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ESTIMATION OF THE MINIMUM RANGE OF ECONOMICALLY EXPLORABLE SANDSTONE-TYPE URANIUM DEPOSITS

by

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ABSTRACT

Much valuable exploration time and funds are often wasted on exhaustive examination of noneconomic mineral deposits. If exploration teams were provided with some guidelines as to the required specifications for deposits to be viable, those deposits which obviously cannot meet the specifications would be left after a minimum amount of examination.

In this paper a method is developed for estimating the specifications of the sandstone-type uranium deposits which will be the minimum that can meet a company's economic requirements for investment. The method is based on Net Present Value analysis. From available cost data and deposit specifications for typical mines, the method involves calculating the uranium oxide selling price necessary for a mining project based on the deposit to achieve the company's required discounted cash flow rate of return on investment. The deposit is assumed to have been drilled to only a minor extent during exploration drilling. The range of deposits achieving the company's required discounted cash flow rate of return on investment at the selected selling price is then graphed. The graphs provide the complete range of deposits with the minimum required specifications. A FORTRAN

computer program capable of accepting a wide variety of inputs was written by the author to perform the necessary calculations.

A major portion of the paper is devoted to discussion of the variations of costs with the depth and thickness of uranium ore deposits. A large amount of unpublished data was collected for this purpose. From this discussion is developed the use of an ore depth to thickness ratio for simplifying the method.

In developing this method the author assumed that the deposits would be mined by either standard open pit or room and pillar underground techniques. In-situ leaching is briefly discussed and some comparative data presented. However, a lack of comprehensive cost data for this latter technique prevented the method from being expanded to include it.

The accuracy of the results derived from the method is limited to a major extent by the accuracy of available cost data. The author therefore believes that the best procedure for application will be to calculate upper and lower limits for the minimum range of economic deposits. Deposits fall-below the lower limit would be left immediately, those above the upper limit would be almost certain to prove viable, while the marginal deposits between the limits would warrant further prudent examination and evaluation.

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ACKNOWLEDGEMENTS

It is necessary at this time to acknowledge the assistance which I received from a large number of sources in the writing of this paper. All people who provide assistance are greatfully thanked. However, the number of people involved is too large for all to be acknowledged individually. Also, many important sources of assistance cannot be acknowledged derectly in order to preserve confidentiality.

My advisor, Dr. Petrick provided many helpful suggestions and much encouragement throughout the writing of this paper. The other members of my thesis committee, Dr. Berry and Dr. Mather, also provided advice when needed.

Thr most important assistance was furnished by Mr. John Klemenic, Director of the Supply Analysis Division, Department of Energy, Grand Junction Office, during many long conversations. Therefore, he is especially thanked. His colleagues, Mr. J. Fred Facer, Jr., Mr. Alfredo Gurmendi and Mr. Frederick C.J. Lu also provided much valuable assistance.

Companies which allowed me to tour their uranium mining operations were Union Carbide Corp., Kerr-McGee Corp., Rocky Mountain Energy Co., Cotter Corp. and Exxon Co. Viewing these operations and conversing directly with the company staff members vastly improved my understanding of uranium mining operations.

Many staff members of both Wyoming Minerals Corp. and Western Nuclear Inc. also provided important information about their specialities. Dr. Douglas J. Lootens, Associate, Dames & Moore, is thanked for discussing with me his published work.

Those who deserve true loving thanks though, are my parents, without whose support this paper may not have been written.

INTRODUCTION

For a mineral exploration program to have the best chance of discovering an ore body, a high rate of turnover of mineral prospects must be achieved, while maintaining an adequate level of examination of each prospect. It has been the author's personal experience that in reality such efficiency is rarely if ever achieved.

Exploration programs often get bogged down in continuing the examination of prospects which should have been left long before. Rarely does the field team, or even the exploration management in the head office, have much concept of what specifications would be required for an economic deposit of the mineral being explored for. Therefore, the exploration team is left to analyse prospects while not knowing what they are required to find.

The field geologist generally resolves this dilemma by perceiving his duty as being to find and completely examine any occurrences of the desired mineral. The end result is that much valuable time and effort is wasted in almost every facet of the exploration program. Drilling sometimes continues on a deposit until a head office analysis shows that the deposit is more than an order of magnitude below the minimum requirements.

To aid in returning some initiative to the field geologist, the author chose to devote this paper to deriving a method of estimating the required specifications of the minimum range of economically explorable deposits for one mineral commodity. The mineral commodity chosen was uranium because of the recent boom in exploration for it. Sandstone-type deposits were chosen because these are the most common mode of occurrence of uranium within the United States.

John Klemenic (Director, Supply Analysis Division, Department of Energy, Grand Junction Office, Colorado) has already presented important work in this field. In February 1972 he presented a paper showing a relationship between the mine primary development cost* and ore depth to thickness ratio(1). In October of both 1972 and 1974 he presented papers analysing the economics of mining uranium from deposits with ore depth to thickness ratios of 24 for the open pit case and 76 for the underground case(2,3). The author, in developing the method presented in this paper, has expanded on Klemenic's work so that the method is suitable for a wide range of ore depth to thickness ratios. However, the method has only been developed for conventional mining techniques.

An example set of calculations has been performed by the author based on Klemenic's cost and mill recovery

*Throughout this paper 'cost' is often used in place of the accounting term 'expenditure'.

estimates. These estimates were the most comprehensive and reliable available.

In the first section, the author describes the geology of sandstone-type uranium deposits and the exploration procedures applied to find them. In the second section, each of the stages involved in typical open pit and underground mining projects are described, together with an analysis of available cost data. The author found it necessary to collect a large amount of unpublished cost data for this analysis. The use of the ore depth to thickness ratio as a basis for cost variations is derived by comparing the variation of costs with both depth and thickness changes. In the third section the in-situ leach process for mining uranium is also discussed for comparative purposes only. The necessary calculations are then explained in the fourth section and illustrated by the application of the author's estimates and assumptions. After an explanation of how the author envisaged that the results would be best applied, a set of conclusions is drawn. Some results of the author's calculations are presented in graphical form in Appendix D to illustrate the range of graphs which can be plotted.

A FORTRAN computer program was written by the author to perform the necessary calculations. It therefore represents a major portion of the method developed in this paper.

For input data on a given tonnage and grade of deposit, the program will generate the uranium oxide selling price necessary for specified deposit depth to thickness ratios in order to yield various discounted cash flow rates of return on investment. The program is capable of accepting a wide variety of input variables. Within the program all important steps have been described by the generous use of Comment statements. A User's Guide is included in Appendix B, followed by the program in Appendix C.

GEOLOGY OF SANDSTONE-TYPE URANIUM DEPOSITS

The sandstone beds containing uranium deposits were deposited among ancient stream channels, generally flood plains, or along ancient sandy beaches and offshore bars. Mudstones are usually interlayered with these sandstones. In the United States, the geological age of these beds can range from the Devonian to Upper Tertiary. In these beds uranium deposits chiefly occur as two types, either "roll-front" bodies or tabular bodies.

Roll-front deposits are elongate bodies, crescent shaped in cross-section and typically ten to fifty feet in height (Figures 1 and 2). They occur intermittently along the often miles-long interface between altered (oxidized) and unaltered (reduced) sandstone. Oxidizing ground waters moving downdip are thought to have carried uranium or remobilized that already contained in the sandstones. The uranium minerals then precipitated when the waters encountered reducing conditions. The concave edge of the deposit therefore faces updip and occurs at the downdip edge of the oxidized zone of the sandstone bed. Roll-front deposits occur chiefly in Tertiary and Cretaceous beds in Wyoming, South Dakota and south Texas.

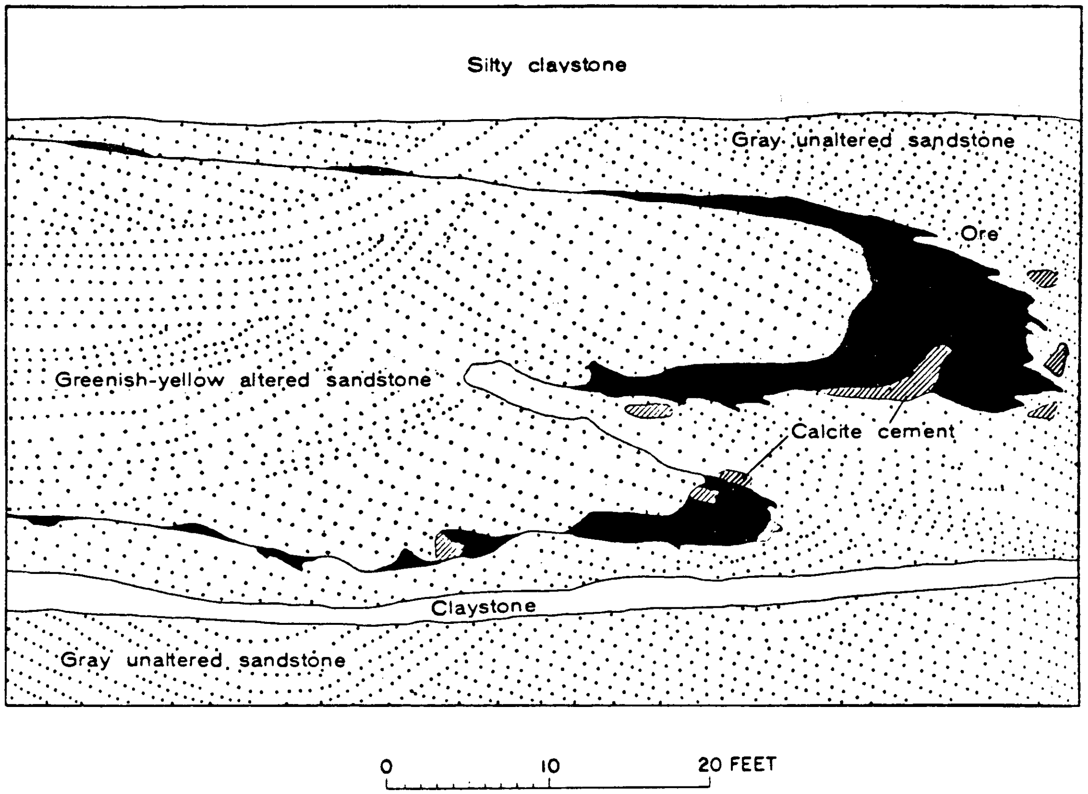


FIGURE 1 - CROSS-SECTION NORMAL TO THE STRIKE OF AN ALTERED SANDSTONE TONGUE

Source: Harshman[4]

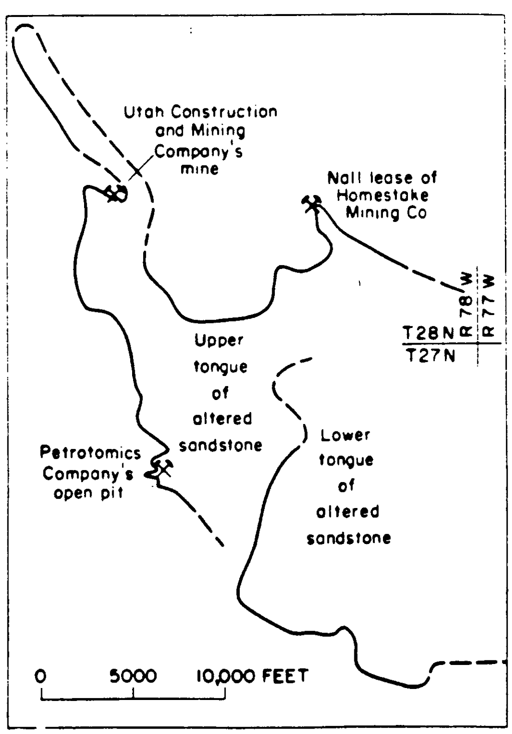


FIGURE 2 - PLAN MAP SHOWING OUTLINE OF UPPER AND LOWER ALTERED SANDSTONE TONGUES, SHIRLEY BASIN, WYOMING

Source: Harshman[4]

The tabular bodies are discrete masses enveloped in reduced sandstone which are in turn scattered through oxidized sandstones (Figure 3). There are two commonly held theories as to the origin of these deposits. The first is that the uranium was originally deposited along with the sandstone when reducing conditions prevailed, with later remobilization being important only in detail. The second theory is that oxidizing solutions containing dissolved uranium are believed to have precipitated uranium minerals on encountering pockets of sandstone rich in reducing organic matter.

Tabular deposits occur mainly in Jurassic and Triassic beds in Colorado, Utah, Arizona and New Mexico. In the Uravan Mineral Belt of western Colorado, the tabular uranium ore bodies contain vanadium concentrations of economic importance averaging in the order of 1 percent vanadium pentoxide (V_2O_5). These uranium deposits concentrate in the sands at the downstream end of the point bars of ancient stream channels.

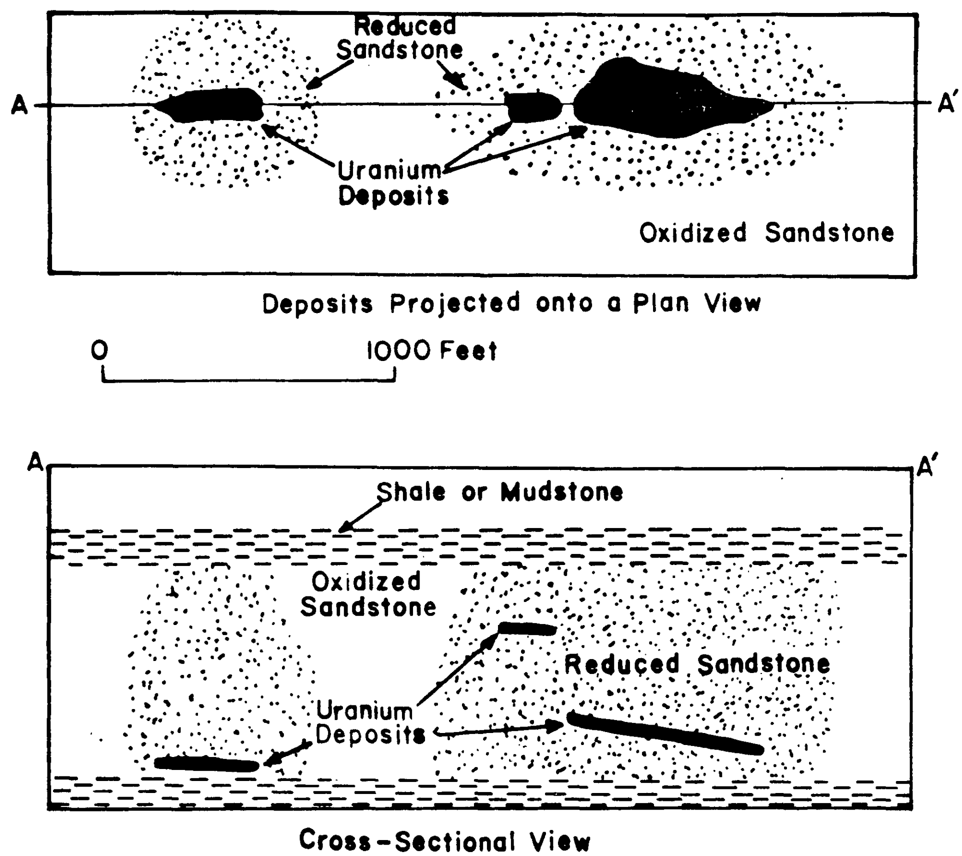


FIGURE 3 - PLAN AND CROSS-SECTIONAL VIEWS OF TABULAR URANIUM DEPOSITS

Modified from U.S. Geol. Survey(5)

COST ESTIMATES FOR CONVENTIONAL MINING PROJECT STAGES

This section briefly describes each stage in the life cycle of the typical sandstone-type uranium mining project and presents the cost data obtained by the author. The use of the ore depth to thickness ratio is derived by comparing the variation of costs for both depth and thickness changes. Estimates and assumptions used in the author's example set of calculations are developed for each stage.

Exploration

Regional scale prospecting for uranium deposits is normally begun by the use of an airborne gamma-survey. Anomalies detected are then located on the ground by the use of either a portable geiger counter, scintillation counter or gamma-ray spectrometer. A search for surrounding anomalies of lesser intensity is also carried out. However, this method is only capable of detecting surface outcrops and due to the thoroughness of previous exploration within the United States, the finding of further outcropping sandstone-type ore bodies will be rare. Buried deposits must be located by the use of careful geological mapping and drilling.

Buried roll-front type deposits are located by the following method. After locating a favorable sandstone bed, drilling is carried out on wide spacings (half mile or greater) along widely spaced lines, until the interface between reduced and oxidized sandstone is straddled. The interface is delineated using moderately spaced drill holes (e.g. 1,500 ft. centers). If an anomalous concentration of uranium is detected the deposit is outlined by the use of more closely spaced holes (e.g. 400 ft. centers). If the results prove favorable, the deposit is typically drilled on 200 ft. centers, possibly closing down later to 100 ft. centers.

In order to locate buried tabular type deposits, geological mapping and wide-spaced drilling is first carried out to find and outline favorable ancient channels, sandy beaches or offshore bars. This is followed by moderately spaced drilling to locate deposits which are then explored by closely spaced drilling similar to that described above.

A compilation and analysis of nationwide uranium exploration and development drilling expenditures is published annually by the Grand Junction Office of the Department of Energy[6,7]. These figures are categorized into: 1. Land Acquisition, 2. Surface Exploration Drilling, 3. Surface Development Drilling and 4. Other Expenditures. However, details of deposits found are not given.

For the author's calculations, Klemenic's exploration cost estimates have been used. Klemenic states the following:

> 'The assumed unit exploration costs are no more than subjective "guesses" as there is very little actual data available. The figures are meant to reflect industry-wide averages including failures. There was no attempt to assign discovery rates, unit drilling costs, or depth of ore.' (3).

For Klemenic's October, 1972 paper he assumed property acquisition to be carried out over years 1 through 9 of the project and exploration drilling over years 3 through 6 (Appendix A, Tables A1 and A2)[(2)]. For his 1974 analysis, he reduced the period of property acquisition to years 1 through 5 and also for these years included the category of Field Expense (Appendix A, Tables A3 - A10)[(3)]. When using the 1972 figures, the author applied the reduced period to property acquisition.

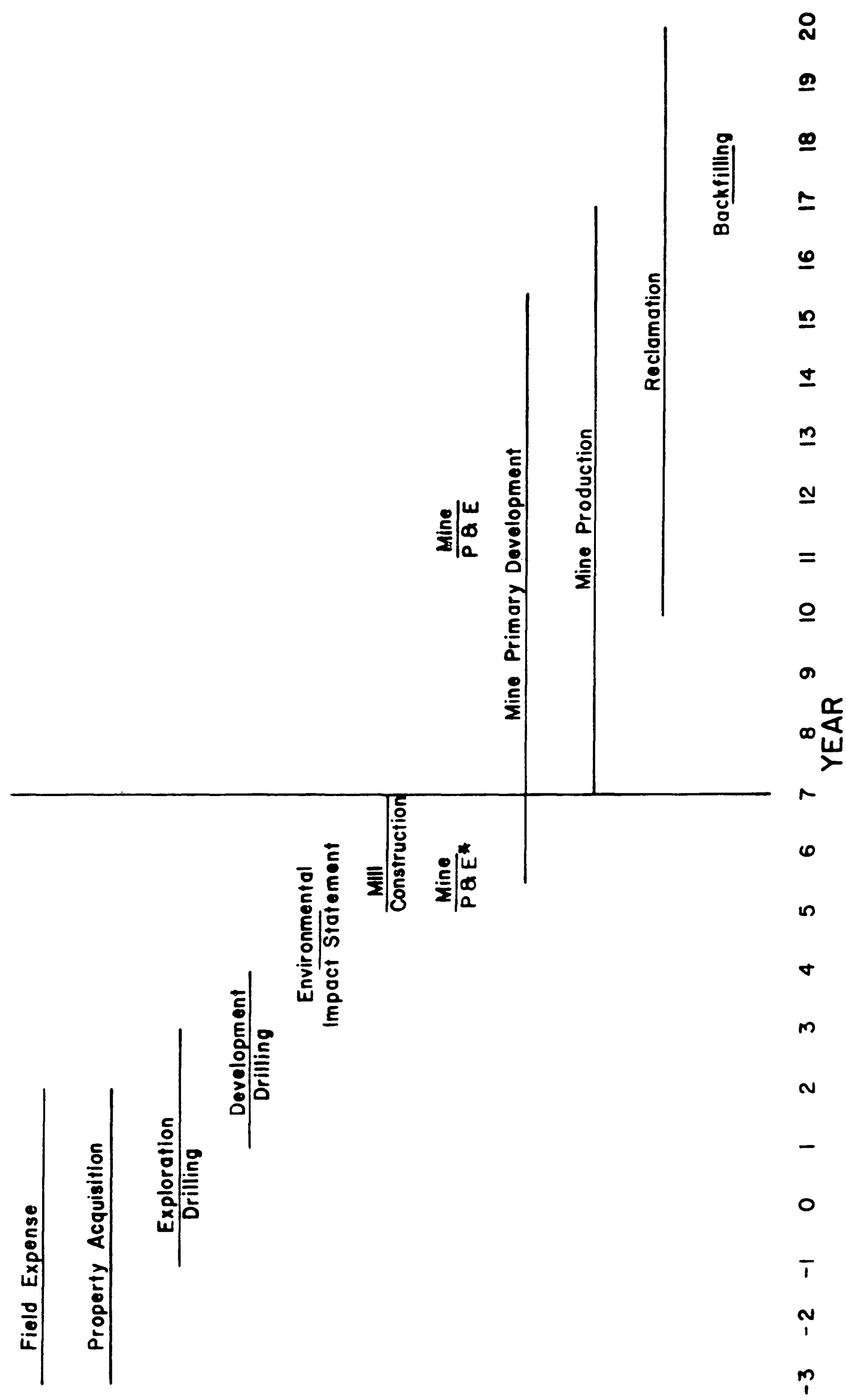
Other assumptions applied by the author are as follows:

1. The uranium deposit(s) in the prospect has (have) already been drilled to a limited extent. This implies that the user of the results of the calculations has some idea of the potential of the prospect for comparison. Therefore, the author set the year 0 for his calculations at the end of Klemenic's first year of exploration drilling (Figures 4 and 5)

2. Property acquisition expenditures do not vary with depth or thickness of deposit.

3. Field expenses are assumed to vary in direct proportion to drilling costs during the drilling phases of a

FIGURE 4 – TIME DIAGRAM FOR THE OPEN PIT CASE



*Mine Plant and Equipment

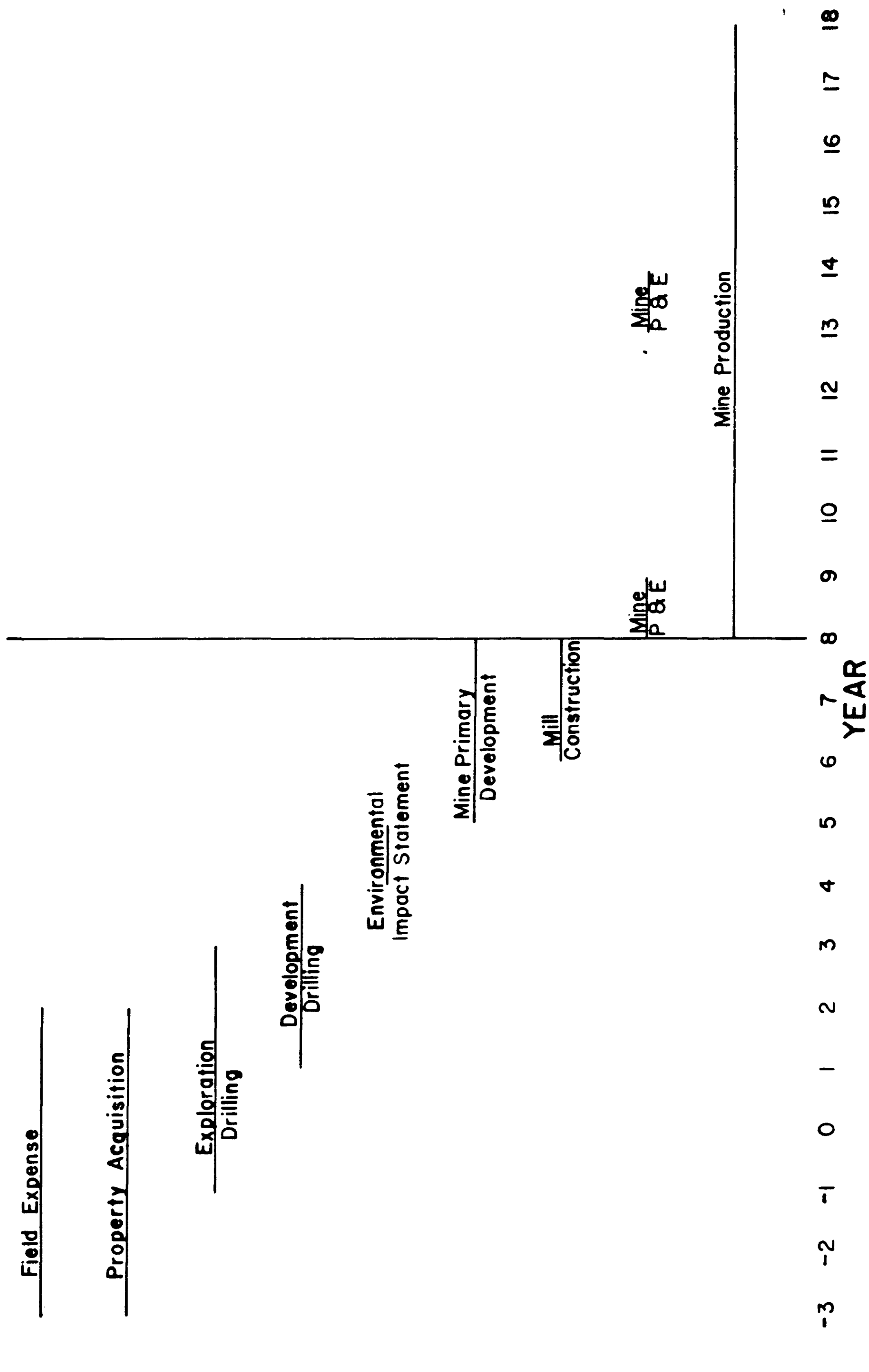


FIGURE 5 - TIME DIAGRAM FOR THE UNDERGROUND CASE

project. This assumption was supported by personal communication with the Exploration Manager of a uranium mining company[8]. This source advised that field expenses are generally 20 to 25 percent of drilling costs on a large drilling program.

4. The drilling costs including development drilling and field expenses in Klemenic's data are assumed valid for an ore depth to thickness ratio of 56. This is the average of the depth to thickness ratios weighted by ore tonnage for the mines included in his study[1]. This adjustment was only made to avoid the discrepancy of these costs for an open pit case being the same for an underground case of much larger depth to thickness ratio.

5. Total drilling costs vary in direct proportion to depth to thickness ratio. Personal communication with industry sources found this to be correct for variation of deposit thickness. Solid data is not available for variation of total expenditures with depth.

Estimated typical rotary drilling costs for a large uranium exploration or development drilling program in a sandstone environment are listed in Table 1[9]. These figures indicate that rotary drilling costs for a typical project increase by approximately 80 percent on an average cost per foot basis as drill hole depth increases from 500

TABLE 1

Estimated Rotary Drilling Costs[9]

Total Depth of Hole (Feet)	Average Cost per Foot (Dollars)
500	2.50
1,000	3.00
1,500	3.70
2,000	4.50
2,500	5.30
3,000	6.10
3,500	6.90
4,000	7.60

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to 2,000 ft. However, cost increases of 300 percent to 400 percent are common due to difficult drilling conditions, such as gravel beds, being encountered. Personal communication with company representatives found that to offset these cost increases, the number of holes drilled are generally reduced by 50 percent to 75 percent[8]. Therefore, a directly proportional relationship for drilling costs with depth appears to be a reasonable approximation.

It is general practice to gamma-log all holes and to have a drift survey made if the depth exceeds 200 ft. The previously referred to Exploration Manager estimated gamma-logging costs average 25 cents per ft. over a large drilling program and the drift survey costs average 15 cents per ft. These costs are only minor when compared to the drilling costs. It is also normal practice to core the zone(s) of interest in approximately 10 percent of the holes. Coring does not add significantly to total drilling costs, even though on a per foot basis it is much more expensive than open hole drilling.

6. The Wholesale Price Index for Industrial Commodities was assumed to be an appropriate index to apply to all exploration phase costs when adjusting Klemenic's cost data to September 1977; the date for which the author's calculations have been made. However, the industry wide exploration costs published annually by the Department of Energy show a much higher rate of increase than the index during the 1973 through 1975 period[6,7].

Personal communication with Klemenic and the previously referred to manager of a major drilling company indicated that the high rate of increase in drilling costs during that period was due to three factors:

(a) A large increase in the proportion of uranium exploration being carried out in hard rock environments (e.g. metamorphic and granitic rocks)

(b) An increase in the average depth of holes drilled in sandstone environments

(c) A shortage of drilling rigs[9,10].

The drilling company manager stated that the company's 1977 prices, when compared with its 1973 prices, had only kept abreast with inflation. Similarly, the Department of Energy statistics show that average land acquisition costs fell dramatically from \$4.80 per acre in 1975 to \$2.92 per acre in 1976. The 1973 average was \$2.67 per acre. The dramatic fall in 1976 was not due to a slackening of demand for land, but because of large acquisitions in new areas of interest where land was relatively inexpensive.

The following summarizes the more important estimates and assumptions presented during this discussion of the exploration stage:

1. The project life cycles are as shown in Figures 4 and 5.

2. Drilling costs and field expenses vary in direct proportion to the ore depth to thickness ratio.

3. Property acquisition expenditures do not vary with depth or thickness of deposit.

4. Klemenic's drilling cost and field expense estimates are valid for an ore depth to thickness ratio fo 56.
5. The Wholesale Price Index for Industrial Commodities is the appropriate index to apply to all exploration phase costs when adjusting Klemenic's cost data to September 1977.

<u>Development</u>

The development stage of a project consists principally of development drilling, mine primary development and mill construction.

During the development drilling phase it is usual practice to drill the deposit(s) on approximately 50 ft. centers. However, approximately 100 ft. centers will be used for deep deposits (e.g. below 1,500 ft.) or where difficult drilling conditions are encountered. The variation of drilling costs with depth to thickness ratio has been previously discussed under the heading Exploration.

Klemenic shows the development drilling as being carried out during years 5 through 7 for his examples. This is concurrent with a large portion of the exploration drilling. The concurrence is partially due to variations within industry in the method of defining the phase of drilling being undertaken.

The final feasibility study and the initial engineering design are normally carried out during the development phase. Due to a lack of adequate information, the author has not separated the costs of these activities, nor the administrative costs, from Klemenic's cost categories.

Also during this phase, the environmental impact statement should be submitted and applications made for the many permits necessary to establish and operate a uranium mine and mill complex. Because of the long time base necessary for collection of data for the environmental impact statement, it is preferable that the company initiate this collection process immediately after a potential ore body is discovered.

Personal communication with company and government officials (October 1977) indicated that usually the start-up for uranium mining projects is being delayed at least two years. This results from the time required to prepare and obtain approval of the environmental impact statement and other permits. The most optimistic of these individuals estimated that increasing familiarity with the regulations will soon reduce delays to a total of one year for a typical project.

For the purpose of calculations, the author has assumed that this estimate of a one year delay will be correct. The author simplified these calculations by assuming that all delays are incurred prior to mine primary development and mill construction. Klemenic did not include such a delay. At the time of his study, environmental regulations were not as stringent as now.

Estimates of total cost for the environmental impact statement process were obtained from John F. Facer, Jr., Chief of the Analysis Branch, Department of Energy, Grand Junction, Colorado[(11)] and the Corporate Attorney of a uranium mining company[(12)]. These estimates indicated that the cost would be about $150,000 for a 500 ton per day open pit mine and mill complex, increasing on a logarithmic scale to $500,000 for a 10,000 ton per day complex. The latter source estimated that the cost for an equivalent underground case would be about two-thirds that for the open pit case. For the calculations the author has assumed that the cost does not vary significantly with ore depth to thickness ratio, and that it is incurred during the one year delay.

Mine primary development for open pit mining is shown in Klemenic's examples (Appendix A) as beginning about one and one half years prior to start-up and continuing at a uniform cost until about one and one half years before completion of mining. This mainly involves stripping the overburden to expose the ore. Stripping ratios of greater than

40:1 are becoming common.

The overburden will usually consist of relatively unconsolidated sandstone, mudstone and conglomerate beds. Therefore most mines use bulldozers with rippers to break up the overburden and scrapers to remove it. Where more competent beds are encountered, a combination of shovels, front-end loaders and haulage trucks is employed.

Often the overburden stripping is contracted out at a fixed charge per unit volume. This effectively distributes the equipment costs over the life of the stripping operation. Klemenic has followed this practice by including the equipment costs and the direct costs of stripping together under the category of mine primary development.

The variation of open pit primary development costs with ore depth to thickness ratio has been shown by Klemenic as approximating to a directly proportional relationship[(1)]. The author has applied this in the calculations.

For underground mines, the mine primary development is shown in Klemenic's examples as taking place in the three years prior to start-up. During this period shaft sinking and development drifting are carried out and a number of production faces are readied for mining. It is common practice for this work to also be contracted out.

As in the open pit case, Klemenic shows a directly proportional relationship of underground mine primary development cost to ore depth to thickness ratio(1). This relationship is supported by the capital cost estimates of Dr. Douglas J. Lootens (Associate, Dames & Moore) for hypothetical underground mines in New Mexico(13). His examples are shown in Table 2. Lootens advised the author that these mines could all be assumed to have the same ore thickness(14).

Klemenic's examples assume that the company must construct a mill within about five miles of the mine, with its capacity matched to that of the mine. Though, other possibilities are:

1. The company has the milling done at an already established mill within the district. This requires that there be spare milling capacity, equivalent to that of the mine production available for the life of the mine. Such a possibility is not very likely for production rates of greater than 500 ton per day.

2. The company becomes a partner in, or contracts capacity in large mill to be established at a central location suitable to the two or more parties involved. This has the obvious advantage of economies of scale, mainly for production of below 2,000 ton per day. However, it also has the obvious requirement that another company within the region be bringing a mine into production at almost the same time.

TABLE 2

Capital Cost Estimates for Hypothetical Underground Uranium Mines in New Mexico[13]

Depth (Feet)	Tons per Day Milled	Capital Cost (1975 Dollars) (000)
2,000	500	11,200
	1,000	12,397
	2,000	14,602
4,000	500	25,048
	1,000	26,353
	2,000	28,010

In both of the above cases, the advantages are rapidly offset by haulage costs for production rates of 1,000 ton per day or higher. For these production rates the author considers it unlikely that the ore would be hauled more than twenty miles. Therefore, the construction of a new mill at the mine appears appropriate for these cases, although it may prove an inappropriate assumption for the 500 ton per day case. For the purpose of the calculations, the author has followed Klemenic's assumption that a new mill must be established specifically for the mine in all cases.

Mill construction is shown in Klemenic's examples as occurring during the two years immediately prior to the start-up of operations.

To adjust Klemenic's development period costs to September 1977, the author applied the Marshall and Swift Mining and Milling Equipment Cost Index. This is generally considered the best index for mine primary development costs. Klemenic considered the Chemical Engineering Plant Cost Index a more appropriate index for the uranium mill construction cost[10]. However, discussion with industry personnel indicated that new reulations pertaining to uranium mills have brought on some significant construction cost increases in recent years. Therefore, the author considered application of the higher rate of escalation provided by the first index to be more appropriate.

In summary, the important estimates and assumptions presented during this discussion of the project development stage are:

1. Both open pit and underground mine primary development costs vary in direct proportion with ore depth to thickness ratio.

2. The company must construct a new mill specifically for the mine within about five miles of the mine.

3. Obtaining approval of the environmental imapact statement and other permits delay mine primary development and mill construction by one year.

4. The total cost of researching and obtaining acceptance of the environmental impact statement is incurred during this one year delay. The cost is $150,000 for a 500 ton per day open pit mine and mill complex, increasing on a logarithmic scale to $500,000 for a 10,000 ton per day complex. The cost for an equivalent underground case is two-thirds that for the open pit case. These costs do not vary with ore depth to thickness ratio.

5. The Marshall and Swift Mining and Milling Equipment Cost Index is the appropriate index to apply to the mine primary development and mill construction costs when adjusting them to September 1977.

Operation

Both open pit and underground mining usually are carried out on a very selective basis. This is because the ore zone can contain layers of ore and waste as little as one foot thick. Even within the one layer of ore important variations in grade can occur within a distance of a few feet laterally.

Open Pit: In the open pit case, the production drilling of the ore is typically on about 6 ft. centers with the holes being gamma-logged or sampled, then lightly blasted in order to loosen the ore. A combination of backhoes and trucks are generally used for the actual mining of the ore. For more consolidated ore bulldozers will be used for ripping, followed by front-end loaders in place of the backhoes. Often an assistant with a scintillometer or equivalent instrument, closely monitors the grade of rock being loaded. The truck load may also be scanned on leaving the pit. This is in order to maintain a relatively constant grade of mill feed. Material below the mill feed cut-off grade, but still containing a significant concentration of uranium, may be sent to a specially prepared stockpile for leaching, or even stockpiled for mill feed in later years when the grade requirement is lowered. True waste will be sent to the overburden dump or used as backfill in mined out areas along with the overburden currently being stripped. Large volumes of waste will be stripped by a similar method to that being employed for the overburden.

Klemenic assumed for his examples that the ore will be mined out over 10 years. This may not prove typical for the larger deposit sizes considered. Instead of applying fractions of years of mining as Klemenic has in years 9 and

19, the author simplified the calculations by assuming the start-up does not begin until the following full year. The author also delayed the start of mine primary development by the same amount.

Klemenic's examples show the mine plant and equipment being purchased in the year in which overburden stripping begins. This indicates that most of this equipment is used to assist on pre-production stripping. The equipment is replaced after 6 years. For adjusting mine plant and equipment costs to September 1977, in both the open pit and underground cases the author has applied the Marshall and Swift Mining and Milling Equipment Cost Index.

Personal communication with Klemenic, Gurmendi and three mining industry personnel found that the depth of the deposit will have an insignificant effect on the mining costs[10,15]. Haulage costs will increase by up to 10 cents per ton for an increase in deposit depth of 300 ft. This is also too minor to warrant taking into account for deposit depths within the limits which industry presently considers for open pit mining of sandstone-type uranium deposits (i.e., a maximum of about 1,000 ft.).

The author had little success in obtaining information as to how the mining costs will vary with deposit thickness. This was probably because the variation is insignificant except in the case of very thin ore. In partial support of this contention, the General Manager of a uranium mining company suggested that due to the highly selective nature of the mining methods employed, there will probably be no noticable difference in the cost with variation of thickness unless the layer to be mined is less than 3 ft. thick[16]. The author therefore chose not to vary the mining costs with ore depth to thickness ratio.

Underground: Room and pillar mining is almost invariably used for the underground mining of sandstone-type uranium deposits. It has the disadvantage that 20 to 30 percent of the geological ore reserves must be left in the pillars. However, methods which recover up to 100 percent of the geological reserves, such as the longwall or panel retreat methods, are much more expensive and tend to require a rather particular set of conditions[13].

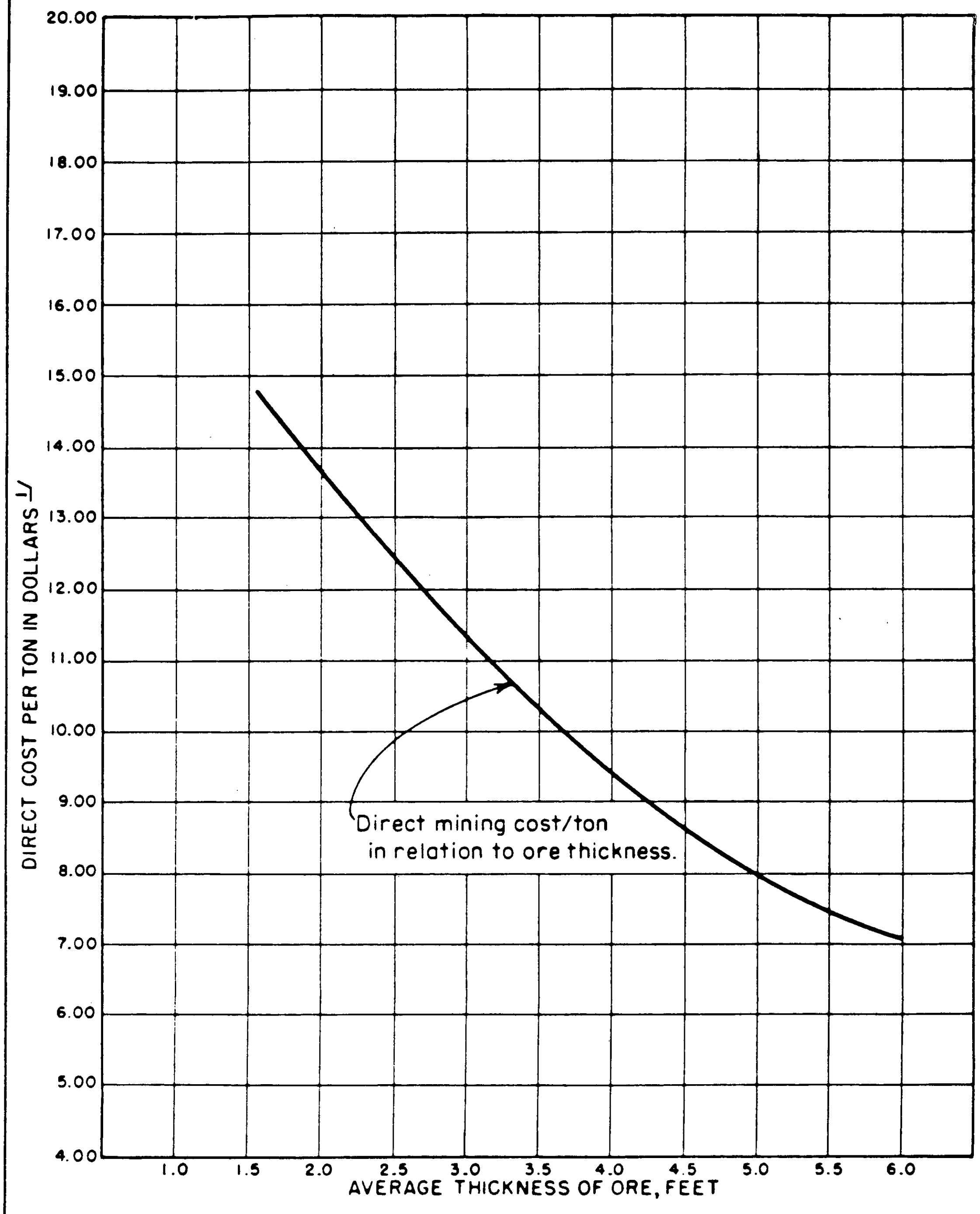
The underground mining is also carried out on a very selective basis. Blast holes and muck piles in production areas are probed every shift for the gamma-ray readings. The mining is typically not highly mechanized, with air-leg drills and slushers being in general use. Even when the ore is less than 5 ft. thick, it is common practice to use separate blasts for the ore and the waste so that each can be mucked separately.

Dare showed the relationship of direct mining costs to ore thickness at ten of Union Carbide Nuclear Company's lease operations in the Uravan mineral belt of Western Colorado (Figure 6)[17]. This relationship indicates that the direct mining costs increase rapidly with decreasing ore thickness for thicknesses below 6 ft.

A comprehensive private industry study of room and pillar uranium mining indicated an average increase of 14 percent in mining costs upon halving the thickness of the deposit[18]. This study was made for deposits 6 to 30 ft. thick with depths from 500 to 2,000 ft. and production rates of from 150,000 ton per year to 1 million ton per year. The total range given for the increase in mining costs upon halving deposit thickness was from 6 to 25 percent.

The same study also indicated that as the deposit depth increases from 500 ft. to 1,000 ft., the mining costs increase by from 2 to 5 percent, with the average increase being 3 percent. Over the 1,000 ft. to 2,000 ft. interval, the range of increase was from 18 to 32 percent with an average of 26 percent.

A study completed by the Department of Energy during 1977 indicated that the unit mining costs will increase by 11 percent as the deposit depth increases from 1,000 ft. to 2,000 ft. An increase of 20 percent was indicated over the



1/ BASED ON COSTS OF LESSEES PLUS THE ESTIMATED AVERAGE COST OF $0.45 A TON TO COVER SUPPLIES FURNISHED BY UCN.

FIGURE 6 - RELATIONSHIP OF MINING COST TO ORE THICKNESS AT TEN SALT-WASH LEASE OPERATIONS OF UNION CARBIDE NUCLEAR CO. IN THE URAVAN MINERAL BELT

Source: Dare(17)

2,000 ft. to 4,000 ft. interval. This latter result was composed of an increase of 10 percent over the 2,000 to 3,000 ft. interval and 9 percent over the 3,000 to 4,000 ft. interval. This study was for deposits of from approximately 6 to 30 ft. thick and production rates of from 500 to 1,000 ton per day per shaft. The results were from an in depth study of data obtained during 1976 from uranium mines in New Mexico(15).

The figures given by Lootens for hypothetical New Mexico mines show an approximate 30 percent increase in mine operating costs when deposit depth increases from 2,000 to 4,000 ft.(13). This is for mines of 500, 1,000 and 2,000 ton per day milled. Personal communication with Lootens found that he expects there to be only a small increase in mine operating costs over the depth interval of 500 to 2,000 ft.(14).

After analysing the above results, the author chose to vary the underground mining costs by 12 percent of the change in depth to thickness ratio. This figure will not be valid for thicknesses of less than 6 ft.

Operating Cost Indexation: The author has applied the Wholesale Price Index for Industrial Commodities to all operating costs to index them to September 1977.

KLemenic is of the opinion that in recent years the operating costs for underground uranium mines have generally increased by a much larger amount than given by applying the above index. This due to both a shortage of skilled labor and the implementation of recent strict Mine Enforce- and Safety Administration (MESA) regulations[10]. However, after further discussion with Klemenic and with a previously referred to General Manager[16], the author decided that the only permanent cost increase above that obtained from the index would be due to the MESA regulations. Based on the experience of the General Manager, the author estimated this permanent amount to be 5 percent of operating costs.

Klemenic advised the author that he considers mill operating costs have kept in line with inflation. Although there have been some recent operating cost increases due to strict regulations, his opinion is that these have been offset by increases in efficiency in other areas of the plant. The Mill Manager for a recently completed mill in Wyoming confirmed this opinion by stating that he did not consider there was any significant over all cost increase at his mill due to operating regulations imposed in recent years[19].

Mill Recovery: The mill recoveries given by Klemenic in his 1972 and 1974 papers (Table 3) have been applied by the author. Personal communication with both Klemenic and Facer found that they considered these estimates to be valid for mills presently being constructed[10,11]. The author

TABLE 3

Mill Recoveries

Ore Grade ($\% \ U_3O_8$)	Mill Recovery (%)
0.25	95.0*
0.20	94.0*
0.10	92.5
0.05	87.5
0.025	77.5
0.01	47.5

Source: Klemenic (1974, p. 4)[3];
*Klemenic, telephone conversation, November 1977[10].

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received conflicting advice from members of industry as to whether Klemenic's recoveries were too high or too low, particularly for the lower range of ore grades. Klemenic suggested that this conflict would be due to variations in ore composition. The author also notes that there has been little experience in the milling of low grade ores (i.e., 0.05 percent U_3O_8 and below) either by industry or government.

Summary: The main estimates and assumptions presented during the above discussion of the project operating stage are the following:

1. The direct mining costs for underground mining increase by 12 percent on doubling of the ore depth to thickness ratio provided the thickness is not less than 6 ft.

2. No other operating period costs vary significantly with ore depth to thickness ratio. This includes the direct mining costs for open pit mining for deposits not less than 3 ft. thick.

3. The appropriate indexes for adjusting Klemenic's operating period costs to September 1977 are:

   (a) The Marshall and Swift Mining and Milling Equipment Cost Index. This is applied to the mine plant and equipment cost.

   (b) The Wholesale Price Index for Industrial Commodities. This is applied to mining, haulage and milling costs.

4. There has been a permanent 5 percent increase in the direct mining cost for underground mines because of MESA regulations.

5. The mill recovery estimates provided by Klemenic (Table 3) shall remain valid for future mills.

Backfilling, Reclamation and Salvage

The regulations governing the backfilling and reclamation of mining areas are still undergoing rapid development. Furthermore, the interpretation of such regulations can take many years to settle. Therefore, any estimates made in these areas must be subject to a high degree of uncertainty. However, it is fortunate that the accuracy of these estimates will only have a small effect on the results of calculations. This is because the cost of reclamation is small compared to other costs, and also the major backfilling cost occurs at the end of the project so that its present value at year 0 is small at currently used industry discount rates.

For the open pit case, the author has assumed that the overburden from the first three years of stripping will be stockpiled. Further stripping will be used to backfill the mined out portions of the pit.

The Manager of Environmental Services for a uranium mining company advised the author that it is appropriate to assume that half of the final pit volume will need to be backfilled at a cost of 50 cents per cubic yard of backfill. He estimated that the cost of reclamation of the mined out portions after backfilling (e.g. contouring and revegetation) would be about \$6,000 per acre. Surrounding land of twice this area would also require reclamation at a cost of about

$3,000 per acre[20]. These estimates concurred with those obtained from other sources. However, Frederick C. J. Lu (Chief of the Supply Branch, Department of Energy, Grand Junction Office, Colorado) advised the author that reclamation costs could have a very wide range depending on factors such as climate and topography (e.g. $2,000 to $12,000 per acre)[21].

In applying the above estimates, the author assumed the following:

1. The backfilling of the final pit takes place in the year after mining ceases.
2. A 500 ton per day operation with a depth to thickness ratio of 24 will require 4,500,000 cubic yards of backfill. This estimate was based on an average pit floor width of 100 yards and the average ore thickness of 8.3 ft. for Klemenic's open pit data[1].
3. The cost of backfilling the final pit varies directly with both ore tonnage and depth to thickness ratio. There is no cost incurred for the backfilling which takes place during the operating years.
4. Reclamation of the backfilled portion of the mine and the surrounding area begins in the fourth year of mining and continues at a constant cost per year until three years after mining ceases.

5. The total mined area over the ten year life of a 500 ton per day operation will be 100 acres. This is assumed to vary on a one to one basis with tonnage.

The surface area of the pit will vary inversely with ore thickness. However, calculations made by the author show that this variation does not have a significant effect on the results of this study.

The salvage value of the mill and mining equipment was taken to equal the reclamation cost of the mill site and tailings pond. In the case of an underground mine, this also includes any other site reclamtion costs.

IN-SITU LEACHING

Due to a lack of necessary cost data, the author has not carried out comparative calculations for in-situ leaching (also called solution mining). However, this mining method has been forecast to rapidly grow in importance for the mining of shallow uranium deposits which are uneconomical to mine by conventional methods. Projections of the U.S. Energy Research and Development Administration made in 1976 show that by 1982 uranium oxide produced from in-situ leaching could compose more than 7 percent of the total United States production[22]. Therefore, for completeness it is necessary to briefly describe this alternative mining method, the situations in which it can be applied, and its major advantages and disadvantages. Sources of currently available cost data and two examples of marginally economic deposits are also presented.

The method requires that a pattern of injection wells and recovery wells be drilled into the uranium bearing sandstone formation (Figures 7,8). The method also requires that the formation be an aquifer below the water table. An acid or alkaline leach solution, together with an oxidizing agent is then introduced into the formation either by gravity or under pressure, depending on the permeability of the

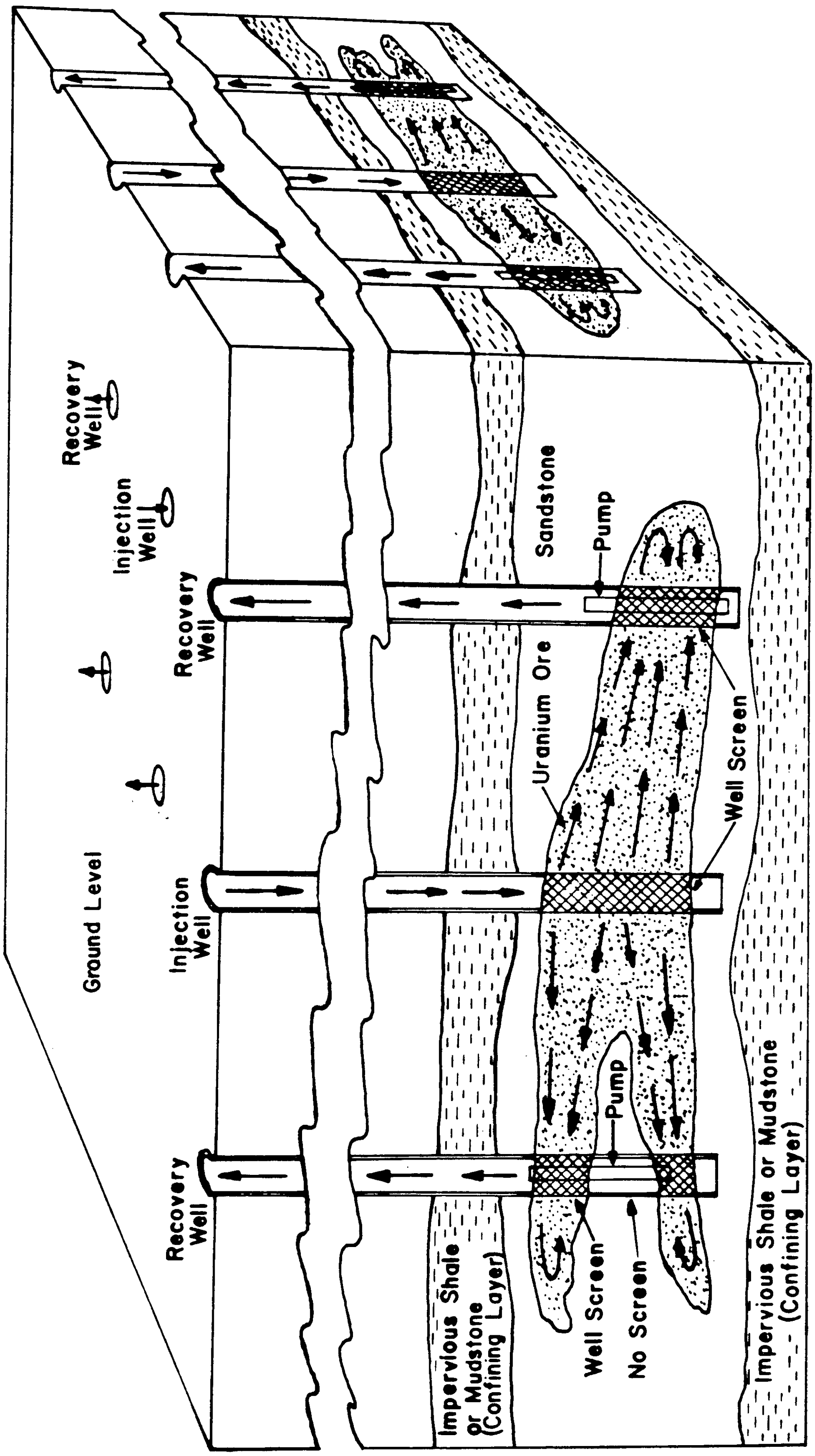

FIGURE 7 - SCHEMATIC OF THE IN-SITU LEACHING TECHNIQUE

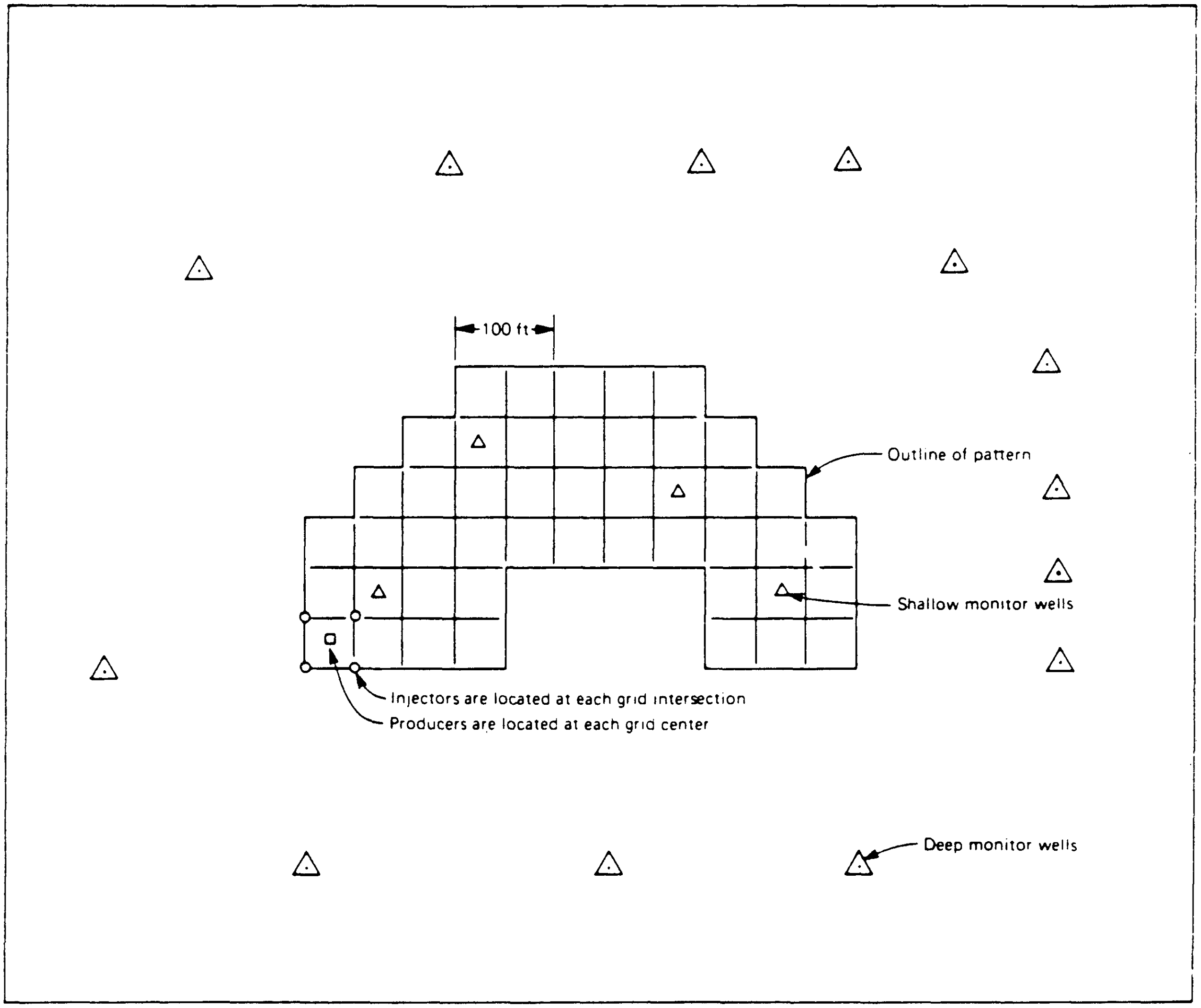

FIGURE 8 - THE URANIUM LEACHING PATTERN EMPLOYED AT CLAY WEST, TEXAS, BY THE ATLANTIC RICHFIELD - U.S. STEEL - DALCO JOINT VENTURE

Source: White(23)

sandstone. Pumping at the recovery wells produces a zone of low pressure towards which the leach solution migrates. The pregnant solution, after being pumped to the surface, then goes to a conventional uranium recovery system.

Frank states that for a uranium deposit to be suitable for in-situ leaching it should exhibit the following characteristics:

1. Be amenable to chemical leaching
2. Have adequate permeability
3. Be located below the natural water table
4. Be confined by impervious layers
5. Be within economical limits of drilling[24].

Presently in-situ leaching is considered as an alternative mining method for uranium deposits which are marginal or uneconomical to mine by conventional methods. Further refinement of the techniques involved could also make it an attractive alternative for the mining of richer deposits in future years.

The maximum depth for which this method can currently be applied occurs between 500 and 1,000 ft., depending on formation conditions[25,26]. All casing and most other piping used in the system is PVC in order to minimize corrosion and to keep corrosion products out of the solutions. Below the depth limitations mentioned, the formation pressures are such that the PVC casing will be crushed. Alternatives are presently too expensive to be viable.

Both the exploration phase and development drilling phase can be considered the same as for a conventional project[26]. There will also be a considerable delay before construction can begin while approval of the necessary permits is obtained. Because the mining method requires injecting foreign substances into an aquifer, many more permits are required than for a conventional uranium mining operation. A detailed economic evaluation of the regulations involved has been carried out by Tittes[27].

Before construction of a full scale operation, a pilot operation will be established. This will be operated for at least one year in order to test the properties of the uranium bearing formation. During this period, a method must also be developed for restoring the formation to the required condition after the leaching has ceased.

Because of the relative simplicity involved in the construction of a commercial scale operation, there will be a saving of about two years in the overall lead time from discovery to start-up when compared to that for a conventional operation. The operating life of the project will obviously depend on the extent of the ore deposit, the size of the well pattern used, and how often the pattern is shifted.

Factors which determine the size and rate of movement of the well pattern are the grade and thickness of the ore deposit, the rate of injection and the effectiveness of the leach solution and the level of uranium recovery considered most economical.

Frank has presented a detailed cost model for a typical project[(24)]. This model indicates that large savings will be obtained in the mill construction and primary development costs when compared with an equivalent open pit project. However, the direct operating costs shown are very similar in both cases.

Surface reclamation on the site of a well pattern after it has ceased operation involves negligible costs. Similarly, the mill and associated structures and pondages damage only a relatively small area and therefore incur only a minor reclamation cost. However, restoring the aquifer to the required standard is proving to be an important cost.

The aquifer restoration is presently an area of great uncertainty. The problem is that large quantities of foreign minerals can precipitate in the aquifer during the leaching process. Present regulations require these to be removed. Tittes estimated the cost of restoration to be about 50 cents per pound of U_3O_8. Although this estimate is much higher than Frank's 5.3 cents per pound, it could easily prove to be much too low. The extent of the problem can be gained from the following statement of Facer:

> "Although some companies chose in-situ leaching because it seemed to involve fewer environmental problems than conventional mining, this may prove to have been an illusion. Restoration of an in-situ leach well field to something approaching its original condition has not been demonstrated on a large scale." (28).

Some additional disadvantages cited by Frank are:

1. Potential toxic spills
2. Loss of leach solutions
3. Possibility of the contamination of ground water
4. Disposal of waste solutions
5. Presence of radium and selenium.

A further problem is that of the level of uranium recovery. The level that will be achieved is often very uncertain and present results are significantly below those obtained from open pit mining. The property of leach solutions to follow the path of least resistance, such as through fractures or loose sands, is often a major cause of low recoveries. Because the intensity of such geological features as jointing can vary greatly from block to block within the one ore deposit, recoveries can fluctuate widely from one well pattern to the next. Therfore, one must be generous in estimating the size and grade of deposit required for a profitable operation.

Little data has been released to date upon which to base calculations of the minimum economical range of deposits for in-situ leaching. However, the author considers that such calculations will be easily made once access to enough cost data is obtained with which to derive trends of cost catagories with deposit size, grade, depth and thickness variation. The most detailed cost studies which the author found available are by the previously referred to Frank and by Bhappu et al[24,29,30,31]. Also, a computerized cost model is presently under development for the Electric Power Research Institute, Palo Alto, California[32].

For comparison with conventional mining, the author obtained from industry personnel the following two examples of deposits currently considered as marginally economic for in-situ leaching:

(a) 7+ million ton of 0.02 - 0.03 percent U_3O_8, thickness 50 ft., depth 500 ft., with a recovery level of 40 - 50 percent.

(b) 1+ million ton of 0.12 percent U_3O_8, thickness 6 in., depth 200 ft., with a recovery level of 70 percent.

These figures, although based on real deposits, contain some subjective adjustments. The tonnage requirements are for a production rate of 250,000 pounds of U_3O_8 per year.

The above description and data for in-situ leaching have been included as a brief introduction to this relatively new mining method. Further investigation of this topic is beyond the scope of this thesis.

DERIVATION OF THE MINIMUM ECONOMICAL RANGE

In this section the method used by the author for deriving the minimum economical range of explorable deposits is described. The method is based on calculating the price per pound of uranium oxide necessary for a large set of example deposits to be economical. The obtainable selling price for the uranium oxide is then estimated and the range of deposits requiring that exact price is plotted.

A FORTRAN computer program was written by the author to calculate the minimum price necessary for a given tonnage and grade of deposit to be economically explorable at each specified ore depth to thickness ratio. This program is based on the Net Present Value (NPV) method of financial analysis[33]. It calculates the price necessary for the project NPV to be zero at the minimum required Discounted Cash Flow Rate of Return (DCFROR) on investment. Copies of both the program and User's Guide have been included in Appendices B and C.

The format of the input data files for the program was designed for use with Klemenic's data and the previously described results of the author's research. However, provided the same format is followed, the program has been written flexibly enough so that other assumptions can be used in most cases in place of the author's. Furthermore, within the program all important steps have been described by the generous use of Comment statements. This has been done so that the user can easily modify the program if he wishes, or apply sections of it to other purposes.

The following are previously mentioned or implied as-aumptions and estimates which the author has applied in producing the example set of results:

1. Klemenic's 1972 and 1974 cost data are the most accurate and comprehensive available. Therefore, all of the calculations have been based on these.

2. The deposit has only recently been discovered. Therefore, the author's year 0 is the beginning of Klemenic's year 4.

3. To adjust the cost data to September 1977 the appropriate indexes are the following:

   a) The Marshall and Swift Mining and Milling Equipment Cost Index. This is applied to mine primary development, plant and equipment, and mill construction expenditures.

   b) The Wholesale Price Index for Industrial Commodities. This is applied to all other cost categories.

4. There has been a permanent 5 percent increase in the direct mining cost for underground mines because of MESA regulations.

5. The cost of researching and obtaining acceptance of the environmental impact statement is $150,000 for a 500 ton per day open pit operation, increasing on a logarithmic scale to $500,000 for 10,000 ton per day. For equivalent underground operations the costs will be two-thirds of these.

6. Obtaining acceptance of the environmental impact statement and other permits will delay the start of the mine primary development and mill construction by one year.

7. On doubling of the ore depth to thickness ratio the following costs increase by the given amounts:

   a) Field expense, drilling, mine primary development, and open pit backfilling: double.

   b) Underground mining: 12 percent.

   c) No other costs vary significantly with ore depth to thickness ratio.

8. The drilling costs and field expenses used by Klemenic are assumed valid for an ore depth to thickness ratio of 56.

9. A mill will be constructed within a distance of about 5 miles of the deposit. This mill will be used only for processing ore from this deposit.

10. The mill recoveries given by Klemenic (Table 3) will remain applicable to future projects.

11. The mined out portions of an open pit will be backfilled at no cost using overburden stripped from the developing portion of the mine. In the year following completion of mining, half the volume of the remaining pit will be backfilled. For a 500 ton per day operation with an ore depth to thickness ratio of 24, this will require 4,500,000 cubic yards be backfilled at 50 cents per cubic yard. The cost of backfilling varies directly with tonnage.

12. The surface reclamation for an open pit begins 3 years after the start of mining and continues for 3 years after mining ceases. The cost of this is \$6,000 per acre for the backfilled pit surface and \$3,000 per acre for surrounding land or twice this area. The total mined area over the 10 year life of a 500 ton per day operation was assumed to be 100 acres. This was varied in direct proportion to tonnage.

13. The net salvage value for all operations was assumed to be zero after reclaiming the tailings pond, mill site, etc.

14. For underground mines, 25 percent of geological ore reserves will be left as pillars.

The following are assumptions not previously mentioned which the author has used in his calculations:

1. The company carrying out the project does not have income external to the project against which to take tax deductions. This is typical of a small exploration company.
2. The effective tax rate for combined federal, state and local taxes will be 50 percent.
3. All present tax regulations will remain unchanged. This includes the 10 percent investment tax credit for plant and equipment, and the 22 percent depletion allow-for uranium mining.
4. The depreciation method applied was double declining balance - switching to straight line when beneficial to do so.
5. The mill depreciation life is 20 years. This was applied on the advice of the previously mentioned General Manager of a uranium mining company[16]. The depreciation life of the mine plant and equipment is the number of years for which they are in operation.
6. The total purchase price of the mine plant and equipment is eligible for investment tax credit, while

only 70 percent of the mill construction cost is eligible. This latter figure was derived from a study of Looten's breakdown of mill construction costs[13].

7. The working capital is added in the year before the start-up of operations. It is 20 percent of the annual operating cost, including the continuing open pit mine primary development cost.

8. The applicable royalty rate is 12 percent of the mine mouth ore value. Klemenic suggested that this would be more representative than the more commonly used 6 to 8 percent of gross revenue[10]. The mine mouth ore value is gross revenue minus the costs of milling and haulage.

9. An escalation rate of 5 percent per annum was applied to all costs and the uranium oxide selling price. A confidential private industry study showed that both capital and operating costs within the uranium industry increased by an average of 4 percent over the 20 years to 1970. Although the inflation rate has been falling since the bout of high inflation in the mid-1970's, the author considered that such a low escalation rate for mining industry costs cannot be maintained again in the future. The author also assumed that a constant relationship will be maintained between the selling price and the costs.

The uranium oxide selling price generated by the program is that required at the date to which the input costs have been indexed. For the author's calculations the date was September 1977.

The price generated is for the uranium oxide contained within the final yellowcake product. It is common for the yellowcake to assay 85 to 90 percent U_3O_8.

Furthermore, the program makes no allowance for additional revenue generated from the possible by-products of vanadium and molybdenum. Although, one could obtain an approximate adjustment by subtracting an expected weighted price for these by-products from the uranium oxide price generated by the program.

The analysis carried out by the program ignores the fact that the project would most likely be financed by a large percentage of borrowed funds. The author did not provide for loan finance because of the belief that before a project is undertaken it should be proved viable without the benefit of leveraging.

Example graphs plotted from the results of the author's calculations are contained in Appendix D. Because there is an infinite number of graphs which could be plotted from these results, the author has only selected those which he considered illustrative. For the case where the user wishes to plot a large number of graphs from the program output with the aid of a computer plotting program, the author designed the output format for use with such programs (Table 4).

Users of either the program or results should be careful to understand that if a price or cost escalation rate is applied, the DCFROR will not be in constant dollar terms. To illustrate this most important fact, the same investment environment will be used as was applied in generating the example graphs. This environment contains escalation rates of 5 percent per annum for both the gross revenue and costs. In such a situation, a DCFROR of 15 percent per annum will be approximated by a constant dollar DCFROR of 9.5 percent. Those wishing to pursue this matter further are referred to the work of Stermole[33].

Prediction of future cost escalation rates within the uranium mining industry will require forecasting inflation rates for the economy as a whole, then combining this forecast with any special developments forecast to occur within the industry. The escalation rate for the uranium oxide selling price will be dependent on both the escalation rate of uranium mining industry costs, and the balance of uranium oxide supply and demand. Factors influencing the United States supply and demand are discussed in papers presented at the Uranium Industry Seminar, which is held annually in

TABLE 4

An Example of the Computer Program Output

```
 1.0,-1.0,-1.0
OPEN PIT MINE AT       SEPT 1977
     MARSHALL & SWIFT INDEX (MINING & MILLING)    526.6
     WHOLESALE PRICE INDEX (INDUSTRIALS)          197.8

INPUT COSTS VALID AT  JAN. 1974
     MARSHALL & SWIFT INDEX (MINING & MILLING)    348.0
     WHOLESALE PRICE INDEX (INDUSTRIALS)          132.2

TOTAL ORE RESERVES (MILLION S.TON)                  7.300
AVERAGE ORE GRADE (URANIUM OXIDE %)                  .050
ORE PRODUCTION RATE (TON PER CALENDAR DAY)       2000.
MINE OPERATING LIFE (YEARS)                        10
YEARS LEAD TIME, YEAR 0 TO START-UP                 7

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)      7,  7
     PRICE ESCALATION RATES (PERCENT)               5.00, 5.00
     COST  ESCALATION RATES (PERCENT)               5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.
```

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	10.00	19.94
10.0	12.00	20.59
10.0	15.00	21.66
20.0	10.00	24.53
20.0	12.00	25.27
20.0	15.00	26.47
40.0	10.00	33.78
40.0	12.00	34.68
40.0	15.00	36.15
60.0	10.00	43.05
60.0	12.00	44.12
60.0	15.00	45.87
80.0	10.00	52.34
80.0	12.00	53.58
80.0	15.00	55.59
100.0	10.00	61.64
100.0	12.00	63.03
100.0	15.00	65.31

```
 -1.0
```

October by the Department of Energy's Grand Junction Office, Colorado. International supply and demand factors are presented in the Mining Annual Review[34]. It is beyond the scope of this paper to pursue the methods of forecasting price and cost escalation rates any further.

Once a set of the graphs has been plotted, the author envisages that the prices shown will be updated regularly by applying the increase in the Wholesale Price Index for Industrial Commodities to them. Although this will only give an approximate update, it should be well within the tolerances of the study for a period of at least 2 years, unless some significant changes occur in the assumptions used (e.g. the depletion allowance is reduced, or forecast escalation rates are altered).

Large variations in costs within the more important cost categories will obviously have a very significant effect on the specifications of the minimum required deposits. Examples of the effect of cost variations are shown in Figures 6 and 11 of Appendix D.

Because cost variations can have such a significant effect, the author envisaged that rather than just graphing the minimum deposit requirements based on the best estimates, the user will also graph the minimum deposit requirements based on lower possible cost estimates. These latter estimates could for example be the estimated lower 75 percent confidence limit for each major cost category. This latter set of graphs would show the range of deposits which are the absolute minimum requirements. In doing this, one would of course need to examine the possibility of in-situ leaching by another method of analysis.

A set of graphs derived similarly from a set of upper cost estimates could also be plotted if one wished. In this way, the user would have a zone of deposits defined by upper and lower limits. Deposits which are sure to fall above this zone could be allocated a large amount of additional funds with relative confidence that the project will prove viable. Deposits which are thought to fall within the zone should only be worked on prudently so as to assure that valuable exploration time and funds are not wasted.

In estimating such limits it would also be possible to include variations in mill recovery or other factors which would give either beneficial or nonbeneficial effects. However, it should be kept in mind that one only needs to combine a few important factors with a low probability of occurrence in order to get almost complete certainty of nonoccurrence. Setting the limits at unreasonable levels could waste valuable exploration time and funds.

Costs, mill recovery and the project lead time will vary dependent on the region in which a project is located. It would therefore be best to modify the estimates to suit the region. The sources upon which Klemenic based his calculations were primarily underground mines in New Mexico and open pits in Wyoming[10]. Even if using Klemenic's data as the basis of estimates for projects within these two regions, one would be wise to update these data as more modern information becomes available.

CONCLUSIONS

1. The objective of this thesis was to develop a method for estimating the minimum range of economically explorable sandstone-type uranium deposits. The method was to assume that the deposit had already been found, but further exploration work was required to fully delineate it. Models were to be developed for both the open pit and underground case, but not for in-situ leaching. The objective has been fulfilled by developing a flexible algorithm.

2. To illustrate the method an example set of results was generated. However, a flexible FORTRAN computer program and a user's guide have been included in Appendices B and C so that the user can generate a set of results based on his own cost data, forecasts, and company requirements. The program and user's guide have been written in a style designed for ease of understanding and usage. Therefore, a user with a knowledge of FORTRAN should have little difficulty modifying the program if necessary.

3. The author considers that the results will be best presented graphically. Examples of various possible graphical presentations are shown in Appendix D. For large amounts of plotting, the use of computer graphics will be

desirable. Therefore, the format of the program output was designed for ease of reading by plotting programs.

4. The method was only designed to provide an 'order of magnitude estimate' of the company's minimum requirements. An attempt to accurately model the deposits was considered pointless due to the large number of variables and unknown factors. However, the accuracy of the results will be highly dependent on the accuracy of the input data. It is possible therefore that if the user based the company's minimum deposit requirements on the best available cost estimates and forecasts, economically viable deposits could be rejected. As the solution to this problem, the author envisaged that the user would generate a set of results for the absolute minimum range of deposits acceptable by the company for further investigation. This set of results would be based on an estimated confidence limit for advantageous variation of a few important input data. Similarly, an upper range could be plotted by applying estimates for unfavorable variations.

5. Costs, mill recovery and lead time could vary significantly between districts. Consequently, the input data should be modified to suit the characteristics of the district to which the results will be applied.

6. The results obtained will be useful in providing both the exploration geologist and management with an idea of what deposit specifications must be found for a deposit to be viable. Once a deposit has been located, these results could be used on a continuing basis during the early drilling phase as a guide to the potential of the deposit and to show how much further tonnage or higher grade ore needs to be found for the deposit to fall above the minimum requirements. The geologist in charge of the drilling, knowing the minimum deposit requirements, could plan the drilling pattern so that after a minimum amount of work a decision would easily be made as to whether the deposit warranted any further consideration. If the deposit fell above the upper range of deposits, a large amount of additional funds could be allocated with relative confidence of the deposit proving viable. However, if the deposit fell in the zone between these upper and lower ranges, further work would need to be prudently carried out to assure valuable exploration time and funds were not wasted.

7. The main advantage of the method presented in this thesis will be the increase in property turnover allowed by less time being spent examining nonviable deposits.

8. If a set of calculations were desired for the case where a marginal deposit could be mined in series or in

conjunction with a nearby deposit or deposits, the method need only be repeated with slight modification. In this case, little or no mill construction cost would be included, together with reduced primary development cost if it is thought that a shaft or some other portion of the primary development cost may be shared. This example indicates that if computer facilities were readily available, a sensitivity analysis could be carried out for special circumstances.

9. To avoid a set of graphs plotted from the results becoming rapidly outdated, the uranium oxide selling price could be simply adjusted by a cost index relatively appropriate to uranium mining and milling costs. An example of such an index would be the Wholesale Price Index for Indusrial Commodities. Such adjustments would only be appropriate for a few years, particularly if economic factors change.

10. One must be cautioned against using this method indiscriminately. The author suggests some limitations on its usage:

a) The method is designed primarily for evaluation of prospects during the exploratory phase.

b) It may prove an inappropriate method for depths greater than 500 ft. for deposits being considered for open pit mining, or 4,000 ft. for underground mining. This is because for depths greater than those stated, the author could not test the assumptions upon which the method is based.

c) It should not be applied to deposits of average thickness less than 3 ft. for open pit mining, nor less than 6 ft. for underground mining. This is because the mining cost will probably increase rapidly for thicknesses less than these (Figure 6).

11. The author did not attempt to develop an equivalent model for in-situ leaching. Once sufficient cost data is acquired, the author believes it will prove a relatively simple matter to derive the required model. Until this is achieved, deposits which are geologically unsuitable for in-situ leaching will in most cases be easily recognized, thereby negating much of this limitation.

12. The author believes that models for other mineral commodities could be developed if a sufficient amount of cost data were acquired. The author's method would be most easily adapted for flatly dipping, stratiform or stratabound deposits.

APPENDIX A

Example Cost Data for the Life Cycle of Typical Uranium Mining Projects

by John Klemenic

TABLE A1

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON OPEN PIT MINING 1/ OPERATIONS AT 0.20% U_3O_8 IN ORE

	500 T/CD		1000 T/CD		2000 T/CD		3000 T/CD		5000 T/CD		Years Incurred
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	
Costs:											
Capital											
Acquisition	0.60	0.158	0.60	0.158	0.60	0.158	0.60	0.158	0.60	0.158	1- 9
Exploratory Drilling	2.00	0.526	2.00	0.526	2.00	0.526	2.00	0.526	2.00	0.526	3- 6
Development Drilling	1.00	0.263	1.00	0.263	1.00	0.263	1.00	0.263	1.00	0.263	5- 7
Mine Primary Development	6.10	1.605	5.40	1.421	5.20	1.368	5.15	1.355	5.05	1.329	8-1[illegible]
Mine Plant & Equipment	0.20 2/	0.053	0.20	0.053	0.20	0.053	0.20	0.053	0.20	0.053	9,14
Mill Construction	2.40	0.632	1.95	0.513	1.60	0.421	1.40	0.368	1.20	0.316	8, 9
Total Capital	12.30	3.237	11.15	2.934	10.60	2.789	10.35	2.723	10.05	2.645	
Operating:											
Mining	2.40	0.632	2.40	0.632	2.40	0.632	2.40	0.632	2.40	0.632	9-19
Hauling	0.65	0.171	0.65	0.171	0.65	0.171	0.65	0.171	0.65	0.171	9-19
Milling	6.05	1.592	4.65	1.224	3.85	1.013	3.45	0.908	3.30	0.868	9-19
Royalty	1.35	0.355	1.35	0.355	1.35	0.355	1.35	0.355	1.35	0.355	9-19
Total Operating	10.45	2.750	9.05	2.382	8.25	2.171	7.85	2.066	7.70	2.026	
Total Costs	22.75	5.987	20.20	5.316	18.85	4.960	18.20	4.789	17.75	4.671	

	After Taxes	After Taxes	After Taxes	After Taxes	After Taxes
Cash Flow Rate of Return, %:3/					
At $6/#	< 0	3.7	5.2	6.2	7.0
At $7/#	4.5	8.6	9.8	10.5	11.1
At $8/#	8.9	12.1	13.0	13.6	14.2
At $9/#	12.2	14.8	15.5	16.3	16.8
At $10/#	14.8	17.1	18.0	18.5	19.0
Breakeven Payout Time, Yrs. 4/					
At $6/#	19.8	17.4	16.4	15.9	15.5
At $7/#	16.9	15.1	14.3	14.1	13.8
At $8/#	14.8	13.7	13.1	12.9	12.7
At $9/#	13.5	12.8	12.3	12.2	12.0
At $10/#	12.7	12.2	11.8	11.6	11.5

1/ At 24 Depth to Thickness Ratio

2/ Some of the mining equipment is included under "Primary Development"

3/ Continuous Cash Flow - Continuous Compounding

4/ Zero Discounting

Source: Klemenic 1972 (2)

TABLE A2

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON UNDERGROUND MINING 1/ OPERATIONS AT 0.25% U_3O_8 IN ORE

	500 T/CD		1000 T/CD		2000 T/CD		3000 T/CD		5000 T/CD		Years
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs:											
Capital											
Acquisition	0.75	0.158	0.75	0.158	0.75	0.158	0.75	0.158	0.75	0.158	1- 9
Exploratory Drilling	2.50	0.526	2.50	0.526	2.50	0.526	2.50	0.526	2.50	0.526	3- 6
Development Drilling	1.25	0.263	1.25	0.263	1.25	0.263	1.25	0.263	1.25	0.263	5- 7
Mine Primary Development	4.15	0.874	3.25	0.684	2.75	0.579	2.60	0.547	2.35	0.495	8-10
Mine Plant & Equipment	0.90	0.189	0.70	0.147	0.55	0.116	0.50	0.105	0.45	0.095	11,16
Mill Construction	2.40	0.505	1.95	0.411	1.60	0.337	1.40	0.295	1.20	0.253	9,10
Total Capital	11.95	2.515	10.40	2.189	9.40	1.979	9.00	1.894	8.50	1.790	
Operating:											
Mining	14.00	2.947	12.00	2.526	11.00	2.316	10.50	2.211	10.10	2.126	11-20
Hauling	0.80	0.168	0.80	0.168	0.80	0.168	0.80	0.168	0.80	0.168	11-20
Milling	6.20	1.305	4.80	1.011	4.00	0.842	3.60	0.758	3.45	0.727	11-20
Royalty	1.75	0.368	1.75	0.368	1.75	0.368	1.75	0.368	1.75	0.368	11-20
Total Operating	22.75	4.788	19.35	4.073	17.55	3.694	16.65	3.505	16.10	3.389	
Total Costs	34.70	7.303	29.75	6.262	26.95	5.673	25.65	5.399	24.60	5.179	

	500 T/CD After Taxes	1000 T/CD After Taxes	2000 T/CD After Taxes	3000 T/CD After Taxes	5000 T/CD After Taxes
Cash Flow Rate of Return, % 2/					
At $6/#	< 0	< 0	1.5	3.0	4.3
At $7/#	< 0	3.6	6.4	7.3	8.2
At $8/#	3.4	7.6	9.4	10.2	11.0
At $9/#	7.1	10.2	11.9	12.6	13.3
At $10/#	9.8	12.4	13.9	14.6	15.2
Breakeven Payout Time, Yrs. 3/					
At $6/#	-	-	19.3	18.5	17.8
At $7/#	-	18.2	16.8	16.3	15.9
At $8/#	18.4	16.3	15.4	15.1	14.7
At $9/#	16.5	15.1	14.5	14.2	14.0
At $10/#	15.4	14.4	13.9	13.6	13.4

1/ At 76 Depth to Thickness Ratio

2/ Continuous Cash Flow - Continuous Compounding

3/ Zero Discounting

Source: Klemenic 1972 (2)

TABLE A3

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON OPEN PIT MINING1/ OPERATIONS AT 0.10% U_3O_8 IN ORE

	500 T/D		1,000 T/D		2,000 T/D		3,000 T/D		5,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs											
Capital:											
Field Expense	0.30	0.162	0.30	0.162	0.30	0.162	0.30	0.162	0.30	0.162	1-5
Property Acquisition	0.24	0.130	0.24	0.130	0.24	0.130	0.24	0.130	0.24	0.130	1-5
Exploration Drilling	0.40	0.216	0.40	0.216	0.40	0.216	0.40	0.216	0.40	0.216	3-6
Development Drilling	0.12	0.065	0.12	0.065	0.12	0.065	0.12	0.065	0.12	0.065	5-7
Mine Primary Development	5.98	3.232	5.98	3.232	5.64	3.049	5.92	2.984	5.30	2.865	8-17
Mine Plant & Equipment 2/	0.20	0.108	0.20	0.108	0.19	0.103	0.19	0.103	0.19	0.103	8,14
Mill Construction	2.36	1.276	1.92	1.033	1.56	0.843	1.38	0.746	1.18	0.638	8,9
Total Capital	9.60	5.189	9.16	4.946	8.45	4.568	8.15	4.406	7.73	4.179	
Operating:											
Mining	1.80	0.973	1.55	0.838	1.40	0.757	1.30	0.703	1.15	0.622	9-19
Hauling	0.75	0.405	0.75	0.405	0.75	0.405	0.75	0.405	0.75	0.405	9-19
Milling	6.59	3.562	5.28	2.854	4.22	2.281	3.70	2.000	3.13	1.692	9-19
Royalty 3/	0.525	0.284	0.475	0.257	0.416	0.225	0.394	0.213	0.370	0.200	9-19
Total Operating	9.66	5.224	8.05	4.354	6.79	3.668	6.14	3.321	5.40	2.919	
Total Costs	19.26	10.413	17.21	9.300	15.24	8.236	14.29	7.727	13.13	7.098	
Cash Flow Rate of Return, % 4/											
Price:											
$8/# U_3O_8	-		-		-		1.6		5.1		
$9	-		-		4.1		6.7		9.7		
$10	-		3.5		8.4		10.7		13.4		
$11	2.8		7.3		12.1		14.1		16.4		
$12	6.5		10.5		15.1		16.9		18.8		
$13	9.5		13.3		17.8		19.2		20.7		
$14	12.2		15.8		19.8		21.0		22.4		
$15	14.7		18.0		21.6		22.7		24.0		

1/ At 24 ore depth to ore thickness ratio.
2/ Some of the mining equipment is included under "Primary Development".
3/ At prices required to generate 15% cash flow rate of return.
4/ Continues cash-flow - continues compounding.

Source: Klemenic 1974 (3)

TABLE A4

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON OPEN PIT MINING1/ OPERATIONS AT 0.05% U_3O_8 IN ORE

	1,000 T/D		2,000 T/D		3,000 T/D		5,000 T/D		10,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs											
Capital:											
Field Expense	0.09	0.103	0.09	0.103	0.09	0.103	0.09	0.103	0.09	0.103	1-5
Property Acquisition	0.07	0.080	0.07	0.080	0.07	0.080	0.07	0.080	0.07	0.080	1-5
Exploration Drilling	0.12	0.137	0.12	0.137	0.12	0.137	0.12	0.137	0.12	0.137	3-6
Development Drilling	0.04	0.046	0.04	0.046	0.04	0.046	0.04	0.046	0.04	0.046	5-7
Mine Primary Development	5.28	6.034	4.80	5.486	4.58	5.234	4.58	5.234	4.44	5.074	8-17
Mine Plant & Equipment 2/	0.18	0.206	0.17	0.194	0.16	0.183	0.16	0.183	0.16	0.183	8,14
Mill Construction	1.81	2.069	1.47	1.680	1.30	1.486	1.12	1.280	0.91	1.040	8,9
Total Capital	7.59	8.675	6.76	7.726	6.36	7.269	6.18	7.063	5.83	6.663	
Operating:											
Mining	0.80	0.914	0.70	0.800	0.65	0.743	0.60	0.686	0.55	0.629	9-19
Hauling	0.75	0.857	0.75	0.857	0.75	0.857	0.75	0.857	0.75	0.857	9-19
Milling	5.10	5.829	4.04	4.617	3.52	4.023	2.95	3.371	2.29	2.617	9-19
Royalty 3/	0.163	0.186	0.140	0.160	0.133	0.152	0.121	0.138	0.109	0.124	9-19
Total Operating	6.81	7.786	5.63	6.434	5.05	5.775	4.42	5.052	3.70	4.227	
Total Costs	14.40	16.461	17.39	14.160	11.41	13.044	10.60	12.115	9.53	10.890	

Cash Flow Rate of Return, %:4/	1,000 T/D	2,000 T/D	3,000 T/D	5,000 T/D	10,000 T/D
Price:					
$12/ U_3O_8	-	-	-		5.5
13	-	-	-	4.2	9.1
14	-		3.5	7.7	12.4
15	-	3.9	7.5	10.8	15.2
17.50	3.9	11.0	13.3	17.2	21.1
20	10.1	16.7	18.7	22.3	25.4
22.50	15.2	21.4	23.0	26.1	28.7
25	19.6	25.2	26.2	29.1	31.5

1/ At 24 ore depth to ore thickness ratio.
2/ Some of the mining equipment is included under "Primary Development"
3/ At price required to generate 15% cash-flow rate of return.
4/ Continuous cash-flow - continuous compounding.

Source: Klemenic 1974(3)

TABLE A5

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON OPEN PIT MINING1/ OPERATIONS AT 0.025% U_3O_8 IN ORE

	2,000 T/D		3,000 T/D		5,000 T/D		10,000 T/D		20,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs											
Capital:											
Field Expense	0.030	0.077	0.030	0.077	0.030	0.077	0.030	0.077	0.030	0.077	1-5
Property Acquisition	0.020	0.052	0.020	0.052	0.020	0.052	0.020	0.052	0.020	0.052	1-5
Exploration Drilling	0.035	0.090	0.035	0.090	0.035	0.090	0.035	0.090	0.035	0.090	3-6
Development Drilling	0.010	0.026	0.010	0.026	0.010	0.026	0.010	0.026	0.010	0.026	5-7
Mine Primary Development	4.80	12.371	4.56	11.75	4.22	10.87	4.22	10.87	4.22	10.876	8-17
Mine Plant & Equipment 2/	0.17	0.438	0.16	0.425	0.16	0.405	0.15	0.389	0.15	0.386	8,14
Mill Construction	1.42	3.660	1.26	3.247	1.08	2.782	0.88	2.266	0.71	1.830	8,9
Total Capital	6.48	16.704	6.08	15.667	5.55	14.302	5.34	13.770	5.17	13.337	
Operating:											
Mining	0.55	1.417	0.50	1.288	0.45	1.159	0.40	1.031	0.35	0.902	9-19
Hauling	0.75	1.933	0.75	1.933	0.75	1.933	0.75	1.933	0.75	1.933	9-19
Milling	3.96	10.208	3.43	8.840	2.86	7.371	2.20	5.670	1.67	4.304	9-19
Royalty 3/	0.057	0.148	0.053	0.137	0.047	0.122	0.042	0.107	0.039	0.100	9-19
Total Operating	5.32	13.706	4.73	12.198	4.11	10.585	3.39	8.741	2.81	7.239	
Total Costs	11.80	30.410	10.81	27.865	9.66	24.887	8.73	22.511	7.98	20.576	
Cash Flow Rate of Return, %: 4/											
Price:											
$20/# U_3O_8	-		-		-		-		-		
25	-		-				6.8		11.6		
30			5.1		10.9		16.0		20.6		
35	8.5		12.9		18.4		23.3		27.6		
40	15.1		19.3		24.6		29.2		32.6		
45	20.7		24.8		29.6		33.4		37.6		
50	25.7		29.5		33.5		37.2		40.2		
55	29.8		33.0		36.9		40.4		43.3		

1/ At 24 ore depth to ore thickness ratio.
2/ Some of the mining equipment is included under "Primary Development".
3/ At price required to generate 15% cash flow rate of return.
4/ Continuous cash flow - continuous compounding.

Source: Klemenic 1974(3)

TABLE A6

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON OPEN PIT MINING1/ OPERATIONS AT 0.01% U_3O_8 IN ORE

	5,000 T/D		10,000 T/D		20,000 T/D		30,000 T/D		50,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs											
Capital:											
Field Expense	0.006	0.063	0.006	0.063	0.006	0.063	0.006	0.063	0.006	0.063	1-5
Property Acquisition	0.004	0.042	0.004	0.042	0.004	0.042	0.004	0.042	0.004	0.042	1-5
Exploration Drilling	0.008	0.084	0.008	0.084	0.008	0.084	0.008	0.084	0.008	0.084	3-6
Development Drilling	0.002	0.021	0.002	0.021	0.002	0.021	0.002	0.021	0.002	0.021	5-7
Mine Primary Development	4.22	44.421	3.84	40.421	3.84	40.421	3.84	40.421	3.84	40.421	8-17
Mine Plant & Equipment	0.15	1.579	0.14	1.474	0.14	1.474	0.14	1.474	0.14	1.474	8, 14
Mill Construction	1.06	11.158	0.87	9.158	0.70	7.368	0.62	6.526	0.53	5.579	8, 9
Total Capital	5.45	57.368	4.87	51.263	4.70	49.473	4.62	48.631	4.53	47.684	
Operating:											
Mining	0.35	3.684	0.30	3.158	0.30	3.158	0.25	2.632	0.25	2.632	9-19
Hauling	0.75	7.895	0.75	7.895	0.75	7.895	0.75	7.895	0.75	7.895	9-19
Milling	2.79	29.368	2.13	22.421	1.60	16.842	1.33	14.000	1.05	11.053	9-19
Royalty 3/	0.016	0.165	0.014	0.143	0.012	0.129	0.012	0.122	0.011	0.114	9-19
Total Operating	3.91	41.112	3.19	33.617	2.66	28.024	2.34	24.649	2.06	21.694	
Total Costs	9.36	98.480	8.06	84.880	7.36	77.497	6.96	73.280	6.59	69.368	

Cash Flow Rate of Return, %: 4/	5,000 T/D	10,000 T/D	20,000 T/D	30,000 T/D	50,000 T/D
Price:					
$80/# U_3O_8	-	-	3.2	6.5	9.8
100	2.2	10.4	15.8	19.0	22.3
120	12.2	20.0	25.4	28.6	32.0
140	20.2	28.0	33.4	36.2	39.1
150	23.8	31.7	36.5	39.3	42.2
160	27.2	34.6	39.4	42.2	45.1
180	33.1	39.8	44.7	47.4	50.2
200	37.8	44.6	49.4	51.9	54.6

1/ At 24 ore depth to ore thickness ratio.
2/ Some of the mining equipment is included under "Primary Development".
3/ At price required to generate 15% cash flow rate of return.
4/ Continuous cash flow - continuous compounding.

Source: Klemenic 1974 (3)

TABLE A7

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON UNDERGROUND MINING 1/ OPERATIONS AT 0.10% U_3O_8 IN ORE

	500 T/D		1,000 T/D		2,000 T/D		3,000 T/D		5,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/ Rec.	Incurred
Costs											
Capital:											
Field Expense	0.30	0.162	0.30	0.162	0.30	0.162	0.30	0.162	0.30	0.162	1-5
Property Acquisition	0.24	0.130	0.24	0.130	0.24	0.130	0.24	0.130	0.24	0.130	1-5
Exploration Drilling	0.40	0.216	0.40	0.216	0.40	0.216	0.40	0.216	0.40	0.216	3-6
Development Drilling	0.12	0.065	0.12	0.065	0.12	0.065	0.12	0.065	0.12	0.065	5-7
Mine Primary Development	4.10	2.216	4.10	2.160	3.50	1.892	3.19	1.724	2.96	1.600	8-10
Mine Plant & Equipment	0.78	0.422	0.78	0.422	0.64	0.346	0.59	0.319	0.55	0.297	11,16
Mill Construction	2.36	1.270	1.92	1.040	1.56	0.840	1.38	0.750	1.18	0.640	9.10
Total Capital	8.30	4.481	7.86	4.195	6.76	3.651	6.22	3.366	5.75	3.110	
Operating:											
Mining	7.90	4.271	7.00	3.784	6.30	3.406	5.90	3.189	5.50	2.973	11-20
Hauling	0.75	0.405	0.75	0.405	0.75	0.405	0.75	0.405	0.75	0.405	11-20
Milling	6.59	3.562	5.28	2.854	4.22	2.281	3.70	2.000	3.13	1.692	11-20
Royalty 2/	0.703	0.380	0.642	0.347	0.566	0.306	0.524	0.283	0.470	0.254	11-20
Total Operating	15.94	8.618	13.67	7.390	11.84	6.398	10.87	5.877	9.85	5.324	
Total Costs	24.24	13.099	21.53	11.585	18.60	10.049	17.09	9.243	15.60	8.434	
Cash Flow Rate of Return, % 3/											
Price:											
$ 8/ U_3O_8	-		-		-		-				
9	-		-		-				6.2		
10	-		-				4.7		8.9		
11	-				5.2		7.6		10.9		
12	-		3.3		7.9		10.1		12.7		
13			6.0		10.1		11.9		14.3		
14	4.5		8.3		11.9		13.4		15.7		
15	6.8		10.3		13.4		14.9		17.0		

1/ At 76 ore depth to ore thickness ratio.
2/ At price to generate 15% cash flow rate of return.
3/ Continuous cash flow - continuous compounding.

Source: Klemenic 1974 (3)

TABLE A8

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON UNDERGROUND MINING 1/ OPERATIONS AT 0.05% U_3O_8 IN ORE

	1,000 T/D		2,000 T/D		3,000 T/D		5,000 T/D		10,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs:											
Capital:											
Field Expense	0.09	0.103	0.09	0.103	0.09	0.103	0.09	0.103	0.09	0.103	1-5
Property Acquisition	0.07	0.080	0.07	0.080	0.07	0.080	0.07	0.080	0.07	0.080	1-5
Exploration Drilling	0.12	0.137	0.12	0.137	0.12	0.137	0.12	0.137	0.12	0.137	3-6
Development Drilling	0.04	0.046	0.04	0.046	0.04	0.046	0.04	0.046	0.04	0.046	5-7
Mine Primary Development	2.66	3.040	2.36	2.700	2.13	2.434	1.98	2.263	1.82	2.080	8-10
Mine Plant & Equipment	0.48	0.549	0.42	0.48	0.36	0.411	0.34	0.389	0.30	0.343	11,10
Mill Construction	1.81	2.070	1.47	1.680	1.30	1.480	1.12	1.280	0.91	1.040	9,10
Total Capital	5.27	6.025	4.57	5.226	4.11	4.691	3.76	4.298	3.35	3.829	
Operating											
Mining	4.70	5.371	4.20	4.800	3.90	4.457	3.55	4.057	3.15	3.600	11-20
Hauling	0.75	0.857	0.75	0.857	0.75	0.857	0.75	0.857	0.75	0.857	11-20
Milling	5.10	5.829	4.04	4.617	3.52	4.023	2.95	3.371	2.29	2.617	11-20
Royalty 2/	0.191	0.218	0.171	0.195	0.151	0.171	0.137	0.156	0.120	0.137	11-20
Total Operating	10.74	12.275	9.16	10.469	8.32	9.508	7.39	8.441	6.31	7.211	
Total Costs	16.01	18.300	13.73	15.695	12.43	14.199	11.15	12.379	9.66	11.040	
Cash Flow Rate of Return, %: 3/											
Price:											
$12/# U_3O_8	-		-		-		-		5.5		
13	-		-		-		3.1		8.2		
14	-		-				5.9		10.6		
15	-				4.4		8.3		12.7		
17.50			6.7		10.0		13.3		16.5		
20	5.8		11.8		14.3		16.7		19.5		
22.50	10.0		16.0		17.2		19.4		22.1		
25	13.6		19.1		19.7		21.8		24.3		

1/ At 76 ore depth to ore thickness ratio.
2/ At price required to generate 15% cash flow rate of return.
3/ Continuous Cash Flow - Continuous Compounding.

Source: Klemenic 1974 (3)

TABLE A9

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON UNDERGROUND MINING1/ OPERATIONS AT 0.025% U_3O_8 IN ORE

	2,000 T/D		3,000 T/D		5,000 T/D		10,000 T/D		20,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs:											
Capital:											
Field Expense	0.03	0.077	0.03	0.077	0.03	0.077	0.03	0.077	0.03	0.077	1-5
Property Acquisition	0.02	0.052	0.02	0.052	0.02	0.052	0.02	0.052	0.02	0.052	1-5
Exploration Drilling	0.035	0.090	0.035	0.090	0.035	0.090	0.035	0.090	0.035	0.090	3-6
Development Drilling	0.01	0.026	0.01	0.026	0.01	0.026	0.01	0.026	0.01	0.026	5-7
Mine Primary Development	2.20	5.671	1.67	4.305	1.52	3.918	1.29	3.325	1.14	2.939	8-10
Mine Plant & Equipment	0.37	0.954	0.27	0.695	0.23	0.593	0.21	0.541	0.19	0.490	11,16
Mill Construction	1.42	3.670	1.26	3.250	1.08	2.780	0.88	2.270	0.71	1.840	9,10
Total Capital	4.09	10.540	3.29	8.495	2.93	7.536	2.48	6.381	2.14	5.514	
Operating											
Mining	3.01	7.759	2.81	7.243	2.55	6.573	2.31	5.954	2.05	5.284	11-20
Hauling	0.75	1.933	0.75	1.933	0.75	1.933	0.75	1.933	0.75	1.933	11-20
Milling	3.96	10.208	3.43	8.840	2.86	7.371	2.20	5.670	1.67	4.304	11-20
Royalty 2/	0.062	0.160	0.055	0.138	0.047	0.122	0.040	0.104	0.034	0.089	11-20
Total Operating	7.78	20.060	7.05	18.154	6.21	15.999	5.30	13.661	4.50	11.610	
Total Costs	11.87	30.600	10.34	26.649	9.14	23.535	7.78	20.042	6.64	17.124	
Cash Flow Rate of Return, %: 3/											
Price:											
$20/# U_3O_8			-		-				8.6		
25			-		4.9		11.1		16.3		
30			7.2		11.8		17.2		21.1		
35	7.8		12.9		17.0		21.3		25.0		
40	12.6		17.4		20.6		24.7		28.2		
45	16.4		20.6		23.7		27.7		31.0		
50	19.2		23.4		26.4		30.2		33.4		
55	21.7		25.1		28.8		32.5		35.6		

1/ At 76 ore depth to ore thickness ratio.
2/ At price required to generate .5% cash flow rate of return.
3/ Continuous Cash Flow - Continuous Compounding.

Source: Klemenic 1974 (3)

TABLE A10

TYPICAL OVERALL ECONOMICS FOR A FUTURE CYCLE OF URANIUM CONCENTRATE PRODUCTION BASED ON UNDERGROUND MINING1/ OPERATIONS AT 0.01% U_3O_8 IN ORE

	5,000 T/D		10,000 T/D		20,000 T/D		30,000 T/D		50,000 T/D		Years'
	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	$/T	$/# Rec.	Incurred
Costs:											
Capital:											
Field Expense	0.006	0.063	0.006	0.063	0.006	0.063	0.006	0.063	0.006	0.063	1-5
Property Acquisition	0.004	0.042	0.004	0.042	0.004	0.042	0.004	0.042	0.004	0.042	1-5
Exploration Drilling	0.008	0.084	0.008	0.084	0.008	0.084	0.008	0.084	0.008	0.084	3-6
Development Drilling	0.002	0.021	0.002	0.021	0.002	0.021	0.002	0.021	0.002	0.021	5-7
Mine Primary Development	1.06	11.182	1.06	11.182	1.06	11.182	0.99	10.444	0.87	9.178	5-10
Mine Plant & Equipment	0.16	1.688	0.16	1.688	0.16	1.688	0.14	1.477	0.12	1.266	11,16
Mill Construction	1.06	11.190	0.87	9.110	0.70	7.400	0.62	6.550	0.53	5.620	9,10
Total Capital	2.30	24.270	2.11	22.190	1.94	20.480	1.77	18.681	1.54	16.274	
Operating:											
Mining	2.00	21.098	1.80	18.988	1.60	16.878	1.50	15.824	1.40	14.769	11-20
Hauling	0.75	7.912	0.75	7.912	0.75	7.912	0.75	7.912	0.75	7.912	11-20
Milling	2.79	29.368	2.13	22.421	1.60	16.842	1.33	14.000	1.05	11.053	11-20
Royalty 2/	0.014	0.147	0.012	0.128	0.011	0.113	0.010	0.101	0.008	0.089	11-20
Total Operating	5.55	58.525	4.69	49.449	3.96	41.745	3.59	37.837	3.21	33.823	
Total Costs	7.85	82.795	6.80	71.639	5.90	62.225	5.36	57.518	4.75	50.097	
Cash Flow Rate of Return, %: 3/											
Price:											
$ 80/# U_3O_8			7.2		12.6		16.2		19.9		
100	10.3		15.8		19.9		22.6		26.2		
120	17.3		21.5		25.1		27.8		31.3		
140	22.2		26.0		29.4		32.1		35.5		
150	24.2		28.0		31.4		34.0		37.4		
160	26.1		29.8		33.2		35.8		39.2		
180	29.6		33.2		36.5		39.0		43.2		
200	38.6		36.3		39.4		41.9		45.1		

1/ At 76 ore depth to ore thickness ratio.
2/ At price required to generate 15% cash flow rate of return.
3/ Continuous cash flow - continuous compounding.

Source: Klemenic 1974(3)

APPENDIX B

Program User's Guide

PRICE.FOR USER'S GUIDE

The following is the User's Guide for the program PRICE.FOR written in January 1978 by Trevor R. Ellis, Department of Mineral Economics, Colorado School of Mines.

Introduction

The program generates the minimum dollar per pound price of uranium oxide contained in yellowcake concentrate, necessary for economical continuation of exploration and development work towards conventional mining of a sandstone-type uranium deposit.

For input data on a given tonnage and grade of deposit, the program will generate the uranium oxide selling price necessary for specified deposit depth to thickness ratios (D/T) in order to yield various Discounted Cash Flow Rates or Return on project investment.

The major purpose for this program is to generate the required yellowcake selling price for a wide range of deposits. It is envisaged that the output data will later be graphed. The graphs will then provide the minimum range of economically explorable sandstone-type uranium deposits at a given uranium oxide selling price and minimum required project Discounted Cash Flow Rate of Return (DCFROR) on investment.

The following are the more important options:

1. Either open pit or underground conventional mining, but not in-situ leaching (solution mining).

2. Either immediate expensing of costs against income from outside the project or accumulation for expensing when income from the mine becomes available.

3. Either straight line depreciation or double declining balance depreciation - switching to straight line when beneficial.

4. Up to two periods of both price escalation and cost escalation. These do not have to be matching in any way.

If an escalation rate is applied, it is necessary to understand that the DCFROR will not be in constant dollar terms. For example, if both the price and cost escalation rates are 5 percent per annum, a DCFROR of 15 percent will be approximated by a constant dollar DCFROR of 9.5 percent (reference: Stermole, Franklin J., Economic Evaluation and Investment Decision Methods, 2nd Ed., Investment Evaluations Corp., Golden, Colo., 1977, pp. 164-177).

5. A much quicker, but slightly less accurate method of price calculation is used (primarily in the subroutine 'INCOME') for the underground case if all of the following apply:

a) No escalation.

b) No investment tax credit.

c) Straight line depreciation.

d) No mine development after the mine start-up.

Expensed exploration costs are not recaptured from the depletion allowance in this case. Also, mine plant and equipment purchases are assumed equally spaced, and mill depreciation life is assumed to be at least as long as the mine life.

General points to note are:

1. No mill depreciation is taken after the end of the last year of mining.

2. No salvage is taken. Mill salvage is considered to offset mill site reclamation.

3. Year 0 costs are assumed to be zero.

Major Steps in the Program

The Main Program: This requests the name of the file containing the file names to be typed in. Over two steps it reads in the names of the files required for the first run. Also, data contained in two of the input files is read in.

The first of these two files contains general data applicable to most operations and therefore rarely requires changing (e.g. taxation specifications and cost indexes). The second one contains the cost data specific to the tonnage and grade of deposit being analyzed on the first run.

The main program then adjusts the input data into forms appropriate for use in the subroutines. It then selects either the subroutine 'UNDER' for underground mining or 'OPEN' for open pit mining.

The main program is returned to at the end of the run to write out the results and for automatic rerunning if more data files are listed in the file of file names. The file containing the general data is not reread if automatic rerunning takes place.

Subroutine UNDER and Subroutine OPEN: One of these two subroutines is chosen dependent on whether the mine is underground or open pit.

Two file names are transferred from the main program in the variables 'PUGGEN' and 'PUGYRS'. The data are read in from these two files. These data are specific to either the open pit or underground style of operation.

The first of these files contains parameters for the type of operation (e.g. percentage variation of cost with D/T and reclamation cost rates). The second specifies the time cycles for individual activities within the project. Average annual costs for the respective time periods are calculated from these data.

The subroutine 'DEDUCT' is called and returned from. Then the do-loop which varies the D/T of the deposit is initiated and the cost data adjusted to the applicable ratio. Finally, the subroutine 'INVEST' is called.

Subroutine DEDUCT: The revenue and cost escalation factors are calculated for each year. Then the depreciation and investment tax credit are determined.

Either the subroutine 'OPEN' or 'UNDER' is returned to.

Subroutine INVEST: This subroutine allocates all costs to their applicable years, escalates them if necessary and calculates the annual investment cash flows. It also determines the applicable cost depletion allowance and the total exploration cost write-off to be deducted from the depletion allowance. It then calls the subroutine 'NETPV'.

Subroutine NETPV: This subroutine has a do-loop which varies the DCFROR. It does all of its calculations within this do-loop.

Based on the investment cash flows and the DCFROR, a number of simplifying assumptions are used to derive a starting price for the uranium oxide selling price. the subroutine 'INCOME' is then called to calculate the present value at the year 0 for the income cash flows generated during the operating years of the mine based on this price.

The present value of the investment cash flows is subtracted from the present value of the income cash flows to provide the Net Present Value. The price is adjusted and passed through the subroutine 'INCOME' until the Net Present Value approximates to 0. When this occurs interpolation is carried out to obtain the final price which is accepted as the required price for the specified D/T and DCFROR.

The do-loop control statements are then returned to for the procedure to be rerun for the given range of DCFROR's at the given range of D/T's. This occurs until either a specified maximum price is encountered, or the appropriate negative Flag occurs.

These negative flags are essential for the successful operation of the output to file sequence on return to the main program. The output sequence occurs immediately after exiting from the do-loop which varies the D/T.

Subroutine INCOME: This subroutine initially calculates the annual income cash flows for the years in which mining takes place based on the price forwarded from the subroutine 'NETPV'.

These calculations are done by two methods:

1. The cash flows are individually calculated for each year. This is the method generally used.
2. The cash flow for the first year is taken as being the same as those for all of the operating years. This simplified method is only used for the underground case

and then only if all of the limitations mentioned in part 5. of the Introduction apply.

The cash flows are present valued and this value returned to the subroutine 'NETPV'.

<u>File Structure</u>

1. <u>File of File Names</u> (e.g. FILE.NAM)

This file contains the names of the input and output files. It is the name of this file which must be entered when the program requests 'ENTER THE NAME OF THE FILE CONTAINING THE FILE NAMES:'.

Example File	Variable Name
PARAM1.GEN	INPRAM
P03000.025	INCOST
C03000.025	OUTFIL
PIT01.GEN	PUGGEN
PIT01.YRS	PUGYRS
1.0	FLAG(4)
U02000.050	INCOST
G02000.050	OUTFIL
UG01.GEN	PUGGEN
UG01.YRS	PUGYRS
-1.0	FLAG(4)

The variables named in this file are as follows:

INPRAM: The file containing the general parameters applicable to most deposits.

INCOST: The cost data file.

OUTFIL: The results output file.

PUGGEN: The parameters file specific to underground or open pit.

PUGYRS: The file containing the time ranges of the activities.

FLAG(4): A positive number indicates the program is to be rerun using the files listed below. A negative number stops the program.

2. General Parameter File (e.g. PARAM1.GEN)

This file contains parameters which will require the least alteration. These figures are appropriate to both the open pit and underground cases.

Example File	Variable Name
JAN. 1974	DATEC
SEPT 1977	DATEP
348.0	PARAM(1)
526.6	PARAM(2)
132.2	PARAM(3)
197.8	PARAM(4)
12.0	PARAM(5)
22.0	PARAM(6)
50.0	PARAM(7)
10.0	PARAM(8)
70.0	PARAM(9)
56.0	PARAM(10)
100.0	PARAM(11)
50.0	PARAM(12)
20	MILDEP
-1.0	FLAG(1)
10.0,12.0,15.0,-1.0	ROR
0.01,0.025,0.05,0.10,0.20,0.25	AVEGRD
47.5,77.5,87.5,92.5,94.0,95.0	MILREC
10.0,7.0,6.0,5.0	ESCALN

The following describes the variables named in this file:

DATEC: The date at which the cost inputs were taken.

DATEP: The date at which the price output is valid.

PARAM(1): The Marshall and Swift Mining and Milling Equipment Cost Index (M&S Index) at DATEC.

PARAM(2): The M&S Index at DATEP.

PARAM(3): The Wholesale Price Index for Industrial Commodities (WPI Index) at DATEC.

PARAM(4): The WPI Index at DATEP.

PARAM(5): Royalty (percentage of mine mouth ore value).

PARAM(6): Percentage depletion allowance.

PARAM(7): Effective tax rate (percent) for combined federal state and local taxes.

PARAM(8): Investment tax credit (percent).

PARAM(9): The percentage of mill cost qualifying for investment tax credit.

PARAM(10): The D/T of field expense and drilling cost input.

PARAM(11): The maximum price to be calculated in dollars per pound of uranium oxide.

PARAM(12): The increments to be used in calculating the price (cents per pound). Since interpolation is applied, 50.0 is recommended.

MILDEP: Mill depreciation life in years. Integer.

FLAG(1): A positive number for straight line depreciation; a negative for double declining balance - switching to straight line when beneficial.

ROR: Required DCFROR. Dimension 6. A negative flag is required. The values should be entered in the order of lowest to highest.

AVEGRD: The average ore grade in percent U_3O_8. Dimension 6.

MILREC: The percentage mill recovery. Dimension 6. Real. These values must be in the order which matches that of the average ore grade.

ESCALN: Escalation rate in percent. Dimension 4. The first and second period revenue escalation rates are (1) and (2) respectively, while (3) and (4) are the first and second periods of cost escalation.

3. <u>Cost Data File</u> (e.g. P02000.050)

Example File	Variable Name
1.0	FLAG(2)
2000.0	TPD
0.05	AVGRD
0.036	COSTPT(1)
0.028	COSTPT(2)
0.09	COSTPT(3)
0.04	COSTPT(4)
4.80	COSTPT(5)
0.17	COSTPT(6)
1.47	COSTPT(7)
0.70	COSTPT(8)
0.75	COSTPT(9)
4.04	COSTPT(10)

The following describes the variables named in this file:

FLAG(2): A negative number for underground; positive for open pit.

TPD: Mine daily capacity in short tons per calendar day.

AVGRD: The average ore grade in percent U_3O_8.

All of the following COSTPT are in average dollars per short ton of ore produced during the mine life.

COSTPT(1): Field expense.

COSTPT(2): Property acquisition.

COSTPT(3): Exploration drilling.

COSTPT(4): Development drilling.

COSTPT(5): Mine primary development.

COSTPT(6): Mine plant and equipment. Most of the plant and equipment cost for mine primary development is assumed to be included in the mine primary development cost.

COSTPT(7): Mill construction.

COSTPT(8): Mining.

COSTPT(9): Haulage.

COSTPT(10): Milling.

4. <u>Open Pit General Parameters File</u> (e.g. PIT01.GEN)

Example File	Variable Name
-1.0	FLAG(3)
24.0	PITDAT(1)
100.0	PITDAT(2)
100.0	PITDAT(3)
100.0	PITDAT(4)
50.0	PITDAT(5)
50.0	PITDAT(6)
2.0	PITDAT(7)
6000.0	PITDAT(8)
3000.0	PITDAT(9)
269018.0	PITDAT(10)
576079.0	PITDAT(11)
10.0,15.0,10.0,25.0,30.0, 35.0,40.0,-1.0,	DTREQ

The variables named in this file are described below:

FLAG(3): Negative when external taxable income is available, positive when none.

PITDAT(1): D/T of the open pit input cost data.

PITDAT(2): The increase in field expense and drilling cost with doubling of D/T (percent).

PITDAT(3): The increase in mine primary development cost with doubling of D/T (percent).

PITDAT(4): The increase in backfilling cost with doubling of D/T (percent).

PITDAT(5): The percentage of the final pit to be backfilled.

PITDAT(6): The backfilling cost in cents per cubic yard.

PITDAT(7): The amount of area surrounding the pit to be reclaimed in terms of times pit surface area.

PITDAT(8): The reclamation cost for the pit surface after backfilling (dollars per acre).

PITDAT(9): The reclamation cost for the land surrounding the pit (dollars per acre).

PITDAT(10) and PITDAT(11): The coefficients for the environmental impact statement cost formula, which is of the form: PITDAT(10) * ALOG10(TPD) - PITDAT(11).

DTREQ: The D/T's for which the price is required. Dimension 40. Five numbers per line. Order from smallest to largest. The last number must be a negative flag.

5. Underground General Parameters File (e.g. UG01.GEN)

Example File	Variable Name
-1.0	FLAG(3)
76.0	UGDATA(1)
100.0	UGDATA(2)
100.0	UGDATA(3)
12.0	UGDATA(4)
5.0	UGDATA(5)
179345.0	UGDATA(6)
384053.0	UGDATA(7)
25.0	UGDATA(8)
25.0,30.0,35.0,40.0,50.0,	DTREQ
60.0,70.0,80.0,90.0,100.0	
120.0,140.0,160.0,180.0,200.0,	
240.0,280.0,320.0,360.0,400.0,	
-1.0,	

The following describes the variables mentioned in the above file:

FLAG(3): A negative number when external taxable income is available; positive when none.

UGDATA(1): The D/T of the underground input cost data.

UGDATA(2): The percentage increase in the field expense and drilling cost with doubling of D/T.

UGDATA(3): The percentage increase in mine primary development cost with doubling of D/T.

UGDATA(4): The percentage increase in mining cost with doubling of D/T.

UGDATA(5): The percentage increase in mining cost due to MESA regulations.

UGDATA(6) and UGDATA(7): The coefficients for the environmental impact statement cost formula, which is of the form: UGDATA(6) * ALOG10(TPD) - UGDATA(7).

DTREQ: The D/T's for which the price is required. Dimension 40. Five numbers per line. Order from smallest to largest. The last number must be a negative flag.

6. <u>Expenditure Years File</u> (e.g. PIT01.YRS)

NYEAR has a second dimension of 2. Except in the case of NYEAR(10,2) and MPEQYR, the number on the left is the first year in which expenditures are made and the that on the right the last year. The exceptions mentioned are described below. NYEAR and MPEQYR are integers.

Example File	Variable Name
1,2	NYEAR(1,2)
1,2	NYEAR(2,2)
1,3	NYEAR(3,2)
2,4	NYEAR(4,2)
6,7	NYEAR(5,2)
8,17	NYEAR(6,2)
5,5	NYEAR(7,2)
18,18	NYEAR(8,2)
11,20	NYEAR(9,2)
7,7	NYEAR(10,2)
5.5,15.5	YRPDV(2)
6,10,13,-1	MPEQYR

The following describes the variables listed in the above file:

NYEAR(1,2): Field expense.

NYEAR(2,2): Property acquisition.

NYEAR(3,2): Exploration drilling.

NYEAR(4,2): Development drilling.

NYEAR(5,2): Mill construction.

NYEAR(6,2): Operating costs.

NYEAR(7,2): Environmental impact statement.

NYEAR(8,2): Backfilling after mining ceases.

NYEAR(9,2): Reclamation. The final year of reclamation is assumed to be the final year for the entire project in the open pit case.

NYEAR(10,2): The end of the first period of price escalation is (10,1) and (10,2) is the end of the first period of cost escalation. In each case the figure 0 should be inserted if no escalation is to be applied.

YRPDV: Mine primary development. Dimension 2. Since this is real, it should be noted that, for example, 8.0 will be taken as the beginning of year 8 as opposed to the entire year 8 period as in the case of an integer 8 used in NYEAR.

MPEQYR: Mine plant and equipment purchase dates. Dimension 5. One line only. Enter from earliest to latest. A negative flag is required. It is assumed that mine plant and equipment is not purchased before the mine primary development begins.

The underground case is as above except that items NYEAR(8,2) and NYEAR(9,2) do not exist. Blank lines should not be left in their place.

7. Output File (e.g. C03000.025)

An example of an output file is shown as Table 4 on page 55.

The output file format has been designed for ease of reading by plotting programs. FLAG(2), FLAG(3) and FLAG(1) are written in this order in the top left hand corner.

A -1.0 is written in the column for 'Depth to Thickness Ratio' if a negative flag is encountered for DTREQ. Similarly, a -1.0 is written in the column for 'Price' if the calculations were stopped due to the maximum price being exceeded.

APPENDIX C

The FORTRAN Computer Program

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C                                     PRICE.FOR
C                                     ---------
C
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C   COLORADO SCHOOL OF MINES, JANUARY 1978.
C
C
C     THIS PROGRAM GENERATES THE MINIMUM DOLLAR PER POUND PRICE OF
C   URANIUM OXIDE (YELLOWCAKE) NECESSARY FOR ECONOMICAL CONTINUATION
C   OF EXPLORATION AND DEVELOPMENT WORK TOWARDS CONVENTIONAL MINING OF
C   SANDSTONE-TYPE URANIUM DEPOSIT.
C
C     THE PROGRAM CONTAINS SIX SUBROUTINES.  THESE ARE OPEN, UNDER,
C   DEDUCT, INVEST, NETPV AND INCOME.
C
C     DETAILS ABOUT THE PROGRAM, INPUTS REQUIRED, AND VARIABLE
C   NAMES ARE LISTED IN THE USER'S GUIDE, APPENDIX B.
C
C
C                                   MAIN PROGRAM
C
C
C     INITIALLY A REQUEST IS MADE FOR THE NAME OF THE FILE CONTAINING THE
C   FILE NAMES TO BE TYPED.  FROM THIS FILE THE NAMES OF THE FILES
C   REQUIRED FOR THE FIRST RUN ARE READ.  THE DATA CONTAINED IN THE
C   GENERAL PARAMETER FILE AND THE COST FILE ARE THEN READ IN.
C
C     THE INPUT DATA IS ADJUSTED FOR USE IN THE SUBROUTINES.  THEN THE
C   SUBROUTINE 'UNDER' IS SELECTED FOR UNDERGROUND MINING OR 'OPEN' FOR
C   OPEN PIT MINING.
C
C     THE MAIN PROGRAM IS RETURNED TO AT THE END OF THE RUN TO WRITE OUT
C   THE RESULTS AND FOR AUTOMATIC RERUNNING IF MORE FILES ARE LISTED.
C
C
      DIMENSION ANCOST(11), AVEGRD(6), COSTPT(11), ESC(4), ESCALN(4),
     1          FLAG(7), PARAM(12), ROR(6), ROR1(6), DTREQ(40),
     2          NYEAR(10,2), PRICE(5,40)
C
      REAL  MILREC(6)
C
      DOUBLE PRECISION  INFILE, INPRAM, INCOST, OUTFIL, DATEC,
     1          DATEP, PUGGEN, PUGYRS
C
      TYPE 10
 10   FORMAT (' ENTER THE NAME OF THE FILE CONTAINING THE ',
     1 'FILE NAMES:        ',$)
      ACCEPT 20, INFILE
 20   FORMAT (A10)
      OPEN (UNIT = 1, FILE = INFILE)
C
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C     READ FROM THE FILE CONTAINING THE FILE NAMES, THE NAME OF THE
C   GENERAL PARAMETER FILE:
C
      READ (1,20) INPRAM
C
C     DATEC = DATE COST INPUTS TAKEN; DATEP = DATE PRICE OUTPUT VALID
C   FOR.
C
      OPEN (UNIT = 2, FILE = INPRAM)
      READ (2,20) DATEC, DATEP
C
C     INPUT 12 GENERAL PARAMETERS:
C
      READ (2,40) (PARAM(I), I = 1,12)
 40   FORMAT (F)
C
C     DIVIDE BOTH PERCENTAGES AND CENTS BY 100:
C
      DO 50 I = 5,9
        PARAM(I) = PARAM(I) / 100.
 50   CONTINUE
      PARAM(12) = PARAM(12) / 100.
C
C     INPUT MILL DEPRECIATION LIFE (YEARS):
C
      READ (2,55) MILDEP
 55   FORMAT (I)
C
C     INPUT FLAG(1).  POSITIVE FOR STRAIGHT LINE DEPRECIATION; NEGATIVE
C   FOR DOUBLE DECLINING BALANCE WITH SWITCHING TO STRAIGHT LINE WHEN
C   BENEFICIAL.
C
      READ (2,40) FLAG(1)
C
C     INPUT THE REQUIRED RATES OF RETURN (PERCENT) WITH A NEGATIVE FLAG:
C
      READ (2,60) (ROR1(I), I = 1,6)
 60   FORMAT (6F)
C
C     DIVIDE THESE PERCENTAGES BY 100:
C
      DO 70 I = 1,6
        ROR(I) = ROR1(I) / 100.
 70   CONTINUE
C
C     READ IN THE POSSIBLE AVERAGE ORE GRADES (MAX 6):
C
      READ (2,60) (AVEGRD(I), I = 1,6)
C
C     READ IN THE CORRESPONDING MILL RECOVERIES (MAX 6):
C
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      READ (2,60) (MILREC(I), I = 1,6)
C
C     INPUT THE PRICE AND COST ESCALATION RATES, THEN DIVIDE BY 100:
C
      READ (2,60) (ESCALN(I), I = 1,4)
C
      FLAG(5) = 0.
      DO 80 I = 1,4
        IF (ESCALN(I) .GT. 0.) FLAG(5) = 2.
        ESCALN(I) = 1. + ESCALN(I) / 100.
 80   CONTINUE
C
      CLOSE (UNIT = 2)
C
C     FROM THE FILE OF FILE NAMES INPUT THE NAMES OF THE COST DATA
C   FILE, OUTPUT DATA FILE AND THE GENERAL DATA FILES.
C
C     LINE 100 IS RETURNED TO AUTOMATICALLY FROM LINE 700 IN ORDER
C   TO RERUN THE PROGRAM USING NEW INPUT DATA.
C
 100  READ (1,20) INCOST, OUTFIL, PUGGEN, PUGYRS
C
C     FROM THE COST DATA FILE INPUT THE COST DATA:
C
      OPEN (UNIT = 3, FILE = INCOST)
C
      READ (3,40) FLAG(2), TPD, AVGRD
      READ (3,40) (COSTPT(I), I = 1,10)
C
      CLOSE (UNIT = 3)
C
C     SET ALL PRICE TO 0.
C
      DO 110 I = 1,5
        DO 110 J = 1,40
          PRICE(I,J) = 0.0
 110  CONTINUE
C
C     ADJUST THE COST DATA UP FROM THE DATE TAKEN (DATEC) TO THE NEW
C   DATE (DATEP) BY APPLYING THE PRICE INDICES.
C
C     APPLY THE MARSHALL & SWIFT MINING AND MILLING INDICES TO THE
C   COSTS OF MINE PRIMARY DEVELOPMENT, PLANT & EQUIPMENT, AND MILL
C   CONSTRUCTION:
C
      DO 120 I = 5,7
        COSTPT(I) = COSTPT(I) * PARAM(2) / PARAM(1)
 120  CONTINUE
C
C     APPLY THE WHOLESALE PRICE INDICES FOR INDUSTRIAL COMMODITIES TO
C   THE REMAINING COST DATA (COSTPT)
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C
      DO 130 I = 1,4
        COSTPT(I) = COSTPT(I) * PARAM(4) / PARAM(3)
 130  CONTINUE
C
      DO 140 I = 8,10
        COSTPT(I) = COSTPT(I) * PARAM(4) / PARAM(3)
 140  CONTINUE
C
C    MULTIPLY ALL COST DATA BY TPD*365 TO GET THE EQUIVALENT
C  ANNUAL COST (ANCOST), WHERE TPD IS MINE DAILY CAPACITY AND 365
C  IS THE NUMBER OF DAYS IN A YEAR.
C
      DO 150 I = 1,10
        ANCOST(I) = COSTPT(I) * TPD * 365.
 150  CONTINUE
C
C    MATCH THE APPROPRIATE MILL RECOVERY TO THE ORE GRADE (WHERE
C  'RECMIL' IS THE APPLICABLE MILL RECOVERY):
C
      DO 160 I = 1,6
        IF (AVGRD .EQ. AVEGRD(I)) RECMIL = MILREC(I)
 160  CONTINUE
C
      RECMIL = RECMIL / 100.
      AVGRD = AVGRD / 100.
C
C    SELECT EITHER THE SUBROUTINE APPROPRIATE FOR AN
C  UNDERGROUND OR OPEN PIT MINE:
C
 180  IF (FLAG(2) .GT. 0.) GO TO 210
C
 200  CALL UNDER (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM, PUGGEN,
     1           PUGYRS, RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR, PRICE,
     2           TOTTON)
C
      GO TO 300
C
 210  CALL OPEN (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM, PUGGEN,
     1           PUGYRS, RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR, PRICE,
     2           TOTTON)
C
C    AFTER RETURN FROM THE SUBROUTINES WRITE THE RESULTS TO THE OUTPUT
C  FILE:
C
 300  OPEN (UNIT = 7, FILE = OUTFIL)
C
      WRITE (7,305) FLAG(2), FLAG(3), FLAG(1)
 305  FORMAT (F4.1,',',F4.1,',',F4.1)
C
      IF (FLAG(2) .GT. 0.0) GO TO 320
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C
      WRITE (7,310) DATEP
 310  FORMAT ('UNDERGROUND MINE AT  ',A10)
      GO TO 340
C
 320  WRITE (7,330) DATEP
 330  FORMAT ('OPEN PIT MINE AT',6X,A10)
C
 340  WRITE (7,350) PARAM(2)
 350  FORMAT (5X,'MARSHALL & SWIFT INDEX (MINING & ',
     1  'MILLING)',4X,F5.1)
C
      WRITE (7,360) PARAM(4)
 360  FORMAT (5X,'WHOLESALE PRICE INDEX (INDUSTRIALS)',
     1  10X,F5.1)
C
      WRITE (7,370) DATEC
 370  FORMAT (/,'INPUT COSTS VALID AT  ',A10)
C
      WRITE (7,350) PARAM(1)
      WRITE (7,360) PARAM(3)
C
      WRITE (7,380) TOTTON
 380  FORMAT (/,'TOTAL ORE RESERVES (MILLION S.TON)',16X,F7.3)
C
      AVGRD  = AVGRD * 100.
      WRITE (7,390) AVGRD
 390  FORMAT ('AVERAGE ORE GRADE (URANIUM OXIDE %)',17X,F5.3)
C
      WRITE (7,400) TPD
 400  FORMAT ('ORE PRODUCTION RATE (TON PER CALENDAR DAY)',
     1  6X,F6.0)
C
      WRITE (7,405) MLIFE
 405  FORMAT ('MINE OPERATING LIFE (YEARS)',24X,I2)
C
      LEADYR = NYEAR(6,1) - 1
      WRITE (7,407) LEADYR
 407  FORMAT ('YEARS LEAD TIME, YEAR 0 TO START-UP',16X,I2)
C
      WRITE (7,410) NYEAR(10,1), NYEAR(10,2)
 410  FORMAT (/,'FIRST PERIOD PRICE AND COST ESCALATION (YEARS)',5X,
     1  I2,',',I3)
C
      DO 411 I = 1,4
 411    ESC(I) = (ESCALN(I) - 1.) * 100.
C
      WRITE (7,412) (ESC(I), I = 1,4)
 412  FORMAT (5X,'PRICE ESCALATION RATES (PERCENT)',14X,F5.2,',',
     1  F5.2,/,5X,'COST  ESCALATION RATES (PERCENT)',14X,F5.2,',',F5.2)
C
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      IF (FLAG(3) .GT. 0.0) GO TO 420
C
      WRITE (7,415)
 415  FORMAT (/,'OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE ',
     1  'TAX BENEFITS AGAINST.')
C
      GO TO 435
C
 420  WRITE (7,430)
 430  FORMAT (/,'OTHER INCOME IS NOT AVAILABLE FOR TAKING IMMEDIATE ',
     1  'TAX BENEFITS AGAINST.')
C
 435  IF (FLAG(1) .LT. 0.) GO TO 445
C
      WRITE (7,440)
 440  FORMAT (/,'STRAIGHT LINE DEPRECIATION.')
C
      GO TO 450
C
 445  WRITE (7,447)
 447  FORMAT (/,'DOUBLE DECLINING BALANCE DEPRECIATION WITH ',
     1  'SWITCHING TO STRAIGHT LINE.')
C
 450  WRITE (7,455)
 455  FORMAT (/,8X,'DEPTH TO',10X,'RATE OF RETURN',8X,
     1  'REQUIRED PRICE',/5X,'THICKNESS RATIO',8X,'(PERCENT)',
     2  13X,'$/LB U. OX.',/)
C
C    DO-LOOP TO PRINT OUT PRICE FIGURES:
C
 460  DO 505 J = 1,40
        DO 500 I = 1,6
C
C    LEAVE DO-LOOP IF NEGATIVE DTREQ FLAG ENCOUNTERED.  THEN
C  WRITE A NEGATIVE FLAG IN THE DTREQ COLUMN:
C
          IF (DTREQ(J) .LT. 0.0) GO TO 510
C
C    LEAVE BLANK LINE AFTER EACH GROUP OF ROR'S BY USING THE
C  NEGATIVE ROR FLAG:
C
          IF (ROR1(I) .GE. 0.) GO TO 480
            WRITE (7,470)
 470        FORMAT (' ')
C
        GO TO 505
C
 480        WRITE (7,490) DTREQ(J), ROR1(I), PRICE(I,J)
 490    FORMAT (9X,F5.1,16X,F5.2,16X,F6.2)
C
C    LEAVE THE DO-LOOP IF MAXIMUM PRICE LIMIT ENCOUNTERED
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```
C  FOR ROR(1):
C
           IF (PRICE(1,J).GE.PARAM(11)) GO TO 530
C
 500  CONTINUE
 505  CONTINUE
C
C
 510  WRITE (7,520)
 520  FORMAT (10X,'-1.0')
C
      GO TO 600
C
 530  WRITE (7,540)
 540  FORMAT (52X,'-1.0')
C
 600  CLOSE (UNIT = 7)
C
C     READ IN FLAG TO DECIDE WHETHER TO RETURN FOR MORE FILES:
C
 700  READ (1,40) FLAG(4)
      IF (FLAG(4) .GT. 0.0) GO TO 100
C
      STOP
      END
```

```
C                              SUBROUTINE OPEN
C                              ---------------
C
C
C     THIS SUBROUTINE IS SIMILAR TO THE SUBROUTINE UNDER.  IT:
C
C      (1)  READS FROM TWO FILES CONTAINING DATA APPROPRIATE TO
C      UNDERGROUND MINING.  THESE FILES CONTAIN:
C            (A)  GENERAL DATA;
C            (B)  TIME PERIODS FOR THE VARIOUS ACTIVITIES.
C
C      (2)  ADJUSTS DRILLING AND EXPLORATION COSTS TO THE DEPTH TO
C      THICKNESS RATIO (D/T) OF THE OTHER INPUT DATA.
C
C      (3)  ADJUSTS ALL COSTS TO AVERAGE ANNUAL COSTS FOR THEIR
C      RESPECTIVE TIME PERIODS.
C
C      (4)  CALLS THE SUBROUTINE 'DEDUCT' AND IS RETURNED TO FROM IT.
C
C      (5)  INITIATES THE DO-LOOP WHICH VARIES THE D/T AND ADJUSTS
C      THE COSTS TO THE RESPECTIVE D/T'S.  YRCOST'S ARE AVERAGE ANNUAL
C      COSTS CORRECTED TO THEIR RESPECTIVE TIME PERIODS AND D/T.
C
C
      SUBROUTINE OPEN (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM,
     1          PUGGEN, PUGYRS, RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR,
     2          PRICE, TOTTON)
C
      DOUBLE PRECISION  PUGGEN, PUGYRS
C
      DIMENSION  ANCOST(11), COSTES(35), DEPREC(35), DTREQ(40),
     1          ESCALN(4), FLAG(7), JYRPDV(2), KP(5), MPEQYR(5),
     2          NYEAR(10,2), PARAM(12), PITDAT(11), PRICE(5,40),
     3          REVESC(35), ROR(6), TAXCR(35), YRCOST(13), YRPDV(2)
C
C     READ IN THE GENERAL DATA:
C
      OPEN (UNIT = 8, FILE = PUGGEN)
C
      READ (8,20) FLAG(3), (PITDAT(I), I = 1,11)
 20   FORMAT (F)
C
C     READ IN THE REQUIRED DEPTH TO THICKNESS RATIOS:
C
      DO 30 I = 1,40
 30     DTREQ(I) = 0.0
      READ (8,40, END = 50) (DTREQ(I), I = 1,40)
 40   FORMAT (5F)
 50   CONTINUE
C
      CLOSE (UNIT = 8)
```

```
C
C     READ IN THE TIME PERIOD DATA:
C
      OPEN (UNIT = 9, FILE = PUGYRS)
C
      READ (9,60)  ((NYEAR(I,J), J = 1,2), I = 1,10)
 60   FORMAT (2I)
C
      READ (9,80) (YRPDV(I), I = 1,2)
 80   FORMAT (2F)
C
      READ (9,100) (MPEQYR(I), I = 1,5)
 100  FORMAT (5I)
C
      CLOSE (UNIT = 9)
C
C     COUNTER NYRS IS USED TO DETERMINE HOW MANY TIMES PLANT &
C   EQUIPMENT IS PURCHASED FOR THE MINE.
C
      DO 110 I = 1,5
        IF (MPEQYR(I) .LE. 0) GO TO 120
        NYRS = I
 110  CONTINUE
C
C     DIVIDE PERCENTAGES AND CENTS BY 100:
C
 120  DO 130 I = 2,6
        PITDAT(I) = PITDAT(I) / 100.
 130  CONTINUE
C
C     ADJUST THE COSTS OF FIELD EXPENSE AND DRILLING TO THE DEPTH TO
C   THICKNESS RATIO OF THE OTHER DATA:
C
      KP(1) = 1
      KP(2) = 3
      KP(3) = 4
C
      DTADJ = 1. +((PITDAT(1) - PARAM(10)) / PARAM(10)) * PITDAT(2)
      DO 140 I = 1,3
        ANCOST(KP(I)) = DTADJ * ANCOST(KP(I))
 140  CONTINUE
C
C     CALCULATE THE ANNUAL COST OF PREPARING THE ENVIRONMENTAL IMPACT
C   STATEMENT AND OBTAINING THE APPROPRIATE PERMITS:
C
      YRCOST(13) = PITDAT(10) * ALOG10(TPD) - PITDAT(11)
      YRCOST(13) = YRCOST(13) / (NYEAR(7,2) - NYEAR(7,1) + 1)
C
C     DETERMINE THE MINE OPERATING LIFE:
C
      MLIFE = NYEAR(6,2) - NYEAR(6,1) + 1
```

```
C
C     ANNUAL BACKFILLING COST:
C
       ANCOST(11) = 9000000. * TPD * PITDAT(5) * PITDAT(6) / 500.
       ANCOST(11) = ANCOST(11) / (NYEAR(8,2) - NYEAR(8,1) + 1)
C
C     ANNUAL RECLAMATION COST:
C
       YRCOST(12) = 100. * TPD  * (PITDAT(8) + PITDAT(9) *
      1          PITDAT(7)) / 500.
       YRCOST(12) = YRCOST(12) / (NYEAR(9,2) - NYEAR(9,1) + 1)
C
C     DETERMINE THE COST OF THE PLANT & EQUIPMENT PURCHASES IN
C   EACH OF THE PURCHASING YEARS:
C
       YRCOST(6) = ANCOST(6) * MLIFE / NYPS
C
C     CALCULATE THE FRACTIONS OF YEARS OF MINE PRIMARY
C   DEVELOPMENT AT BOTH THE BEGINNING AND END OF DEVELOPMENT:
C
       JYRPDV(1) = IFIX(YRPDV(1))
       JYRPDV(2) = IFIX(YRPDV(2))
       YFPDV = YRPDV(1) - JYRPDV(1)
       YLPDV = YRPDV(2) - JYRPDV(2)
       JYRPDV(1) = JYRPDV(1) + 1
       JYRPDV(2) = JYRPDV(2) + 1
C
C     TOTAL TONNAGE OF ORE IN MILLION SHORT TONS:
C
       TOTTON = TPD * 365. * MLIFE / 1000000.
C
C     ADJUST THE REMAINING ANCOST'S TO TRUE AVERAGE ANNUAL COSTS FOR
C    THEIR RESPECTIVE TIME PERIODS:
C
       DO 150 I = 1,4
 150     ANCOST(I) = ANCOST(I) * MLIFE / (NYEAR(I,2) - NYEAR(I,1) + 1)
       YRCOST(2) = ANCOST(2)
       ANCOST(5) = ANCOST(5) * MLIFE / (YRPDV(2) - YRPDV(1))
       YRCOST(7) = ANCOST(7) * MLIFE / (NYEAR(5,2) - NYEAR(5,1) + 1)
       DO 160 I = 8,10
 160     YRCOST(I) = ANCOST(I)
C
C     OPERATING COST = MINING + HAULAGE + MILLING:
C
       YRCOST(8) = YRCOST(8) + YRCOST(9) + YRCOST(10)
C
C
       CALL DEDUCT  (ANCOST, ESCALN, FLAG, MILDEP, MLIFE, MPFQYR,
      1          NYEAR, NYRS, PARAM, YRCOST, YRPDV, COSTES, DEPREC,
      2          LASTYR, REVESC, TAXCR)
C
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```
C     THIS DO-LOOP IS THE MAIN ONE OF THE PROGRAM.  IT PROVIDES
C   A NEW REQUIRED DEPTH TO THICKNESS RATIO (DT) ON EACH LOOP.
C
 200  DO 300 KDT = 1,40
C
      NDT = KDT
C
C    THIS IF STATEMENT PROVIDES THE RETURN TO THE MAIN PROGRAM:
C
      IF (DTREQ(NDT) .LT. 0.0) GO TO 400
C
      DT = DTREQ(NDT)
C
C     ADJUST THE FOLLOWING COSTS TO THE NEW DEPTH TO THICKNESS RATIO:
C
C   (A)  FIELD EXPENSE, EXPLORATION DRILLING, DEVELOPMENT DRILLING:
C
      DTADJ = 1. + ((DT - PITDAT(1)) / PITDAT(1)) * PITDAT(2)
      DO 210 I = 1,3
        YRCOST(KP(I)) = DTADJ * ANCOST(KP(I))
 210  CONTINUE
C
C   (B)  MINE PRIMARY DEVELOPMENT:
C
      YRCOST(5) = (1.+((DT-PITDAT(1))/PITDAT(1))*PITDAT(3))*ANCOST(5)
C
C   (C)  BACKFILLING OF PIT:
C
      YRCOST(11)=(1.+((DT-PITDAT(1))/PITDAT(1))*PITDAT(4))*ANCOST(11)
C
C
      CALL INVEST (AVGRD, COSTES, DEPREC, DTREQ, FLAG, JYRPDV, KP,
     1          LASTYR, MLIFE, MPEQYR, NDT, NYEAR, NYRS, PARAM, RECMIL,
     2          REVESC, ROR, TAXCR, TPD, YFPDV, YLPDV, YRCOST, PRICE)
C
C     TEST TO DETERMINE IF THE PRICE FOR ROR(1) WAS EQUAL TO OR
C   GREATER THAN THE MAXIMUM ALLOWED:
C
      IF (PRICE(1,NDT) .GE. PARAM(11)) GO TO 400
C
 300  CONTINUE
C
 400  RETURN
C
      END
```

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C                             SUBROUTINE UNDER
C                             ----------------
C
C
C     THIS SUBROUTINE IS SIMILAR TO THE SUBROUTINE OPEN.  IT:
C
C      (1)  READS FROM TWO FILES CONTAINING DATA APPROPRIATE TO
C      UNDERGROUND MINING.  THESE FILES CONTAIN:
C            (A)  GENERAL DATA;
C            (B)  TIME PERIODS FOR THE VARIOUS ACTIVITIES.
C
C      (2)  ADJUSTS DRILLING AND EXPLORATION COSTS TO THE DEPTH TO
C      THICKNESS RATIO (D/T) OF THE OTHER INPUT DATA.
C
C      (3)  ADJUSTS ALL COSTS TO AVERAGE ANNUAL COSTS FOR THEIR
C      RESPECTIVE TIME PERIODS.
C
C      (4)  CALLS THE SUBROUTINE 'DEDUCT' AND IS RETURNED TO FROM IT.
C
C      (5)  INITIATES THE DO-LOOP WHICH VARIES THE D/T AND ADJUSTS
C      THE COSTS TO THE RESPECTIVE D/T'S.  YRCOST'S ARE AVERAGE ANNUAL
C      COSTS CORRECTED TO THEIR RESPECTIVE TIME PERIODS AND D/T.
C
C
      SUBROUTINE UNDER (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM,
     1          PUGGEN, PUGYRS, RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR,
     2          PRICE, TOTTON)
C
      DOUBLE PRECISION  PUGGEN, PUGYRS
C
      DIMENSION  ANCOST(11), COSTES(35), DEPREC(35), DTREQ(40),
     1          ESCALN(4), FLAG(7), JYRPDV(2), KP(5), MPEQYR(5),
     2          NYEAR(10,2), PARAM(12), PRICE(5,40), REVESC(35), ROR(6),
     3          TAXCR(35), UGDATA(8), YRCOST(13), YRPDV(2)
C
C     READ IN THE GENERAL DATA:
C
      OPEN (UNIT = 8, FILE = PUGGEN)
C
      READ (8,20) FLAG(3), (UGDATA(I), I = 1,8)
 20   FORMAT (F)
C
C     READ IN THE REQUIRED DEPTH TO THICKNESS RATIOS:
C
      DO 30 I = 1,40
 30     DTREQ(I) = 0.0
      READ (8,40, END = 50) (DTREQ(I), I = 1,40)
 40   FORMAT (5F)
 50   CONTINUE
C
      CLOSE (UNIT = 8)
```

```
C
C     READ IN THE TIME PERIOD DATA:
C
      OPEN (UNIT = 9, FILE = PUGYRS)
C
      READ (9,60) ((NYEAR(I,J), J = 1,2), I = 1,7)
      READ (9,60) (NYEAR(10,J), J = 1,2)
 60   FORMAT (2I)
C
C     SET VALUE FOR NYEARS (I = 8,9) TO 0:
C
      DO 70 I = 8,9
        DO 70 J = 1,2
          NYEAR(I,J) = 0
 70   CONTINUE
C
      READ (9,80) (YRPDV(I), I = 1,2)
 80   FORMAT (2F)
C
      READ (9,100) (MPEQYR(I), I = 1,5)
 100  FORMAT (5I)
C
      CLOSE (UNIT = 9)
C
C     COUNTER 'NYRS' IS USED TO DETERMINE HOW MANY TIMES PLANT &
C   EQUIPMENT IS PURCHASED FOR THE MINE.
C
      DO 110 I = 1,5
        IF (MPEQYR(I) .LE. 0) GO TO 120
        NYRS = I
 110  CONTINUE
C
C     DIVIDE PERCENTAGES BY 100:
C
 120  DO 130 I = 2,5
        UGDATA(I) = UGDATA(I) / 100.
 130  CONTINUE
      UGDATA(8) = UGDATA(8) / 100.
C
C     ADJUST COSTS OF FIELD EXPENSE AND DRILLING TO THE DEPTH
C   TO THICKNESS RATIO OF THE OTHER DATA:
C
      KP(1) = 1
      KP(2) = 3
      KP(3) = 4
C
      DTADJ = 1. + ((UGDATA(1) - PARAM(10)) / PARAM(10)) * UGDATA(2)
C
      DO 140 I = 1,3
        ANCOST(KP(I)) = DTADJ * ANCOST(KP(I))
 140  CONTINUE
```

```
C
C     INCREASE MINING COST BY 'MESA' PROPORTION (I.E. THE AMOUNT
C   OF ADDITIONAL COST BROUGHT ON BY THE RECENT STRICT SUPERVISION
C   BY THE 'MINE ENVIRONMENTAL SAFETY ADMINISTRATION').
C
      ANCOST(8) = ANCOST(8) * (1. + UGDATA(5))
C
C     CALCULATE THE ANNUAL COST OF PREPARING THE ENVIRONMENTAL IMPACT
C   STATEMENT AND OBTAINING THE APPROPRIATE PERMITS:
C
      YRCOST(13) = UGDATA(6) * ALOG10(TPD) - UGDATA(7)
      YRCOST(13) = YRCOST(13) / (NYEAR(7,2) - NYEAR(7,1) + 1)
C
C     DETERMINE MINE OPERATING LIFE:
C
      MLIFE = NYEAR(6,2) - NYEAR(6,1) + 1
C
C     DETERMINE THE COST OF THE PLANT & EQUIPMENT PURCHASES IN
C   EACH OF THE PURCHASING YEARS:
C
      YRCOST(6) = ANCOST(6) * MLIFE / NYRS
C
C     CALCULATION OF FRACTIONS OF YEARS OF MINE PRIMARY DEVELOPMENT
C   AT BOTH THE START AND END OF PRIMARY DEVELOPMENT:
C
      JYRPDV(1) = IFIX(YRPDV(1))
      JYRPDV(2) = IFIX(YRPDV(2))
      YFPDV = YRPDV(1) - JYRPDV(1)
      YLPDV = YRPDV(2) - JYRPDV(2)
      JYRPDV(1) = JYRPDV(1) + 1
      JYRPDV(2) = JYRPDV(2) + 1
C
C     THIS IF STATEMENT STOPS ANY DETAILED CALCULATION IN THE SUBROUTINE
C   INCOME CAUSED BY ROUND OFF ERROR WHEN THE MINE PRIMARY DEVELOPMENT
C   STOPS AT MINE START-UP.
C
      IF (YLPDV .LE. 0.001) YLPDV = 0.0
C
C     TOTAL TONNAGE (MILLION S.TONS) OF ORE IN THE GROUND, INCLUDING
C   PILLAR REQUIREMENTS:
C
      TOTTON = TPD * 365. * MLIFE / ((1. - UGDATA(3)) * 1000000.)
C
C     ADJUST THE REMAINING ANCOST'S TO TRUE AVERAGE ANNUAL COSTS FOR
C    THEIR RESPECTIVE TIME PERIODS:
C
      DO 150 I = 1,4
 150    ANCOST(I) = ANCOST(I) * MLIFE / (NYEAR(I,2) - NYEAR(I,1) + 1)
      YRCOST(2) = ANCOST(2)
      ANCOST(5) = ANCOST(5) * MLIFE / (YRPDV(2) - YRPDV(1))
      YRCOST(7) = ANCOST(7) * MLIFE / (NYEAR(5,2) - NYEAR(5,1) + 1)
```

```
      YRCOST(9) = ANCOST(9)
      YRCOST(10) = ANCOST(10)
      YRCOST(11) = 0.
      YRCOST(12) = 0.
C
C
      CALL DEDUCT  (ANCOST, ESCALN, FLAG, MILDEP, MLIFE, MPEQYR,
     1             NYEAR, NYRS, PARAM, YRCOST, YRPDV, COSTES, DEPREC,
     2             LASTYR, REVESC, TAXCR)
C
C    THIS DO-LOOP IS THE MAIN ONE OF THE PROGRAM.  IT PROVIDES
C  A NEW REQUIRED DEPTH TO THICKNESS RATIO (DT) ON EACH LOOP.
C
 200  DO 300 KDT = 1,40
C
      NDT = KDT
C
C    THIS STATEMENT PROVIDES THE RETURN TO THE MAIN PROGRAM:
C
      IF (DTREQ(NDT) .LT. 0.0) GO TO 400
C
      DT= DTREQ(NDT)
C
C    ADJUST THE FOLLOWING COSTS TO THE NEW DEPTH TO THICKNESS RATIO:
C
C  (A) FIELD EXPENSE, EXPLORATION DRILLING, DEVELOPMENT DRILLING:
C
      DTADJ = 1. + ((DT - UGDATA(1)) / UGDATA(1)) * UGDATA(2)
      DO 210 I = 1,3
        YRCOST(KP(I)) = DTADJ * ANCOST(KP(I))
 210  CONTINUE
C
C  (B) MINE PRIMARY DEVELOPMENT:
C
      YRCOST(5) = (1.+((DT-UGDATA(1))/UGDATA(1))*UGDATA(3))*ANCOST(5)
C
C  (C) MINING:
C
      YRCOST(8) = (1.+((DT-UGDATA(1))/UGDATA(1))*UGDATA(4))*ANCOST(8)
C
C    OPERATING COST = MINING + HAULAGE + MILLING:
C
      YRCOST(8) = YRCOST(8) + YRCOST(9) + YRCOST(10)
C
C
      CALL INVEST (AVGRD, COSTES, DEPREC, DTREQ, FLAG, JYRPDV, KP,
     1             LASTYR, MLIFE, MPEQYR, NDT, NYEAR, NYRS, PARAM, RECMIL,
     2             REVESC, ROR, TAXCR, TPD, YFPDV, YLPDV, YRCOST, PRICE)
C
C    ON RETURN FROM THE SUBROUTINES TEST TO DETERMINE IF THE PRICE
C  FOR THE ROR(1) WAS EQUAL TO OR GREATER THAN THE MAXIMUM ALLOWED:
```

```
C
      IF (PRICE(1,NDT) .GE. PARAM(11)) GO TO 400
C
 300  CONTINUE
C
 400  RETURN
C
      END
```

```
C                                   SUBROUTINE DEDUCT
C                                   -----------------
C
C
C     THIS SUBROUTINE CALCULATES FOR EACH YEAR:
C
C      (1)  THE ESCALATION FACTORS APPLICABLE TO BOTH REVENUES
C      (REVESC) AND COSTS (COSTES);
C
C      (2)  THE INVESTMENT TAX CREDIT (TAXCR);
C
C      (3)  THE DEPRECIATION (DEPREC).
C
C
       SUBROUTINE DEDUCT  (ANCOST, ESCALN, FLAG, MILDEP, MLIFE, MPEQYR,
     1           NYEAR, NYRS, PARAM, YRCOST, YRPDV, COSTES, DEPREC,
     2           LASTYR, REVESC, TAXCR)
C
       DIMENSION  ANCOST(11), COSTES(35), DEPREC(35), DPCMIL(35),
     1           DPCPEQ(35), ESCALN(4), FLAG(7), LIFE(4), MPEQYR(5),
     2           NYEAR(10,2), PARAM(12), REVESC(35), TAXCR(35),
     3           YRCOST(13), YRPDV(2)
C
C     A POSITIVE FLAG(6) INDICATES THAT KLEMENIC'S EXPLORATION TIME
C   PERIODS ARE BEING USED BEGINNING AT THE END OF HIS YEAR 3.  IT
C   THEN CAUSES THE PAST 3 YEAR'S COSTS TO BE TAKEN INTO ACCOUNT IN
C   TAXATION DEDUCTION CALCULATIONS IN THE SUBROUTINE 'INVEST'.
C
         FLAG(6) = 0.
       IF ((NYEAR(1,2) - NYEAR(1,1) .EQ. 1) .AND. (NYEAR(2,2) -
     1       NYEAR(2,1) .EQ. 1) .AND. (NYEAR(3,2) - NYEAR(3,1) .EQ. 2))
     2       FLAG(6) = 2.
C
C     FIND THE PROJECT FINAL YEAR (LASTYR):
C
       LASTYR = NYEAR(9,2)
       IF (NYEAR(6,2) .GT. NYEAR(9,2)) LASTYR = NYEAR(6,2)
C
C     SET COSTES AND REVESC TO 1.0 AND DEPREC AND TAXCR TO 0.:
C
       DO 10 I = 1,35
         COSTES(I) = 1.
         DEPREC(I) = 0.
         REVESC(I) = 1.
         TAXCR(I) = 0.
 10    CONTINUE
C
C     IF THE INDICATORS ARE APPROPRIATE, GO TO THE SIMPLIFIED STRAIGHT
C   LINE METHOD FOR THE UNDERGROUND CASE.  A POSITIVE FLAG(7) INDICATES
C   THAT THE SIMPLIFIED METHOD IS BEING USED.
C
```

```
      FLAG(7) = 0.
        Y = FLOAT(NYEAR(6,1))
      IF ((FLAG(2) .LT. 0.) .AND. (FLAG(1) .GT. 0.) .AND. (FLAG(5)
     1       .EQ. 0.) .AND. (PARAM(8) .EQ. 0.) .AND. (YRPDV(2) - 0.001
     2       .LT. Y - 1.)) GO TO 600
C
C     DECIDE IF ESCALATION IS REQUIRED AT ALL:
C
      IF (FLAG(5) .EQ. 0.) GO TO 100
C
C     DECIDE IF PRICE ESCALATION IS REQUIRED:
C
      IF (NYEAR(10,1) .EQ. 0) GO TO 50
C
C     CALCULATE THE PRICE ESCALATION FACTORS FOR THE FIRST AND THEN
C   THE SECOND PERIOD:
C
        N = NYEAR(10,1)
        IF (N .GT. LASTYR) N = LASTYR
        REVESC(1) = ESCALN(1)
      DO 20 I = 2,N
 20     REVESC(I) = REVESC(I-1) * ESCALN(1)
C
      IF (N .EQ. LASTYR) GO TO 50
        N = N + 1
      DO 30 I = N, LASTYR
 30     REVESC(I) = REVESC(I-1) * ESCALN(2)
C
C     DECIDE IF COST ESCALATION IS REQUIRED:
C
 50   IF (NYEAR(10,2) .EQ. 0) GO TO 100
C
C     CALCULATE THE COST ESCALATION FACTORS FOR THE FIRST AND THEN
C   THE SECOND PERIOD:
C
        N = NYEAR(10,2)
        IF (N .GT. LASTYR) N = LASTYR
        COSTES(1) = ESCALN(3)
      DO 60 I = 2,N
 60   COSTES(I) = COSTES(I-1) * ESCALN(3)
C
      IF (N .EQ. LASTYR) GO TO 100
        N = N + 1
      DO 70 I = N, LASTYR
 70     COSTES(I) = COSTES(I-1) * ESCALN(4)
C
C     DETERMINE THE USEFUL LIFE OF THE PLANT & EQUIPMENT (P&E) FOR EACH
C   PURCHASE.  THIS IS USED FOR DETERMINING THE INVESTMENT TAX CREDIT
C   AND DEPRECIATION.  IT IS ASSUMED THAT NO P&E IS PURCHASE BEFORE THE
C   FIRST DEVELOPMENT YEAR.
C
```

```
 100  DO 110 I = 1,4
 110    LIFE(I) = 0
C
      IF (NYRS .EQ. 1) GO TO 125
C
      DO 120 I = 2, NYRS
 120    LIFE(I-1) = MPEQYR(I) - MPEQYR(I-1)
C
 125  LIFE(NYRS) = NYEAR(6,2) - MPEQYR(NYRS) + 1
C
C     CALCULATE THE INVESTMENT TAX CREDIT FOR BOTH THE MILL AND P&E:
C
      IF (PARAM(8) .EQ. 0.) GO TO 200
C
C  (A)  MILL  (CRMILL = MILL TAX CREDIT):
C
        CRMILL = 0.
      DO 130 I = NYEAR(5,1), NYEAR(5,2)
 130    CRMILL = YRCOST(7) * COSTES(I) * PARAM(8) * PARAM(9)
     1          + CRMILL
C
C  (B)  MINE P&E:
C
C     BASED ON THE USEFUL LIFE THE FRACTION OBTAINED OF THE MAXIMUM
C  PERCENTAGE ALLOWABLE IS CALCULATED.
C
      DO 140 I = 1, NYRS
        FRAC = 1.
        IF (LIFE(I) .LE. 6) FRAC = 2. / 3.
        IF (LIFE(I) .LE. 4) FRAC = 1. / 3.
        TAXCR(MPEQYR(I)) = YRCOST(6) * COSTES(MPEQYR(I)) * FRAC
     1          * PARAM(8)
 140  CONTINUE
C
C     ALLOCATE THE INVESTMENT TAX CREDIT DUE TO THE MILL TO THE FIRST
C  OPERATING YEAR:
C
      TAXCR(NYEAR(6,1)) = TAXCR(NYEAR(6,1)) + CRMILL
C
C
C     STRAIGHT LINE DEPRECIATION:
C
 200  IF (FLAG(1) .LT. 0.) GO TO 300
C
C  (A)  MILL  (DEPMIL = ANNUAL MILL DEPRECIATION):
C
C     THE DEPRECIATION IS ACCUMULATED FOR DEDUCTION WHEN MINING BEGINS.
C
        DEPMIL = 0.
      DO 210 I = NYEAR(5,1), NYEAR(5,2)
 210    DEPMIL = YRCOST(7) * COSTES(I) + DEPMIL
```

```
C
      DEPMIL = DEPMIL / MILDEP
C
C  (B)  MINE P&E (DEPPEC = ANNUAL P&E DEPRECIATION):
C
      DO 220 I = 1, NYRS
          DEPPEC = YRCOST(6) * COSTES(MPEQYR(I)) / LIFE(I)
            N = MPEQYR(I) - 1
        DO 220 J = 1, LIFE(I)
          DEPREC(N+J) = DEPPEC
 220  CONTINUE
C
C    SUM THE MILL AND MINE P&E DEPRECIATION OVER THE OPERATING YEARS:
C
        N = NYEAR(6,2) - NYEAR(6,1) + 1
        K = NYEAR(6,2)
      IF (MILDEP .LT. N) K = NYEAR(6,1) + MILDEP - 1
C
      DO 230 I = NYEAR(6,1), K
 230    DEPREC(I) = DEPREC(I) + DEPMIL
C
      GO TO 500
C
C    DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT
C  LINE WHEN BENEFICIAL:
C
C  (A)  MILL  (DPCMIL = ANNUAL MILL DEPRECIATION):
C
 300  DO 310 I = 1,35
 310    DPCMIL(I) = 0.
C
C    CALCULATE THE TOTAL MILL COST (COSTML):
C
        COSTML = 0.
      DO 320 I = NYEAR(5,1), NYEAR(5,2)
 320    COSTML = YRCOST(7) * COSTES(I) + COSTML
C
C    DETERMINE THE ANNUAL DEPRECIATION OVER THE OPERATING YEARS:
C
        N = NYEAR(6,2) - NYEAR(6,1) + 1
        K = NYEAR(6,2)
      IF (MILDEP .LT. N) K = NYEAR(6,1) + MILDEP - 1
C
      DO 330 I = NYEAR(6,1), K
          J = I
          M = MILDEP - (I - NYEAR(6,1))
        COSTML = COSTML - DPCMIL(I-1)
C
C    CHECK IF STRAIGHT LINE DEPRECIATION IS THE GREATEST AND ACCEPT IT
C  IF IT IS:
C
```

```
          DEPMIL = COSTML / M
          DPCMIL(I) = COSTML * 2. / MILDEP
        IF (DEPMIL .GE. DPCMIL(I)) GO TO 340
 330  CONTINUE
C
      GO TO 400
C
 340  DO 350 I = J, K
 350    DPCMIL(I) = DEPMIL
C
C  (3)  P&E  (DPCPEQ = P&E ANNUAL DEPRECIATION):
C
 400  DO 410 I = 1,35
 410    DPCPEQ(I) = 0.
C
      DO 440 I = 1, NYRS
          COSTPE = YRCOST(6) * COSTFS(MPEQYR(I))
        DO 420 K = 1, LIFE(I)
            J = K
            M = LIFE(I) - K + 1
            N = MPEQYR(I) + K - 1
          IF (K .EQ. 1) GO TO 415
          COSTPE = COSTPE - DPCPEQ(N-1)
C
C    ACCEPT STRAIGHT LINE DEPRECIATION IF IT IS GREATEST:
C
 415      DEPPEQ = COSTPE / M
          DPCPEQ(N) = COSTPE * 2. / LIFE(I)
        IF (DEPPEQ .GE. DPCPEQ(N)) GO TO 430
 420  CONTINUE
C
      GO TO 440
C
 430  DO 440 L = J, LIFE(I)
        N = MPEQYR(I) + L - 1
        DPCPEQ(N) = DEPPEQ
 440  CONTINUE
C
C    SUM THE MILL AND P&E DEPRECIATION:
C
      DO 450 I = 1, LASTYR
 450    DEPREC(I) = DPCMIL(I) + DPCPEQ(I)
C
C    IF THERE IS NO INCOME AVAILABLE AGAINST WHICH TO EXPENSE THE
C  DEVELOPMENT YEAR EXPENDITURES, THEN SUM THE INVESTMENT TAX CREDIT
C  FOR THESE YEARS INTO THE FIRST OPERATING YEAR:
C
 500  IF ((FLAG(3) .LT. 0.) .OR. (PARAM(8) .EQ. 0.)) GO TO 520
        N = NYEAR(6,1) - 7
        IF (N .LE. 0) N = 1
        J = NYEAR(6,1) - 1
```

```
      DO 510 I = N, J
        TAXCR(NYEAR(6,1)) = TAXCR(NYEAR(6,1)) + TAXCR(I)
 510  CONTINUE
C
 520  RETURN
C
C
C     SIMPLIFIED STRAIGHT LINE DEPRECIATION:
C
 600  FLAG(7) = 2.
C
C     PLANT & EQUIPMENT DEPRECIATION:
C
      DEPPEQ = ANCOST(6) * MLIFE / (NYEAR(6,2) - MPEQYR(1) + 1)
C
C     MILL DEPRECIATION:
C
        M = MILDEP
      IF (MILDEP .LT. MLIFE) M = MLIFE
      DEPMIL = ANCOST(7) * MLIFE / M
C
C     ALLOCATE TO YEARS:
C
      DO 610 I = MPEQYR(1), NYEAR(6,2)
 610    DEPREC(I) = DEPPEQ
C
      DO 620 I = NYEAR(6,1), NYEAR(6,2)
 620    DEPREC(I) = DEPREC(I) + DEPMIL
C
      RETURN
C
      END
```

```
C                               SUBROUTINE INVEST
C                               -----------------
C
C
C     THIS SUBROUTINE ALLOCATES ALL COSTS TO THEIR APPLICABLE YEARS,
C   ESCALATES THEM IF NECESSARY AND CALCULATES THE ANNUAL INVESTMENT
C   CASH FLOWS.  IT ALSO DETERMINES THE APPLICABLE COST DEPLETION
C   ALLOWANCE AND THE TOTAL EXPLORATION COST WRITE-OFF TO BE DEDUCTED
C   FROM THE DEPLETION ALLOWANCE.  IT THEN CALLS THE SUBROUTINE 'NETPV'.
C
C
      SUBROUTINE INVEST (AVGRD, COSTES, DEPREC, DTREQ, FLAG, JYRPDV,
     1          KP, LASTYR, MLIFE, MPEQYR, NDT, NYEAR, NYRS, PARAM,
     2          RECMIL, REVESC, ROR, TAXCR, TPD, YFPDV, YLPDV, YRCOST,
     3          PRICE)
C
      DIMENSION  COST(13,35), CFINV(35), COSTES(35), DEPREC(35),
     1          DTREQ(40), FLAG(7), JYRPDV(2), KP(5), MPEQYR(5),
     2          NYEAR(10,2), PARAM(12), PRICE(5,40), REVESC(35), ROR(6)
     3          TAXCR(35), YRCOST(13)
C
C     RESET THE ANNUAL COST MATRIX COST(13,35) AND THE ANNUAL
C   INVESTMENT CASH FLOWS CFINV(35) TO 0.
C
      DO 10 J = 1,35
          CFINV(J) = 0.0
        DO 10 I = 1,13
          COST(I,J) = 0.0
 10   CONTINUE
C
C     FOR A NEGATIVE FLAG(3), TAKE THE TAX BENEFIT OF EXPENSING
C   INTANGIBLE FIELD EXPENSE, DRILLING AND BACKFILLING
C   EXPENDITURES AGAINST OTHER COMPANY INCOME IN THE NON-OPERATING
C   YEARS.
C
      IF (FLAG(3) .GT. 0.0) GO TO 30
C
      KP(4) = 11
      KP(5) = 13
C
      EXTAX = 1. - PARAM(7)
C
      DO 20 I = 1,5
        YRCOST(KP(I)) = YRCOST(KP(I)) * EXTAX
 20   CONTINUE
C
C     ALLOCATE COSTS TO THEIR APPLICABLE YEARS:
C
C   (A)  FIELD EXPENSE, PROPERTY ACQUISITION, EXPLORATION AND
C        DEVELOPMENT DRILLING:
C
```

```
 30   DO 40 I = 1,4
        DO 40 J = NYEAR(I,1), NYEAR(I,2)
             COST(I,J) = YRCOST(I)
 40   CONTINUE
C
C  (B)  MILL CONSTRUCTION:
C
      DO 50 J = NYEAR(5,1), NYEAR(5,2)
        COST(7,J) = YRCOST(7)
 50   CONTINUE
C
C  (C)  RECLAMATION IF OPEN PIT:
C
      IF (FLAG(2) .LT. 0.0) GO TO 60
      DO 60 J = NYEAR(9,1), NYEAR(9,2)
        COST(12,J) = YRCOST(12)
 60   CONTINUE
C
C  (D)  MINE PRIMARY DEVELOPMENT:
C
C   1)  FIRST YEAR COST:
C
      COST(5,JYRPDV(1)) = YRCOST(5) * (1. - YFPDV)
C
C   2)  FINAL YEAR COST:
C
      COST(5,JYRPDV(2)) = YRCOST(5) * YLPDV
C
C   3)  INTERMEDIATE YEARS:
C
      M = JYRPDV(2) - 1
      N = JYRPDV(1) + 1
      IF (N .GT. M) GO TO 70
      DO 70 J = N, M
        COST(5,J) = YRCOST(5)
 70   CONTINUE
C
C    IF OTHER INCOME IS AVAILABLE EXPENSE PRIMARY DEVELOPMENT COSTS
C  BEFORE START-UP:
C
      IF (FLAG(3) .GT. 0.0) GO TO 90
C
      N = NYEAR(6,1) - 1
      DO 80 J = JYRPDV(1), N
        COST(5,J) = COST(5,J) * EXTAX
 80   CONTINUE
C
C    SIMILARLY, EXPENSE THE RECLAMATION COSTS FROM AFTER MINING
C  CEASES:
C
      IF (FLAG(2) .LT. 0.0) GO TO 90
```

```
C
      N = NYEAR(6,2) + 1
      DO 90 J = N, NYEAR(9,2)
        COST(12,J) = COST(12,J) * EXTAX
 90   CONTINUE
C
C  (E)  PLANT & EQUIPMENT:
C
      DO 100 N = 1,NYRS
          J = MPEQYR(N)
          COST(6,J) = YRCOST(6)
 100  CONTINUE
C
C  (F)  OPERATING COST:
C
      DO 120 I = 8,10
        DO 120 J = NYEAR(6,1), NYEAR(6,2)
          COST(I,J) = YRCOST(I)
 120  CONTINUE
C
C  (G)  ENVIRONMENTAL IMPACT STATEMENT:
C
      DO 130 J = NYEAR(7,1), NYEAR(7,2)
 130    COST(13,J) = YRCOST(13)
C
C  (H)  BACKFILLING:
C
      IF (FLAG(2) .LT. 0.) GO TO 150
      DO 140 J = NYEAR(8,1), NYEAR(8,2)
 140    COST(11,J) = YRCOST(11)
C
C
C     ESCALATE ALL COSTS:
C
 150  IF (FLAG(5) .EQ. 0.) GO TO 160
      DO 160 J = 1, LASTYR
        DO 160 I = 1,13
          COST(I,J) = COST(I,J) * COSTES(J)
 160  CONTINUE
C
C
C     SUM THE INVESTMENTS TO GET THE ANNUAL INVESTMENT CASH
C  FLOWS (CFINV(J)):
C
C  (A)  FOR YEAR 1 TO THE YEAR BEFORE START-UP:
C
      N = NYEAR(6,1) - 1
      DO 210 J = 1,N
        DO 210 I = 1,7
            CFINV(J) = CFINV(J) + COST(I,J)
 210  CONTINUE
```

```
C
C   ADDITION OF WORKING CAPITAL IN THE YEAR BEFORE START-UP:
C
C     THE WORKING CAPITAL IS TAKEN TO BE 20% OF THE THIRD
C   OPERATING YEAR'S OPERATING COSTS.  THIS IS TO AVOID THE CASE
C   WHERE MINE PRIMARY DEVELOPMENT JUST CARRIES OVER INTO THE EARLY
C   PART OF THE MINING PERIOD.
C
      WC = 0.2 * (COST(5,NYEAR(6,1)+2) + COST(8,NYEAR(6,1)+2))
      CFINV(NYEAR(6,1)-1) = CFINV(NYEAR(6,1)-1) + WC
C
C     ADD ENVIRONMENTAL IMPACT STATEMENT COST:
C
      DO 215 J = NYEAR(7,1), NYEAR(7,2)
 215    CFINV(J) = CFINV(J) + COST(13,J)
C
C  (B)  ADDITION OF PLANT & EQUIPMENT COST IN THE OPERATING
C         YEARS:
C
      DO 220 J = NYEAR(6,1), NYEAR(6,2)
        CFINV(J) = COST(6,J)
 220  CONTINUE
C
C  (C)  ADDITION OF RECLAMATION AND BACKFILLING FOR THE OPEN PIT
C         CASE IN THE YEARS AFTER MINING CEASES:
C
      IF (FLAG(2) .LT. 0.0) GO TO 230
C
      N = NYEAR(6,2) + 1
      DO 230 J = N, NYEAR(9,2)
        CFINV(J) = COST(11,J) + COST(12,J)
 230  CONTINUE
C
C     IF OTHER INCOME IS AVAILABLE, SUBTRACT THE DEPRECIATION AND
C   INVESTMENT TAX CREDIT FOR PLANT & EQUIPMENT IN THE YEARS BEFORE
C   START-UP:
C
      IF ((FLAG(3) .GT. 0.).OR.(MPEQYR(1) .GE. NYEAR(6,1))) GO TO 300
      N = NYEAR(6,1) - 1
      DO 240 J = MPEQYR(1), N
 240    CFINV(J) = CFINV(J) - DEPREC(J) * PARAM(7) - TAXCR(J)
C
C
C     CALCULATION OF THE 'DEFERRED DEVELOPMENT WRITE-OFF' (DEFDWO)
C   IN THE CASE WHERE INTANGIBLE DRILLING AND DEVELOPMENT COST WERE
C   NOT EXPENSED AGAINST OTHER COMPANY INCOME IN THE YEARS BEFORE
C   START-UP.  THE DEFERRED DEVELOPMENT WRITE-OFF AMORTIZES
C   THESE INTANGIBLE EXPENDITURES OVER THE OPERATING LIFE OF THE
C   MINE:
C
 300  DEFDWO = 0.0
```

```
      IF (FLAG(3) .LT. 0.0) GO TO 400
C
      N = NYEAR(6,1) - 1
      DO 310 J = NYEAR(4,1), N
        DEFDWO = DEFDWO + COST(4,J) + COST(5,J) + DEPREC(J)
 310  CONTINUE
      DEFDWO = DEFDWO / MLIFE
C
C
C     CALCULATION OF THE EFFECTS OF THE INVESTMENT PERIOD COSTS ON
C   THE DEPLETION ALLOWANCE:
C
C     CALCULATE TOTAL COST DEPLETION ALLOWANCE (COSDPN):
C
 400  COSDPN = 0.
      N = NYEAR(3,2)
      IF (NYEAR(2,2) .GT. NYEAR(3,2)) N = NYEAR(2,2)
C
      DO 410 J = 1,N
          COSDPN = COSDPN + COST(2,J)
        IF (FLAG(3) .LT. 0.) GO TO 410
          COSDPN = COSDPN + COST(1,J) + COST(3,J)
 410  CONTINUE
C
C     ADD THE COST DEPLETABLE ITEMS OF YEARS -2 TO 0 IF KLEMINIC'S TIME
C   PERIODS ARE BEING USED (IN YEARS -1 AND -2 COST ESCALATION IS NOT
C   ADJUSTED FOR):
C
      IF (FLAG(6) .EQ. 0.) GO TO 420
        COSDPN = COSDPN + YRCOST(2) * 3.
      IF (FLAG(3) .LT. 0.) GO TO 420
        COSDPN = COSDPN + YRCOST(1) * 3. + YRCOST(3)
C
 420  EXPCS = 0.
      IF ((FLAG(7) .GT. 0.) .OR. (FLAG(3) .GT. 0.)) GO TO 500
C
C     SUM THE EXPENSED EXPLORATION COSTS (EXPCS) FOR SUBTRACTION FROM
C   THE DEPLETION ALLOWANCE:
C
      DO 440 J = 1,N
 440    EXPCS = EXPCS + COST(1,J) + COST(3,J)
C
      IF (FLAG(6) .EQ. 0.) GO TO 450
C
      EXPCS = EXPCS + YRCOST(1) * 3. + YRCOST(3)
 450  EXPCS = EXPCS / EXTAX
C
C
 500  CALL NETPV (AVGRD, CFINV, COSDPN, COST, DEFDWO, DEPREC, DTREQ,
     1           EXPCS, FLAG, LASTYR, MLIFE, NDT, NYEAR, PARAM, RECMIL,
     2           REVESC, ROR, TAXCR, TPD, PRICE)
```

```
C
      RETURN
C
      END
```

```
C                                SUBROUTINE NETPV
C                                ----------------
C
C
C     A DO-LOOP WITHIN THIS SUBROUTINE VARIES THE ROR.  IT PASSES A
C   STARTING PRICE FOR URANIUM OXIDE TO THE SUBROUTINE 'INCOME'
C   WHICH RETURNS THE PRESENT VALUE OF THE RESULTANT CASH FLOWS.  THE
C   PRESENT VALUE OF THE INVESTMENT CASH FLOWS IS SUBTRACTED FROM THIS
C   TO GIVE THE NET PRESENT VALUE (NPV).  ITERATION AND FINAL INTER-
C   POLATION TAKES PLACE TO DERIVE THE PRICE WHICH GIVES NPV = 0.
C   THIS PRICE IS THE REQUIRED URANIUM OXIDE PRICE, IN YEAR 0 DOLLARS,
C   NECESSARY FOR THE PROJECT TO BE VIABLE.
C
C
      SUBROUTINE NETPV (AVGRD, CFINV, COSDPN, COST, DEFDWO, DEPREC,
     1          DTREQ, EXPCS, FLAG, LASTYR, MLIFE, NDT, NYEAR, PARAM,
     2          RECMIL, REVESC, ROR, TAXCR, TPD, PRICE)
C
      DIMENSION  CFINV(35), COST(13,35), DEPREC(35), DTREQ(40),
     1          FLAG(7), NYEAR(10,2), PARAM(12), PRICE(5,40),
     2          REVESC(35), ROR(6), TAXCR(35)
C
      REAL  NPV1, NPV2
C
C
C     THIS IS THE STARTING POINT OF THE DO-LOOP WHICH VARIES THE ROR:
C
      DO 300 I = 1,6
        IF (ROR(I) .LT. 0.0) GO TO 400
          DCF = ROR(I)
C
C     PRESENT VALUE THE INVESTMENT CASH FLOWS:
C
      PVINV = 0.0
      DO 10 J = 1, LASTYR
        PVINV = PVINV + CFINV(J) / ((1. + DCF)**J)
 10   CONTINUE
C
C     DECISION AS TO THE REQUIREMENT FOR A NEW STARTING PRICE
C   CALCULATION:
C
C     FOR ROR(1), IF THE CHANGE IN D/T IS WITHIN THE LIMITS OF 0.9 AND
C   1.2 TIMES THE PREVIOUS D/T, PRICE(1,NDT-1) IS USED AS THE STARTING
C   PRICE ON WHICH TO BASE THE PRICE INCREMENTS.  FOR THE OTHER ROR'S,
C   IF THE DIFFERENCE BETWEEN ROR(I) AND ROR(I-1) IS POSITIVE AND LESS
C   THAN 5%, THE PREVIOUS PRICE IS USED AS THE STARTING PRICE.
C
      IF (I .EQ. 1) GO TO 20
      DIFF = (ROR(I) - ROR(I-1)) * 100.
      IF ((DIFF .LE. 5.0) .AND. (DIFF .GT. 0.0)) GO TO 30
C
```

```
 20   IF((I .EQ. 1) .AND. (NDT .GT. 1) .AND. (DTREQ(NDT) /
     1       DTREQ(NDT-1) .LE. 1.2) .AND. (DTREQ(NDT) / DTREQ(NDT-1)
     2       .GE. 0.9)) GO TO 40
C
C    CALCULATION OF A NEW STARTING PRICE FOR URANIUM OXIDE IN $/LB:
C
C    THIS INVOLVES CALCULATING THE ANNUAL INCOME CASH FLOWS REQUIRED
C  TO GIVE NPV = 0 WHEN PRESENT VALUED TO MATCH THE PRESENT VALUE
C  OF THE INVESTMENT CASH FLOWS.  FROM THESE ANNUAL CASH FLOWS AND
C  USING SIMPLIFYING ASSUMPTIONS, AN APPROXIMATION OF THE PRICE REQUIREC
C  TO GENERATE THE INCOME CASH FLOWS IS CALCULATED.
C
C    FUTURE VALUE PVINV TO THE BEGINNING OF THE MINE START-UP YEAR:
C
      FV = PVINV * (1. + DCF) ** (NYEAR(6,1) - 1)
C
C    DETERMINATION OF THE ANNUAL INCOME CASH FLOW REQUIREMENT (AF):
C
      AF = FV * (DCF * (1. + DCF)**MLIFE) / (((1. + DCF)**MLIFE) -1.)
C
C    CALCULATION OF THE NECESSARY ANNUAL GROSS REVENUE (GREV):
C
C    THIS IS DONE FOR THE THIRD OPERATING YEAR.  THERE IS ASSUMED TO BE
C  NO ROYALTY PAYMENT AND THE DEPLETION ALLOWANCE TO BE 50% OF TAXABLE
C  INCOME BEFORE DEPLETION IS TAKEN.  'COSTOP' IS THE OPERATING COST ANI
C  INCLUDES MINE PRIMARY DEVELOPMENT.
C
      N = NYEAR(6,1) + 2
      COSTOP = COST(8,N) + COST(5,N)
      GREV = (AF - DEPREC(N) - DEFDWO) * (1. - 0.5 * PARAM(7))
     1      + COSTOP + DEPREC(N) + DEFDWO
C
C    'SPRICE' IS THE STARTING PRICE IN YEAR 0 DOLLARS:
      SPRICE = GREV / (2000. * RECMIL * AVGRD * TPD * 365.
     1          * REVESC(N))
      GO TO 100
C
 30   SPRICE = PRICE(I-1,NDT)
      GO TO 100
C
 40   SPRICE = PRICE(1,NDT-1)
C
C
 100  CALL INCOME (AVGRD, CFINV, COSDPN, COST, DCF, DEFDWO, DEPREC,
     1          EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL, REVESC,
     2          SPRICE, TAXCR, TPD, PVCF)
C
      NPV1 = 0.
      NPV2 = PVCF - PVINV
C
C    DETERMINE WHETHER TO USE POSITIVE OR NEGATIVE PRICE INCREMENTS,
```

```
C  OR WHETHER TO ACCEPT THE PRICE AS IS.
C
      IF (NPV2) 200, 240, 220
C
C    POSITIVE INCREMENTS USED:
C
 200  SPRICE = SPRICE + PARAM(12)
C
      CALL INCOME (AVGRD, CFINV, COSDPN, COST, DCF, DEFDWO, DEPREC,
     1          EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL, REVESC,
     2          SPRICE, TAXCR, TPD, PVCF)
C
      NPV1 = NPV2
      NPV2 = PVCF - PVINV
      IF (NPV2) 200, 240, 210
C
C    INTERPOLATE:
C
 210  SPRICE = SPRICE - (NPV2 / (NPV2 - NPV1)) * PARAM(12)
      GO TO 240
C
C    NEGATIVE INCREMENTS USED:
C
 220  SPRICE = SPRICE - PARAM(12)
C
      CALL INCOME (AVGRD, CFINV, COSDPN, COST, DCF, DEFDWO, DEPREC,
     1          EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL, REVESC,
     2          SPRICE, TAXCR, TPD, PVCF)
C
      NPV1 = NPV2
      NPV2 = PVCF - PVINV
C
      IF (NPV2) 230, 240, 220
C
C    INTERPOLATE:
C
 230  SPRICE = SPRICE + (NPV2 / (NPV2 - NPV1)) * PARAM(12)
      GO TO 240
C
 240  PRICE(I,NDT) = SPRICE
C
C    TEST IF THE MAXIMUM PRICE TO BE CALCUALTED HAS BEEN EQUALLED
C  OR EXCEEDED FOR ROR(1):
C
      IF (PRICE(1,NDT) .GE. PARAM(11)) GO TO 400
C
 300  CONTINUE
C
 400  RETURN
C
      END
```

```
C                    SUBROUTINE INCOME
C                    -----------------
C
C
C     THIS SUBROUTINE CALCULATES THE ANNUAL INCOME CASH FLOWS
C   DERIVED FROM THE URANIUM OXIDE PRICE TRANSFERRED FROM THE
C   SUBROUTINE 'NETPV'.  IT THEN PRESENT VALUES THESE TO YEAR 0 AND
C   RETURNS THIS VALUE TO THE SUBROUTINE 'NETPV'.
C
C
      SUBROUTINE INCOME (AVGRD, CFINV, COSDPN, COST, DCF, DEFDWO,
     1           DEPREC, EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL,
     2           REVESC, SPRICE, TAXCR, TPD, PVCF)
C
      DIMENSION  CFINV(35), COST(13,35), DEPREC(35), FLAG(7),
     1           NYEAR(10,2), PARAM(12), REVESC(35), TAXCR(35)
C
      REAL  LOSSF, LOSSF1
C
      PVCF = 0.
C
C     DECIDE WHETHER ONLY THE SIMPLIFIED METHOD IS REQUIRED FOR THE
C   UNDERGROUND CASE:
C
      IF (FLAG(7) .GT. 0.) GO TO 400
C
      CSDPN = COSDPN
      EXPLCS = EXPCS
      KOUNT = -1
      LOSSF1 = 0.
      SUMTAX = 0.
      TAXCRF = 0.
C
C     THIS METHOD CALCULATES EACH YEAR'S CASH FLOWS INDIVIDUALLY:
C
 10   DO 200 J = NYEAR(6,1), NYEAR(6,2)
C
      KOUNT = KOUNT + 1
      DEPL = 0.
C
C     LOSS CARRIED FORWARD:
C
      LOSSF = LOSSF1
C
C     DETERMINE THE ANNUAL GROSS REVENUE:
C
      GREV = SPRICE * REVESC(J) * 365. * 2000.0 * TPD * RECMIL * AVGRD
C
C     DETERMINE THE ANNUAL ROYALTY BASED ON THE MINE MOUTH ORE VALUE
C   (I.E. GROSS REVENUE LESS HAULAGE AND MILLING COSTS):
C
```

```
      ROYAL = PARAM(5) * (GREV - COST(9,J) - COST(10,J))
C
C     GROSS REVENUE AFTER ROYLTY:
C
      GRAR = GREV - ROYAL
C
C     TAXABLE INCOME BEFORE DEPLETION:
C
      TIBDPL = GRAR - COST(8,J) - COST(5,J) - COST(12,J) - DEPREC(J)
     1          -DEFDWO - LOSSF
C
C     CALCULATE PERCENTAGE DEPLETION ALLOWANCE:
C
      IF ((TIBDPL .LE. 0.) .OR. (PARAM(6) .EQ. 0.)) GO TO 20
      DEPLPC = PARAM(6) * GRAR
C
C     UPPER LIMIT ON PERCENTAGE DEPLETION ALLOWANCE:
C
      DEPL = 0.5 * TIBDPL
C
C     SELECT THE SMALLER OF THE ABOVE TWO:
C
      IF (DEPLPC .LT. DEPL) DEPL = DEPLPC
C
C     SELECT THE LARGER OF COST AND PERCENTAGE DEPLETION ALLOWANCE:
C
 20   IF (CSDPN .EQ. 0.) GO TO 30
C
      DEPLCS = CSDPN / (MLIFE - KOUNT)
      IF (DEPLCS .GT. DEPL) DEPL = DEPLCS
C
C     FIND THE BALANCE OF COST DEPLETION ALLOWANCE TO BE CARRIED FORWARD:
C
      CSDPN = CSDPN - DEPL
      IF (CSDPN .LT. 0.) CSDPN = 0.
C
C     RECOVER EXPENSED EXPLORATION COSTS FROM THE DEPLETION ALLOWANCE
C   AND CARRY FORWARD ANY BALANCE:
C
 30   IF ((EXPLCS .EQ. 0.) .OR. (DEPL .EQ. 0.)) GO TO 40
C
      DEPL = DEPL - EXPLCS
      IF (DEPL .GE. 0.) EXPLCS = 0.
      IF (DEPL .GE. 0.) GO TO 40
      EXPLCS = ABS(DEPL)
      DEPL = 0.
C
C     TAXABLE INCOME AFTER DEPLETION ALLOWANCE TAKEN:
C
 40   TIADPL = TIBDPL - DEPL
C
```

```
C     IF NO ESPENSING OF TAX LOSSES AGAINST OTHER INCOME IS POSSIBLE,
C   CALCULATE THE LOSS CARRIED FORWARD FOR FUTURE DEDUCTION (LOSSF1):
C
      IF (FLAG(3) .LT. 0.) GO TO 50
      IF (TIADPL .GE. 0.) LOSSF1 = 0.
      IF (TIADPL .GE. 0.) GO TO 50
      LOSSF1 = ABS(TIADPL)
      TIADPL = 0.
C
C     TAX:
C
 50   TAX = TIADPL * PARAM(7)
C
C     IF OTHER INCOME IS AVAILABLE EACH YEAR'S INVESTMENT TAX CREDIT
C   IS TAKEN IMMEDIATELY.  OTHERWISE A MAXIMUM OF 50% OF THE TAX
C   LIABILITY IS TAKEN WITH THE REMAINING INVESTMENT TAX CREDIT
C   BEING CARRIED FORWARD FOR FUTURE DEDUCTION.
C
      IF (PARAM(8) .EQ. 0.) GO TO 100
      IF (FLAG(3) .GT. 0.) GO TO 60
C
      TAX = TAX - TAXCR(J)
      GO TO 100
C
 60   TAXCRF = TAXCR(J) + TAXCRF
      IF ((TAX .EQ. 0.) .OR. (TAXCRF .EQ. 0.)) GO TO 100
        TAX1 = TAX - TAXCRF
      IF (TAX1 .GE. 0.5 * TAX) GO TO 70
        TAX = 0.5 * TAX
        TAXCRF = TAXCRF - TAX
      GO TO 100
C
 70     TAX = TAX1
        TAXCRF = 0.
C
C     NET INCOME CASH FLOW:
C
 100  CASHFL = TIADPL - LOSSF1 - TAX + DEPRFC(J) + DEFDWO + LOSSF
     1          + DEPL
C
C     PRESENT VALUE THE NET INCOME CASH FLOW AND SUM WITH THE
C   PREVIOUS TOTAL:
C
      PVCF = PVCF + CASHFL / ((1.0 + DCF) ** J)
C
C     IF NO EXTERNAL TAXABLE INCOME IS AVAILABLE THEN SUM THE TAX FOR
C   THE FINAL 3 OPERATING YEARS.  THIS WILL BE USED FOR TAKING TAX
C   LOSSES AGAINST ON THE BACKFILLING AND RECLAMATION IN THE 3 YEARS
C   AFTER MINING CEASES.
C
      IF ((NYEAR(6,2) - J .GE. 2) .AND. (FLAG(3) .GT. 0.) .AND.
```

```
     1            (FLAG(2) .GT. 0.)) SUMTAX = SUMTAX + TAX
C
 200  CONTINUE
C
      IF ((FLAG(3) .LT. 0.) .OR. (FLAG(2) .LT. 0.) .OR. (SUMTAX
     1            .EQ. 0.)) GO TO 300
C
C    RECOVER THE ABOVE MENTIONED TAX LOSS FROM BACKFILLING AND RECLA-
C  MATION UP TO THE VALUE OF 'SUMTAX'.  PRESENT VALUE THAT RECOVERED.
C
        M = NYEAR(6,2) + 1
        N = NYEAR(6,2) + 3
C
      DO 210 J = M,N
          TXLOSS = CFINV(J) * PARAM(7)
          SUMTX = SUMTAX - TXLOSS
          IF (SUMTX .LT. 0.) SUMTX = 0.
          TXGAIN = SUMTAX - SUMTX
        PVCF = PVCF + TXGAIN / (1. + DCF) ** J
        IF (SUMTX .EQ. 0.) GO TO 300
          SUMTAX = SUMTX
 210  CONTINUE
C
 300  RETURN
C
 400  CONTINUE
C
C    THE FOLLOWING SIMPLIFIED CALCULATION FOR THE UNDERGROUND CASE
C  ASSUMES CONSTANT INCOME CASH FLOWS FOR THE ENTIRE MINE OPERATING
C  LIFE (I.E. BOTH CONSTANT REVENUES AND COSTS):
C
      GREV = SPRICE * 365. * 2000.0 * TPD * RECMIL * AVGRD
        J= NYEAR(6,1)
      ROYAL = PARAM(5) * (GREV - COST(9,J) - COST(10,J))
      GRAR = GREV - ROYAL
      TIBDPL = GRAR - COST(8,J) - DEPREC(J) - DEFDWD
      IF (PARAM(6).EQ. 0.) GO TO 410
      DEPLPC = PARAM(6) * GRAR
      DEPL = 0.5 * TIBDPL
      IF (DEPL .LT. 0.) DEPL = 0.
      IF (DEPLPC .LT. DEPL) DEPL = DEPLPC
 410  CSDPN = COSDPN / MLIFE
      IF (CSDPN .GT. DEPL) DEPL = CSDPN
      TIADPL = TIBDPL - DEPL
C
C    ANNUAL NET INCOME CASH FLOWS:
C
      ANCF = TIADPL * (1. - PARAM(7)) + DEPREC(J) + DEFDWD + DEPL
C
C    PRESENT VALUE THESE CASH FLOWS FROM OVER THE MINE LIFE
C  TO THE BEGINNING OF THE START-UP YEAR:
```

```
C
      PVANCF=ANCF*((1.+DCF)**MLIFE-1.)/(DCF*(1.+DCF)**MLIFE)
C
C     PRESENT VALUE PVANCF TO YEAR 0:
C
      PVCF = PVANCF / (1. + DCF) ** (NYEAR(6,1) - 1)
C
      RETURN
C
      END
```

APPENDIX D

Example Plots

from Program Output

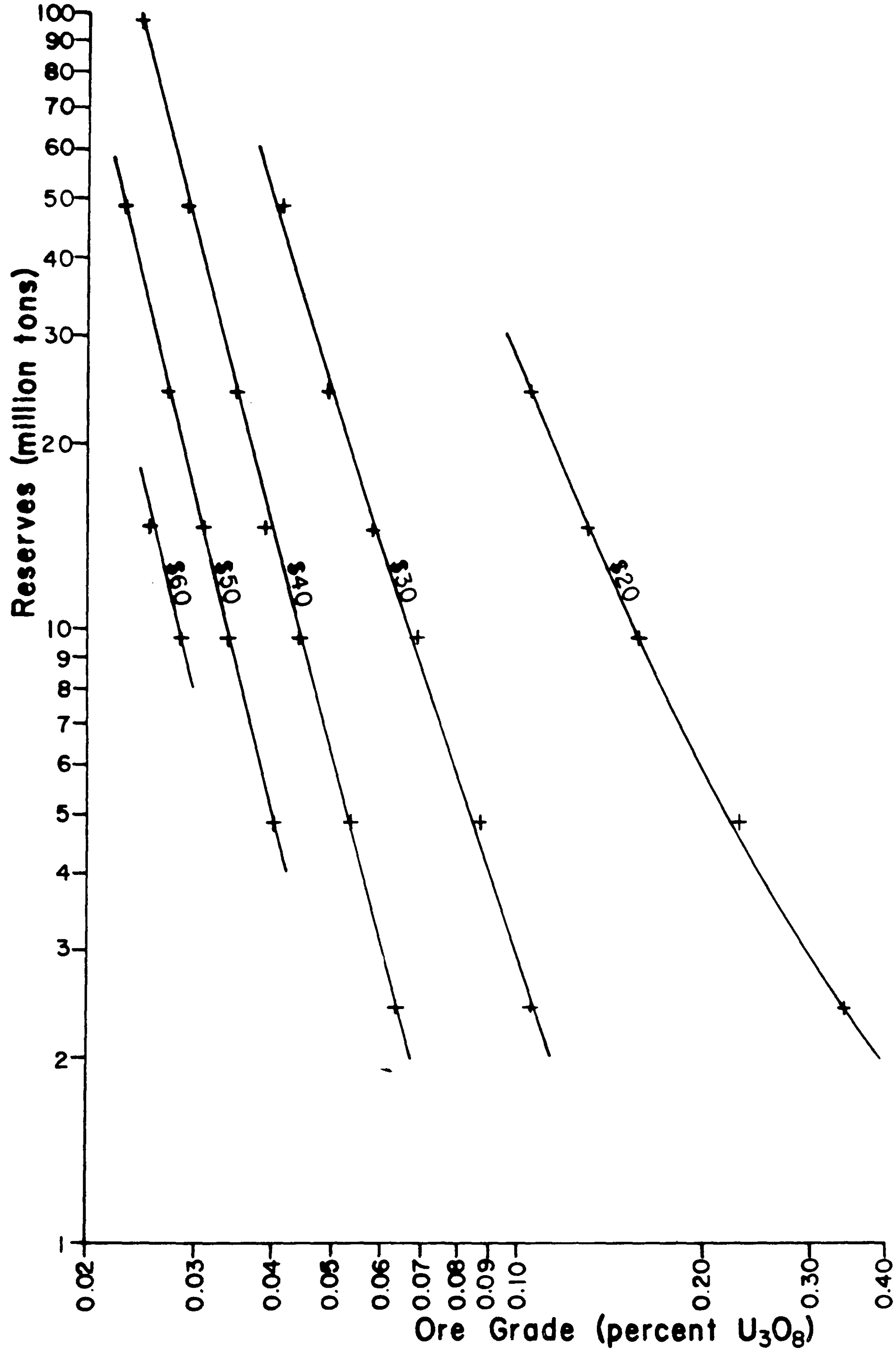

FIGURE DI - RESERVES VERSUS ORE GRADE
UNDERGROUND MINE 15% DCFROR
80 ORE DEPTH TO THICKNESS RATIO
VARYING U_3O_8 SELLING PRICE (DOLLARS PER POUND)

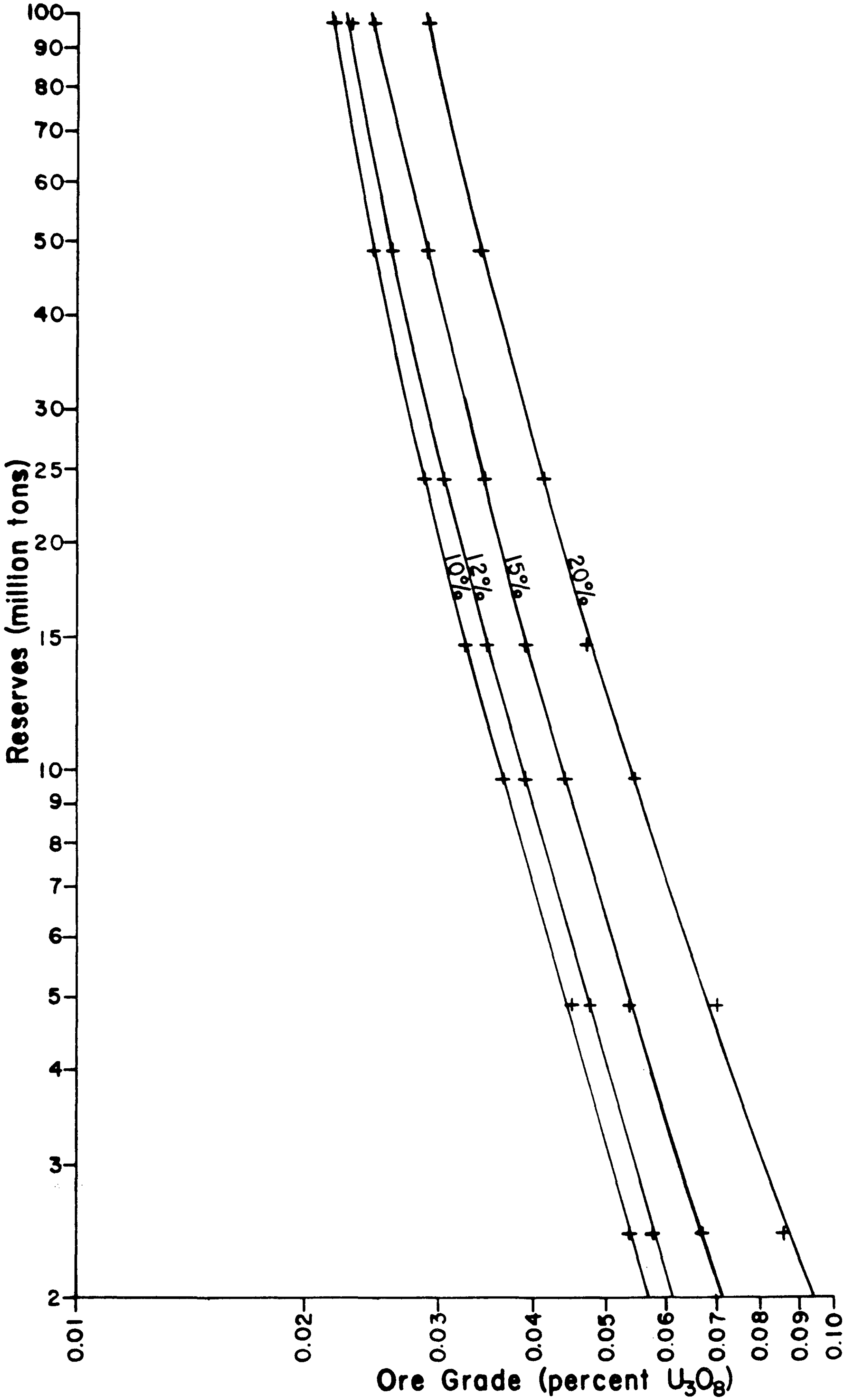

FIGURE D2 - RESERVES VERSUS ORE GRADE VARYING DCFROR
UNDERGROUND MINE $40/LB. U_3O_8
80 ORE DEPTH TO THICKNESS RATIO

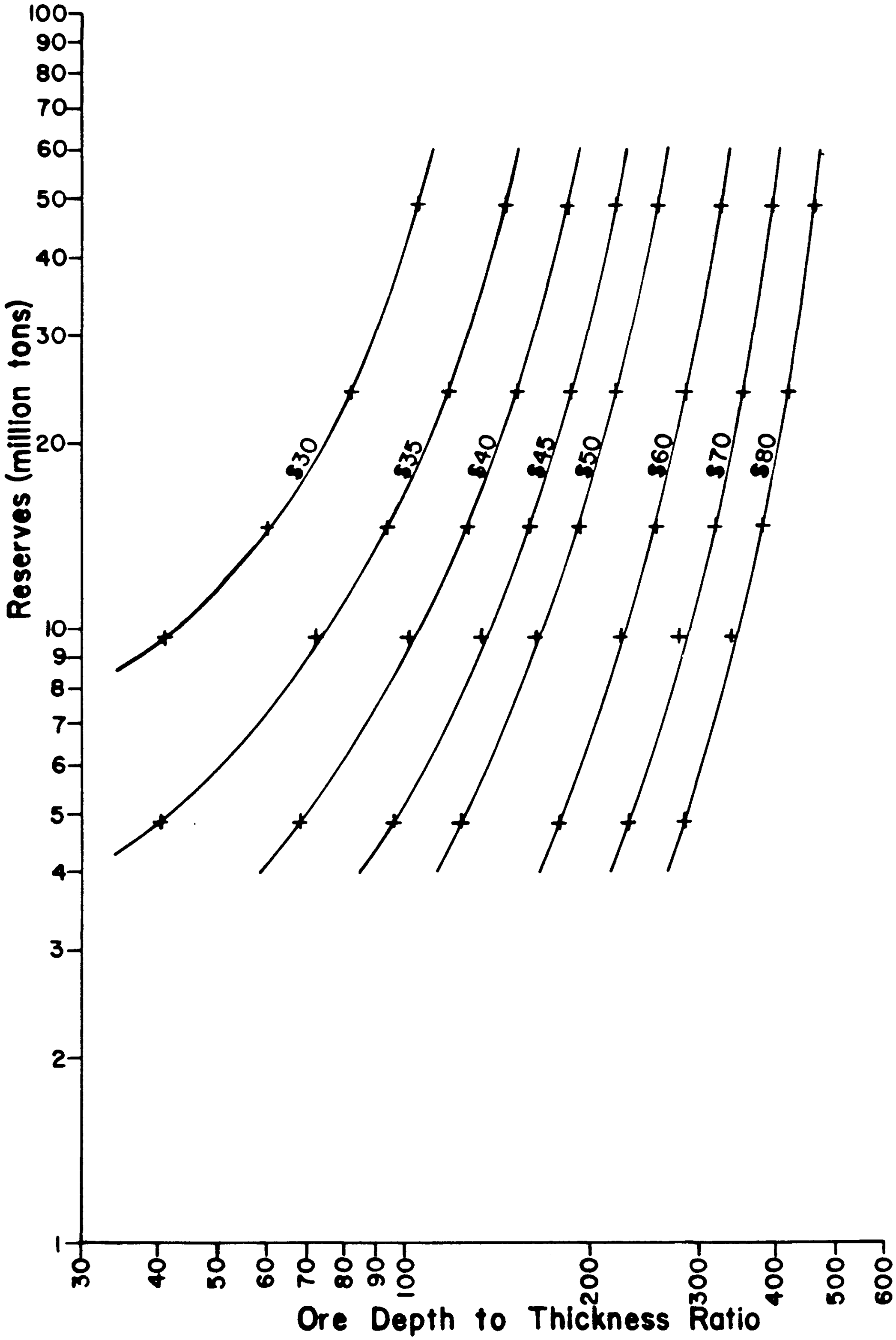

FIGURE D3 - RESERVES VERSUS ORE DEPTH TO THICKNESS RATIO
UNDERGROUND MINE 0.050% U_3O_8 15% DCFROR
VARYING U_3O_8 SELLING PRICE (DOLLARS PER POUND)

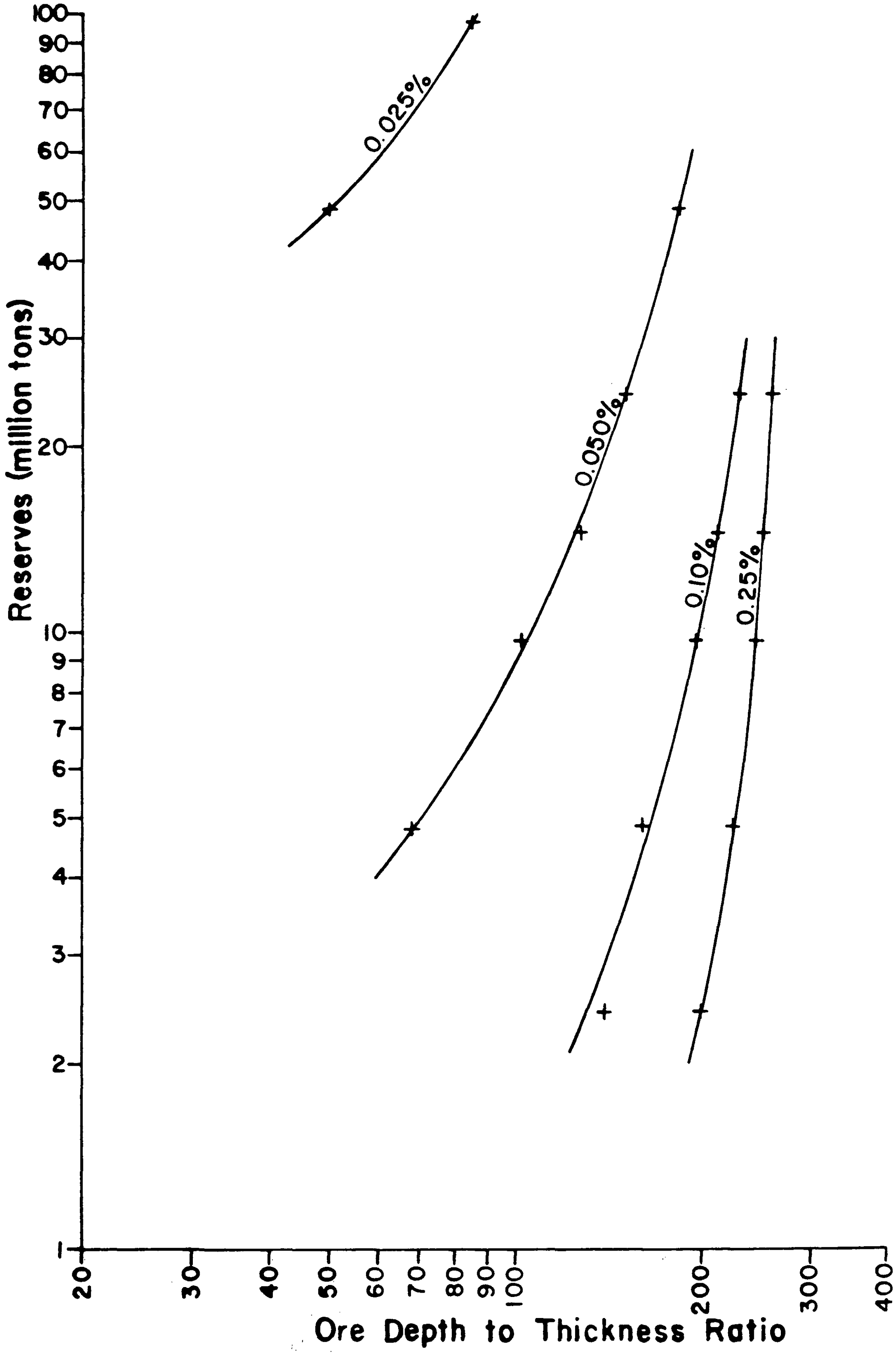

FIGURE D4 - RESERVES VERSUS ORE DEPTH TO THICKNESS RATIO
UNDERGROUND MINE $40/LB. U_3O_8 15% DCFROR
VARYING ORE GRADE (PERCENT U_3O_8)

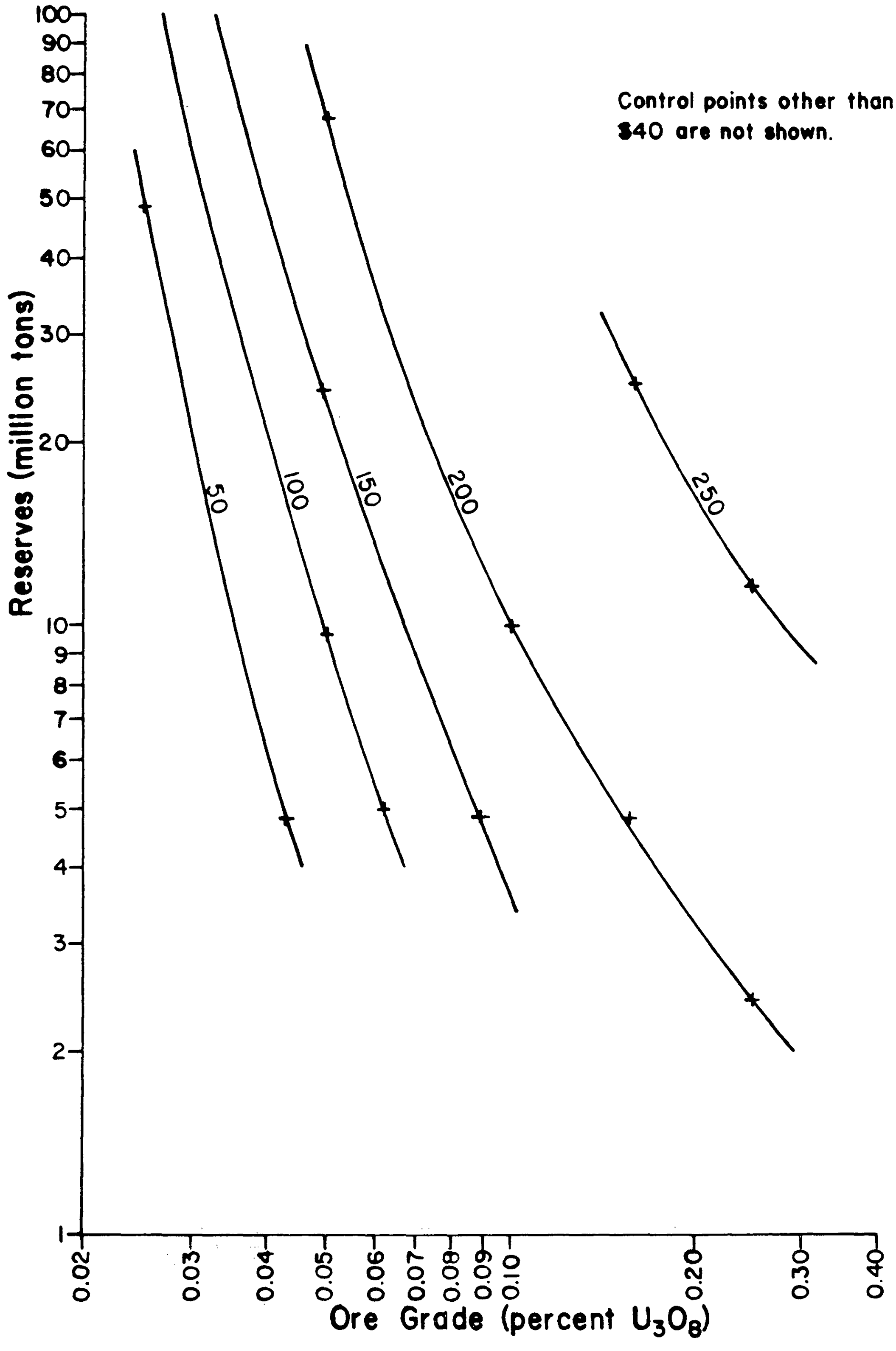

FIGURE D5 - RESERVES VERSUS ORE GRADE

UNDERGROUND MINE $40/LB. U_3O_8 15% DCFROR

VARYING ORE DEPTH TO THICKNESS RATIO

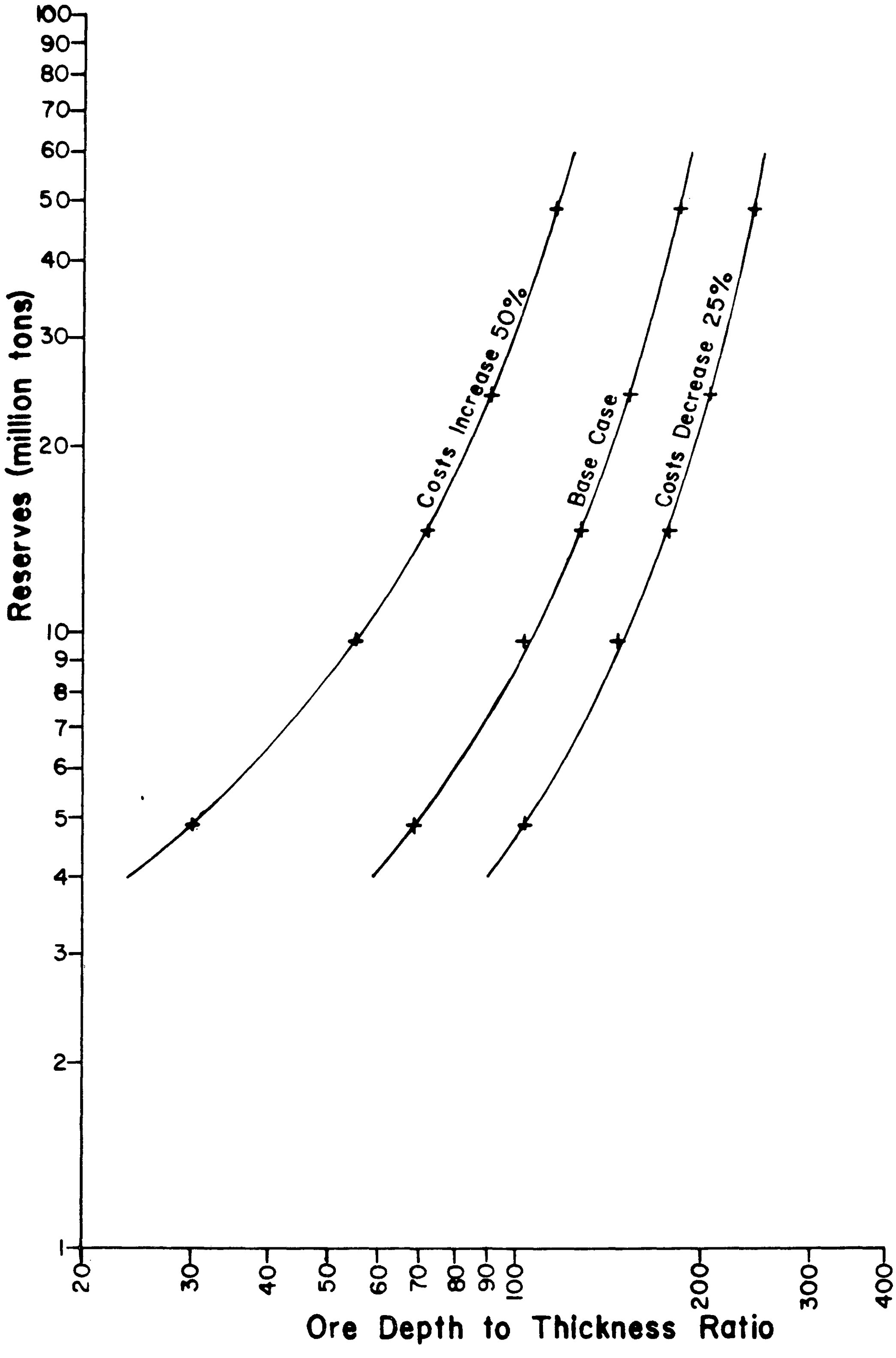

FIGURE D6 - RESERVES VERSUS ORE DEPTH THICKNESS RATIO
UNDERGROUND MINE 0.050% U_3O_8
$40/LB. U_3O_8 15% DCFROR
VARYING MINE PRIMARY DEVELOPMENT AND MINING COSTS

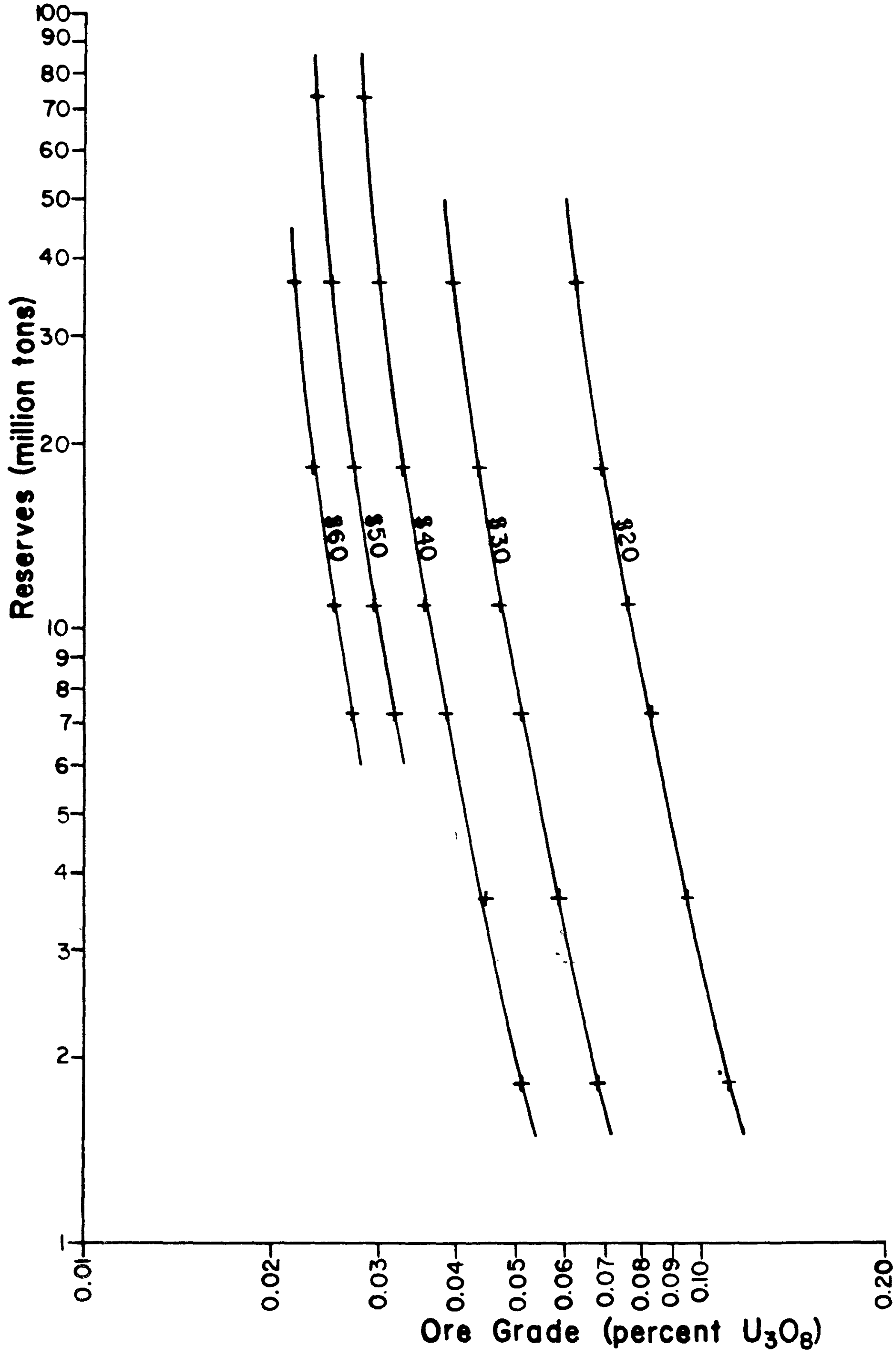

FIGURE D7 - RESERVES VERSUS ORE GRADE
OPEN PIT MINE 15% DCFROR
25 ORE DEPTH TO THICKNESS RATIO
VARYING U_3O_8 SELLING PRICE (DOLLARS PER POUND)

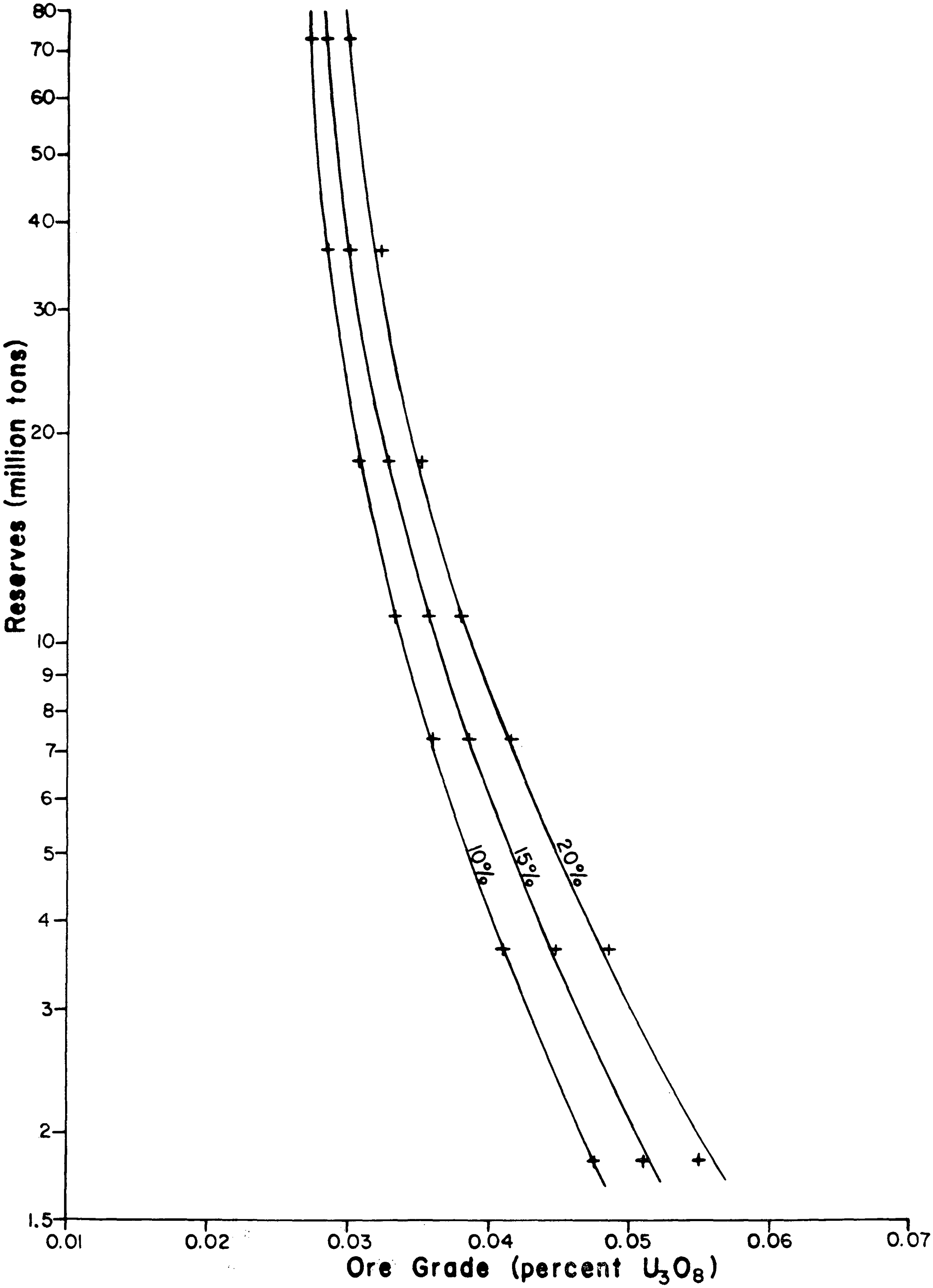

FIGURE D8 - RESERVES VERSUS ORE GRADE VARYING DCFROR

OPEN PIT MINE $40/LB. U_3O_8

25 ORE DEPTH TO THICKNESS RATIO

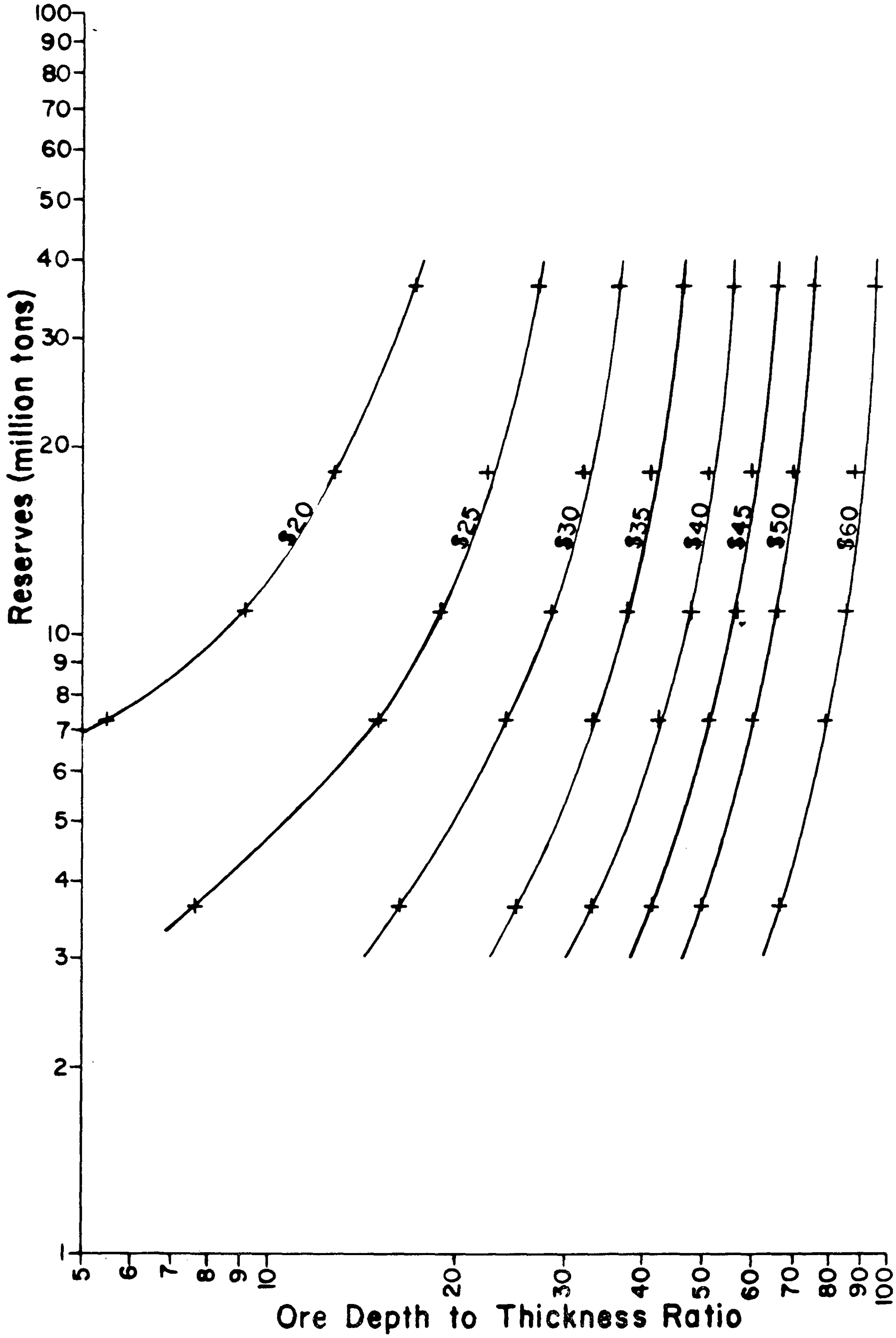

FIGURE D9 - RESERVES VERSUS ORE DEPTH TO THICKNESS RATIO
OPEN PIT MINE 0.050% U_3O_8 15% DCFROR
VARYING U_3O_8 SELLING PRICE (DOLLARS PER POUND)

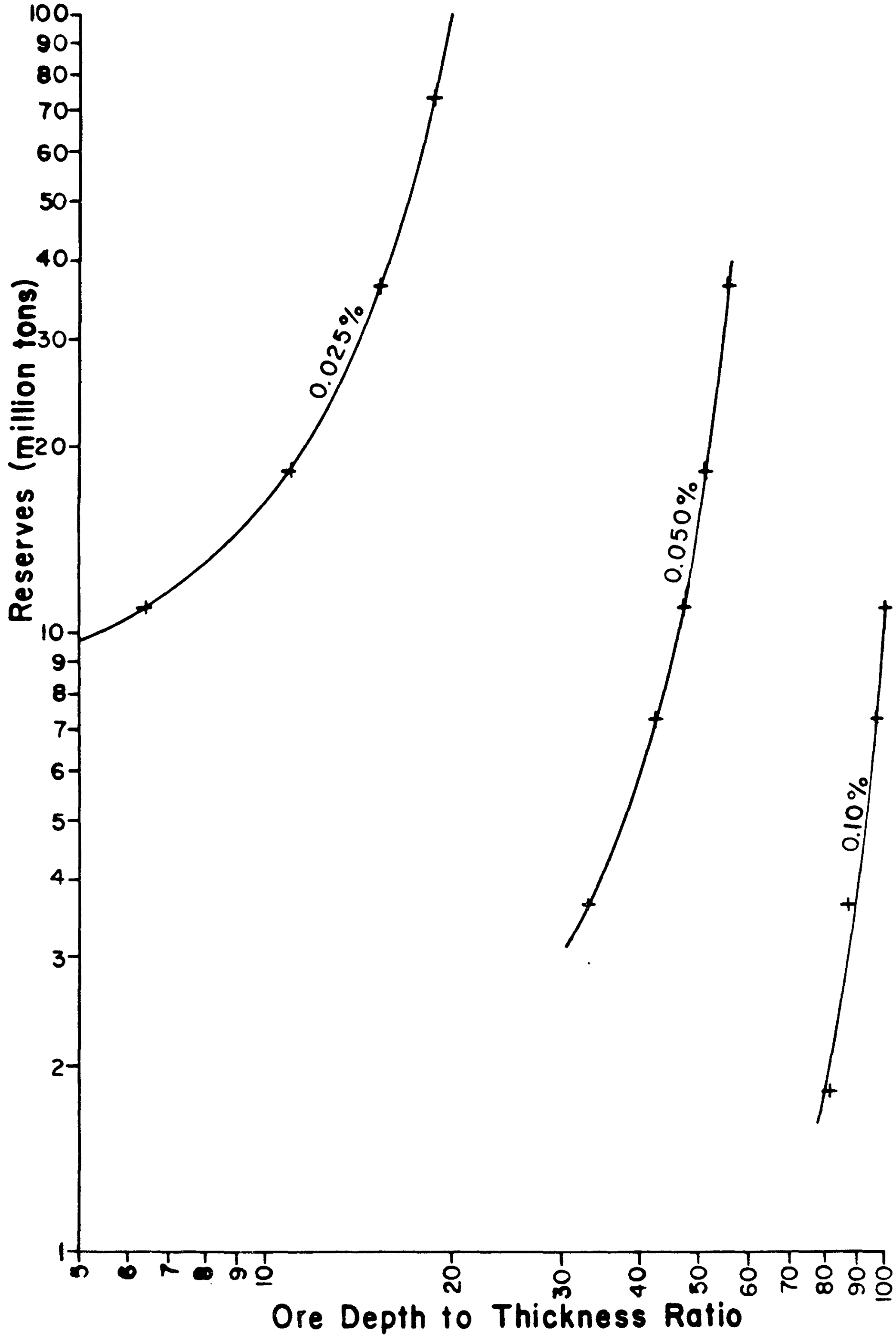

FIGURE D10 - RESERVES VERSUS ORE DEPTH TO THICKNESS RATIO
OPEN PIT MINE $40/LB. U_3O_8 15% DCFROR
VARYING ORE GRADE (PERCENT U_3O_8)

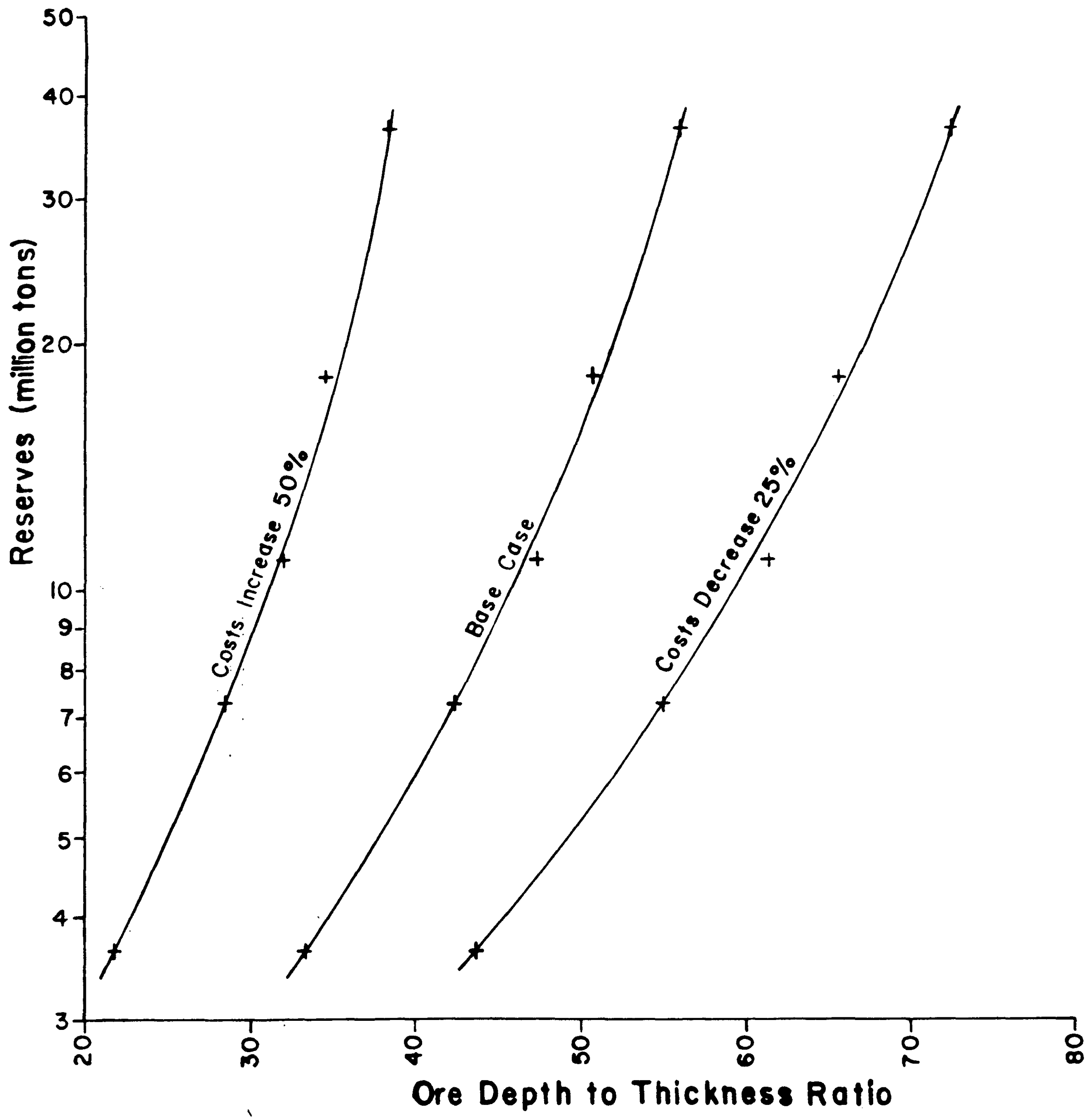

FIGURE DII - RESERVES VERSUS ORE DEPTH TO THICKNESS RATIO
OPEN PIT $40/LB. U_3O_8 0.050% U_3O_8 15% DCFROR
VARYING MINE PRIMARY DEVELOPMENT AND MINING COSTS

REFERENCES CITED

1. John Klemenic, Development of Nomograph of Cost of Open Pit Primary Development as a Function of Size, Depth and Thickness, USAEC Grand Junction Office, Colo., Feb. 24, 1972.

2. ____________, Examples of Overall Economics in a Future Cycle of Uranium Concentrate Production for Assumed Open Pit and Underground Mining Operations, USAEC Grand Junction Office, Oct. 20, 1972.

3. ____________, An Estimate of the Economics of Uranium Concentrate Production from Low Grade Sources, USAEC Grand Junction Office, Colo., Oct. 22, 1974.

4. E.N. Harshman, Uranium Deposits of the Shirley Basin, Wyoming, Ore Deposits in the United States - 1933/1967, AIME, New York, 1968, pp. 849-856.

5. U.S. Geol. Survey, Nuclear Energy Resources - A Geological Perspective, Washington D.C., U.S. Govt. Printing Office, INF-74-14., 1974.

6. USERDA, Uranium Exploration Expenditures in 1976 and Plans for 1977-78, Grand Junction Office, Colo., GJO-103(77), May, 1977.

7. ______, Statistical Data of the Uranium Industry, Grand Junction Office, Colo., GJO-100(74-77), Jan. 1, 1974-77.

8. Exploration Manager, uranium mining company, telephone conversation, Nov. 1977.

9. Manager, major drilling company, telephone conversation, Nov. 1977.

10. John Klemenic, Director of the Supply Analysis Division, Department of Energy, Grand Junction Office, Colo., telephone conversations, Sept. 1977 - Feb. 1978; interview, Oct. 1977.

11. J.F. Facer, Jr., Chief of the Analysis Branch, Department of Energy, Grand Junction, Colo., interview, Oct. 1977; telephone conversations, Nov. 1977 and Feb. 1978.

12. Corporate Attorney, uranium mining company, telephone conversation, Oct. 1977

13. Douglas J. Lootens, Uranium Production Methods and Economic Considerations, Engineering Bulletin 49, Dames & Moore, Los Angeles, Sept. 1977, pp. 23-34.

14. Douglas J. Lootens, Associate, Dames & Moore, Park Ridge, Ill., letter, Nov. 1977; telephone conversations, Jan. 1978.

15. Alfredo Gurmendi, Production Analyst in Mining, Department of Energy, Grand Junction, Colo., telephone conversations, Oct. 1977 and Jan. 1978.

16. General Manager, uranium mining company, telephone conversations, Nov. 1977 - Jan. 1978.

17. W.L. Dare, Uranium Mining Practices and Costs at Ten Salt Wash Lease Operations of Union Carbide Nuclear Co., Bureau of Mines, Information Circular 7922, 1959.

18. Confidential private industry report, 1973.

19. Manager, uranium mill, Wyoming, brief conversation, Oct. 1977.

20. Manager of Environmental Services, uranium mining company, telephone conversation, Nov. 1977

21. Frederick C.J. Lu, Chief of the Supply Branch, Department of Energy, Grand Junction, Colo., telephone conversations, Oct. 1977 and Feb. 1978.

22. John Klemenic, Analysis and Trends in Uranium Supply, USERDA Grand Junction Office, Colo., GJO-108(76), Oct. 1976, pp. 227-269.

23. Lane White, In-Situ Leaching Opens New Uranium Reserves in Texas, Engineering and Mining Journal, McGraw-Hill, v.176, No. 7, July 1975, pp. 73-81.

24. J.N. Frank, Cost Model for Solution Mining of Uranium, USERDA Grand Junction Office, Colo., GJO-108(76), Oct. 1976, pp. 201-225.

25. Chemical Engineer, experimental in-situ leach operation, brief conversation, Oct. 1977.

26. W.H. Ford, Geologist, Wyoming Minerals Corp., AIME presentation, Colorado School of Mines, Golden, Colo., Jan. 25, 1978.

27. Pamela R. Tittes, An Economic Evaluation of Regulatory Agency Requirements for Uranium Solution Mining in the United States, Master's Thesis, Colorado School of Mines, Golden, Colo., T-1935, 1977.

28. J.F. Facer, Jr., Uranium Production Trends, U.S. Department of Energy, Grand Junction Office, Colo., Oct. 1977.

29. R.B. Bhappu, Economic Evaluation of Inplace Leaching and Solution Mining Situations, Seminar on Inplace Leaching and Solution Mining, Mackay School of Mines, Univ. of Nev., Reno, Nev., Nov. 1975.

30. H.J. Winters, C.K. Chase and R.B. Bhappu, Technical Notes: Economic Evaluation for In Situ Extraction of Uranium, Mountain States Research and Development, Tuscon, Arizona, Nov, 1975.

31. F.M. lewis and R.B. Bhappu, Evaluating Mining Ventures Via Feasibility Studies, Mining Engineering, AIME, v. 27, No. 10, Oct. 1977, pp. 48-54.

32. George Toth, Uranium Solution Mining Cost Model, AIME Annual Meeting presentation, Denver, Colo., Feb. 28, 1978.

33. F.J. Stermole, Economic Evaluation and Investment Decision Methods, 2nd Ed., Investment Evaluations Corporation, Golden, Colo., 1977.

34. B.C.J. Lloyd, Uranium Mining Annual Review, Mining Journal Ltd, London, June 1977, pp. 92-94.