INCREMENTAL FINANCIAL ANALYSIS
APPLIED TO
OPEN PIT ORE HANDLING

by

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ABSTRACT

This study presents general methods of solving three major haulage problems encountered in the surface mining of extensive orebodies. Mathematical modeling and incremental financial analysis are used to represent the situations considered, and find optimal solution functions.

The first problem involves choosing between investing in rail haulage facilities and continuing the use of truck haulage. A mathematical model is used to find the financial consequences of varying ore reserves, production rates, depreciation periods, and tax rates by means of a computer program. The model takes into account the timing of expenditures and revenues by the use of discounting, the decision criterion used being the discounted cashflow rate of return, as obtained by a cost-saving investment.

In the case where a loading pocket might have to serve several orebodies, it is shown that the optimal location of the loading pocket is such that the sum of the present cost of haulage from all orebodies for their economic lives is at a minimum.

A preliminary analysis showing how to find the optimum depth to which orebodies should be mined is presented, with suggestions for further research.

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Finally, the author thanks his wife, Beatrice, for her patience and understanding during this period of study.

This study presents a general solution to a problem which is quite common, in different forms, in surface mining. In order to deal with a realistic situation, the author has taken for consideration the general features of the Iron Ore

Company of Canada's Carol Lake operation. Those details that are included have all been taken from published material; their sources are indicated in the text. All costs, tonnages, and other data whose sources are not given are entirely synthetic, and should under no circumstances be construed to be directly applicable to the operations of the Iron Ore Company of Canada, or any other organization.

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INTRODUCTION

General

The purpose of this study is to describe a method of approach to a haulage problem encountered in the surface mining of extensive orebodies, where both truck and rail haulage systems are used.

The general features of the situation considered are taken from the Iron Ore Company of Canada's mining operation at Carol Lake, Labrador. This company mines several large iron ore deposits which, being near the surface, require little or no stripping.

Briefly, the sequence of operations is as follows: after benches in the open pits are drilled and blasted, the ore is loaded into trucks (approximately 100 tons each) by large shovels. These trucks haul the ore to a conveniently located loading pocket. The loading pocket consists of an ore - pass (which acts as a storage bin with a large surge capacity), leading to a chain gate, and a pan feeder. Directly below the ore pass is a tunnel, through which automated trains haul the ore from the loading pocket to the processing mill, some miles away.

At the processing mill, the ore is concentrated, pelletized, and then shipped by rail to the seaport, for subsequent shipment by sea to various parts of the world.

More details on the general aspects of the operation are given by Harris, Carr, and Calder (1965); Pfleider and Dufresne (1961); Selleck and Pfleider (1968); and the Engineering & Mining Journal (1968, and 1969).

The question considered in more detail in this work concerns the decision as to whether a new loading pocket should be built to reduce haulage costs, when an open pit is being mined some distance from an existing loading pocket. Figure 1 shows some of the orebodies in the area considered; the position and size of the areas being worked are rather approximate.

This is part of a more general decision : choosing which orebodies to mine, in what order, and to what depth. A number of very extensive orebodies is involved, so it may not be economic to extract more than a portion of each one, at least in the foreseeable future.

In the operation described above, the specific decision concerning the loading pocket is of critical economic significance, as a choice must be made between (1) a very large investment (of the order of several million dollars) in tunnel driving and in the installation of automatic railroad facilities and (2) the cost of hauling several million tons of ore every year by truck over a distance that could amount to a number of miles.

The possibility of using one loading pocket to service more than one open pit (not necessarily more than one ore-

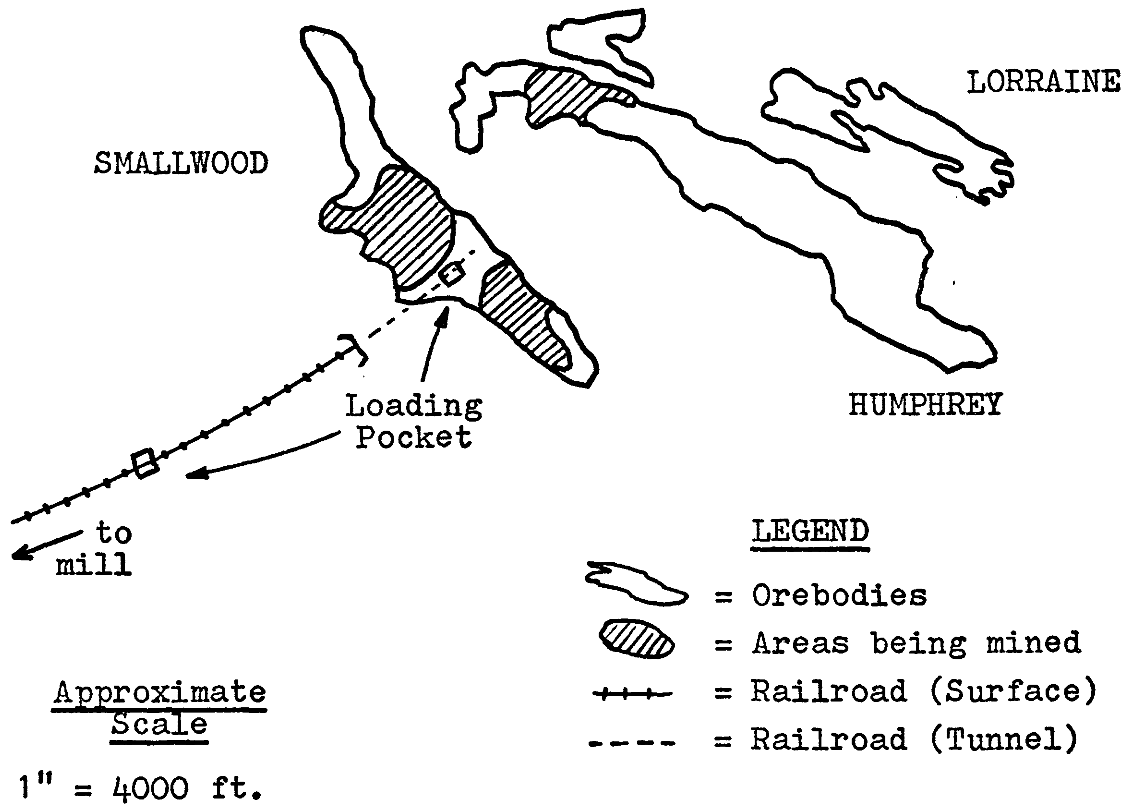


Figure 1. Distribution of Smallwood, Humphrey and Lorraine Orebodies.

(Selleck and Pfleider, 1968)

body) is also discussed, and a method of choosing its optimal location is presented.

The general method of analysis used in this study is applicable to a large variety of mining haulage problems. Several examples of haulage systems which use ore passes are given by Selleck and Pfleider (1968), and a number of different haulage combinations are described in "Case Studies of Surface Mining", edited by Hartman (1969). With the present trend towards large open pit operations, and the projected future trend towards deeper pits and surface - underground combinations, problems of the kind described in this study will become more prominent. Mines in which ore is extracted through several shafts from flat, bedded deposits are also amenable to a similar analysis; as operations progress further and further from existing shafts, new shafts are sunk to reduce haulage, ventilation, and other costs.

The relevance of this analysis to these essentially similar problems should be apparent, and its adaption should be conceptually simple, particularly as the preservation of a certain generality has been an objective in this study.

Problem Definition and Method of Approach

In the preceding section, the most general problem that must be solved was described as deciding which orebodies to mine, in what order, and to what depth. This involves complex long-term planning, and the solutions that are obtained must constantly be revised as more complete data become available, and as economic factors change. As mining of the various orebodies progresses, the possibility of abandoning a pit that has been worked to some depth, and starting a new one should be constantly reviewed.

The deeper a pit becomes, the greater the production cost per ton of ore. The more obvious reasons for this cost increase are : longer haulage distances, a higher stripping ratio (if applicable), and higher de-watering costs (greater influx of water, and pumping head). As an orebody may extend to great depth, it is clear that it will be more profitable in many cases to start mining a new orebody before exhausting the old one.

In order to decide whether to start mining a new orebody, the effect of the extra haulage distance to the concentrating plant must be considered. At this point, the choice between road and rail haulage arises in the case considered in this study. If rail haulage is to be used, a loading pocket must be built, and a tunnel driven to the bottom of the loading pocket.

The most important factors affecting this decision involve:

Capital Costs: Loading Pocket, Tunnel, and Railroad.

Haulage Costs: Truck and Rail.

Tonnage Data: Ore Reserves, Annual Production.

Fiscal Conditions: Tax Rate, Depreciation, Depletion.

One other essential factor has a direct bearing on this decision : the distance between the open pit and an existing loading pocket to which ore would have to be hauled by truck if a new loading pocket were not built. If a new loading pocket were built, ore would be hauled by rail, at a very much lower cost per ton.

Thus the building of a new loading pocket would result in a reduction in haulage costs. At the Humphrey orebody, shown in Figure 1, ore is being mined and hauled by truck to the loading pocket near the center of the Smallwood orebody. It is necessary to decide whether to build a loading pocket, say in the center of the Humphrey deposit, and to extend the tunnel to this point.

Best estimates of the factors mentioned above must be made for the open pit in the Humphrey orebody, and for possible future production from that portion of the orebody not being mined at present. The truck haulage distance avoided and the extra rail haulage involved are found. From these data, the investment required and the savings expected may be computed (for the years in which they apply) and the profitability of the investment may be calculated.

In order to make the mathematical model easier to apply, and to examine the effect of this factor, the distance between the old, and the projected new loading pocket has been taken as the independent variable (for a given set of other factors), the dependent variable being the rate of return.

This choice results in the generation of distance - rate of return curves for given sets of capital and haulage costs, reserve and production data, and fiscal conditions.

Profitability Criterion

Choice of Criterion.- In order to decide whether the investment in a railroad extension and a loading pocket is attractive, it must be compared with other investment opportunities available to the firm. To do this, some basis of comparison must be chosen.

Numerous profitability criteria are described in the financial literature (for example, Bierman and Schmit, 1960) and the relative merits of each are discussed. The main methods are: Payback; Accounting, or Book Rate of Return; and several methods which use discounting, the most important being Net Present Value, and Discounted Cashflow (D.C.F.) rate of return.

It is generally recognized that as profitability criteria, only techniques that take into account the time element are realistic. Thus only the Net Present Value and D.C.F. rate of return methods will be discussed. The payback period is not really a profitability criterion, rather it is a time or liquidity concept. Its value is in assessing the time an investment is exposed to risk, the payback period being the length of time required for the cash proceeds produced by the investment to equal the original cash outlay.

Description.- The term 'discounting' is basically the opposite of 'compounding'. Just as a sum of money P borrowed today would require a repayment of $P(1+r)$ one year hence (at an interest rate r), a sum P to be received one year hence would be worth $P/(1+r)$ today. Similarly a sum P to be received two years hence is worth $P/(1+r)^2$ today, and the same sum to be received n years hence is worth $P/(1+r)^n$ today. The use of discounting permits the reduction of future cashflows, whether outlays (negative), or inflows (positive), to a present value, just as compounding leads to a terminal value.

Thus if an investment leads to cashflows of A, A_2, A_3.. ..A_n arising at the end of years 1,2,3, ...n, then for a discount rate r, the Net Present Value (N.P.V.) may be expressed as:

$$NPV = \sum_{i=1}^{i=n} \frac{A_i}{(1+r)^i}$$

where: A_i = Net cash flow at the end of year i.

r = Firm's cost of capital.

n = Life of the project.

The discount rate, r, used in this case is the firm's cost of capital. If the Net Present Value is positive, then the project is expected to earn enough to repay the capital invested, plus interest at the discount rate used, and produce a cash surplus at the end of the project's life equal to the Net Present Value computed above.

The other discounting criterion referred to, the D.C.F. rate of return, uses the Net Present Value concept but instead of using a value of r equal to the firm's cost of capital, the value of r which results in a Net Present Value of zero is found. This value of r is the D.C.F. rate of return, often referred to as the 'yield' of the investment.

An excellent and thorough description of these methods is given by Merrett and Sykes (1963). A good shorter treatment is that of Groundwater (1967).

The advantage of the D.C.F. rate of return is that it is expressed as a percentage, which, being strictly comparable with the yield of securities, gives a meaningful idea of profit as a proportion of capital investment.
Net Present Value is meaningless unless the capital investment involved is also stated.

The main objective of any investment is to improve the firm's overall rate of return. This does not necessarily mean that a new investment should always earn a rate of return higher than the firm's average, although this is an objective. If money is invested at a low interest rate in a bank, or in government securities, reinvestment of this money in a capital project earning a higher rate will improve the firm's overall rate, whether or not the rate earned by the project itself is higher than the overall rate.

When comparing investment opportunities, the level of risk involved must be considered. The main sources of risk are given by Merrett and Sykes (1963), pp. 176-177, as:

(i) Risk from undertaking insufficient numbers of similar investments;
(ii) risk from misinterpretation of data;
(iii) risk from bias in the data and in its assessment;
(iv) risk from a changing external economic environment invalidating much of the usefulness of past experience; and
(v) risk from errors of analysis.

Risks from sources (ii) , (iii), and (iv) will vary with different projects, and the estimated risk must be considered in conjunction with the rate of return. The usual procedure is to set a 'target' rate of return which the company attempts to achieve with new projects, which involve little risk, and to expect proportionally higher rates of return from projects involving greater risks. Much work has been done recently on Risk Analysis, particularly on the use of simulation techniques, of which a general description is given by Hertz (1964), and an application to mining investment by O'Brian (1969).

Sources of Data

The cost data used to test the mathematical model are fictitious; however, they are within the range of costs actually incurred at different mines.

Comprehensive tables of operating costs are given by Michaelson and Hammes (1968), and by Pfleider and Wheaton (1968). The latter reference quotes the haulage cost per ton of ore at the Smallwood orebody of the Carol Lake operation as 18¢, for a combination of a mean truck haul distance of 6000 feet, and a rail haul distance of 6 miles. The former reference quotes costs for open pit copper mines varying very considerably, from about 3¢ to 26¢ per ton-mile, for different truck sizes and haul profiles.

Using Pfleider and Wheaton's average haul distances, and the haulage costs used in this study, the total haulage cost per ton of ore is 17¢ compared with the quoted total cost of 18¢.

METHOD OF ANALYSIS

Mathematical Model

For the solution of the haulage problem described above, a mathematical model was derived to represent the system in question.

The major factors which determine whether a new loading pocket should be built are given below, with the notation used in the derivation, and in the computer program.

Notation.-

RES = Ore reserves (tons)

ANPROD = Annual Production (tons)

N = Depreciable Life of Asset (years)

T = Tax Rate

CV1 = Truck Haulage Cost ($/ton-mile)

CV2 = Rail Haulage Cost ($/ton-mile)

CC1 = Cost of loading Pocket ($)

CC2 = Set-up (fixed) Expense for Building Tunnel ($)

CC3 = Tunnel and Railroad Cost ($/mile)

C1 = Total Capital Investment ($)

D1 = Length of Proposed Tunnel (miles)

This study describes a viable method of solving this problem, and shows the effect of variation of the physical and cost factors, instead of giving an answer for one specific situation. Consequently, the following assumptions and/or simplifications have been made.

Assumptions.-

1. That the length of the proposed tunnel is the same as the distance that the ore would have to be hauled by truck if the tunnel were not built. In practice the latter distance could be greater, but if this were the case, the model could be adjusted very simply.

2. That the truck and rail haulage costs are fixed in terms of dollars per ton-mile, and that these costs may be considered as actual cash flows. If, in the case of truck haulage, a large number of trucks were involved, and trucks were replaced at a constant rate, then for a given haul grade, the cost per ton would be very nearly a linear function of the distance hauled. Although these conditions are seldom met in practice, a detailed analysis of truck haulage costs is not considered an essential part of this report. In fact the use of a linear haulage cost function makes the general results of this study clearer. Also, the depreciation and ownership cost of truck haulage amounts to only a small percentage of the operating and maintenance cost.

3. Consideration of depletion allowances is not necessary in this case, but they can be easily incorporated if required.

4. That the cost of the loading pocket and the tunnel

may be considered as a capital expenditure. It may be possible to expense a certain proportion of this cost for tax purposes, a step which would of course be to the firm's advantage.

It should be noted that assumptions 1, 3, and 4 are conservative in that they will cause the profitability of the investment to be slightly underestimated.

Derivation.-

The objective of this model is to find the rate of return generated by investing in a railroad and tunnel extension, and a new loading pocket. This investment reduces operating costs, as ore haulage by rail is considerably less expensive than by truck. The notation defined above is used.

The annual saving in haulage costs, after tax, is:

$$ANBEN = ANPROD \cdot D1 \cdot (CV1 - CV2) \cdot (1 - T)$$

Since ANPROD is assumed constant, ANBEN is a constant annuity, which continues for the economic life of the orebody, N1 = RES/ANPROD. The present value of an annuity of ANBEN, for N1 years, is:

$$PVBEN = \frac{ANBEN}{(1+R)} + \frac{ANBEN}{(1+R)^2} + \frac{ANBEN}{(1+R)^3} \cdots + \frac{ANBEN}{(1+R)^{N1}}$$

where R is the discount rate. This equation (which is a geometric progression) can be simplified as follows. Multiplying both sides of the equation by 1/(1+R), then

$$\frac{PVBEN}{(1+R)} = \frac{ANBEN}{(1+R)^2} + \frac{ANBEN}{(1+R)^3} \cdots + \frac{ANBEN}{(1+R)^{N1}} + \frac{ANBEN}{(1+R)^{N1+1}}$$

If this second equation is subtracted from the first, only the first term of the series PVBEN, and the last term of the series PVBEN/(1+R) fail to cancel out, leaving:

$$PVBEN - \frac{PVBEN}{(1+R)} = \frac{ANBEN}{(1+R)} - \frac{ANBEN}{(1+R)^{N1+1}}$$

Multiplying both sides by (1+R), the expression simplifies to:

$$PVBEN(1+R) - PVBEN = ANBEN - ANBEN(1+R)^{-N1}$$

therefore

$$PVBEN \,.\, R = ANBEN(1-(1+R)^{-N1})$$

whence

$$PVBEN = ANBEN.\frac{1-(1+R)^{-N1}}{R}$$

The total capital investment is:

$$C1 = CC1 + CC2 + (D1.CC3)$$

As this investment is depreciated, it generates tax allowances during the depreciation period. If sum of the years' digits depreciation is used, the tax allowances will be:

$$TA = \frac{C1.T.N}{D} + \frac{C1.T.(N-1)}{D} + \frac{C1.T.(N-2)}{D} \ldots + \frac{C1.T.(1)}{D}$$

where D = the sum of the years' digits (1+2+3+4......+N). Thus the present value of the series of tax allowances is:

$$PVTA = \frac{C1.T}{D} \cdot \left[\frac{N}{(1+R)} + \frac{N-1}{(1+R)^2} + \frac{N-2}{(1+R)^3} \ldots + \frac{1}{(1+R)^N}\right]$$

Then the total net present value of the investment and the cashflows it generates is:

TNPV = - C1 + PVBEN + PVTA

The value of R which discounts TNPV to zero is the D.C.F. rate of return. As R cannot be obtained directly from the relationship above, an iterative procedure is used, successive values of R being tried until the value which reduces TNPV to zero is found. A description of the procedure used is given in the following section.

Computer Program

The program was written to allow the calculation of rate of return with respect to tunnel distance for any desired set of major cost and tonnage data. The program follows the derivation above quite closely.

A generalized block diagram of the program is given in Figure 2, and a typical output in Table 1. A listing of the program, which was written in FORTRAN IV, is given in Appendix 1.

The input data (ore reserves, annual production, allowable depreciation period, tax rate) are read from data cards, with the costs being defined by DATA statements. Thus any variation of orebody characteristics and fiscal conditions could be evaluated by reading different data cards.

To compute the D.C.F. rate of return, a very low initial value of R (discount rate) is taken, and the net present value of the cashflows (NPV) is found. This trial value of R

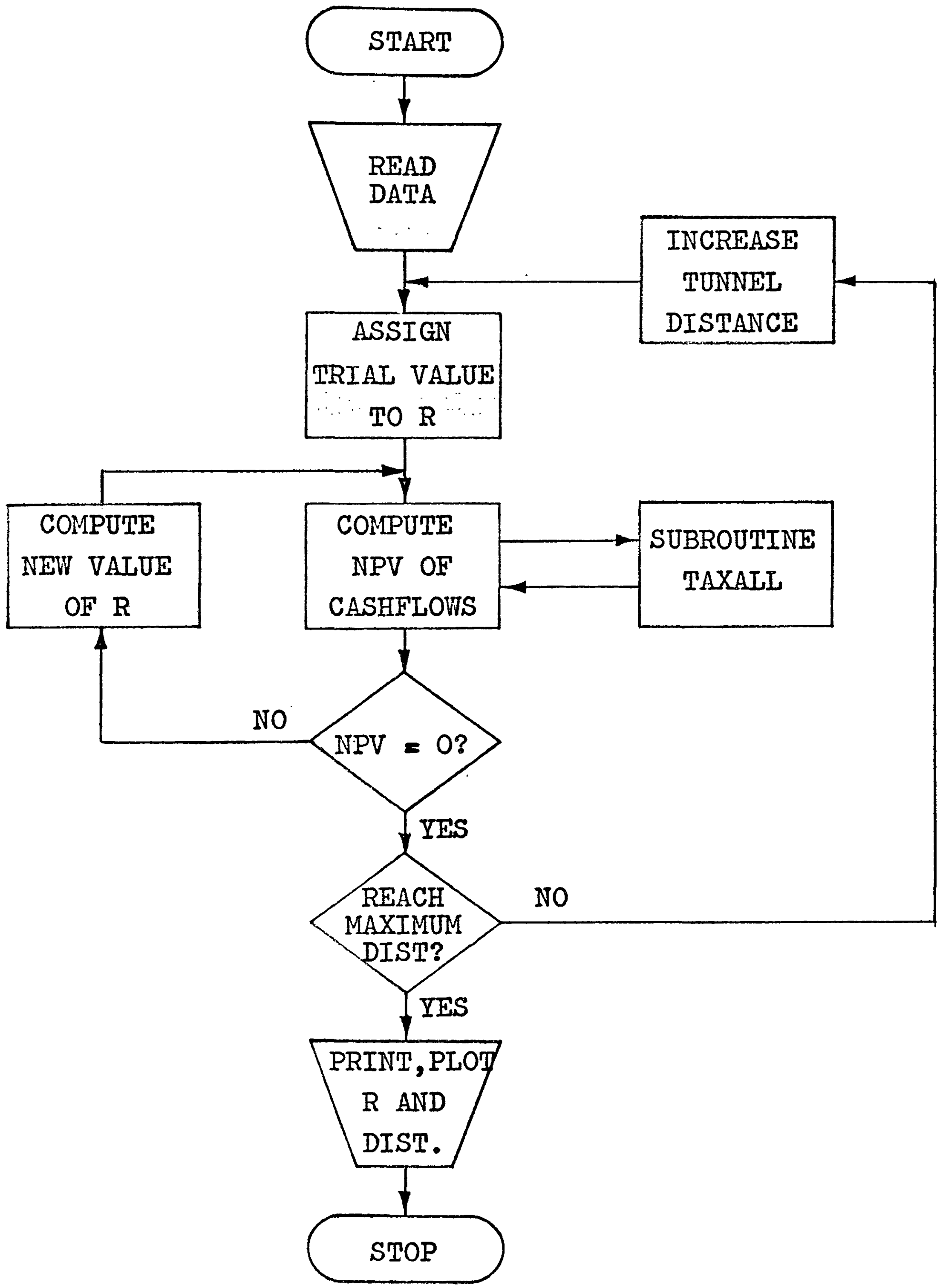


Figure 2. Block Diagram of the Computer Program

RESERVES = 250 MILLION TONS, ANNUAL PRODUCTION = 15 MILLION TONS
DEPRECIABLE LIFE OF INVESTMENT = 7 YEARS, TAX RATE = 50 PERCENT

D1 (MILES)	C1	PVTA	R
0,25	4980000,0	2109600,8	0.043
0,50	6960000,0	2562160,9	0.082
0,75	8940000,0	3083101,5	0.101
1,00	10920000,0	3617428,4	0.113
1,25	12900000,0	4175640,7	0.120
1,50	14880000,0	4722941,9	0.126
1,75	16860000,0	5282358,2	0.130
2,00	18840000,0	5827010,8	0.134
2,25	20820000,0	6398167,8	0.136
2,50	22800000,0	6961905,4	0.138
2,75	24780000,0	7518328,0	0.140
3,00	26760000,0	8067539,0	0.142
3,25	28740000,0	8636988,2	0.143
3,50	30720000,0	9202788,2	0.144
3,75	32700000,0	9764964,5	0.145
4,00	34680000,0	10323543,1	0.146
4,25	36660000,0	10878548,2	0.147
4,50	38640000,0	11430004,7	0.148
4,75	40620000,0	12015703,8	0.148
5,00	42600000,0	12561796,5	0.149

Table 1. Example of Computer Output.

must be lower than any anticipated actual value. In obtaining NPV, the present value of tax allowances at this discount rate is found using 'SUBROUTINE TAXALL'. This subroutine takes into account the depreciation method, (straight line, double declining balance, or sum of the years' digits, with provision for initial allowances, resale values, etc.) and the depreciation period.

If the value of NPV calculated above is positive, (which in the first case it should be), R is increased by a large amount (e.g. 10%, the initial value being of the order of -30%), and the process repeated, until the NPV becomes negative. At this point, the cycle is continued, but R is now decreased by a smaller amount (e.g. 1%) until NPV becomes positive. R is then increased by an amount equal to the precision required (e.g. 0.1%). When R finally becomes negative, the DCF rate of return has been found, this being the value which discounts the NPV to zero.

This procedure seems complex, but is in fact a simple way of obtaining the rate of return, which can not be obtained directly in terms of NPV. Several other iterative methods are available, for example the Newton-Raphson and False-Position procedures. The choice of method involves a trade-off between computing time and programming complexity.

Once the rate of return has been found for the first tunnel distance, the result is printed, then the tunnel distance is increased, and the cycle repeated until the maximum required tunnel distance is reached.

The use of incremental analysis results in a series of values of rate of return corresponding to different tunnel distances. This allows the rate of return for various possible tunnel distances to be obtained immediately in a practical situation. In order to obtain a general idea of the variation of rate of return with tunnel distance for each data set, a simple line-plotting subroutine was used. It is not included here as it is not strictly relevant, but immediate plotting of the results is a considerable aid in their rapid interpretation.

CHOICE OF ULTIMATE MINING DEPTH

This section, which deals with the choice of ultimate mining depth for each orebody, is intended to present the problem, and a general solution, and hopefully provoke futher research.

Deciding when to start mining a new orebody is a cyclic problem, which consists of choosing the optimum life for each successive orebody. This optimum life must be between zero years and the time required to extract the entire orebody. The production rate is assumed to be constant.

In order to present a general analysis, all orebodies are assumed to be the same distance apart, and the proposed open pits are of the same size and shape. These assumptions are made to simplify the analysis, but do not alter the logic of the derivation.

The haulage cost per ton is approximately a linear function of pit depth, for a constant haul grade. The depth of the pit is approximately a geometric function of the pit's life. This is because successive benches are of lower tonnage, and therefore, at a constant production rate, they are extracted in progressively shorter times. The exact nature of this relationship depends on the pit geometry.

A simple model will be derived below in general terms and a numerical example is worked out in Appendix II.

The following notation will be used:

H = Extra Haulage Cost per ton for hauling ore by rail from loading pocket at new orebody to existing loading pocket.

Tmax = Maximum Life of orebody (if all ore extracted)

Topt = Optimum Life of orebody

T_t = Trial Life of orebody

C1 = Capital expenditure for new loading and rail facilities.

R = Discount Rate

T = Time in years

f(T) = Cost of hauling ore from bench being worked to loading pocket

A = Haulage Cost per ton from existing loading pocket to plant

The general relationship between f(T) and T is shown below for two cycles.

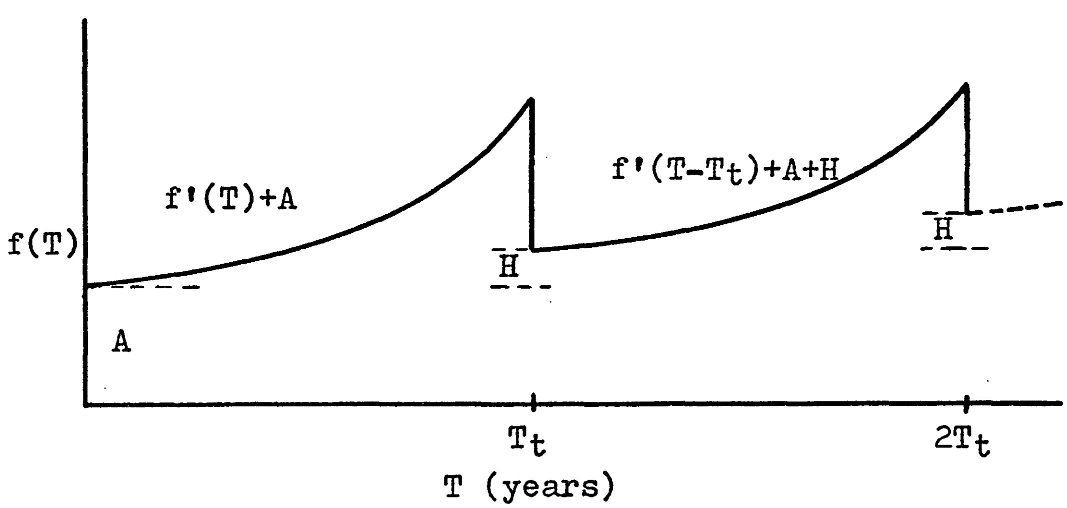


Figure 3. Haulage Cost - Time Relationship

This curve shows f(T) for a trial value of pit life, T_t. The objective is to find the optimum life, Topt, which reduces the present cost of f(T) and the investments C1 to a minimum at a given discount rate.

From the figure above, the cost function f(T) may be expressed as:

$$f(T) = \begin{cases} f'(T) + A, & 0 \leq T \leq T_t \\ f'(T-T_t) + A + H, & T_t \leq T \leq 2T_t \\ f'(T-2T_t) + A + 2H, & 2T_t \leq T \leq 3T_t \\ f'(T-3T_t) + A + 3H, & 3T_t \leq T \leq 4T_t \\ \text{etc.} & \end{cases}$$

Hence f(T) may be represented by the following sum, where n is the cycle number, and N the total number of cycles.

$$\sum_{T=1}^{T=N.T_t} f(T) = \sum_{n=1}^{N} \sum_{T=1}^{T_t} \left[f'(T) + (n-1)H + A \right]$$

This sum is of actual haulage costs; since we wish to obtain the total present cost, the capital investment (C1) made at the beginning of each cycle must be considered and discounting used.

$$P.V. \sum_{T=1}^{T=N.T_t} f(T) = \sum_{n=1}^{N} \frac{C1}{(1+R)^{(n-1)T_t}} + \sum_{T=1}^{T_t} \frac{f'(T) + A + (n-1)H + TA}{(1+R)^{T+(n-1)T_t}}$$

In order to calculate the total present cost of haulage, or P.V.$\sum$f(T), each cycle should be considered separately, within the boundary conditions applying to T. The values of TA (tax allowances), and C1 (capital investment) should only be considered in the years during which they apply.

It now remains to find Topt, which is the value of T_t which makes the total present cost a minimum. Two main approaches are available: the function could be differentiated with respect to T, and the derivative equated to zero. This would give the values of T_t corresponding to the function's maxima and minima. The minima could then be confirmed by seeing whether the second derivative of the function is positive or negative. The second approach is to represent the function by a computer program, and calculate the present value of the sum for values of T_t from zero to Tmax. The minimum value found will correspond to the optimum life of each orebody.

The latter approach is best for several reasons. It may not be possible to differentiate the cost function; if it is possible, the derivative will be quite complex. The use of a computer program allows the consideration of different, or irregular cost functions for different orebodies. Variations in the values of H and C1 may also be taken into account.

The details of the numerical example are given in Appendix II, with the necessary calculations. The total present haulage cost for 40 years' production was computed for three values of T_t (5,8, and 10 years). The maximum life of each deposit was $10\frac{1}{4}$ years.

The figure below shows the relationship between the total present cost and the trial values of orebody life. A definite minimum exists, corresponding in this particular example to a life of approximately 8 years.

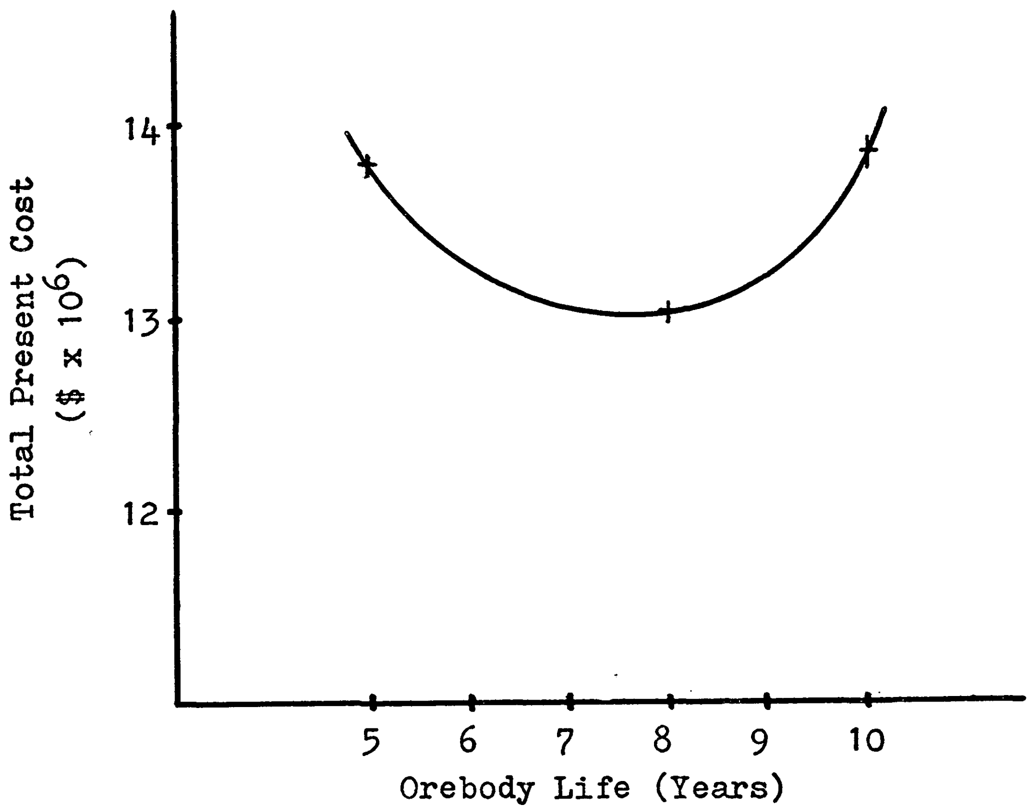


Figure 4. Total Present Cost vs. Orebody Life.

The optimum depth corresponding to the optimum orebody life may be found by referring to Figure 6. The cumulative tonnage is equal to the product of the optimum life and the production rate, and the corresponding value of depth may be read off the abscissa.

OPTIMAL LOCATION OF LOADING POCKET

In the case where the ore from several orebodies is to be hauled to one central loading pocket, and then hauled by rail to the plant, the best location for the loading pocket must be decided upon.

The objective is to locate the loading pocket so that the present cost of all future haulage costs is at a minimum. The following characteristics of each deposit will influence the optimal loading pocket location:

Production Rate : The yearly haulage cost is proportional to the tonnage hauled.

Ore Reserves : The total haulage cost for a given deposit is proportional to the total tonnage hauled, i.e. the deposit's ore reserves.

Haulage Cost : The haulage cost per ton is the product of the haulage cost per ton-mile, and the distance hauled. The capital cost of the haulage system should also be considered.

A convenient way to approach this problem is to derive a weight for each deposit, and then find the minimum of the sum of weighted distances. In other words, the sum of the product of each deposit's weight and haulage distance must be minimized. If this weight is the present cost of all future haulage costs per mile, then we are able to minimize the total present cost of haulage. The following notation is used:

M = Deposit Number (1, 2, 3, ...)

$R(M)$ = Ore Reserves of deposit (M), tons

$P(M)_i$ = Production from deposit (M) in year i, tons

$D(M)$ = Distance between deposit (M) and optimal location of loading pocket, miles

$CH(M)$ = Haulage cost from deposit (M) to loading pocket, \$/ton-mile

R = Discount Rate (cost of Capital)

$W(M)$ = Weight assigned to deposit (M)

$CC(M)$ = Capital Cost of haulage system from deposit (M) to loading pocket, \$/mile.

The notation above is also applicable for haulage between the loading pocket and the plant, 'Reserves' being replaced by total tonnage hauled, 'Production' being equivalent to the annual tonnage hauled, etc. Thus a weight may be computed to apply to the plant.

The total cost of haulage from deposit (M), in year i is:

$$TCH(M)_i = CH(M).P(M)_i.D(M)$$

The productive life of deposit (M) is:

$N(M) = R(M)/P(M)$, where $P(M)$ = average annual production. $P(M)_i$ is assumed constant $(=P(M))$

The total present cost of haulage from deposit (M) is:

$$TPCH = \sum_{i=1}^{i=n(M)} \left[CC(M)_i + \frac{CH(M).P(M)}{(1+R)^i} . D(M) \right]$$

But TPCH = W(M).D(M)

Therefore, on the assumption that all roads and the railway are built in the first year, and with the inclusion of depreciation allowances, the weight W(M) is:

$$W(M) = CC(M) - PVTA(M) + CH(M).P(M). \frac{1-(1+R)^{-N(M)}}{R}$$

where : $\frac{1-(1+R)^{-N(M)}}{R}$ is the present value of an annuity of $1.00 for N(M) years. (See introduction for derivation of this formula). PVTA(M) is the present value of the depreciation allowances (after tax) arising from the investment of CC(M). This may be found by using SUBROUTINE TAXALL given in Appendix 1.

The derivation above is simplified to illustrate the concept involved; more realistic conditions are easily dealt with by taking into account varying production from each orebody, and the timing of investments, which will seldom all take place in one year. The great advantage of this method is that the effect of these variations is automatically reflected in the weights through the use of discounting. Once these weights have been found, the optimal location of the loading pocket is easily determined by one of several well-known methods. A good account of these methods is given by Zambo (1968).

In practice, there will often be restrictions on the choice of the loading pocket site. Two approaches are available for solving this problem. Firstly, if the optimal location obtained by the method above is near a feasible site, then this could be taken as an acceptable location; secondly, a more rigorous method is available, where all possible locations and haulage routes from the nearest feasible location are considered. This method uses iterative optimization, and is quite involved where more than three orebodies must be considered. The use of fixed routes rather than assumed straight lines for haulage will be necessary in most situations. The same technique is necessary when existing roads must be used as is commonly the case in industrial distribution problems. Again Zambo (1968) gives a good description of the analytical method used in the solution of this problem.

RESULTS AND DISCUSSION

Example of Rate of Return Calculation

In order to show how the rate of return and payback may be calculated, the set of data in line 1 of Table 2 will be used, for a tunnel distance of 2 miles.

RES = 250 million tons (Ore Reserves)

ANPROD = 15 million tons (Annual Production)

N = 7 years (Depreciation Period)

T = 50% (Tax Rate)

CV1 = $0.14/ton-mile (Truck Haul Cost)

CV2 = $0.0066/ton-mile (Rail Haul Cost)

CC1 = $3.0 million (Loading Pocket Cost)

CC2 = $0.0 (Fixed Tunnel Expense)

CC3 = $7.92 million (Tunnel and Railroad Cost)

D1 = 2.0 miles (Tunnel Distance)

Sum of the years' digits depreciation will be used.

The total capital investment, C1, is:

C1 = CC1 + CC2 + (D1 x CC3)

= 3,000,000 + 0 + (2 x 7,920,000)

C1 = $18,840,000

The life of the orebody, N1 is:

N1 = RES/ANPROD

N1 = 250,000,000/15,000,000

N1 = 16.7 years.

The annual saving in haulage costs after tax is:

$$ANBEN = ANPROD \, . \, D1 \, . \, (CV1 - CV2) \, . \, (1 - T)$$

$$= 15{,}000{,}000 \times 2 \times (0.14 - 0.0066) \times (1 - 0.5)$$

$$= \$2{,}001{,}000.$$

In order to find the present value of the cost savings, PVBEN, a value of R must be chosen as a first approximation. This value should be a 'best guess' of what the actual value is expected to be. Let us choose 15%. In the derivation of the model, PVBEN was equal to:

$$PVBEN = ANBEN \, . \, \frac{1-(1+R)^{-N1}}{R}$$

$$= 2{,}001{,}000 \times \frac{1-(1.15)^{-16.7}}{0.15}$$

$$= \$12{,}046{,}020$$

The present value of tax allowances, PVTA, is:

$$PVTA = \frac{C1.T}{D} \left[\frac{N}{1+R} + \frac{N-1}{(1+R)^2} + \frac{N-2}{(1+R)^3} \ldots + \frac{1}{(1+R)^N} \right]$$

where $D = 1+2+3+4+5+6+7 = 28$

$$PVTA = \frac{18{,}840{,}000 \times 0.5}{28} \left[\frac{7}{1.15} + \frac{6}{(1.15)^2} \ldots + \frac{1}{(1.15)^7} \right]$$

$$= 336{,}430 \times 18.93$$

$$= \$5{,}768{,}620$$

Then the total net present value, when R = 15% is:

$$TNPV = - C1 + PVBEN + PVTA$$

TNPV = - 18,840,000 + 12,046,020 + 5,768,620

= -$1,025,360

Our objective is to find the value of R which reduces TNPV to zero. Our choice of 15% was too high, as the resulting TNPV was negative. Therefore a new value is taken, smaler than the first trial value. Using R = 12%, by exactly the same procedure above, we obtain:

PVBEN = 2,001,000 x 7.08

= $14,167,080

PVTA = 336,430 x 20.30

= $6,829,530

The capital investment, C1, remains the same, so:

TNPV = - 18,840,000 + 14,167,080 + 6,829,530

= $2,156,610

Since this value of TNPV is positive, it has been shown that the true value of R lies between 12 and 15 percent. Repetition of the same procedure using R = 14%, then 13%, and so on, will lead to the true value, 13.4%. This method is tedious when performed by hand, hence it is advisable to use a computer if one is available.

The calculation of the payback period is much simpler. The cost saving, and depreciation allowance for the first year is computed, and subtracted from the capital investment. The calculation is repeated for each successive year, each time subtracting the cost saving and depreciation allowance from the previous balance. When the balance becomes negative, the payback period lies between that year and the previous one, the fraction being found by interpolation.

Results

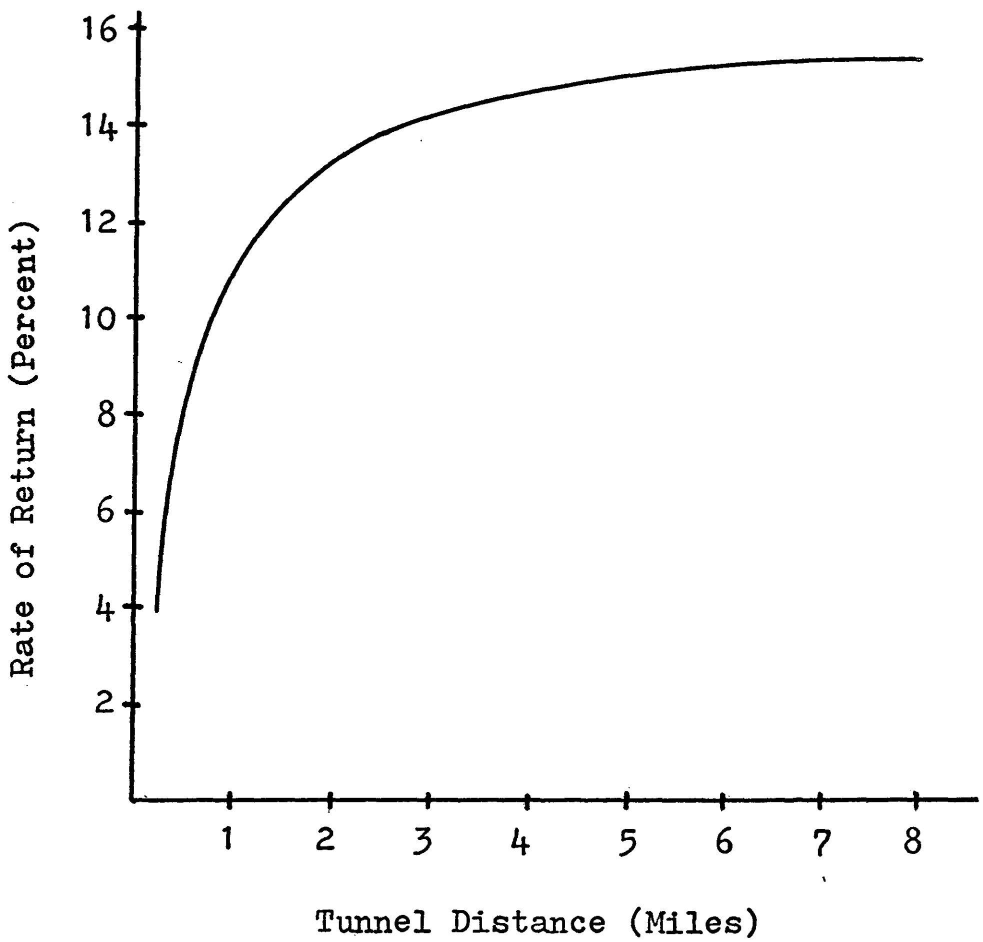
The mathematical model was tested with a set of typical data, and then each major input factor (ore reserves, annual production, depreciation period, and tax rate) was varied individually.

A table giving the rate of return (R) against tunnel distance (D1) was obtained for each set of variables. The amount of the capital investment and the present value of the tax allowances are also shown. A typical computer output is given in Table 1.

The results all show the same general relationship between tunnel distance and rate of return. Initially, at low values of D1, R increases rapidly, but then tends towards an assymptotic value, as shown in Figure 5. This graph portrays the relationship for the standard data below, which were used in the example of rate of return calculation above.

Ore Reserves	=	250 million tons
Annual Production	=	15 million tons
Depreciation Period	=	7 years
Tax Rate	=	50 percent

Figure 5. Relationship between Rate of Return and Tunnel Distance.



	ORE RESERVES	ANNUAL PRODN.	DEPRE- PERIOD	TAX RATE	D.C.F. YEILD WHEN D1 = 1 MILE	2 MILES	6 MILES	PAYBACK D1=2 MLS.
	MILLIONS OF TONS		YEARS	%	%	%	%	YEARS
1	250	15	7	50	11.3	13.4	15.1	5.2
2	500	15	7	50	13.1	15.0	16.5	5.2
3	750	15	7	50	13.3	15.1	16.6	5.2
4	1000	15	7	50	13.3	15.1	16.6	5.2
5	250	10	7	50	8.4	9.9	11.1	7.1
1	250	15	7	50	11.3	13.4	15.1	5.2
6	250	20	7	50	13.9	16.6	18.8	4.2
7	250	15	5	50	11.7	13.9	15.7	4.8
1	250	15	7	50	11.3	13.4	15.1	5.2
8	250	15	10	50	10.7	12.7	14.4	5.7
1	250	15	7	50	11.3	13.4	15.1	5.2
9	250	15	7	60	9.8	11.6	13.1	5.3
10	250	15	7	70	8.2	9.7	10.9	5.5

Table 2. Summary of Results

A complete summary of the results is given in Table 2 with rate of return values corresponding to tunnel distances of 1, 2, and 6 miles, and with the payback period for a 2-mile tunnel.

Discussion

The general form of the relationship between the tunnel distance, D1, and the rate of return, R, may be explained simply when it is remembered that the capital investment is composed of a fixed part, CC1, the loading pocket cost, and a variable part, CC3, the tunnel and railroad cost, which is proportional to D1. Thus when D1 is small, the cost savings are low, but the fixed part, CC1 is still invested, together with a small variable part, CC3; consequently the rate of return is low. As D1 is increased, the cost savings also increase, and CC1 is a smaller proportion of the total invested, so R increases. Finally, for large values of D1, the cost of the loading pocket becomes insignificant when compared with the tunnel cost, and the rate of return is near its assymptotic maximum value.

The practical conclusion to be drawn from this relationship is that the rate of return is very sensitive to changes in D1 when D1 is small, so this distance should be determined as precisely as practicable when a specific situation is being evaluated.

The summary of results emphasizes very clearly that factors which influence the cashflows immediately after the investment are the most important. The increase in rate of return brought about by doubling production rate is over 75% of the lower value. However, the difference in rate of return for ore reserves of 500, and 1000 million tons is about 0.1%. These results are to be expected, as discounting is used, but it is well to emphasize their importance. Note that changes in reserves have no effect on the payback period (after a certain minimum value).

Changes in the depreciation period have a lesser, nevertheless significant effect. Variation in the tax rate has a considerable effect upon the rate of return, but a lesser effect upon the payback period.

CONCLUSIONS

The most important factors influencing the decision whether or not a new loading pocket should be built are the annual ore production rate, and the distance between the proposed loading pocket and the existing one, particularly when this distance is small. The amount of the capital investment, and the relative costs of truck and rail haulage are also of obvious importance.

The mathematical model and computer program provide a method of rapidly evaluating a proposed haulage system, which can be readily adapted to a particular situation.

The general model used to find the optimum life, and hence ultimate depth of each of a series of orebodies provides a preliminary method of solving this problem. Expansion of the model to deal with more complex situations is easily achieved.

The choice of loading pocket location when more than one orebody is involved is shown to be easily solved, the method being applicable to many other situations. The author has not encountered any published material using discounting in this context, yet the use of this technique would appear essential in arriving at a correct solution, particularly in mining situations.

RECOMMENDATIONS

It is suggested that further work be done on other aspects of ore transportation, leading to detailed mathematical models of entire haulage systems, and eventually to the representation of whole mining operations. This is the only way in which available data can be put to maximum use in the long range planning of operations.

This would involve the formulation of mathematical functions for the economic factors considered. For example, the availability of cost functions for truck and rail haulage would have been very useful in this study, and would have obviated a number of assumptions. These particular functions could be approached by regression analysis, or by model building.

The general approach to the problem of finding the optimum ultimate mining depth of orebodies could be expanded. By using a computer program, real situations could be dealt with accurately, if different values of ultimate depth were considered for each orebody.

The three main questions dealt with are all interdependent; the models given in this study could be combined to give a more general solution.

With regard to the use of the results of this study, it is emphasized that the objective has been to describe methods of approaching haulage problems, and that the quantitative results only serve as illustrations. In order to use these methods, a more detailed cost breakdown is required, and the assumptions stated should be carefully considered.

APPENDIX I

COMPUTER PROGRAM

```
C
C     LOADING POCKET EVALUATION PROGRAM
C
C     ***************************************************************************
C                                                                               *
C       RES   = ORE RESERVES (TONS)                                             *
C     ANPROD  = ANNUAL PRODUCTION (TONS)                                        *
C         N   = DEPRECIABLE LIFE OF ASSET                                       *
C         T   = TAX RATE                                                        *
C       CV1   = TRUCK HAULAGE COST ($/TON-MILE)                                 *
C       CV2   = RAIL HAULAGE COST ($/TON-MILE)                                  *
C       CC1   = COST OF LOADING POCKET ($)                                      *
C       CC2   = SET-UP EXPENSE FOR BUILDING TUNNEL ($)                          *
C       CC3   = TUNNEL AND RAILROAD COST ($/MILE)                               *
C        C1   = TOTAL CAPITAL INVESTMENT                                        *
C        D1   = LENGTH OF PROPOSED TUNNEL                                       *
C      ANBEN  = ANNUAL COST SAVING THROUGH BUILDING NEW LOADING POCKET ($)      *
C         R   = DISCOUNTED CASHFLOW RATE OF RETURN                              *
C       PVTA  = PRESENT VALUE OF TAX ALLOWANCES                                 *
C       DINC  = INTERVAL BETWEEN VALUES OF D1 REQUIRED                          *
C      DLAST  = MAXIMUM VALUE OF D1 REQUIRED                                    *
C                                                                               *
C     ***************************************************************************
C
      DIMENSION RNEW(300)
C
C     INPUT DATA (INUM = DATA SET IDENTIFICATION)
    1 READ 10,INUM,RES,ANPROD,N,T
   10 FORMAT(I10,F10.0,F10.0,I10,F10.0)
      DATA CV1,CV2/0.14,0.0066/
      DATA CC1,CC2,CC3/3.0E+6,0.,7.92E+6/
      DATA DINC,DLAST/0.25,10.0/
C
C     PRINT IDENTIFICATION, HEADINGS
      M = (DLAST/DINC)
      IRES = (RES/1.E+6)
```

```
      IANP = (ANPROD/1.E+6)
      IT = 100.*T
      PRINT 34,INUM
      PRINT 35,IRES,IANP,N,IT
      PRINT 40
C
C     COMPUTE RATE OF RETURN FOR EACH VALUE OF D1
   15 D1 = 0.
      K1 = 1
      N1 = (RES/ANPROD)
      DO 100 J=1,M
      D1 = D1 +DINC
   16 R = -.31
   17 R = R + .10
      L =-1
      GO TO 20
   18 R = R - .01
      L = 0
      GO TO 20
   19 R = R + 0.001
      L = 1
   20 C1 = CC1+CC2+CC3*D1
      CALL TAXALL ( 1.,C1,N,T,R,0.,0.,0.,0.,1.,PVTA)
      ANBEN = ANPROD*D1*(CV1-CV2)*(1.-T)
      PVBEN = ANBEN*(1.-(1./(1.+R)**N1))/R
      TNPV = -C1+PVTA+PVBEN
      IF(TNPV)22,45,23
   22 IF(L)18,18,45
   23 IF(L)17,19,19
C     PRINT RESULTS
   45 PRINT 50,D1,C1,PVTA,R
   90 CONTINUE
      RNEW(J) = R
      PRINT 65
   65 FORMAT(1H )
      IF(J-20*K1)100,75,100
```

```
 75 PRINT 80
 80 FORMAT(1H1)
    K1 = K1 + 1
100 CONTINUE
    CALL PLTT 1(RNEW,M)
    GO TO 1
 34 FORMAT(1H1,/)
 35 FORMAT(1H0,21X,14H    RESERVES =,I4,34H MILLION TONS, ANNUAL PRODU
   1CTION =,I3,13H MILLION TONS,/,26X,32HDEPRECIABLE LIFE OF INVESTMEN
   2T =,I3,18H YEARS, TAX RATE =,I3,8H PERCENT,//)
 40 FORMAT(1H0,29X,10HD1 (MILES),10X,2HC1,13X,4HPVTA,13X,1HR,//)
 50 FORMAT(1H ,30X,F6.2,7X,F12.1,4X,F12.1,7X,F8.3)
999 CONTINUE
    STOP
    END
```

```
      SUBROUTINE TAXALL (DEP,C1,N,T,R,D,V,F,S,B,PVTA)
C
C     COMPUTES THE PRESENT VALUE OF TAX ALLOWANCES ARISING FROM
C     A CAPITAL INVESTMENT
C
C     *************************  CALLING SEQUENCE  ***************************
C                                                                            *
C                       CALL TAXALL (DEP,C1,N,T,R,D,V,F,S,B,PVTA)            *
C                                                                            *
C     WHERE                                                                  *
C                                                                            *
C     DEP  = DEPRECIATION METHOD                                             *
C          =-1, STRAIGHT LINE                                                *
C          = 0, (DOUBLE) DECLINING BALANCE                                   *
C          = 1, SUM OF THE YEARS DIGITS                                      *
C     IFLG = PRINT FLAG                                                      *
C          =-1, DO NOT PRINT                                                 *
C          = 0, PRINT PVTA                                                   *
C          = 1, PRINT PVTA AND C1                                            *
C      C1  = INITIAL CAPITAL COST                                            *
C       N  = DEPRECIABLE LIFE OF ASSET                                       *
C       T  = PERCENTAGE TAX RATE                                             *
C       R  = COST OF CAPITAL TO THE FIRM                                     *
C       D  = PERCENTAGE RATE OF DEPRECIATION (DECLINING BALANCE METHOD)      *
C       V  = PERCENTAGE INVESTMENT ALLOWANCE                                 *
C       F  = PERCENTAGE INITIAL ALLOWANCE                                    *
C       S  = RESALE VALUE AT END OF YEAR N                                   *
C       B  = DELAY BETWEEN INCURRING AND PAYING TAXES                        *
C     PVTA = PRESENT VALUE OF TAX ALLOWANCES                                 *
C                                                                            *
C     ************************************************************************
C
      DIMENSION BNUM (50)                                              TAX
  450 IFLG = -1                                                        TAX
      IF(DEP)500,600,700                                               TAX
C                                                                      TAX
```

```
C      **************************                                        TAX
C      STRAIGHT LINE DEPRECIATION                                        TAX
C      G = PERCENTAGE RATE OF DEPRECIATION (STRAIGHT LINE METHOD)        TAX
  500 G = 1./FLOAT(N)                                                    TAX
      Z = (1.-F-G)/G                                                     TAX
      IF(Z-FLOAT(N))510,510,520                                          TAX
  510 AN = Z                                                             TAX
      GO TO 530                                                          TAX
  520 AN=FLOAT(N)                                                        TAX
C     W  = WRITTEN DOWN VALUE AT END OF YEAR N                           TAX
  530 W = C1*(1.-F-(AN*G))                                               TAX
      PVTA = (T*C1/(1.+R)**B)*((V+F+G)+(G*(1.-(1.+R)**AN)/R)+(W-S)/C1*(1 TAX
     1.+R)**N)                                                           TAX
      GO TO 800                                                          TAX
C                                                                        TAX
C     ***************************************                            TAX
C     (DOUBLE) DECLINING BALANCE DEPRECIATION                            TAX
  600 Q=(R+D)/(1.-D)                                                     TAX
      U=(1.-F-D)                                                         TAX
      PVTA=(T*C1/(1.+R)**B)*((V+F+D)+(D*U/(1.-D))*(((1.-(1./(1.+Q)**N))/ TAX
     1Q)+((1./(1.+Q))**N))-S/(C1*(1.+R)**N))                             TAX
      GO TO 800                                                          TAX
C                                                                        TAX
C     ************************************                               TAX
C     SUM OF THE YEARS DIGITS DEPRECIATION                               TAX
  700 DENOM = 0.                                                         TAX
      DO 710 I = 1,N                                                     TAX
      AN=FLOAT(N)                                                        TAX
      AI=FLOAT(I)                                                        TAX
      BNUM(I) = AN-AI+1.                                                 TAX
  710 DENOM = DENOM+AI                                                   TAX
      I = 0                                                              TAX
      DCF = 0.                                                           TAX
      DO 720 J=1,N                                                       TAX
      I = I+1                                                            TAX
      DCF = DCF + ((BNUM(I)/DENOM)/(1.+R)**J)                            TAX
```

```
  720 CONTINUE                                                    TAX
      PVTA =(T*C1/(1.+R)**B)*(DCF-(S/C1*(1.+R)**N))               TAX
C                                                                 TAX
C     PRINT RESULTS
  800 IF(IFLG)820,815,805                                         TAX
  805 PRINT 810,PVTA,C1                                           TAX
  810 FORMAT (1H ,10X,8HPVTA = $,F15.1,5X,6HC1 = $,F18.1)         TAX
      GO TO 820                                                   TAX
  815 PRINT 816,PVTA                                              TAX
  816 FORMAT(1H ,10X,8HPVTA = $,F12.1)                            TAX
  820 CONTINUE                                                    TAX
      RETURN                                                      TAX
      END                                                         TAX
```

APPENDIX II

NUMERICAL EXAMPLE OF CHOICE OF ULTIMATE MINING DEPTH

APPENDIX II

Numerical Example of Choice of Ultimate Mining Depth

A number of identical, separate orebodies are an equal distance from each other. A railroad and loading pocket system is used to haul ore to the mill. When mining starts at a new orebody, the railroad is extended, and a new loading pocket built. In this example this cyclic investment, C1, is taken as $5 million. The orebodies will be mined using conical open pits, with a pit slope of 45^{o}. The radius (R) of the top of the cone is 1,000 feet, and the bench height, B, is 50 feet. The haul grade is 7%, and the haulage cost is $0.14 per ton-mile. Then for a bench D feet below the edge of the pit,

$$\text{Haul distance} = \frac{\text{Depth}}{\text{Haul Grade}} = \frac{D}{0.07}$$

$$\text{Haul Cost/ton} = \text{Haul Distance x Cost/ton-mile} = \frac{D}{0.07} \times \frac{0.14}{5280}$$

If benches are numbered (N= 1, 2, 3...etc) from surface downwards, then the volume of bench N, which is cylindrical, is:

$$\text{Vol (N)} = \pi \,.\, (R-N.B)^{2}.B$$

Using this relationship, and taking the ore density to be 0.1 tons/ft^3, the following table may be obtained.

BENCH	TONNAGE	CUMULATIVE TONNAGE
1	15.7	15.7
2	14.2	29.9
3	12.7	42.6
4	11.4	54.0
5	10.0	64.0
6	8.8	72.8
7	7.7	80.5
8	6.6	87.1
9	5.6	92.7
10	4.8	97.5
11	3.9	101.4
12	3.2	104.6
13	2.5	107.1
14	1.9	109.0
15	1.4	110.4
16	1.0	111.4
17	0.6	112.0
18	0.3	112.3
19	0.1	112.4
20	0.1	112.5

Table 3. Bench Tonnages (Tons x 10^6)

The cumulative tonnage may be plotted against bench number, or depth, to give Figure 6. If each year's production (10 million tons) is marked off on the Tonnage axis, then the average mining depth for each year may be found by taking the corresponding values on the Depth axis.

Since the haulage cost is proportional to depth, the average haul cost for each year may be found, as shown in Table 4.

The total haul cost for each year is then found by multiplying the average cost per ton by the annual production,

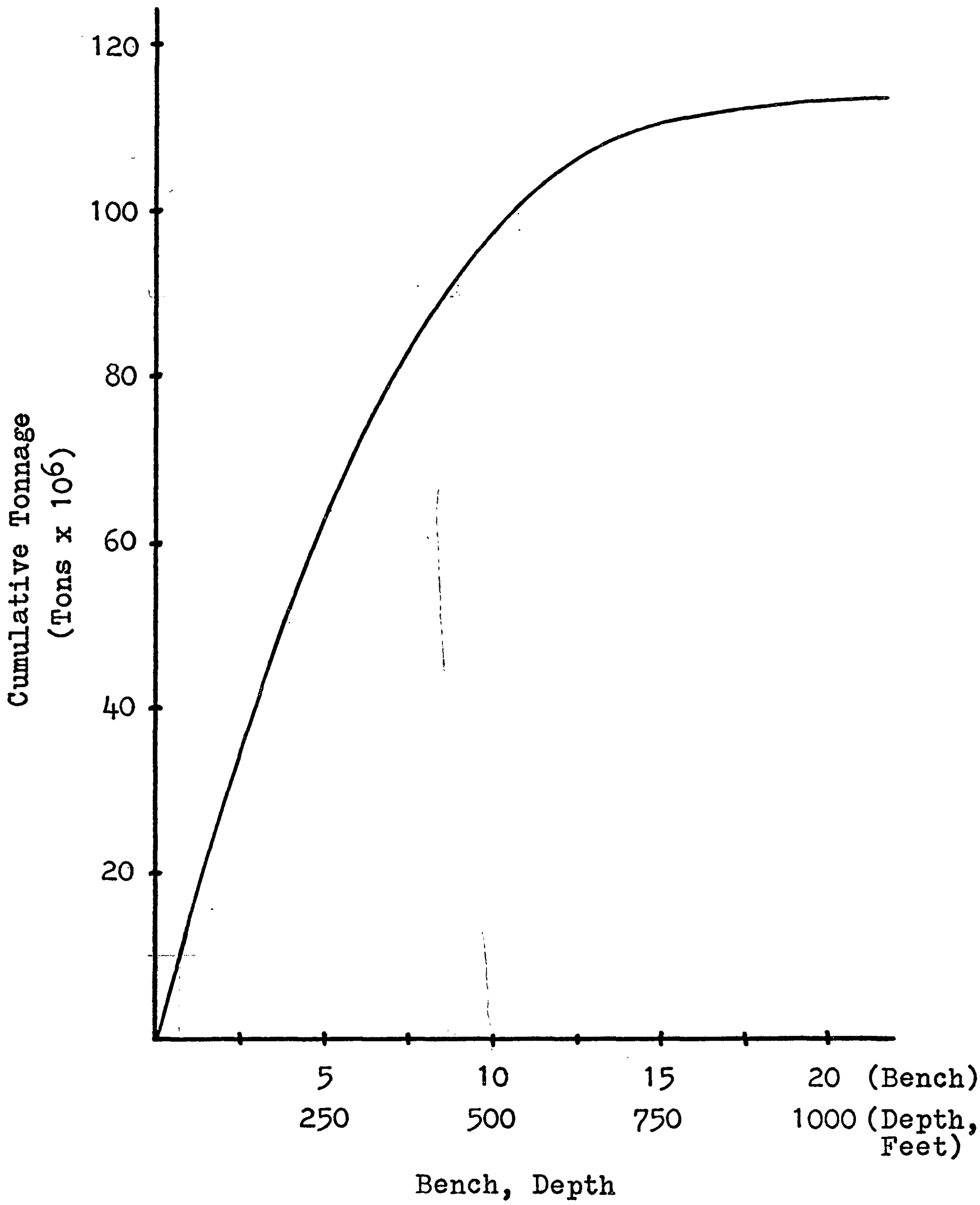


Figure 6. Cumulative Tonnage vs. Bench Number and Depth

and adding H, the annual rail haulage cost, which increases by 1¢/ton for each successive cycle.

YEAR	AVERAGE MINING DEPTH (Feet)	AVERAGE HAUL COST ($/Ton)
1	48	0.018
2	85	0.032
3	124	0.047
4	166	0.062
5	212	0.080
6	264	0.100
7	324	0.122
8	395	0.149
9	480	0.181
10	605	0.228
11	850	0.320

Table 4. Average Yearly Haul Costs

This is how the third columns of Tables 5, 6, and 7 were obtained. The capital outlay of $5 million is made at the beginning of each cycle. In these calculation sheets, cycles are refered to as A, B, C, etc., each corresponding to a different orebody. The tax allowances are computed using sum of the years' digits depreciation (7 year write-off), and a 50% tax rate. The net cashflow is then multiplied by the appropriate discount factor to obtain the net discounted cashflow. The sum of the discounted cashflows is the total net present cost of haulage for the 40 years' production considered.

The procedure was repeated using trial orebody lives (T_t) of 5, 8, and 10 years. Since 40 is an exact multiple of the values of T_t used, the present costs found are strictly comparable.

The cost curve for the three trial values is given in figure 4, the minimum cost corresponding to orebody lives of approximately 8 years.

ORE BODY	YEAR	TOTAL HAUL COST	CAPITAL OUTLAYS & TAX ALLOWANCES	NET CASH FLOW	DISCOUNT FACTOR (R=10%)	DISCOUNTED CASH FLOW
A	0		-5,000	-5,000	1.000	-5,000
A	1	-180	625	445	.909	405
A	2	-320	536	216	.826	197
A	3	-470	446	-24	.751	-18
A	4	-620	357	-263	.683	-180
A	5	-800	-4,732	-5,532	.621	-3,435
B	6	-280	804	524	.564	296
B	7	-420	625	205	.513	105
B	8	-570	446	-124	.467	-58
B	9	-720	357	-363	.424	-154
B	10	-900	-4,732	-5,632	.386	-2,174
C	11	-380	804	424	.350	148
C	12	-520	625	105	.319	33
C	13	-670	446	-224	.290	-65
C	14	-820	357	-463	.263	-122
C	15	-1,000	-4,732	-5,732	.239	-1,370
D	16	-480	804	324	.218	71
D	17	-620	625	5	.198	1
D	18	-770	446	-324	.180	-58
D	19	-920	357	-563	.164	-92
D	20	-1,100	-4,732	-5,832	.149	-869
E	21	-580	804	224	.135	30
E	22	-720	625	-95	.123	-12
E	23	-870	446	-424	.112	-48
E	24	-1,020	357	-663	.102	-68
E	25	-1,200	-4,732	-5,932	.092	-546
F	26	-680	804	124	.084	10
F	27	-820	625	-195	.076	-15
F	28	-970	446	-524	.069	-36
F	29	-1,120	357	-763	.063	-48
F	30	-1,300	-4,732	-6,032	.057	-344
G	31	-780	804	24	.052	1
G	32	-920	625	-295	.047	-14
G	33	-1,070	446	-624	.043	-27
G	34	-1,220	357	-863	.039	-34
G	35	-1,400	-4,732	-6,132	.036	-221
H	36	-880	804	-76	.032	-2
H	37	-1,020	625	-395	.029	-11
H	38	-1,170	446	-724	.027	-20
H	39	-1,320	357	-963	.024	-23
H	40	-1,500	536	-964	.022	-21
						-13,788

Table 5. Calculation of Total Net Present Cost when Tt = 5 years. ($1000's)

ORE BODY	YEAR	TOTAL HAUL COST	CAPITAL OUTLAYS & TAX ALLOWANCES	NET CASH FLOW	DISCOUNT FACTOR (R=10%)	DISCOUNTED CASH FLOW
A	0		-5,000	-5,000	1.000	-5,000
A	1	-180	625	445	.909	405
A	2	-320	536	216	.826	178
A	3	-470	446	-24	.751	-18
A	4	-620	357	-263	.683	-180
A	5	-800	268	-532	.621	-330
A	6	-1,000	179	-821	.564	-463
A	7	-1,220	89	-1,131	.513	-580
A	8	-1,490	-5,000	-6,490	.467	-3,030
B	9	-280	625	345	.424	146
B	10	-420	536	116	.386	45
B	11	-570	446	-124	.350	-43
B	12	-720	357	-363	.319	-116
B	13	-900	268	-632	.290	-183
B	14	-1,100	179	-921	.263	-242
B	15	-1,320	89	-1,231	.239	-300
B	16	-1,590	-5,000	-6,590	.218	-1,437
C	17	-380	625	245	.198	48
C	18	-520	536	16	.180	3
C	19	-670	446	-224	.164	-37
C	20	-820	357	-463	.149	-69
C	21	-1,000	268	-732	.135	-99
C	22	-1,200	179	-1,021	.123	-126
C	23	-1,420	89	-1,331	.112	-149
C	24	-1,690	-5,000	-6,690	.102	-682
D	25	-480	625	145	.092	13
D	26	-620	536	-84	.084	-7
D	27	-770	446	-324	.076	-25
D	28	-920	357	-563	.069	-39
D	29	-1,100	268	-832	.063	-52
D	30	-1,300	179	-1,121	.057	-64
D	31	-1,520	89	-1,431	.052	-74
D	32	-1,790	-5,000	-6,790	.047	-319
E	33	-580	625	45	.043	2
E	34	-720	536	-184	.039	-7
E	35	-870	446	-424	.036	-15
E	36	-1,020	357	-663	.032	-21
E	37	-1,200	268	-932	.029	-28
E	38	-1,400	179	-1,221	.027	-33
E	39	-1,620	89	-1,531	.024	-37
E	40	-1,890		-1,890	.022	-42
						-13,007

Table 6. Calculation of Total Net Present Cost when T_t = 8 years. ($1000's)

ORE BODY	YEAR	TOTAL HAUL COST	CAPITAL OUTLAYS & TAX ALLO-WANCES	NET CASH FLOW	DISCOUNT FACTOR (R=10%)	DIS-COUNTED CASH FLOW
A	0		-5,000	-5,000	1.000	-5,000
A	1	-180	625	445	.909	405
A	2	-320	536	216	.826	178
A	3	-470	446	-24	.751	-18
A	4	-620	357	-263	.683	-180
A	5	-800	268	-532	.621	-330
A	6	-1,000	179	-821	.564	-463
A	7	-1,220	89	-1,131	.513	-580
A	8	-1,490		-1,490	.467	-706
A	9	-1,810		-1,810	.424	-767
A	10	-2,228	-5,000	-7,228	.386	-2,790
B	11	-280	625	345	.350	121
B	12	-420	536	116	.319	37
B	13	-570	446	-124	.290	-36
B	14	-720	357	-363	.263	-95
B	15	-900	268	-632	.239	-151
B	16	-1,100	179	-921	.218	-201
B	17	-1.320	89	-1,231	.198	-244
B	18	-1,590		-1,590	.180	-286
B	19	-1,910		-1,910	.164	-313
B	20	-2,328	-5,000	-7,328	.149	-1,092
C	21	-380	625	245	.135	33
C	22	-520	536	16	.123	2
C	23	-670	446	-224	.112	-25
C	24	-820	357	-463	.102	-47
C	25	-1,000	268	-732	.092	-67
C	26	-1.200	179	-1,021	.084	-86
C	27	-1.420	89	-1,331	.076	-101
C	28	-1,690		-1,690	.069	-117
C	29	-2,010		-2,010	.063	-127
C	30	-2,428	-5,000	-7,428	.057	-423
D	31	-480	625	145	.052	8
D	32	-620	536	-84	.047	-4
D	33	-770	446	-324	.043	-14
D	34	-920	357	-563	.039	-22
D	35	-1.100	268	-832	.036	-30
D	36	-1,300	179	-1,121	.032	-36
D	37	-1,520	89	-1,431	.029	-41
D	38	-1,790		-1,790	.027	-48
D	39	-2,110		-2,110	.024	-51
D	40	-2,528		-2,528	.022	-56
						-13,763

Table 7. Calculation of Total Net Present Cost when T_t = 10 years. ($1000's)

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