  0.0000000E+00  0.0000000E+00
  0.0000000E+00  0.6200000E+01  0.0000000E+00  0.0000000E+00
NOC ATOC DPOC
          1   0.35770    1.00000
VOC(I) PIOC(I)
      33174863.    1.00000
NCC LIFE PDI FWC RIC
          1          20    0.80000    0.25000    0.09000
VCC(I) PICC(I)
     114850000.    1.00000
RINT1 RINT2 NINCR
   0.22000    0.13000     1
TAX ROYAL
   0.50000    1.00000
```

DISCONTED CASH FLOW RATE OF RETURN (LOW LIMIT)

	12.00	11.00	10.00	9.00	8.00	7.00	6.00	5.00	4.00	3.00	2.00	1.00	0.00
100000.	2	1	0	0	0	0	0	0	0	0	0	0	0
300000.	6	8	8	8	8	7	6	3	0	0	0	0	0
500000.	1	4	6	8	10	11	11	10	8	6	3	0	0
700000.	0	0	0	0	4	8	10	11	11	10	8	5	0
900000.	0	0	0	0	0	0	2	6	10	11	10	8	4
1100000.	0	0	0	0	0	0	0	0	1	5	8	7	4
1300000.	0	0	0	0	0	0	0	0	0	0	5	7	6
1500000.	0	0	0	0	0	0	0	0	0	0	0	5	6
1700000.	0	0	0	0	0	0	0	0	0	0	0	0	6
1900000.	0	0	0	0	0	0	0	0	0	0	0	0	5
2100000.	0	0	0	0	0	0	0	0	0	0	0	0	0

APPENDIX III

Flow Sheets of the Processes Proposed for Producing Titanium Metal

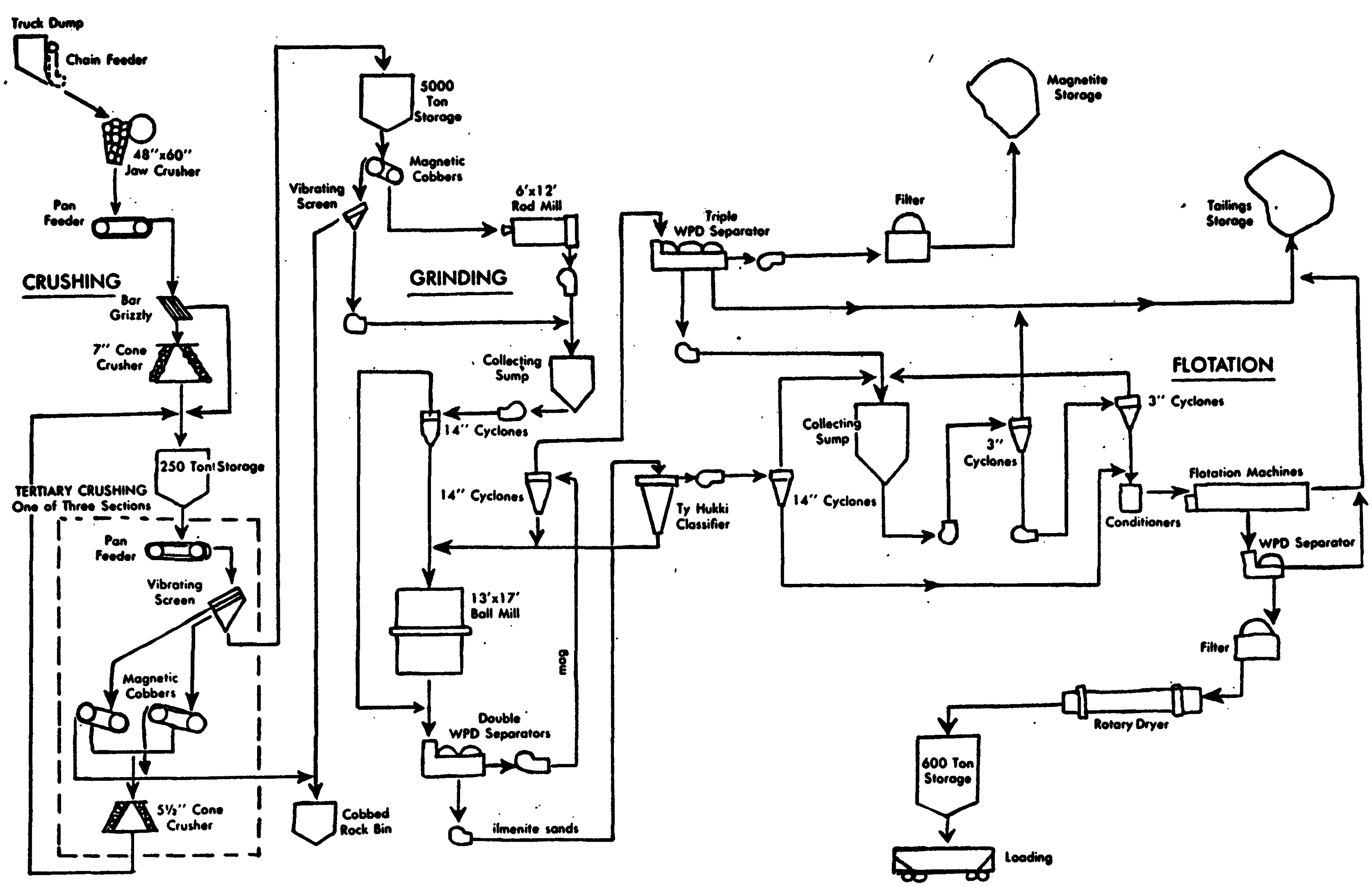

Figure III.1 Flowsheet for All-Flotation Ilmenite

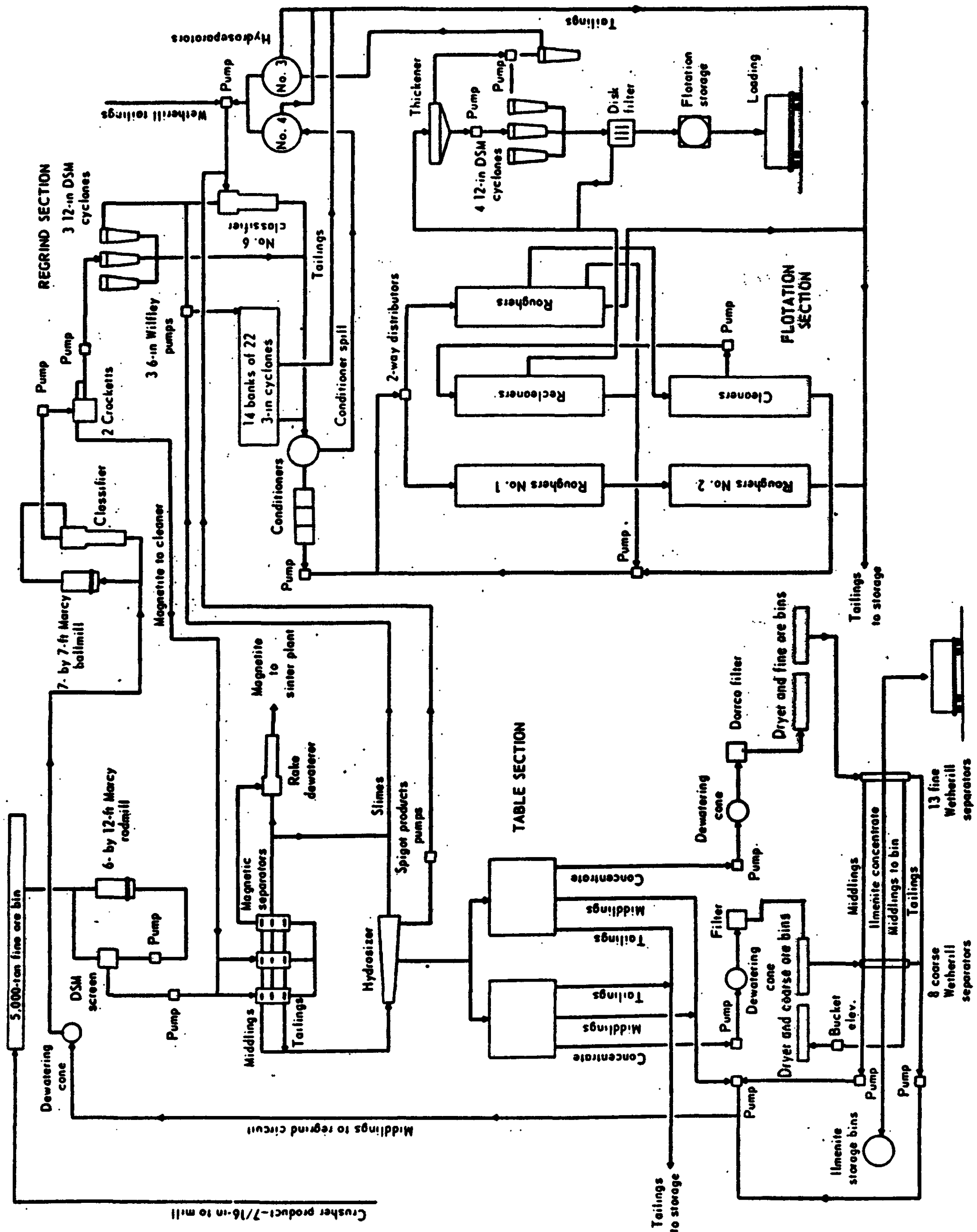

Figure III.2 Flowsheet for Magnetic and Flotation Ilmenite Concentrates

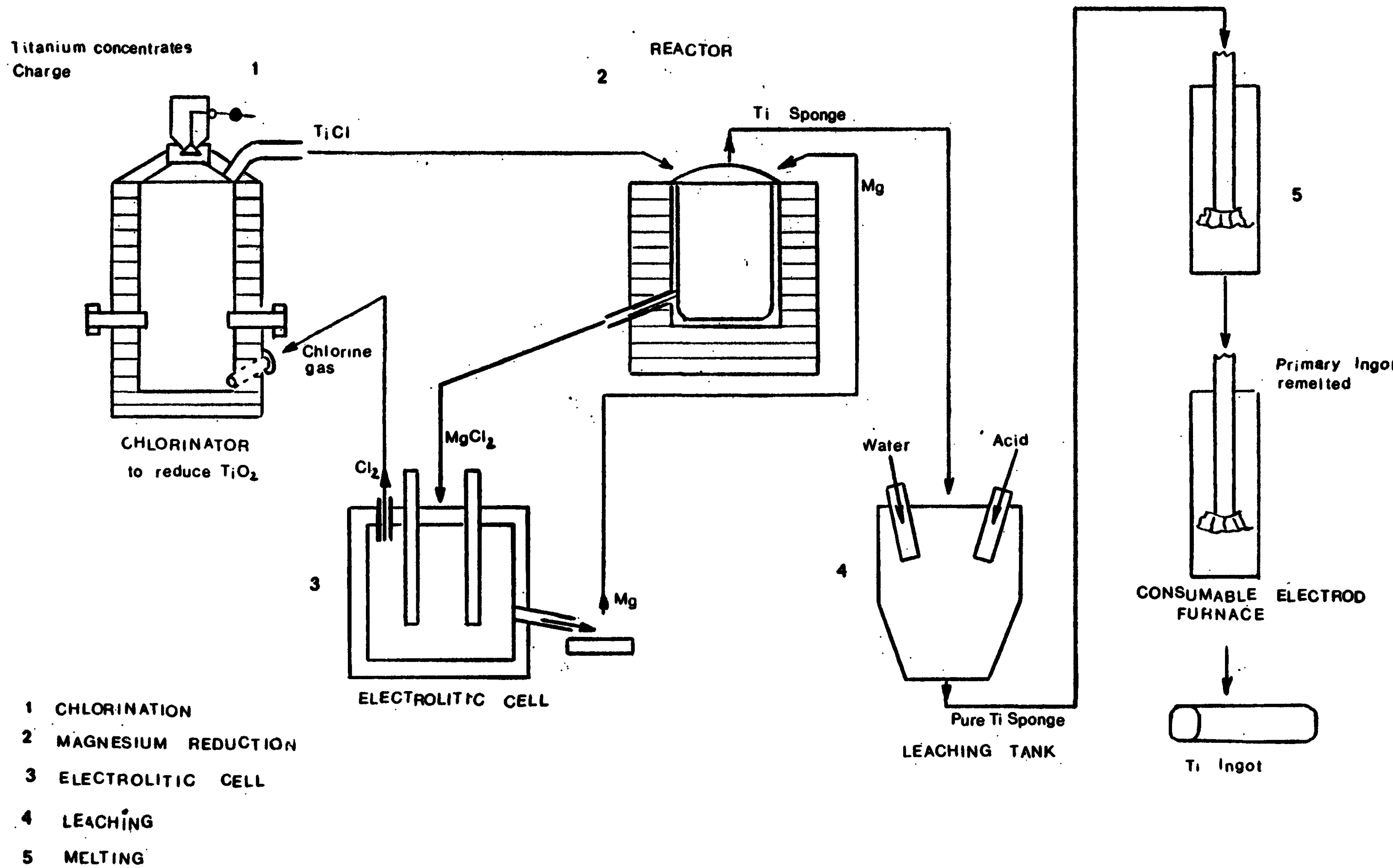

Figure III.3 Flowsheet of the Process for Producing Titanium Metal

APPENDIX IV

Results of the Analysis of Drill Holes

TABLE IV.1

Results of the Analysis of the Drill Holes

Drill Hole	Depth (mts)	Percentage of Ti	Percentage of TiO_2
S-3	8.50	1.58	2.65
S-4	23.40	3.11	5.19
S-5	44.79	2.84	4.73
S-7	31.00	3.12	5.21
S-8	100.00	5.70	9.51
S-9	36.39	5.86	9.78
S-10	48.00	4.68	7.80
S-11	100.00	3.85	6.42
S-12	45.22	4.46	7.47
S-13	36.96	4.85	8.09
S-14	48.00	6.09	10.15
S-15	43.30	6.80	11.35
S-16	40.10	4.86	8.11
S-17	40.00	3.18	5.31
S-18	100.00	4.29	7.16
S-19	40.00	2.74	4.57
S-20	41.48	3.19	5.31
S-21	48.11	4.47	7.46
S-22	37.31	0.99	1.66
S-23	48.00	3.90	6.51

Table IV.1 continued

Drill Hole	Depth (mts)	Percentage of Ti	Percentage of TiO_2
S-24	26.64	0.77	1.28
S-25	25.16	3.19	5.33
S-26	24.86	2.30	3.87
S-27	25.00	0.63	1.05
S-28	33.69	0.73	1.22
S-29	54.13	1.59	2.64
S-30	50.27	0.90	1.49
S-31	37.02	2.58	4.31
S-32	40.21	2.96	4.94
S-33	40.00	0.48	0.81

Source: Ministry of Mines and Hydrocarbons of Venezuela.

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