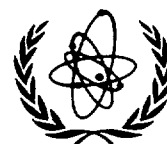IAEA-TECDOC-885

Steps for preparing uranium production feasibility studies: A guidebook

INTERNATIONAL ATOMIC ENERGY AGENCY

STEPS FOR PREPARING URANIUM PRODUCTION FEASIBILITY STUDIES:
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FOREWORD

For over two decades, the IAEA has published a large number of reports dealing with uranium geology, exploration, ore processing and, to a lesser extent, also with mining. Almost all of these reports are strongly emphasizing the technology. The economic evaluation at the various stages of a uranium resource project, from exploration to production, is barely covered. The purpose of the present publication is to provide a brief guideline on the various processes, from pre-evaluation to feasibility study, that can be used to test whether a project has a potential of being viable. This is a necessary exercise that can avoid unnecessary expenditure on a project that has no or little economic potential. History has shown that there were too many projects in the uranium exploration and production fields that have been operating for several years, even decades, only to be abandoned for lack of proper economic evaluation at the appropriate time or stage. As more and more countries are switching from centrally planned to the market economy system, the need for a realistic economic evaluation of a uranium production project in these countries becomes more apparent.

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CONTENTS

1. INTRODUCTION

The search for a mineral commodity, uranium included, and its eventual production is basically an economic activity. A realistic assessment of the economic viability of a project, whether to fulfill domestic needs or to meet world demand, should therefore be carried out as early and as frequently as possible.

Uranium exploration, development and eventual production, form a series of progressive and logical steps. Each step is part of a progression of activities with the objective of obtaining new or additional information from which a crucial decision is to be made. This decision is either to proceed with the project or to stop it, thus the term *GO, NO-GO* decision. Evaluation of the viability of the project must be carried at the various stages of the project development. Any delays in stopping a non-viable project will normally result in unnecessary or wasteful expenditure of resources that could have been spent on other projects which offer a better potential.

This guidebook is primarily aimed at mineral management personnel in developing countries who have little or no experience in preparing feasibility studies in uranium production. It is not a textbook which describes the geology, mining or processing of uranium. This guidebook deals with the philosophy, basic principles and important factors in the various stages of economic evaluation of the project. This guidebook is primarily concerned with small to medium sized mining projects. However, it can also provide useful guidance for the initial study of larger mining projects. More detailed studies of larger projects, however, should be left to well known experts in the field.

While the mineral commodity in question is uranium, the procedures and approaches outlined in this guidebook are generally applicable to the study of other commodities.

2. GENERAL PROJECT DEVELOPMENT PROCEDURE

2.1. PHILOSOPHY

During the last few years, paralleling the general growth of industry in the world, a great number of mining projects have been launched in response to the demand for metals. This has been particularly true for uranium. Among these numerous projects, many have been failures due to a lack of appropriate pre-studies and procedures.

The goal of this guidebook is to explain, in detail, the philosophy of a step-by-step approach combined with a strategy of go, no-go decisions at each step.

Why this cautious approach? The mining industry is very capital intensive and a mine requires a high development cost compared to the yearly output, unlike other industries. In addition, a long time is required from the moment when a decision to explore is taken to the delivery of the first yellow cake production. This means high interest costs on any borrowed money. Accordingly, it is easy to understand that a mining project warrants a cautious, staged and efficient procedure when developing a project for production. Should such a cautious and progressive approach not be followed, there will be high risk of failure upon commencement of operations.

Due to the high expenditures required to develop a mining project, and due to the high interest cost on loans, there is a significant risk of endangering the life, or at least, the growth of the company involved in the project or, at the minimum, to tarnish its image in the mining world.

As far as developing countries are concerned, they often do not have the valuable hard currencies needed to implement their industrial development, and reducing the risks mentioned above is vitally important.

The feasibility study for any proposed uranium recovery project must accurately determine the real cost (including capital) of uranium production at a production rate that matches market demand. A decision to proceed with project construction should only be made if the projected cost of production is substantially less than world market prices. If the forecast production cost is greater than actual or projected market prices, then the project should be deferred. Available funds can then be used to purchase uranium from elsewhere, or applied to more viable or profitable projects.

The feasibility study must accurately and completely describe the proposed project. The mining method, process equipment, infrastructure details and all other facets of the project must be totally resolved and designed in detail. If this definition is lacking, cost over-runs will inevitably occur. The feasibility study must also present evidence to the potential investor that the proposed process will actually work. Proposed mining methods and costs, and mill recovery and cost projections must be accurate and supported by adequate testwork and studies.

The feasibility study is the final examination of a proposed development before financial approval and construction of the facilities. The feasibility study is usually the last in a series (or stepwise production) of studies of increasing accuracy. As will be explained later, it is a serious undertaking that will cost between $100 000 and $1 000 000 depending on the complexity of the project. The feasibility study should only be undertaken if there is a reasonable certainty of a successful outcome. This is logically assessed through an earlier series of less-costly pre-evaluation and pre-feasibility studies which are undertaken as information becomes available.

There are many tasks to be executed during the passage of a project from inception to production. These tasks present opportunities for mistakes, repeated effort, and wasted time. A logical approach to the project will help maximize productivity and ensure a successful uranium production facility. It is the intention of this guidebook to provide specific procedures for the development of a project using a step-by-step procedure. This procedure is combined with a go or no-go decision at each step.

In such a procedure, four steps are considered which will be looked at in detail in the following sections. These steps are chronologically:

— Pre-evaluation
— Pre-feasibility study
— Intermediate assessments
— Feasibility study.

2.2. STUDY STAGES

It is common to progress through a series of three or four studies of increasing accuracy and cost before construction of a project starts. These four phases, the available information, and accuracy of the estimates are outlined in Table I. The study phases can be briefly defined as follows:

— **Pre-evaluation**: First economic study carried out with minimum requirements and by comparison with similar existing operations, more advanced projects, or using general cost curves.

— **Pre-feasibility study**: Economic study based on more specific data for the actual deposit.

— **Intermediate assessment**: Under certain circumstances an intermediate assessment is necessary in order to clarify and define some specific items.

— **Feasibility study**: Final detailed study at the end of which a decision to proceed with or defer construction can be taken.

TABLE I. PHASES IN DEVELOPMENT OF A URANIUM MINING PROJECT

ITEM	TYPE 1 PRE-EVALUATION STUDY	TYPE 2 PRE-FEASIBILITY STUDY	TYPE 3 INTERMEDIATE STUDY	TYPE 4 FEASIBILITY STUDY
SITE/INFRASTRUCTURE				
Geographical location	Assumed	Approximately defined	Defined	Finalized
Plant area required	Assumed	Approximately defined	Defined	Finalized
Maps and surveys	None	If available	Necessary	Detailed
Soil and foundation tests	None	None	Preferable	Essential
Power supply	Assumed	Type defined	Initial design	Detailed
Site access roads	Assumed	Approximate length	Initial alignment	Final alignment
Camp/townsite	Assumed	Approximate size	Defined	Finalized
Site visit by project team	Recommended	Recommended	Essential	Several
OREBODY				
Reserve category	Assumed	Indicated	Proven/probable	Proven/probable
Bulk sampling	None	Possible	Probable	Essential
MINE				
Production rate	Assumed	Preliminary	Finalized	Finalized
Mining method	Assumed	Conceptual	Optimized	Finalized
Mine layout	None	Preliminary	Detailed	Detailed
Equipment selection	Assumed	Preliminary	Detailed	Quotations
Rock mechanics	Some recommended	Some recommended	Preliminary	Essential
Test mining	None	Probable	Probable	Probable
PROCESS PLANT				
Plant throughput	Assumed	Preliminary	Finalized	Finalized
Bench lab. tests	Recommended	Initial	Detailed	Detailed
Pilot plant tests	None	None	Possibly	Probably
Process flowsheet	Assumed	Preliminary	Detailed	Frozen
Equipment sizing	Assumed	Preliminary	Detailed	Final
General arrangements	Area only	Preliminary	Detailed	Final
Piping and instrumentation	None	None	P&ID	P&ID and layout
Electrical	None	None	Single line	Detailed
Concrete and steel	None	None	Preliminary	Detailed
TAILINGS/ENVIRONMENT				
Tailings design	Assumed	Assumed	Preliminary	Defined
Decommissioning plans	Assumed	Assumed	Preliminary	Defined
Effluent treatment tests	None	None	Initial	Completed
Effluent treatment plant	Assumed	Assumed	Preliminary	Finalized
Baseline study	None	None	Started	Complete
Environmental impact statement	None	None	Started	Submitted
Permits	None	None	In preparation	Applied for

Notes: P&ID — Piping and instrumentation diagram

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TABLE I. (cont.)

ITEM	TYPE 1 PRE-EVALUATION STUDY	TYPE 2 PRE-FEASIBILITY STUDY	TYPE 3 INTERMEDIATE STUDY	TYPE 4 FEASIBILITY STUDY
CAPITAL COSTS				
Estimate type	Factored	Preliminary	Detailed	Very detailed
Staff involved	Engineers	Eng.& estimator	Estimators	Estimators
Number of staff	2 to 4	2 to 6	2 to 6	4 to 8
Equipment costs	Factored	Budget quotes	Budget quotes	Firm quotes
Installation costs	Factored	Prel. $/h & h	Study $/h & h	Contract $ & h
Transport costs	Assumed	Allowance	Prelim. study	Quotations
Civil costs	Factored	Area basis	Prel. QTO & $/m	QTO & bid
Buildings and structures	Factored	Volume basis	Prel. QTO & $/t	QTO & bid
Piping — in-plant	Factored	% installed equip.	% installed equip.	Partial QTO
Piping — off-site	Factored	Prel. estimate	Preliminary estimate	QTO & bid
Instrumentation	Factored	% installed equip.	Loop count	QTO & bid
Electrical	Factored	% installed equip.	Prelim. est.	QTO & bid
Construction indirect	Factored	% direct costs	% direct costs	Estimated
EPCM	Factored	% direct costs	% direct costs	Estimated
Contingency — % of all other costs	20 to 25%	15 to 20%	10 to 15%	10%
OPERATING COSTS				
Staff level	Assumed	Preliminary list	Detailed list	Detailed list
Labour rates	Assumed	Assumed	Studied	Preliminary contract
Supplies consumed	Assumed	From lab. data	From lab. data	From lab. data
Supply prices	Assumed	Assumed/quotes	Budget quotes	Firm quotes
Power consumed	Assumed	Preliminary	Detailed est.	Detailed estimate
Power cost/unit	Assumed	estimate	Studied	Firm quote
Marketing costs	Allowance	Assumed	Allowance	Studied
Head office costs	Allowance	Allowance Allowance	Estimated	Detailed est.
PROJECT SCHEDULE	Assumed	General	Detailed	Final
ECONOMIC ANALYSIS				
Taxation	Assumed	Preliminary study	Detailed	Detailed
Revenue/markets	Preliminary	Firm	Firm	Firm/LOI
Discounted cash flow	Very preliminary	Preliminary	Detailed	Detailed
USE OF ESTIMATE	1. Initial evaluation 2. Rank targets	Internal test of project profitability	More detailed internal test of project profitability	Demonstrate project feasibility to investors
ESTIMATE ACCURACY	+50% to -25%	+30% to -15%	+20% to -10%	+15% to -5%
STUDY STATISTICS				
Peak staff involved	1–2	2–8	10–20	20–30
Number of drawings	1–3	10–20	15–50	50–100
Weeks required for study	2–4	4–16	8–20	8–32
STUDY COST — OOO's	$20–50	$50–100	$50–100	$200–750

Notes: QTO — Quantity take-off.
   LOI — Letter of intent to purchase.
   EPCM — Engineering Procurement Construction Management.
   Study costs are cumulative.

2.3. EXAMPLES

The sequence of studies outlined above has been developed over many years of industrial endeavour. Occasionally it is possible to eliminate some of the steps and still succeed. However as a general rule, any attempts to reduce the project development time or costs by taking short-cuts will result in failure. It is instructive to look at some real examples of projects which have ended unsuccessfully. Selected cases are presented below, categorized as to the main problem.

2.3.1. Drilling campaign misinterpretation

In a very promising geological environment, not far from already existing profitable operations located in the same geological context, a company decided to launch a small exploration campaign.

Unfortunately, the first drill holes appeared to intersect exceptionally rich ore shoots. Since these results occurred in a very favourable context, the decision was taken to launch the mine.

The decline was drilled with difficulty due to a large amount of ground water needing preparation of a sump and the establishment of a battery of pumps. This cost a lot. And all that, just to find that by chance the first holes had been drilled in thin veins, rich but with no horizontal continuity. Complementary horizontal drilling proved that the ore, very rich in some places, has no horizontal continuity and therefore was not of economic value.

The mine was closed and reclaimed. Not a pound had been produced but a lot of money had been spent.

2.3.2. Insufficient drilling programme

A company carried on a drilling campaign. The holes were placed at regular intervals and the results were as shown in Fig. 1. It was concluded that the deposit was a system of parallel regular veins dipping at 60°. Mining would be easy and the first analyses showed that the grade was good and that the ore could be treated without difficulties.

A decline, small and short because the orebody was shallow, was drilled. The ore was found as expected. It was then decided to drive a horizontal drift in the vein. The miners were surprised to find that the vein kept turning in the same direction, and at the end, it came back at its departure point.

Then, it became clear that the deposit was made of two closed thalwegs, very irregular in width and dip and therefore not economically minable. So the project was stopped. The amount of money lost was not high but it could have been less, if one or two holes had been drilled at a greater depth to test the assumed geological model (steeply dipping veins).

2.3.3. Poor process development

A government agency discovered that the recirculating solution at a base metal heap leach operation contained significant uranium values. A recovery project was designed and built on a fast-track basis because the government wished to become self-sufficient in uranium at the earliest possible opportunity. The testwork, design and economic studies were based on an assayed solution grade of 20 g U/t of solution. The plant was commissioned and successfully made uranium concentrate at the design rate — for the first month. From that time on, the uranium production rate rapidly declined as the solution grade fell to 10 g U/t. The uranium production cost increased from $18/kg U to $33/kg U since the monthly operating cost was essentially fixed but the production rate decreased by a factor of two. After six months of operation the plant was decommissioned.

CROSS CUT AA

drill holes

surface

overburden

PLAN VIEW

drift

A ———————————— A
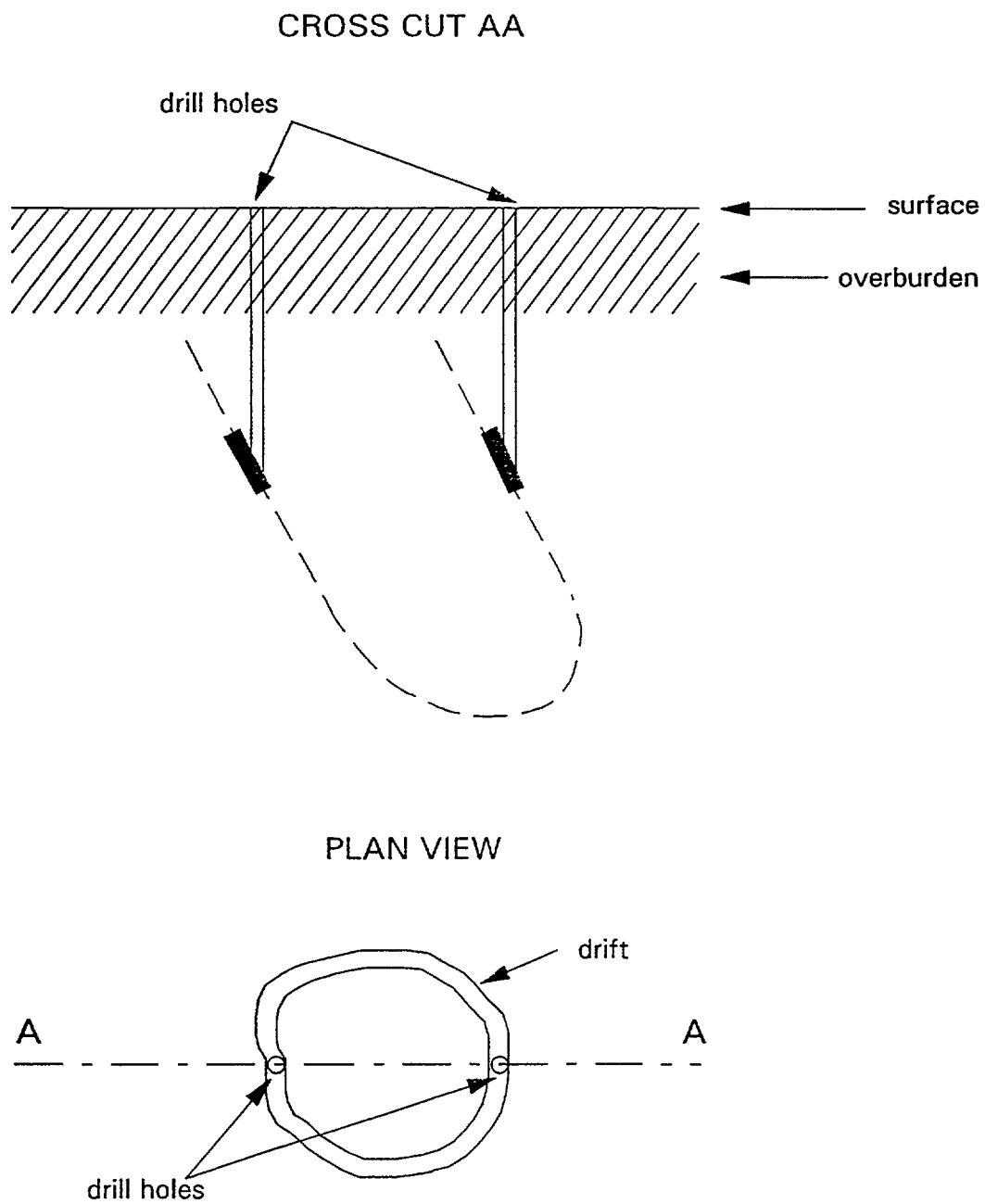
drill holes

FIG. 1. Example of misinterpretation of drill hole result caused by inadequate drilling programme.

The plant was developed in great haste. The developers did not take the time to properly examine all technical aspects of the proposed project. Specifically they failed to properly perform uranium accounting around the heap leach operation. If they had, they would have determined that much of the uranium in solution was a recirculating load due to recycling spent heap leach liquor back onto the heaps.

Processes must be thoroughly tested and understood before they are placed into operating plants. Pilot plant operations may not always be enough. It is also important to perform a sensitivity analysis for any proposed project. The sensitivity analysis examines the impact of feed grade, operating cost and other factors on the economic viability of a proposed project. This analysis will clearly show what factors are most important to the projects success or failure.

2.3.4. Inadequate marketing study

A mining company located a massive, shallow, relatively high grade, lanthanide deposit in a remote part of the world. In their pre-evaluation, they performed geological, mining, and metallurgical testwork and studies and determined that the project was technically possible. They then prepared a pre-feasibility study at an ore processing rate of 1000 t/d. At this tonnage the capital and operating costs per kilogram of product were very low compared to world prices. The company perceived a very profitable operation and started a detailed feasibility study.

The company had progressed some way through the study before they undertook a market analysis. They finally realized that their proposed processing rate would produce twenty times the world demand for the main product of the plant. A sensible ore processing rate would have been 25 to 50 t/d instead of 1000 t/d. The cost of recovering a kilogram of product at the reduced processing rate would obviously be far greater than at the higher processing rate — and higher than world prices. The project was therefore stopped before more money was wasted.

The mining company was late in their review of the market for their product and they could have saved themselves some embarrassment if they had performed the study earlier. However it is fortunate that the mining company did examine the project in a phased fashion. If they had simply built their project on the expectation of low production costs they would have been disappointed as their warehouse accumulated product that could not be sold.

Clearly projects must be designed to produce a quantity of product that can be readily absorbed by the market without causing a major drop in prices. A market study must be an early component in any development sequence.

3. TARGET SETTING

3.1. PROJECT OBJECTIVE

Before a country or company embarks on the risky and expensive activity of exploring and the eventual production of uranium, a clear policy needs to be firmly defined. This policy or target might be based on one of the two following objectives:

(a) To find uranium to meet eventual domestic needs within the framework of the country's nuclear power programme and to be independent of outside sources.

(b) To search for and eventually produce uranium for the world market.

Annual Average Selected Uranium Prices
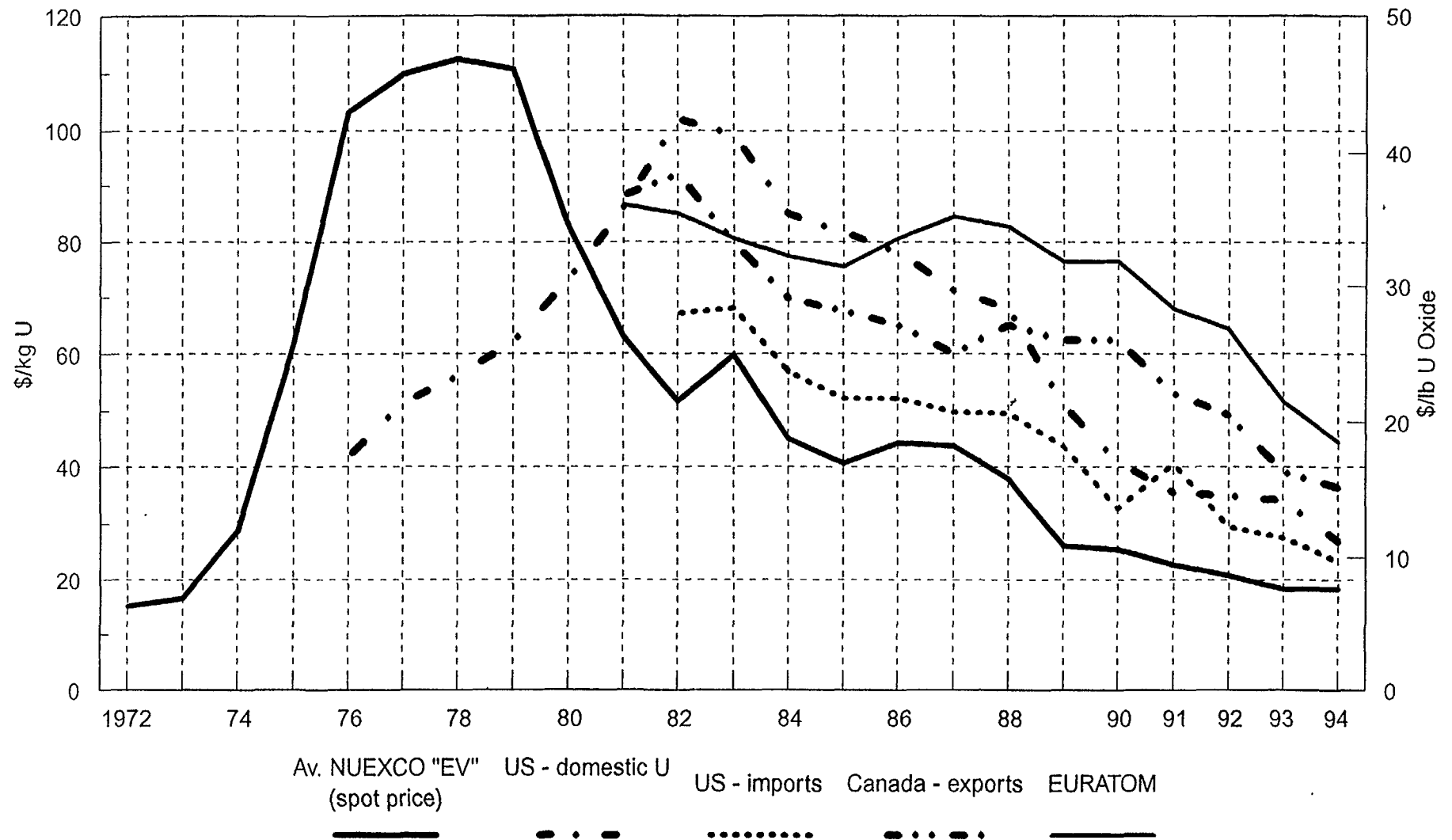
FIG. 2. Historical uranium price development from 1981 to 1994.

Regardless of the selected objective, economic measure should always be applied. The notion that uranium can be produced at any cost because of domestic requirement has been proven to be wrong. Recent developments during the early 1990s in Eastern Europe and elsewhere in relation to their uranium production are good examples. Poor economics forced the closure of many of these production centres. Decision-makers should therefore be aware of the various factors which may effect the economics of such an undertaking. It is to be stressed that the single most important variable in the economic evaluation of a mineral resource project is the price of the commodity in question.

In most cases, the commodity in question is clearly uranium. In such a case, regardless of the project objectives, awareness of supply-demand and market price situation is essential. Where the aim is to produce uranium to compete in the open market, the target should be to produce uranium at a cost that is lower than the current spot market price. Even if the production is to meet domestic requirements, the aim should be a production cost that is lower than a uranium price that can be procured under a long-term contract agreement. Figure 2 shows the historical uranium price development from 1981 to 1994. The often quoted spot price is here noted as the "NUEXCO EV (Exchange Value)".

Where uranium is produced as a co-product (e.g., the Olympic Dam deposit), the economics of the project are strongly influenced by the other commodities, which in this particular case are gold and copper. Where uranium is produced as a by-product of other mineral commodities (phosphate, gold or copper), its potential output is totally dependent on the output and market situation of the primary product. Even in this particular case, the uranium price situation is still a determining factor. Recent closures of almost all uranium extraction plants from phosphoric acid production in the world are caused by the persistent depressed uranium market as noted in Fig. 2.

An added objective for some developing countries is the element of training or man-power development in this type of exercise. This is particularly true for countries with no previous mining industry experience. It needs to be emphasized that training in the evaluation at the different stages of a mineral development project for one commodity are quite similar to that of others. Experience gained through the various stages of feasibility studies of a uranium project can therefore be easily adapted to similar studies of other minerals.

3.2. GEOLOGICAL TARGETS

An important factor in uranium project development is the recognition of appropriate geological targets that might offer attractive grades and tonnages as well as favourable physical and chemical characteristics that would allow the extraction of uranium at reasonable cost. Exploration geologists use a number of recognition criteria to determine the geological favourability of a given region based on its similarity to known uriniferous regions in the world. Important criteria include favorable host rocks, age, major geological structure (craton, continental sedimentary basin, graben system) and other information specific to the various types of uranium deposit. Some countries have adequate geological, geophysical, geochemical and other relevant maps which facilitate the selection of favorable target areas for uranium exploration. However, in a number of countries such information is often incomplete thus making assessment of the country's favourability difficult or nearly impossible. In such a case the needed basic information has to be created.

3.2.1. Uranium deposit classification

Uranium deposits are not distributed at random, but geologically controlled. Each deposit is unique in its geological character, dimension, grade and tonnage. A very good understanding on these characteristics can help geologists in their exploration programmes and the assessment of their economic potential. The information on their host rock, ore mineralogy, associated gangue, ore control, dimension and attitude are essential for the metallurgist and mining engineer in their assessment for the ore processing and mining method considered most appropriate for such a deposit. It needs to be

emphasized that every deposit has its own unique character, therefore over-generalization can lead to erroneous judgement. According to their geological setting, most known uranium deposits in the world can be assigned to one of the following deposit types. Table II is a summary of a descriptive classification developed by IAEA and used in the NEA(OECD)/IAEA joint report "Uranium Resources, Production and Demand" that are published periodically. This classification is approximately arranged in decreasing order of importance in terms of its significance to the current level of uranium production. One should be aware that there are other known uranium deposit classifications used by geologists, often based on the predominant type(s) commonly found in the area(s) or based on the their process of formations (genetic models).

TABLE II. URANIUM DEPOSIT CLASSIFICATION

1. Unconformity related
2. Sandstone
3. Quartz-pebble conglomerate
4. Veins
5. Breccia complex (Olympic Dam type)
6. Intrusive
7. Phosphorite
8. Collapse breccia pipe
9. Volcanic
10. Surficial
11. Metasomatite
12. Metamorphite
13. Lignite
14. Black shale
15. Others

A brief descriptions of each of these types are noted below.

Unconformity related deposits

They are well known for their high grade and large tonnage features, and their spatial relation to major erosional unconformity. Most well known deposits are Proterozoic in age, but younger (Phanerozoic) deposits are also known. These deposits are found primarily in northern Saskatchewan, Canada (Rabbit Lake, Cluff Lake, Key Lake, Cigar Lake, Midwest, MacArthur River, etc.), and the Alligator River area of northern Australia (Ranger, Jabiluka, Nabarlek, etc.). Pitchblende is generally the principal ore mineral. However, they may also contain polymetallic mineral assemblages (U, Ni, Co, As, Cu, Ag, Au).

Sandstone deposits

These deposits generally occur in carbon and/or pyrite-bearing fluvial or marginal marine sandstone. The host rocks are commonly friable and often associated with tuffaceous material. Pitchblende and coffinite are the most common minerals for the unoxidized ore. Secondary uranium minerals such as carnotite, tyuyamunite and uranophane are the usual ore minerals in the oxidized zone. In addition to uranium, the following elements may also be enriched: Mo, Se, Cu and V. An important feature of many of these deposits is a host rock which has good porosity and permeability, and bounded by less permeable horizons. Such a deposit is generally amenable to in situ leaching (ISL) method of uranium production. Deposits of this type occur in sandstone of different ages and were or being mined in: the USA (Wyoming, Grants Mineral Belt, Colorado Plateau, Texas Gulf Coast, etc.); Niger (Arlit, Akouta, Taza, etc.); Kazakhstan (Moynkum, Mynkuduk, Northern

Karamurun, Uvanas, Zarechnoye, etc.); Russia (Khiagdinskoye); Uzbekistan (Bukeenai, Sugraly, Uchkuduk, etc.); as well as in Bulgaria, China, the Czech Republic, Gabon, Hungary, Japan, Pakistan and Romania.

Quartz-pebble conglomerate deposits

These deposits are restricted to conglomeratic rocks of Lower Proterozoic age, generally very large, but very low grade. The most well known deposits of this type where uranium is being mined are in the Elliot Lake district of Canada, and in the Witwatersrand basin of South Africa. In South Africa, uranium is recovered as a by-product of gold. Occurrences of similar type are also known in Brazil (Jacobina) and India.

Vein deposits

This type of deposits occur as steeply dipping veinlets or stockworks generally associated with granitic rocks. As in the sandstone type, they are found in a broader geographical regions of the world. Most well known concentrations are located in the western part of Europe (Czech Republic, France, Germany, Portugal, Spain and Portugal) where many of them are still being mined. They are also known in Canada, China, Kazakhstan, the Russian Federation, USA, Uzbekistan and Zaire. Mineralogically, they are either monometallic, having only uranium ore minerals (uraninite, pitchblende or coffinite) or associated with other metals (polymetallic: U, Co, Ni, As and Ag).

Breccia complex or Olympic Dam type of deposit

It is a unique, large tonnage, low grade deposit occurring in a very extensive breccia zone (20 km^2 in area and 1 km in vertical extent) found in South Australia. Uranium is produced together with Cu and Au. The breccia zone composed dominantly of hematite, granite fragments and quartz. To date, no deposit of this type has ever been found outside Australia. The Acropolis prospect, also in South Australia, is considered of similar type.

Intrusive deposits

These are uranium deposits that are found in intrusive rocks, generally granitic or syenitic in composition. Included in this group are also carbonatite deposits. In most cases, the uranium minerals occur disseminated in the host rock as non-refractory minerals. Almost all present uranium production from this type of deposit comes from the Rössing mine in Namibia. There are other deposits of similar type in Namibia which are not yet exploited (Goanikontes, Ida Dome, Valencia, SJ Claims). Deposits belonging to this varied group are known in: Brazil (Araxa); Cameroon (Lolodorf); Canada (Johan Beetz, Bancroft); Finland (Sokli); Greenland (Kvanefjeld); India (Sevathur); South Africa (Pilanesberg); and USA (Bingham Canyon, Twin Buttes, Yerington).

Phosphorite deposits

Most phosphorite deposits in the world contain around 100 ppm uranium generally in the form of apatite. As such phosphorite deposit is normally classified as non-conventional source of uranium. Where uranium is recovered, it is a by-product of the phosphoric acid production, and therefore entirely dependent on the primary product. Until recently (1993), a fair amount of uranium was produced from this source in the USA (Florida). Phosphorite deposits are rather common all over the world, particularly in rocks of Cretaceous age. Well known are those found in North Africa (Algeria, Morocco, Tunisia) and the Middle East (Jordan, Syria).

Collapse breccia pipe deposits

This is a rather unique deposit that occur in a circular, vertical pipes, 30 to 200 metres in diameter, within a sedimentary sequence. The usual ore mineral is pitchblende normally associated with other sulphide minerals of Cu, Pb and Ag. The State of Arizona, USA, is the only place in the world where deposits of this type are known.

Volcanic deposits

Volcanic uranium deposits can be structurally controlled (as in vein deposits) and/or interbedded (stratabound) within the acidic to intermediate composition volcanic host rocks. Uranium may be accompanied by Mo, Cu, Se, F and other elements normally enriched in these types of host rocks. The most important deposits where uranium is currently produced are located in China (Jiang Xi district), Mongolia (Dornot), and in the Russian Federation (Streltsovsky district). Similar deposits are known in Canada (Michelin), Italy (Novazza), Mexico (Nopal-1, Peña Blanca) and USA (McDermit Caldera).

Surficial deposits

Most deposits belonging to this group are of recent age and are found in surficial depression, solution cavities (karst) or near-surface joints and fracture system. Uranium occurs almost exclusively as secondary minerals (hexavalent stage) or adsorbed on other materials. Because of their relatively young age, they are not easily detected by the conventional radiometric method of surveying. Among the well known examples are: the Yeelirrie/Yilgarn in Southwest Australia; Summerland area, Canada; Langer Heinrich, Namibia; Dusa Mareb, Somalia; and Flodel Creek and Pryor Mountains, USA.

Metasomatite deposits

These are deposits where uranium occurs as fissure filling in alkali silicate metasomatites (albitites, aegirinites and alkali-amphibole rocks). At Zheltye Vody deposit in Ukraine, where uranium is still being mined, it is associated with iron formation. Other examples are: Espinharas, Brazil; and Ross Adam, USA.

Metamorphic deposits

These are uranium concentrations that are found in disseminated form, interbedded within a metamorphic host rocks (metasediments and/or metavolcanics). There is generally no evidence of post metamorphic mineralization. Deposits of this group are found in: Forstau, Austria; and Duddridge Lake, Saskatchewan, Canada.

Lignite deposits

Some lignite deposits are known to contain low concentration of uranium. The uranium may also be concentrated in clay and/or sandstone immediately adjacent to the lignite formation. Deposits of this type are known in the Czech Republic, Germany, Greece, Kazakhstan, Kyrgyzstan, Poland, the Russian Federation, USA and Ukraine.

Black shale deposits

Shale, particularly those rich in organic matter, contain higher uranium then other sedimentary rocks. Some of these are exceptionally enriched to be regarded as a potential ore for uranium. These shales are normally also enriched in Mo, V and Cu. These deposits are normally very extensive, but very low in grade. Well known examples are: the Ranstad deposit of Sweden; Ogcheon Formation in the Republic of Korea; Chattanooga Shale in the USA; Rajsk in Poland; Djantuar deposit in Uzbekistan.

Other deposits

Deposits that can not be classified into one of those discussed earlier are grouped in this category. The uranium deposits in the Todilto Limestone in New Mexico, USA are an example.

As noted earlier, each uranium deposits has its own specific characteristics and geological control. The two most important features that are essential in an economic evaluation are grade and tonnage. A summary description of the grade and tonnage of selected types of uranium deposits in the world is presented in Table III.

TABLE III. GRADE AND TONNAGE OF DIFFERENT TYPES OF URANIUM DEPOSITS

Type of deposits	Grade in %U	Tonnage in tonnes U
Unconformity related		
– Canadian type	0.27–12.2	5 000–100 000
– Australian type	0.17–1.56	3 000–173 000
Sandstone	0.1–0.5	4 000–20 000
Quartz pebble conglomerate	0.03–0.15	8 000–80 000
Vein	0.1–1.0	< 100–105 000
Breccia complex	0.07	305 000
Intrusive	0.03–0.1	< 100–105 000
Collapse breccia pipe	0.3–1.0	< 2 000
Volcanic	0.03–0.2	500–35 000
Surficial	0.03–0.2	< 100–25 000
Metasomatite (albitite)	0.08–0.2	500– > 50 000
Metamorphic	0.005–0.1	< 2 000
Phosphorite	0.005–0.03	> 50 000
Lignite*	0.015–0.2	500–4 000 500
Black shale	0.02–0.1	–75 000

*Kazakhstan reported 2 deposits grading over 0.1% U with tonnage greater than 20 000 each.

Sources:
1. DAHLKAMP, F.J., Uranium Ore Deposits, Springer-Verlag (1993).
2. RUZICKA, V., Unconformity-type Uranium Deposit, in Mineral Deposit Modelling, R.V. Kirkham, W.D. Sinclair, R.I. Thorpe, J.M. Duke (Editors) (1993).
3. INTERNATIONAL ATOMIC ENERGY AGENCY, World Atlas of Uranium Deposits, IAEA-TECDOC (in preparation).

Examination of the present production situation in the world suggests reasonable geological targets that might be considered of interest. Total world uranium production in 1994 is estimated to be in the order of 31 500 tonnes U. Table IV. shows the 26 countries where uranium was produced. One should carefully note that over 98% of this production comes from only 7 different types of deposits from the 15 classes noted above: the high grade, large tonnage, unconformity-related type with 36%; the sandstone type uranium deposits, particularly those that are amenable to in situ leaching method of production, with 33%; the volcanic type accounted for 9%; quartz-pebble conglomerate deposits contributed 7%; the rather unique low grade, but very large tonnage intrusive deposit or the Rössing type with 6%; the more well known vein type deposits with 4%; and the breccia complex or the Olympic Dam type counts for 3%. Although much lower than in the previous years, uranium is also produced as a by-product of phosphate mining.

TABLE IV. ESTIMATED WORLD URANIUM PRODUCTION IN 1994

Country	In tonnes U (estimated)
Argentina	80
Australia	2200
Belgium	40
Brazil	110
Bulgaria	50
Canada	9700
China	780
Czech Republic	540
France	1030
Gabon	650
Germany	50
Hungary	410
India	290
Kazakhstan	2240
Mongolia	120
Namibia	1910
Niger	2970
Pakistan	30
Portugal	30
Romania	120
Russia	2000
South Africa	1690
Spain	260
Ukraine	500
United States of America	1290
Uzbekistan	2600
TOTAL	31690

It needs to be borne in mind that even from these 7 types, for some of them, if they are found today, there is a high probability that they may not be developed under the present market situation, mainly because their low grades that lead to high production cost per kg U. The unconformity related type of deposit, for its normally high grade and large tonnage feature is an obvious target for exploration. Unfortunately this type of deposits are confined to specific geological environment that

are limited to only a few countries. It may be noted that while the grade of sandstone type deposits are usually low (0.1 to 0.3% U) but, because of its amenability to the low production cost and environmentally friendly in situ leaching process, it is considered as a favorable target. As indicated in a number of recent publications, for a good in situ leach project in the USA, the production cost per lb U_3O_8 is between \$10 and \$15. A very competitive figure, even under the present depressed market situation. Another interesting geological target is the volcanic type deposit similar to those found in the Russian Federation and Southern China, that are higher grade than those found in Mexico and Italy. Some of the higher grade, granite related vein deposits typically found in western Europe, are also targets that are worth exploring.

In evaluating the viability of any deposits, it should be noted that it depends a great deal on the economic situation at the time of expected production. As noted above, production cost of \$10 to \$15/lb U_3O_8 represents a good quality in situ leach project of suitable sandstone orebodies. Accurate figure on production cost from different types of deposits and mining methods are not easily available. To give a general idea, Fig. 3 provides an illustration on the different cost of uranium production of selected deposits from presently operating and closed mines, and others that are still in the planning stage. One should be careful in using this information since some of these numbers appears to be too high or too low. It is obvious that other factors have to be taken into consideration where uranium is produced as a co-product (the Olympic Dam case) or by-product (for gold in South Africa; and phosphoric acid in the USA).

3.2.2. Uranium resource classification system

As can be expected, every country tends to develop its own terminology related to the different categories of minable resources. Figure 4 shows the correlations of terms used to classify resources in different countries. For the periodically published NEA(OECD)/IAEA report on "Uranium Resources, Production and Demand" (the "Red Book"), a resource classification for uranium was developed. It combines the degree of confidence in the existence of the resource with its economics. The system consisted of two parts, resource and cost categories.

The following definitions for the different resource categories are taken directly from the latest issue (1993) of the "Red Book".

"Reasonably Assured Resources (RAR)

refers to uranium that occurs in known mineral deposits of delineated size, grade and configuration such that the quantities which could be recovered within the given production cost ranges with currently proven mining and processing technology, can be specified. Estimates of tonnage and grade are based on specific sample data and measurements of the deposits and on knowledge of deposit characteristics. Reasonably Assured Resources have a high assurance of existence."

"Estimated Additional Resources — Category I (EAR-I)

refers to uranium in addition to RAR that is estimated to occur, mostly on the basis of direct geological evidence, in extensions of well-explored deposits, or in deposits in which geological continuity has been established but where specific data, including measurements of the deposits, and knowledge of the deposits' characteristics are considered to be inadequate to classify the resource as RAR." EAR-I category of resources has less reliance than those for RAR.
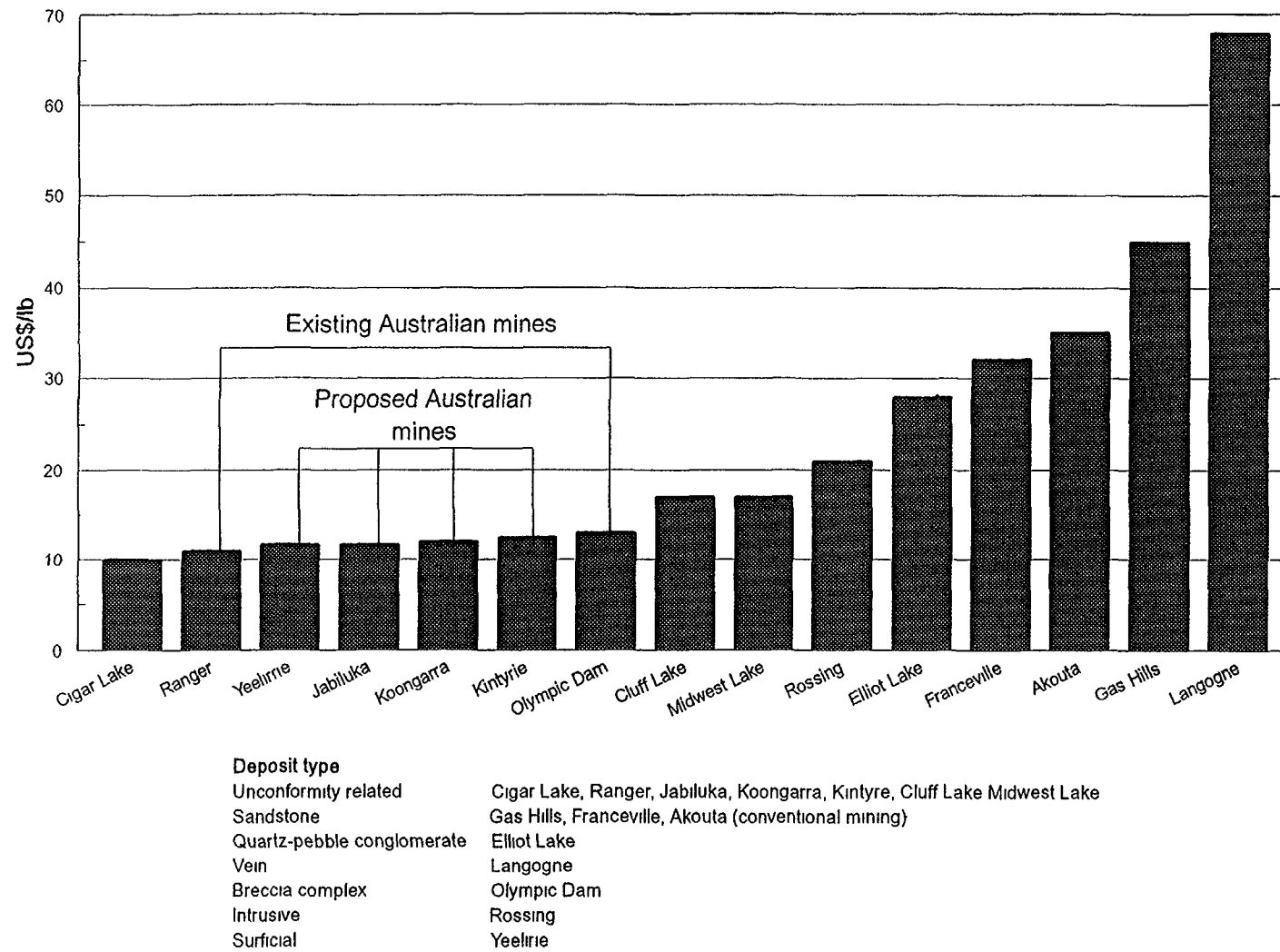
FIG. 3. Production costs of selected uranium deposits.

	KNOWN RESOURCES			UNDISCOVERED RESOURCES	
NEA / IAEA	REASONABLY ASSURED	ESTIMATED ADDITIONAL I	ESTIMATED ADDITIONAL II	SPECULATIVE	
Australia	REASONABLY ASSURED	ESTIMATED ADDITIONAL I	UNDISCOVERED		
Energy, Mines and Resources Canada	MEASURED	INDICATED	INFERRED	PROGNOSTICATED	SPECULATIVE
France	RESERVES I	RESERVES II	PERSPECTIVE I	PERSPECTIVE II	
Fed. Rep. of Germany	PROVEN	PROBABLE	POSSIBLE	PROGNOSTICATED	SPECULATIVE
South Africa	REASONABLY ASSURED	ESTIMATED ADDITIONAL I	ESTIMATED ADDITIONAL II	SPECULATIVE	
United States DOE	REASONABLY ASSURED	ESTIMATED ADDITIONAL	SPECULATIVE		
CIS	A+B	C1	C2	P1	P2	P3
CSFR, Hungary, Romania	REASONABLY ASSURED	ESTIMATED ADDITIONAL I	ESTIMATED ADDITIONAL II	SPECULATIVE	

The terms illustrated are not strictly comparable as the criteria used in the systems are not identical. "Grey zones" in correlation are therefore unavoidable, particularly as the resources become less assured. Nonetheless, the chart presents a reasonable approximation of the comparability of terms.

FIG. 4. Approximate correlation of terms used in major resource classification systems.

refers to uranium in addition to EAR-I that is expected to occur in deposits for which the evidence is mainly indirect and which are believed to exist in well-defined geological trends or areas of mineralization with known deposits. Estimates of tonnage, grade and cost of discovery, delineation and recovery are based primarily on knowledge of deposit characteristics in known deposits within the respective trends or areas and on such sampling, geological, geophysical or geochemical evidence as may be available. Less reliance can be placed on the estimates in this category than on those for EAR-I."

"Speculative Resources (SR)

refers to uranium, in addition to Estimated Additional Resources-Category II, that is thought to exist, mostly on the basis of indirect evidence and geological extrapolations, in deposits discoverable with existing exploration techniques". The existence and size of this category of resources are obviously speculative.

The "Red Book" system employs 4 cost categories: $40/kg U ($15/lb U_3O_8) or less; $80/kg U ($30/lb U_3O_8) or less; $130/kg U ($50/lb U_3O_8) or less; and $260/kg U ($100/lb U_3O_8) or less. Figure 5 shows the interrelation between resource and cost categories. The shaded areas, because of their higher degree of confidence of their existence and low recovery cost are considered as reliable information for a long-term planning or projection in a supply-demand analysis.

4. PRE-EVALUATION

In this guidebook, the first engineering and economic assessment of a mineral discovery is termed the pre-evaluation stage of the study. It is intended to indicate the general merit of a uranium source or deposit. It is often used to prioritize several potential exploration targets so that resources can be focused on the most promising. It must be done as soon as the exploration team has the feeling that they have discovered or intersected something which could be a minable deposit. The pre-evaluation can also be used to reject projects that have little or no worth.

One could argue that only a few projects will be put aside at this stage. That may be correct, but even if the percentage of rejected projects is low, it is worth, to have taken the time for a rough study which could avoid useless expenses.

Moreover, the geology and the morphology of a deposit are often complicated and different from one to another. In this regard, it is very good for the project that the exploration team stops the drilling campaign, to collect, analyse and compare the data and, with the help of the miners and the metallurgists, brainstorm the project. At the end of this process, there will be a better understanding of the project for all parties and costly mistakes will be avoided. A "no-go" decision can be taken at this time, allowing a saving of money which could be used in a better way, for new and more potentially profitable projects, for instance. If a go decision is taken, which will be the case for a majority of the projects, the study will have been of great value to determine the requirements needed for the next step and at the lowest cost. The pre-evaluation process includes a preliminary selection of mining and processing methods based on knowledge of the uranium source. This is followed by factored estimates of the project capital and operating costs, and an initial economic evaluation using realistic marketing/price data.
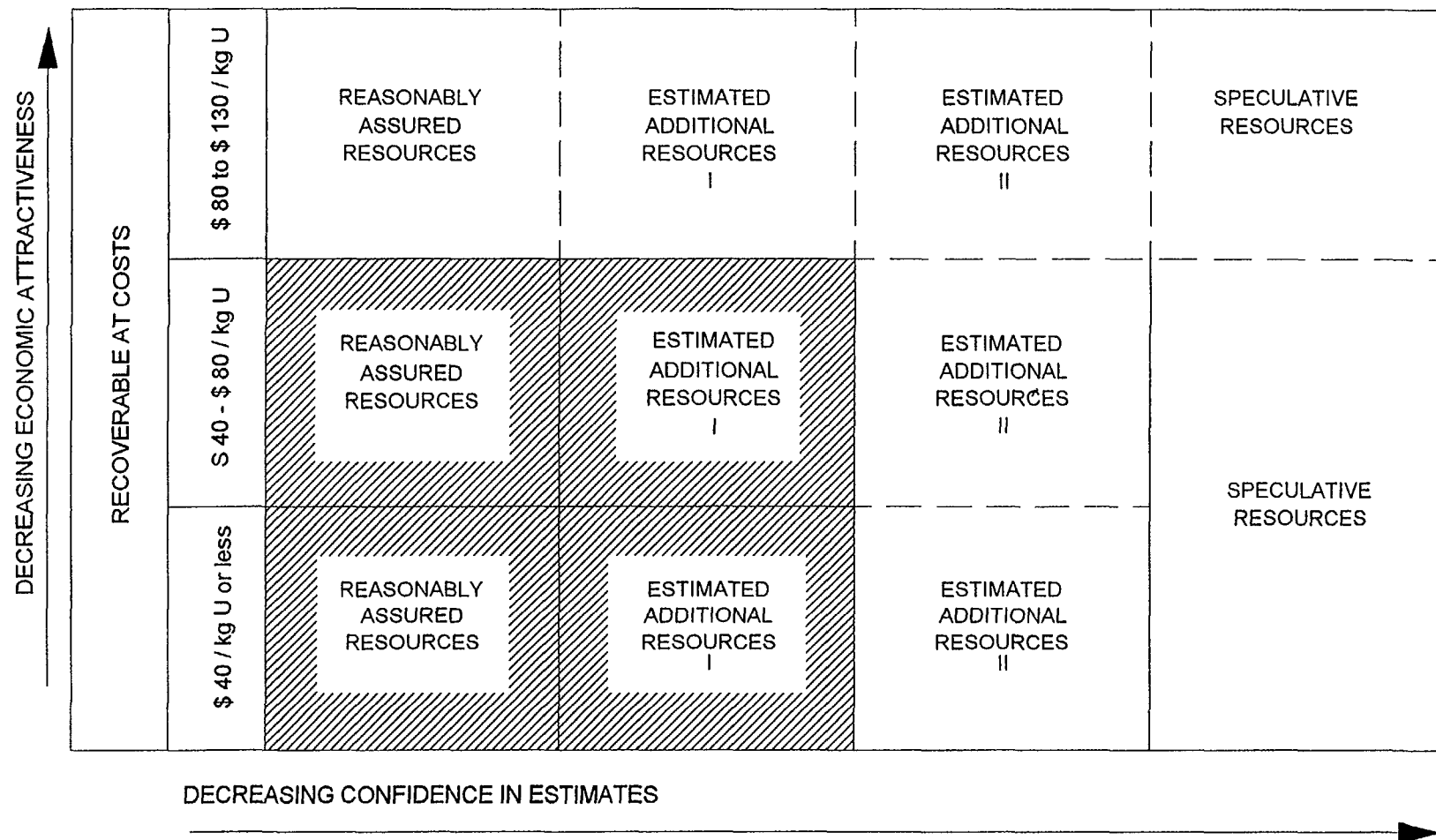
FIG. 5. NEA/IAEA classification scheme for uranium resources.

4.1. REQUIREMENTS

The requirements for a pre-evaluation are detailed below. In general terms, it can be said that those preparing the analysis should have an appreciation of the demand for the product and its market price, preliminary estimates for the tonnage and grade of the deposit or source, some idea of the mineralogy or chemistry, and an appreciation of the local infrastructure and geography. These basic data can then be applied to develop a very preliminary model of the project which can be used for the initial economic analysis. There are number of inexpensive computer software available in the market that can facilitate this type of preliminary assessment.

Every effort should be made to obtain all reasonably available data concerning the proposed project. The more information available, the more accurate the assessment will be.

Any history of earlier mining or mineral recovery efforts in the general area or the prospect should be researched. Old records can often provide information that will be useful in subsequent design work. It is frequently useful to talk to old miners or plant operators. They can often tell the investigators much about conditions underground, the processing route, problems, and other anecdotal material.

The investigators should obtain all the geological information they can obtain. It is often helpful to view drill core or test pits and trenches. These examinations can provide indications of rock strength, hardness, abrasiveness and other factors useful in a preliminary assessment of the mine and mill design.

Although the requirements are minimal, one should not undertake the pre-evaluation stage without the following:

— A minimum of percussion drilling and core drilling and a first estimate of assumed ore reserve and an idea of the radium-uranium correlation (radioactive disequilibrium factor).

— The cores will be useful to make the following tests:

  · Some uniaxial compressive tests,
  · A measure of water content and absolute density,
  · Some laboratory metallurgical tests and analyses.

In addition, it is recommended, especially in the case of vein deposits, to make measurements of the deviation in the drill holes in order to make, a rough determination of the thickness and the dip.

— As much analytical data as possible should be obtained. A uranium assay on its own will tell the investigator very little about the orebody and potential processing routes. It is important to obtain assays for other elements such as molybdenum, vanadium, thorium, gold, etc., all of which can seriously influence the way the ore is processed. Other important assays should be sought including those for sulphur (sulphide) and carbonate. If some preliminary uranium leach test or extraction data can be obtained, it will considerably enhance the accuracy of the pre-evaluation process.

These measurements and tests will be of great interest to the mining engineers to define the most efficient mining method and to the metallurgists to choose the best way for treating the ore.

4.2.   GEOLOGY

Some basic exploration objectives must be met before one enters into the pre-evaluation stage. Presumably a uranium mineralization that might offer a good economic potential, such as a reasonably thick ore intersection in some of the exploration holes, has been found. Having a good idea on the regional and local geological setting of the mineralization, the exploration geologist should come up with a model where the newly discovered prospect may fit into one of the known classes of uranium deposits discussed under Section 3.2.1. This task is easier if one works in a well established uranium district. Thus allowing the use of the most commonly found deposits in that district as a working model. However, in new areas, generally only limited information is known. The geologist should therefore be willing to modify his/her model with newly acquired information. It is quite often the case that because of an inflexible idea on the expected deposit (model), wrong exploration techniques are employed which result in a waste of precious time and resources. Worst of all, the project might have been abandoned prematurely only to find out much later, through the work of others (competing companies), that the model used was wrong. It should be borne in mind that even deposits of the same type are rarely identical.

Having established an appropriate model, through the limited data that were obtained from the exploration work, the geologist should try to define the likely average grade, extent and size of the expected deposit. Depending on the deposit type, prediction on its physical and chemical characteristic may be simple or very complex. Generally, a more uniform, stratabound, but low grade mineralizations (such as quartz-pebble conglomerate, some sandstone, some unconformity related, intrusive, phosphorite and black shale types) are easier to predict than higher grade, structurally controlled orebodies (as in many unconformity related, veins, collapse breccia pipe, volcanic types, etc.). A certain degree of confidence on these characteristics can normally be obtained with fewer samples (around 10 drillholes) in an orebody which is homogeneous with simple configuration. A complex orebody may require twice or more the number of holes that need to be drilled in order to obtain the same level of information. Hence, the exploration cost for a complex deposit is generally much higher.

At this stage, the geologist should also try to collect information considered useful for the metallurgist and mining engineer in their assessment of the most appropriate methods for the processing and mining of the deposit. These include:

- size, geometry and depth of the ore zone,
- grade, its variability (heterogeneity of the deposit), radioactive disequilibrium,
- mineralogy of the ore, including those of the associated elements,
- mineralogy of the gangue materials,
- physical properties of the host and adjacent rocks (grain size, density, massive, semi-consolidated, fractured, friable, porosity, permeability, impermeability of the inclosing rocks, etc.),
- hydrogeological information,
- available exploration data that can be used as baseline information for environmental impact studies (regional and detailed radiometric, and geochemical survey data).

Some of the needed information noted above might be available from previous works in the area. However, most of them will probably need to be created through a systematic survey, surface and sub-surface sampling and analysis. In sampling, effort should always be made that they are as close as possible of being representative of the expected orebody. The confidence level of the expected information collected or assumed at this stage of evaluation is relatively low. The estimate on reserve or resource figures at this stage is obviously very subjective. It is likely to be in the Possible or Estimated Additional Resource — Categories I or II described in Section 3.2.2.

4.3.   MINING CONSIDERATIONS

Evidently, the factors to be considered during mine design are:

— The morphology of the orebody, the thickness for the sedimentary deposits, the thickness and dip for the vein type deposits.

— The ground conditions including compressive strength measurements and, when dealing with clayey material, additional data such as triaxial compressive strength and the internal angle of slip.

— The water inflow which will show, if it is necessary, to dewater with external wells or to use pumps from within the mine workings.

— The radioactivity level calculated from the value of the grade, the mining method, and the water inflow which may bring radon with it and pollute the air of the mine. The ventilation of the mine will be designed accordingly.

These characteristics and considerations are sufficient to make a first attempt at determining the most suitable mining method, at least from a mining point of view. Actually, the choice of mining method is not so simple. The study team must keep in mind that there are interactions and impacts between the mining method and the metallurgical process. So, a project must be taken as a whole and in many cases, the best solution will be a compromise between the mining and the process.

The possible role of physical and mechanical ore sorting techniques should be reviewed. These methods can, at low cost, increase the feed grade to the processing plant. The consequence is that the process cost can be lowered, sometimes by a significant factor. The mining cost might also be reduced because, as the project now has a cheap technique for increasing the grade of the ore, a less selective and consequently more economical mining method can be used.

In the case of uranium mining and milling, the following might apply:

Mining cost using a selective method is estimated at . . . .   100 units per tonne of ore
Milling cost is also estimated at . . . .   100 units per tonne of ore
Sorting cost is estimated at . . . .   5 units per tonne of ore

Assuming that the result of the ore sorting will double the feed grade and that a less expensive mining method can subsequently be used, the above figures can become:

Mining cost by bulk method . . . .   50 units per tonne of ore
Sorter operating . . . .   5 units per tonne of ore
Milling cost . . . .   50 units per tonne of ore

These figures show clearly that a significant increase of the grade through an inexpensive sorting technique can lead to a significant decrease of the total operating cost.

Even if the overall uranium recovery with sorting is lower, say for instance 80% instead of 95%, the total cost per unit produced shifts from:

$$\frac{100+100}{0.95} \qquad \text{to} \qquad \frac{50+50+5}{0.80}$$

or from 211 to 131 units per tonne of production.

The above figures are not meant to be exact, but rather, they show the potential cost reductions made possible by a thorough use of all the techniques in the field of ore sorting, and the usefulness

of joint planning by those involved in the mining and processing. The potential benefits of radiometric sorting, and other pre-concentration techniques, should be carefully reviewed in the pre-evaluation stage.

Another example is provided below which indicates other benefits possible through sorting.

Example:

A narrow vein type ore, with a rather low grade: 1.2 kg U per tonne (0.12% U), seems on first examination to be uneconomic because, due to high carbonate content, hence a high acid consumption: 200 kg, and consequently a processing cost which is too high.

Closer examination of the ore shows that it composes of a competent rock with almost no fines in it when broken. Furthermore, the ore is made of small pitchblende veins interlaced with carbonate, which is a waste. Such an ore calls for radiometric ore sorting. Due to the big difference of grade between the veinlets of ore and waste, radiometric ore sorting can improve the grade by a high factor and reject the acid-consuming carbonates. The reduction in the treatment cost and reduced mining cost through the use of a bulk mining method, turned a marginal deposit into a viable operation.

4.4. PROCESS OPTIONS

Preliminary selection of a process at the pre-evaluation stage is necessary for a reasonably accurate initial assessment. The optimum process is dictated by the nature of the ore and the form of the uranium. Key factors are discussed below and illustrated in Fig. 6.

4.4.1. Primary uranium deposits

There are four broad options for the processing of an ore deposit in which uranium is the main commercial mineral. Radiometric sorting can be used ahead of any of these options. The ore can be either leached in situ; mined and leached in vats or heaps; mined, ground and leached in agitated vessels; or ground and processed by physical means using, for example gravity concentration or froth flotation. There are several subsidiary choices within each main option. For example, conventional crushing, and rod/ball milling might be used or semi-autogenous grinding (SAG) and ball milling. Similarly, either acid or alkaline leaching might be an appropriate system after the ore has been ground.

The main factors determining the process selection are the grade, tonnage, local geology and mineralogy. Several of these factors are treated in detail in an IAEA publication on Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores (1980). The following general guidelines are offered for the selection of a process during the pre-evaluation step.

– Any earlier uranium operations in the general area or in the same geological setting should be used as a guide.

– Permeable, lower grade deposits sandwiched between impermeable strata may be suitable for in situ leaching.

– Lower grade material (less than 2 kg U/t) containing readily leached uranium might be best treated by heap or vat leaching. This can be particularly attractive if the rock contains pyrite which can generate sulphuric acid with bacterial assistance.

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Uranium Source — | In-Situ Deposit | | By-Product Solution |

Mining Method — | Solution | | Conventional | | Hydraulic |

Primary Benefication — | Sorting |

Comminution — | None | | Autogenous | | Crush. & Grinding |

| Heap Leach |

Secondary Benefication — | Benefication or By-Product Recovery |

Leaching System — | Acid | | Alkaline |

Solution Concentration Process — | Filters | | Thickeners | | Desand | | Filters | | Thickeners | | Desand |

Uranium Concentrate Process — | Direct PPT | | SX | | IX | | RIP | | Direct PPT | | IX | | RIP |

Additional Processing — | Dissolve & Reprocess | | SX | | Dissolve & Reprocess | | SX |

| Uranium Concentrate To Market |

Note IX- Ion Exchange
SX- Solvent Extraction
RIP- Resin In Pulp

FIG. 6. Ore processing options for uranium.

- Large, higher grade orebodies (containing more than 2 kg U/t) are probably best treated by grinding and leaching.

- Acid or alkaline leaching systems can be selected after initial tests and studies. In an initial assessment, and without the benefit of leach tests, it can be assumed that alkaline leaching will be preferable *only* if the CO_2 content of the ore is greater than 5%.

- There should be enough date at the pre-evaluation stage to consider coarse ore sorting and other pre-concentration techniques.

4.4.2. By-product uranium

There may be an opportunity to recover uranium from an existing facility processing ore for the recovery of some other material. Several examples of this type of plant include the recovery of uranium from South African gold plants, uranium extraction from copper heap leach solutions in the United States, and uranium recovery from phosphoric acid, in the Southern United States. Beach sand, oil sands and other large tonnage operations may also have the potential to economically yield by-product uranium, such as those found in South Africa, Malaysia and other parts of the world.

There is ample precedence to provide guidelines for the initial selection of a processing route. Copper heap leach solutions have been successfully processed using up-flow ion exchange. Phosphoric acid is best processed using solvent extraction techniques and DEHPA-TOPO as an extractant. The recovery of by-product uranium from ore processing plants would probably follow a route including acid leaching, liquid-solid separation, and ion exchange or solvent extraction. Such a process might be applied before or after the existing recovery processes.

4.5. PROJECT OPTIONS

In addition to selection of the mining and processing routes, there will be several other decisions to be made that will affect the project development.

4.5.1. Production rate

One of the most important criteria to be selected is the uranium production rate. The production rate has a major impact on the capital and operating cost, and the potential profitability of the project. With a very large reserve, the higher the production rate, the lower is the overall cost of production expressed on a per kilogram of product basis and the higher is the project profitability. Yet, as stated previously, there must be a well defined commercial market for the product. However with finite resources, the life of the operation becomes important. This is illustrated in Fig. 7.

Figure 7, taken from a typical project, illustrates the importance of project life and how its selection can influence the profitability of the project. Too high a production rate requires an inordinately high capital cost and, because of the short project life, brings earlier decommissioning costs which adversely affect the cash flow. Conversely, a too low production rate increases the unit operating costs and might make the project uneconomic.

At the pre-evaluation stage, it is suggested that a production life of 10 years be considered. The processing rate may then be computed from the anticipated ore reserve. As an example we might consider a deposit with an expected reserve of about 15 000 t of uranium at a minable grade of 3 kg U/t. The pre-evaluation study would consider a production rate of 1 500 t U/a or an average ore processing rate of about 1 500 t/d. Subsequent stages of the development process would refine this estimate.
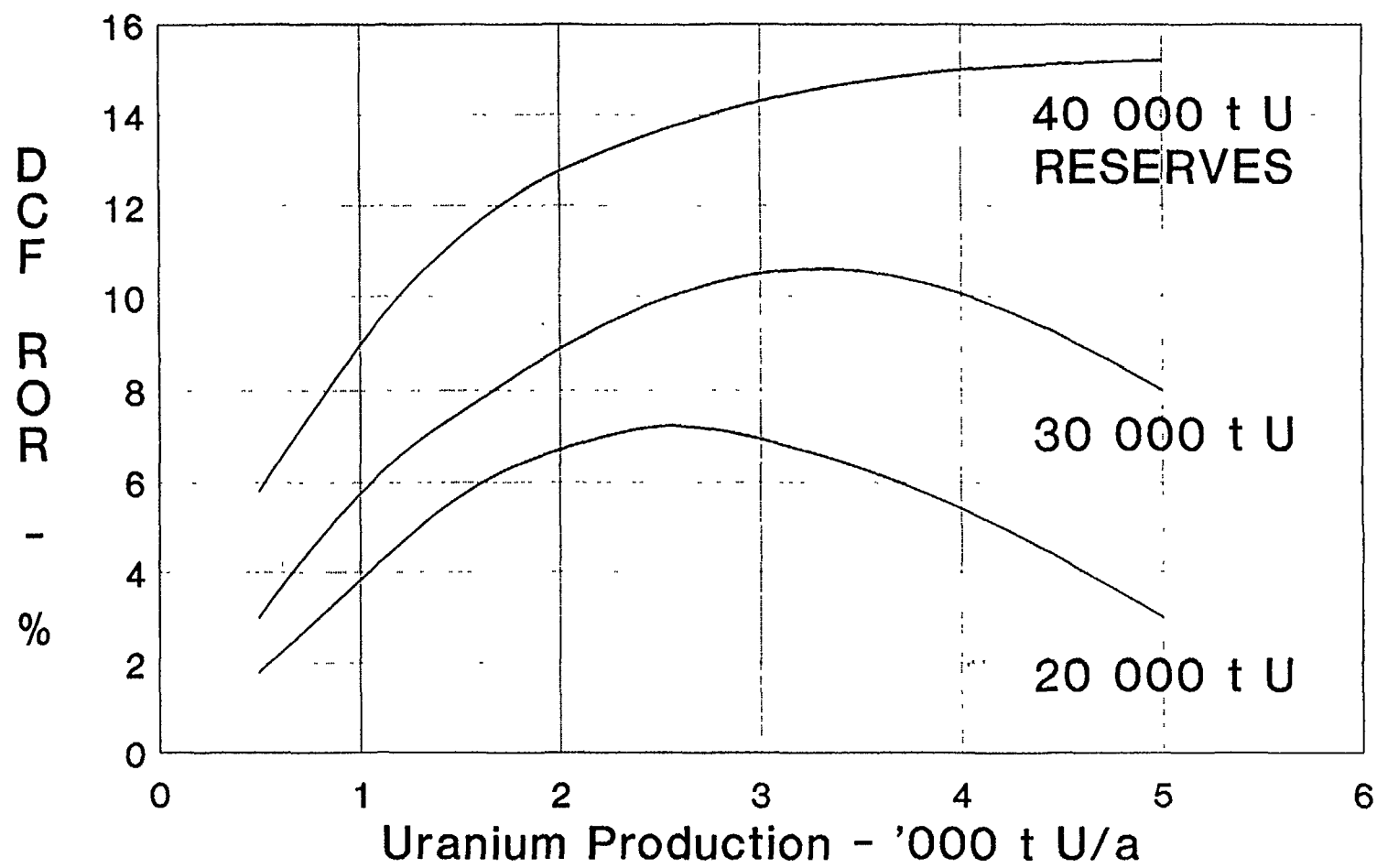
FIG. 7. Effect of reserves and production rate on DCFROR.

It should be noted that the configuration of the orebody might impose practical limits on the production rate. Furthermore, the production rate must reflect market conditions.

In the case of by-product uranium production, it is usual for the processing rate of the primary product to set the maximum production rate for uranium. As an example, a phosphoric acid plant producing acid containing 100 000 t/a of P_2O_5 might produce a weak acid containing 350 kg/m^3 P_2O_5 and 100 g U/m^3. This plant could produce no more than 29 t/a of uranium unless phosphoric acid production were increased.

4.5.2. Development strategies

The location of the uranium deposit with respect to other centres of population in the country can influence the way in which it is developed. If the deposit is within about 100 km from a population centre then it will probably be serviced and manned from that centre. However if the deposit is located at a great distance from an existing town it may be necessary to either establish a new town, or consider a fly-in/fly-out operation. In the later type of operation, the workers are flown to the site where they spend one or more weeks working 12 hours per day. They stay and eat in a company operated camp while they are on site. After spending their time on site, they return to their home for a similar time period. If this type of operation is likely, it should be recognized at an early stage. The capital cost to establish a camp and airstrip are considerable. The cost of operating the camp and the air shuttle service are also significant and must be added to the operating cost estimates.

Another consequence of the remote location must also be computed. A project at the end of a long supply line has special requirements regarding installed and warehoused spare equipment, reagent storage facilities, tankage for diesel and other fuels. This increases both the direct capital cost and the working capital for the project.

In some cases it may make sense to limit the activity at the mine site to mining of the ore. Factors that might, alone or in combination, influence this strategy include a very rugged, remote or inhospitable site, lack of tailings storage area, lack of grid supply of electrical power, and very high value ore. In such cases, ore might be transported to a processing site located in a more convenient area.

4.5.3. Economic evaluation

Except for unusual cases, the data available at the pre-evaluation stage will not be sufficient to allow a rigorous analysis of the economics of the project. Two alternative methods of analysis are available.

4.5.3.1. The comparison approach

The policy recommends that when preparing for the go or no-go decision, one can compare the project to existing operations or more advanced projects. This approach takes into account real experience and gives a good view of what is to be done.

Projects located in the same country

When the projects or the operations to be taken for comparison are located in the same country, the reconciliation of the different costs and other economical data is not too difficult. Some amendments are to be made such as:

the costs of the transport due to the location of the mines,
the labour skills and costs,
the availability of consumables,
the cost of energy and electricity availability, and,
the possibility of contracting parts of the job.

Projects located in different countries

When the projects are located in different countries, it may be dangerous but nevertheless is sometimes necessary to make a simplified mathematic calculation using the currency exchange rates.

It is recommended to send a small team, to the country where the model project is located. The task of the team is to collect the most accurate data on the unit costs, the working legislation, the possibility of contracting or not. When these data have been correctly collected, it is then possible to prepare a good analysis of the economies of the new project according to the specific costs and working conditions for the two projects being compared.

The main data to be collected are:

-   the cost of labour,
-   the cost of energy and the electricity availability,
-   the cost of transport,
-   the cost of the most important other consumables:
    -   explosives,
    -   drilling rods and tools,
    -   spare parts, gas, oil, tires,
    -   timber,
    -   cement and concrete,
    -   acid, ... and other reagents
    -   and others.

Similar data has to be collected for the capital cost, factors peculiar to the location of the project. Data is also required on interest rates and taxes.

Having obtained such data, the economics of the project can be estimated and conclusions and recommendations made.

4.5.3.2. The handbook approach

If it is not possible to have access to data for existing operations, or if the size of the project under study does not bend itself to a comparative approach, a good alternative is to use the handbook approach.

4.6.   COSTING

4.6.1.   **Economics**

Economic analysis at the "pre-evaluation" level serves as a basis to measure the economic acceptability of a prospect project. It requires an appreciation or understanding of the confidence factors that should be applied to this evaluation process. Suffice to say, applying "typical" mine and mill models to a possible deposit type will generate cost estimates that, at best, only approximate what actual costs may be. However, these estimates are still valid and useful for evaluating uranium deposits

on a comparative basis. The procedure effectively helps eliminate subeconomic deposits and target those deposits with the best economic potential. As the evaluation process continues from prefeasibility to feasibility-level studies, results become more quantitative with a correspondingly higher confidence level.

Assessing the economics of uranium deposits requires consideration of multiple variables operating independently as well as in concert. Variables such as location, topography, climate, production capacity, mine and mill methods, metallurgy, marketable products, prices, transportation, taxes, infrastructure, and financing must all be considered when evaluating properties. It is important when doing comparative analysis that these variables be addressed and applied on a common basis.

The cost models and procedures for estimating costs as presented in this document are guidelines. They are based on experience factors as they apply to the economic extraction of uranium from deposits in the United States. It has been left to the reader to make the appropriate adjustments for other countries and economic conditions. These adjustments include costs related to contractual commitments to private mineral owners and financial obligations, such as taxes and royalty payments, restrictions and incentives in accordance with policies, laws and regulations established by the host country. Labour, equipment, and supplies percentages, presented in the text, may be useful for modifying cost estimates to account for site specific conditions.

All values and equation results are presented on average 1990 US dollars.

4.6.2. Exploration

Estimating exploration costs is the first step in the evaluation process. After delineating target areas, a standardized checklist of exploration procedures, costs, and logistical considerations should be developed. In all mineral resource programmes, the exploration phase has the highest element of risk. Most exploration programme does not end up with minable deposit. The exercise, however is still useful since it can provide information on the economic potential of the area or the lack of it.

An accurate information on exploration cost is difficult to obtain. Simply, because most unsuccessful exploration activities are not accountable. It also depends on whether basic regional geological, geophysical and/or geochemical information needed to assess the favourability of the target area are available prior to the actual work. Where such an information is not available, it has to be generated at cost. This information may be made available by the Government or derived from previous exploration works in the area, even for completely different mineral commodities. As an example, early exploration for sandstone type uranium deposit in the USA benefitted a great deal from information generated by previous oil exploration in the area.

Of all exploration costs, drillings are usually the most expensive. For example, a "typical" deposit in the USA would require 1.1 million metres of exploratory drilling and 0.8 million metres of development drilling to develop a reserve of 9 to 10 million kilograms of recoverable uranium oxide. Based on average 1990 US dollars, surface exploratory drilling costs were $13.60 per metre[1] and development drilling costs were $13.03 per metre[1]. These costs included ground surveys, road construction, site preparation, drilling, downhole geophysical surveys, sample collection and analysis, and geological and other support. Therefore, the estimated cost for exploration would be $15 million and development drilling would be $10.4 million. This translates into about $2.50 to $2.80 per kilogram of uranium oxide for all costs related to exploration and development drilling.

Although the following figures are rather old, they are still useful to give a general idea on the cost of finding 1 kg uranium. It may be noted that these figures reflect the cost when uranium exploration activities in the world were at the highest levels and new resources were discovered.

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[1] Costs have been updated and converted to metric units.

Canada (1971 to 1983)	Saskatchewan	$1.22
	(excl. Cigar Lake)	$2.01
	Canada (excl. Saskatchewan)	$10.86
	Total	$2.08
	(excl. Cigar Lake)	$3.23
USA (1972 to 1982)	Average	$12.41
Australia (1967 to 1983)	Average	$0.48

If one can assume that the figure for the USA is about normal (since deposits like Cigar Lake and Olympic Dam are not that common), to find a reasonable size deposit (say with a reserve of about 10 000 tonnes), an exploration expenditure in the order of $100 million to $150 million will be required. Even with such an expenditure, there is still no guarantee that an economic deposit will be found.

4.6.3. Uranium mining

4.6.3.1. Optimization

Once an orebody has been delineated, general assumptions may be made as to methods of ore extraction. Before cost models can be applied to each method, an optimum mine life should be established for the deposit. Optimum production rate can be determined in many ways, but for uranium, probably the most important determinant is market analysis and supply/demand analysis. For pre-evaluation estimations, a very simple rule can be applied as a common estimating basis. Taylor's Rule (1978) is an empirical formula for calculating optimal mine life. The rule states:

$$Life \ (years) \approx 0.2 \times \sqrt[4]{Expected \ ore \ tonnage}$$

4.6.3.2. Mine capital costs

General cost models are presented for open pit and underground mining operations. In situ leach mining operations are addressed in the beneficiation section. The models represent typical sedimentary-hosted uranium ore deposits within the United States. The models represent capital intensive operations, with associated US productivities, labour rates and burden. Ore production rates range from 900 to 5400 metric tonnes per day ore.

Typical open pit and underground capital costs are composed of 25–30% labour, 65–70% equipment, and 5–10% supplies.

Open Pit Model

Open pit mines are limited by the stripping ratio which is directly related to depth. As a general rule, maximum depth is about 150 metres although the grade of the deposit will affect the economics and therefore the ultimate pit depth. The break-even stripping ratio, and ultimate pit depth, is determined when the costs of removing ore plus waste equals the value of the ore minus beneficiation costs.

Two typical open pit methods are employed for overburden and waste removal; shovel and truck operations and scraper operations. Mining of ore, however, is usually by backhoe and truck or loader and truck. Shovel and truck operations are preferable when round-trip waste-hauls start to exceed 1.6 kilometres, or when digging conditions are difficult because of boulders or strongly cemented formations. Drilling and blasting may also be required when some of these conditions are encountered. Scraper operations are preferable when digging unconsolidated or loosely consolidated material. Bulldozers are often used in conjunction with scrapers and may contribute to a sizable percentage of material moved. They may also be used for ripping to eliminate the need for drilling and blasting.

When estimating capital requirements, it becomes readily apparent how many variables can affect the cost estimate. Type and combination of mining equipment, surface facilities, and infrastructure requirements (see Tables A-I.I and A-I.II of Appendix I) are items that need to be considered on a site specific basis. Despite all of the unknowns, a rough estimate can still be determined at this "pre-evaluation" stage. Excluding acquisition, exploration, and infrastructure costs, an estimate can be made using the following equation:

$$Capital\ Cost\ =\ 207.3\ (tonnes)\ +\ 2\ 500\ 000$$

Where: The equation is of the form $Y = Ax + B$;
"Capital Cost" is on average 1990 US dollars;
207.3 is the constant "A";
"Tonnes" is the daily mine capacity of ore plus waste;
2 500 000 is the constant "B".

Underground Model

Underground mines are developed when the depth of the orebody generally exceeds 120 metres up to about 1400 metres. Underground mines typically produce higher grade ore than surface mines which offsets the higher capital and operating costs. Production rates are generally less, typically between 300 to 2300 metric tonnes of ore per day, and mining methods may include modified room-and-pillar, cut and fill, sublevel-open stope, and variations of each method.

General mine plant facilities would be similar to the open pit facilities with three notable exceptions: shafts and headframes, ventilation facilities, and a larger power substation. Excluding shafts and headframes, general surface facilities approximate 25% of total mine capital costs.

Shaft sinking is the largest capital cost item. Depending on depth and number of shafts required, the cost can range from 30 to 50% of total mine capital. Shaft sinking costs can be extremely variable due to rock conditions. Production shafts with 3.9 to 4.3 metre diameters will range from $16 300 to $17 600 per metre. Ventilation shafts, commonly 1.8 metres in diameter, will cost approximately $8800 per metre[2]. When poor ground conditions are encountered, costs can range from $30 000 to $50 700 per metre.

Underground development costs depend on the type of mine plan, amount and type of development required, and mine equipment utilized. Typical development costs, excluding shafts, will include haulage drifts, cutouts, crosscuts, raises, manways, undercuts, and stope preparation. Development costs account for approximately 20% of total mine capital costs.

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[2]   Costs have been updated and converted to metric units.

$$\textit{Capital Cost per mt ore } (\$\times 10^6) = 0.002 \; (\textit{Depth}^{.67} \times \textit{Tonnes}^{.81})$$

Where: The equation is of the form $Z = C(X^A Y^B)$;
"Capital Cost" is on average 1990 US dollars;
0.002 is the constant "C";
"Depth" is the hoisting depth in metres;
"Tonnes" is the daily mine capacity of ore.

4.6.3.3. Mine operating costs

The following mine operating cost equations for the "pre-evaluation" stage of property evaluation may be used to roughly approximate costs.

Open Pit Model

The open pit model is based on a combination of scrapers and dozers for mining of overburden and waste, and loaders and trucks for ore and waste. The strip ratio is 40:1 and the pit depth is approximately 60 metres. Drilling and blasting is not required but ripping with dozers is required. The cost model equation is presented below:

$$\textit{Operating Cost } = 0.004 \; (\textit{Tonnes}) + 34.43$$

Where: The equation is of the form $Y = Ax + B$;
"Operating Cost" is on average 1990 US dollars;
0.004 is the constant "A";
"Tonnes" is the daily mine capacity of ore;
34.43 is the constant "B".

Adjustment factors: Labour: 55%, Equipment: 30%, Supplies: 15%

Underground Model

The underground cost model should approximate a modified room and pillar mining method with pillar extraction, a sublevel open stope mining method, and a cut and fill mining method. The model should also approximate a mining method which uses combinations of the above methods. The cost model equation is presented below:

$$\textit{Operating cost per mt ore } = 75.4 \; (\textit{Depth}^{.1} \times \textit{Tonnes}^{-.2})$$

Where: The equation is of the form $Z = C(X^A Y^B)$;
"Operating Cost" is on average 1990 US dollars;
75.4 is the constant "C";
"Depth" is the hoisting depth in metres;
"Tonnes" is the daily mine capacity of ore;

Adjustment factors: Labour: 65%, Equipment: 5%, Supplies: 30%

4.6.3.4. Decommission/reclamation costs

Decommissioning the mine and mill facilities after the completion of mining are important considerations in this age of environmental awareness. Costs for these activities for a "typical" western United States, high-plains arid-region are in the order of $7500 per hectare on average 1990 US dollars.

4.6.4. Uranium processing and milling

Knowledge of the orebody, the physical properties and characteristics of the host rock and uranium mineralization are needed for the selection of the proper and suitable process method to successfully recover uranium. Important factors having an influence on the uranium processing include the size of the uranium minerals, their arrangement in the host rock matrix and the presence of other minerals and gangue constituents. In addition, the chemical nature of the ore determines the processing method which would be most favourable for the recovery of uranium. Minerals with uranium appearing in the hexavalent form are normally easier to process than those with uranium in the tetravalent form.

To get the uranium in a soluble chemical form, an oxidizing environment has to be created. For this purpose, the acid-leach processing uses a dilute sulphuric acid, while an alkaline carbonate is the reagent in the alkaline-leach circuit.

Where a mine is remotely located in relation to that of the milling facilities, preliminary treatment of the ore at the mine site may be economically justified. In this case, beneficiation techniques may include concentration of the uranium, separation of ore into portions containing, for example, high or low lime, sulphides and carbon.

4.6.4.1. Milling capital costs

Capital investments for milling and processing uranium ores depend largely on the extent of the ore reserves, ore grade, projected ore mined and expected kilograms of uranium to be recovered.

Capital cost for alkaline and acid leaching processes are not significantly different. Therefore, one cost model was developed to be used to estimate capital costs for both of the processing methods.

The capital cost consists of cost items related to purchase of equipment, construction of the milling facilities, labour and transportation. Expenses such as engineering and consultants costs, working capital, startup and recruiting costs are here lumped together as "miscellaneous" costs. If so desired, these costs can also be considered separately in preparing capital cost estimates.

Typical capital investments for uranium processing facilities at different production levels is listed in Table A-I.IV of Appendix I.

The listing also shows a percentage breakdown for the items included in the capital cost. The percentages are illustrations of typical values of the individual costs as indicated on the table.

The following equations can be used for estimating capital costs for uranium milling operations.

Capital Cost

High estimate: $Y = (328 \times 10^3) \times X^{0.6909}$

Low estimate: $Y = (154 \times 10^3) \times X^{0.7033}$

Where: Y = Capital cost ($);

X = Mill capacity (metric tonnes/day) (see also Fig. A-I.1 of Appendix I).

The above capital cost estimates do not include costs such as land acquisition, leasing costs, taxes and expenses for environmental studies, licensing and permits, if so required under the laws of the host country. This type of expenses are additional costs. Due to site specific geographic variations, costs associated with the infrastructure have to be added on to the estimated capital costs also.

4.6.4.2. Milling operating costs

Cost elements contributing to the total operating expenses include:

I. Direct costs:

-   Operating and maintenance labour
-   Operating and maintenance materials and supplies
-   Chemicals
-   Utilities:
    -   · Electric power
    -   · Fuel.

II. Indirect costs:

These are costs which are not directly charged to a specific cost category of the operation but are needed to perform and manage the operation. Indirect costs may include cost items such as:

-   Administration
-   Communications (telephone, mail, teletype, etc.)
-   Travel
-   Taxes (income, property, etc.).

Typical operating costs and a cost breakdown (in %) for uranium milling are shown in Table A-I.V of Appendix V. The percentages are typical values of the listed individual cost items.

The following cost model equations can be used for estimating operating costs:

	Mill Capacity	
	2000 tonnes/day or less	2000 tonnes/day or more
High estimate:	$Y = X^{-0.4165} \times (630)$	$Y = X^{-0.2180} \times (139)$
Low estimate:	$Y = X^{-0.4013} \times (354)$	$Y = X^{-0.1385} \times (48)$

Where: Y = Operating cost ($/metric tonne);
         X = Mill capacity (metric tonnes/day);
         (see Fig. A-I.2 of Appendix I)

Costs which depend on the local conditions and geography, or are required by governmental statutes, have to be estimated separately and are additional costs.

4.6.5. Uranium recovery by in situ leaching (ISL)

In situ solution mining or borehole leaching is a technology, which removes the ore mineral from the host rock through a system of wells, by means of leaching fluids and whereby the mineral values subsequently are recovered in soluble form for further processing. This technique of extracting minerals permits the recovery of mineral resources which are not otherwise available. Like any mining method, in situ leaching has its advantages and its disadvantages. Some of the advantages are development of low grade deposits, low capital cost, low labour requirements, and a short pre-production development time. Disadvantages include restrictive deposit characteristics suitable to leaching, poor ore recovery ranging from 30% to 80%, possible groundwater contamination, and treatment and disposal of large volumes of leach solutions.

The major factors affecting hole patterns and hole spacing are rock-type characteristics, such as permeability, orebody configuration, stratigraphy, and depth to the orebody. The wellfield development patterns should be designed on a site specific basis accounting for these characteristics.

The five-spot, the seven-spot and the line-drive are the three typical wellfield patterns widely used in the United States. Well spacings commonly used between injection and production wells are 10 to 15 metres for five-spot and seven-spot patterns and 5 to 10 metres for line drive patterns. Typical production rates are 40 to 80 litres per minute per production well.

To detect any escaping solution monitor wells are drilled at strategic locations around and outside the orebody. The number of monitor wells required is about 10% of the total wells drilled. The depth of the monitor wells must extend into aquifers below the ore horizon. Monitor wells should be drilled on 60 metre centers in the direction of the ground water flow. Perimeter spacings of monitor wells will usually not exceed 120 metres from the orebody and 120 metre spacing peripheral to the orebody.

Upon termination of the in situ operations, concentration levels in the groundwater of undesirable ions and impurities introduced into the formation through the leaching fluid have to be restored to baseline values or legally acceptable levels. Possible changes of water quality have to be monitored closely in wells strategically located around the area where leaching took place.

4.6.5.1. ISL capital costs

Cost items included in the capital cost for an ISL operation are as follows:

I. Fixed capital costs
   - Purchase of process equipment
   - Plant construction
   - Initial wellfield cost
   - Installation and site improvement
   - Pilot plant cost
   - Restoration system cost
   - Engineering/project management cost

II. Deferred capital costs

III. Contingency costs

Deferred capital costs and contingencies are approximately 4.2% and 10% respectively of the total fixed capital cost.

Deferred capital costs include replacement costs of the initial capital equipment and deferred expenditures for capital additions. Wellfields development includes surface preparation, drilling, casing, cementing and equipping injection, production and monitor wells. PVC pipe ranging from 10 to 15 centimetres in diameter is commonly used as casing. The casing is perforated at the ore intercepts.

Wellfield cost depend on the depth of the hole. Following are typical costs for well depths from 75 to 150 metres.

–	Drilling and well completion	—	$51 to $66 per metre
–	Equipment: Injection wells	—	$500 to $650 per hole
	Production wells	—	$3200 to $4700 per hole
	Monitor wells	—	$1100 to $1450 per hole

General surface requirements vary by the wellfield pattern used, hole spacing, diameter of pipe used, pipe insulation requirements, and distance to the beneficiation plant. Total required number of metres at the surface is needed to estimate the total installation cost, which is approximately $32 per metre and an additional $19 per metre if insulation is required.

Typical capital costs for an ISL operation, and a percentage breakdown for the cost elements, are listed in Table A-I.VI and Table A-I.IV-A, respectively, of Appendix I.

The model equation for ISL capital cost estimations is shown below:

Capital Cost

$$Y = (7.49 \times 10^6) \times (1.0067)^X$$

Where: Y = Capital cost ($);
X = Annual production (10^3 kg U_3O_8).

4.6.5.2. ISL operating costs

The following cost elements are included in in situ mining operations:

I. Direct costs
 – Wellfield cost
 – Manpower cost
 – Chemical (reagent) cost
 – Utilities
 – Operating and maintenance supplies
 – Makeup water cost

II. Indirect costs

These costs, which include general and administration costs (G&A), are not directly charged to a specific cost category. However, they are needed to run the operations.

Typical operating costs and a percentage breakdown of the cost items for in situ solution mining are shown in Table A-I.VII of Appendix I. The G&A costs were assumed at 5% of the direct costs.

For estimating ISL operating costs, the following cost model equations can be used:

	Annual U_3O_8 Production	
	250 000 kg or less	250 000 or more
High estimate:	$Y = X^{-0.2643} \times (1,168)$	$Y = X^{-0.1382} \times (243.5)$
Low estimate:	$Y = X^{-0.2474} \times (859.5)$	$Y = X^{-0.1287} \times (196.5)$

Where: Y = Operating cost ($/kg U_3O_8);
 X = Annual production (10^3 kg U_3O_8).
 (see Fig. A-I.3 of Appendix I)

For the same reasons as discussed previously to estimate operating costs for uranium milling, cost items, depending on the local situation and conditions, have to be estimated separately and are additional costs.

4.7. DISCUSSION OF SIMPLIFIED CASH-FLOW ANALYSIS

When evaluating the merits of a mineral deposit, an economic analysis is prepared in the early Pre-Evaluation and Pre-Feasibility stages and continues through as one of the last steps in a Feasibility study. The analysis is performed to provide decision makers (government or private concerns) with a measure of the potential of the mineral property in economic terms.

One prerequisite for making any investment is an investment criterion which will allow management to compare forecast results from the projects being investigated with a benchmark or required minimum or maximum of that particular criterion. There are numerous criteria or simple financial measures available, but for the purpose of this guidebook, Discounted-Cash-Flow-Rate-of Return (DCFROR) is used.

The DCFROR is the financial measurement that this guidebook uses as its method for criterion for investment decisions. To understand the DCFROR concept (including discounting and the time value for money), we must first discuss the concepts and components of cash flow analysis. A cash flow is the sum of net profits, depreciation, amortization depletion, and deferred deductions. It is the after-tax money that remains available to a company to pay off debts and invest in new projects from sales revenues after paying all of its operating expenses and taxes.

An example of a typical Pre-Evaluation stage (abbreviated) cash flow is shown below:

	REVENUES
less:	mine operating costs (MINE)
	mill operating costs (MILL)
	any and all transport operating costs (TRANS)
	any post processing operating costs (POST)
	leach (if any) operating costs (LEACH)
equals:	BEFORE TAX INCOME (BTINC)
less:	a very general, effective overall tax
equals:	NET INCOME
less:	equity investments (EQI)
equals:	NET UNDISCOUNTED CASH FLOW

At the Pre-Evaluation stage, there may only be a very limited amount of detailed information and performing a very elaborate analysis may not be necessary. However, setting up the structure to perform detailed DCFROR analyses for subsequent Pre-Feasibility and Feasibility stages, is a wise investment.

4.7.1. Brief description of the time value of money concepts

The concepts of "interest" and a "required rate of return" are used throughout the financial community. Interest is the rent paid for the use of someone else's money. The Required Rate of Return is the percentage or return on an investment that a project must make to justify an investment.

Money (i.e. currencies of any denomination) is worth more in the future because of its interest earning power. Alternatively, because of its interest earning power, a future sum of money must be "discounted" at an interest rate, to determine its value today, or more commonly described as its "present worth".

The determination of a present value of a future sum is termed "discounting" (or, present worthing). Cash flows are projected sums of money out into the future. So, when the future value is known, estimated or projected, the present value may be calculated. The time diagram below helps illustrate the formula used for the above discussion.

```
Initial invest.              Salvage value
PV          CF          CF       FV
 I-------------------I-----------------I-----------------I
YEAR        AV, FV      AV, FV     YEAR
```
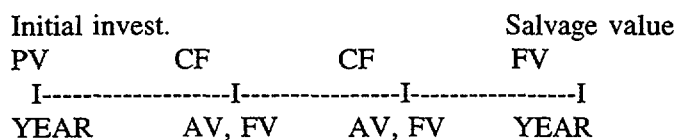
$PV = FV/(1+i)^n$

where: PV = present value
         FV = future value
         AV = annual value
         i = interest rate
         n = number of periods.

Discounted-Cash-Flow-Rate-of-Return (DCFROR) is defined most commonly as the rate of return that makes the present worth of cash flows for an investment equal to the present worth of all after-tax investments.

Net Present Value (NPV) on an after-tax basis equals the present worth of cash flows minus the present worth of after-tax investments calculated at the minimum rate of return (i). On a before-tax basis, project rate of return is the "i" value that makes NPV = 0. The rate to use in the analysis for time value of money calculations, is the rate that is thought to represent other opportunities for the investment of capital. This rate is called either the minimum rate of return, the hurdle rate, the opportunity cost of capital, or simply, the discount rate. On an after-tax basis, this project ROR is the project DCFROR.

When performing DCFROR analysis, one is calculating the unknown interest rate, i, using trial and error to determine the i value which makes project costs equivalent to inflows on a present value basis. This determined DCFROR is compared to rates that represent other opportunities that may exist. Whichever is higher is clearly the better investment choice.

4.7.2. Escalation and inflation

Escalation and inflation are not exactly interchangeable. Escalation refers to changes in price or value at a specific time of specific goods or services. Inflation refers to the change in price or value of some average aggregate grouping of goods (e.g. the Consumer Price Index (CPI) in the USA).

After the actual costs and revenues to be incurred over the life of a project have been predicted, the analysis can be performed in two fashions. Escalated dollars (otherwise known as current dollars, nominal dollars, or inflated dollars), or using constant dollars (which are sometimes referred to as real or deflated dollars). Either approach (constant or current dollars) is correct to use and will always give the same results, as long as comparisons are kept within the same dollars.

4.7.3. Hypothetical base case

The hypothetical base case information and potential Pre-Evaluation stage analysis results are as follows:

- A potential mineral occurrence exists with an estimated 15 000 tonnes of minable Uranium (U_3O_8) which would translate to approximately 4 000 000 tonnes of ore, allowing for a possible life of 12 years, at 320 000 tonnes of ore per year.

- The grade is estimated at 0.30 percent U_3O_8 (with estimated recoveries).

- The geologists suggest that it does not seem like a deep deposit, therefore the strip ratio for potential open-pit mining is estimated to be 10:1.

- For rough estimates for the necessary post mine processing, density of the rock is estimated at 2.5 t/cubic metre.

- Overall estimates for capital and operating costs are $72 000 000 for capital (not including the possible cost of acquisition) and $70.00/t of ore (not including any production royalty or finance charges) for operating costs.

- General effective taxes were estimated at approximately 50%.

Given the above hypothetical information, a preliminary DCFROR analysis was performed, and it was determined that a uranium price/cost of approximately $21.00/lb of U_3O_8 would be necessary to cover all costs and attain a pre-specified rate of return of 20%. Price is the term that is represented in the market place as the value that would be paid for a specific product. For this study, price and cost will be used interchangeably. Since the grade of the material and the capital and operating costs are Pre-Evaluation level gross estimates, a few sensitivity analyses (see Figs 8-A, B, C, D) should, and have been run against these figures to ensure that the determined price falls within or below the current market price, and thereby suggesting whether further investigation and analysis is warranted.

The impact on price while varying the feed grade from 0.20 to 0.50% and holding all other parameters constant, is illustrated in Fig. 8-A. If the feed grade were to decrease from 0.35 to 0.20%, the necessary price to obtain the same pre-specified DCFROR (20%) would increase by approximately 75% to $41/lb.

In Fig. 8-B, the determined price for U_3O_8 increases by about 65% as the capital requirements almost double. It should be noted, that this does not suggest that there is always such a dramatic correlation between capital cost requirements and price. Capital investments for the Pre-Evaluation stage are financed at 100% equity (no borrowed funds).
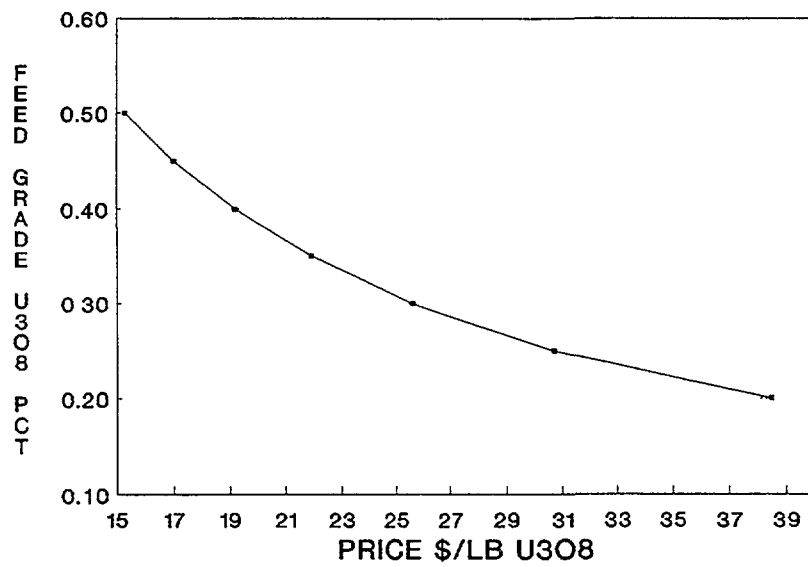
FIG. 8-A. Grade variations vs. price.
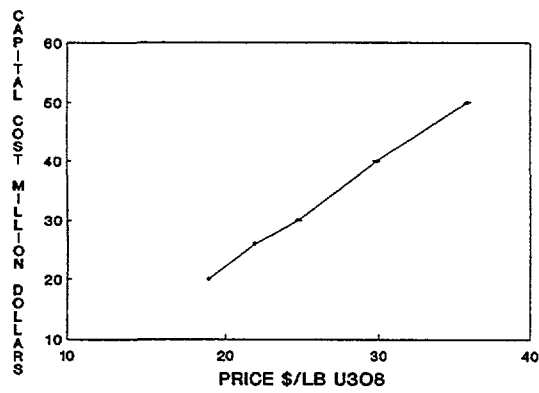
FIG. 8-B. Capital cost vs. price.
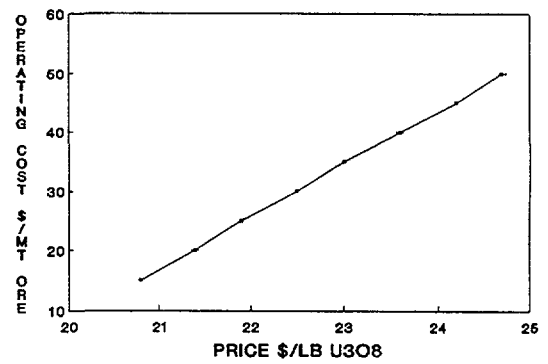
FIG. 8-C. Operating cost vs. price.
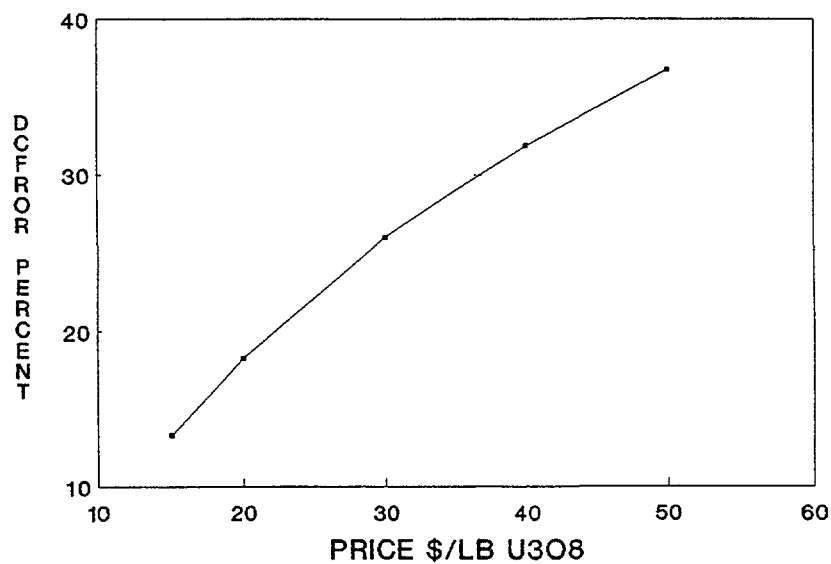
FIG. 8-D. Price Variation vs. DCFROR.

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The impact of operating cost is illustrated in Fig. 8-C. For this particular hypothetical operation, the operating cost seems to have a very small impact upon the overall economics. Again, as in the sensitivity analysis with the capital costs, this should not be perceived as the usual and natural trend. All mineral deposits are *not* alike.

The final sensitivity analysis performed during the Pre-Evaluation stage (Fig. 8-D) was a comparison of price and DCFROR. While the price per pound of U_3O_8 was varied between $15.00 and $50.00 for this particular case, the DCFROR rose from 13.3 to 36.8%.

It should be understood that these sensitivity analyses that have been presented, are portrayals of this specific hypothetical operation only, and they can most definitely be different given different base parameters.

Though the numbers suggest a "GO", the evaluating team should make sure that there is a market for their product. This should be monitored continuously, up to and until the final decision to develop is made.

4.8. CONCLUSION

At the conclusion of a pre-evaluation study, it should be decided whether the project should be allowed to proceed into the next phase or be aborted. Thus a go or no-go decision. The possibility to "revisit" the project should be kept open since some of the parameters used in pre-evaluation may change with time. A change of metal price, for example, may modify cut-off grade, hence possible increase of the minable resources that might reverse the no-go situation to go. A new process or mining techniques can also modify the outcome of a pre-evaluation study.

5. PRE-FEASIBILITY

The prefeasibility is undertaken when a go decision has been taken at the end of the pre-evaluation. At this stage of the procedure, we know that the project under study has some chance to be an economic one and that additional money can reasonably be spent in order to have a better understanding of the project and to improve the accuracy of the cost estimates and other key financial factors.

5.1. REQUIREMENTS

5.1.1. Resource information

Probably one of the most important parameter in any mineral feasibility study is the existence of a minable reserve or resource. The accuracy and precision of information related to its grade and tonnage is becoming more important in the pre-feasibility study as compared to the previous evaluation stage. It is obvious that the degree of confidence on this estimate is dependent upon the reliability of the data from which it is based. It needs emphasizing that no amount of sophisticated mathematical manipulation can compensate for poor quality data.

By the time a project reaches this stage, it is expected that sufficient amount of data have been collected upon which estimation of in-situ or geological resources can be made. The most valuable information is generally generated from the systematic drilling programme which objective was to

define the likely ore characteristic, grade, geometry and limit of the mineralized body. In some cases, a number of development adits might have been available thus allowing a closer investigation into the distribution and nature of the mineralization and at the same time providing access for bulk sampling to be used in metallurgical test. Furthermore, the adits are particularly important for the mining engineer in understanding the characteristic of the host rock for mine planning.

At this stage, the exploration geologist is trying to match the newly acquired information with his/her working model. A good understanding on the characteristic of the mineralized zone can facilitate development plan that will provide a more accurate picture of the orebody that will give better accuracy and precision to the resource estimate.

There are a number of factors that need to be considered in determining what constitutes uranium ore and in estimating ore quantities and grades. Some of these factors are discussed below. It should be further noted that in addition to these factors, uranium deposits of different types imposed its own peculiar characteristics requiring special consideration.

5.1.1.1. Sampling and assaying

One of the most important element in ore reserve estimation are the samples. Aware that the uranium concentration in most deposits varies from place to place a careful approach to sampling the mineralized zone should be designed. Effort should be made that representative samples are collected that will reflect, as close as possible, the actual chemical and physical characteristic of the deposit (geological model). They can be obtained from trenches, pits, adits, but more commonly from drill holes. The density of sampling of the mineralized body generally determined whether one has achieved good representation or not. This however is determined further by the complexity of the orebody. In a relatively simple and uniform (homogeneous) deposit, lower density of sampling (wide-spaced drilling) is normally sufficient to obtain a relatively good estimate of the contained resource. However, in a more complex orebody, such as in vein and structurally controlled deposits, a very dense sampling (close-spaced drilling) may be required. It needs emphasizing that regardless of the method employed, any reserve calculation is basically based on a very small portion of the total deposit.

In uranium exploration, it is quite often that the uranium content of the samples or drill intersections were derived radiometrically (from borehole logging data or radiometric laboratory). Thus the obtained values are basically equivalent uranium (eU) rather than true or chemical uranium. Where the ore is not in equilibrium, the disequilibrium factor should be obtained and appropriate correction applied. More importantly, in such a situation "mapping" of the actual distribution of uranium in the different parts of the orebody will become necessary. While this phenomenon can occur in any type of young deposits, they are more frequently observed in sandstone type uranium deposits of Cenozoic and younger ages. Such a phenomenon can still be observed in older deposits that have been subjected to recent modification. In principle, equilibrium condition should exist where the orebody has not been modified during the past 1.3 million years.

As can be expected, variations in uranium content in the orebody is less critical in low-grade and relatively homogeneous bodies. In such a situation the contact between the designated ore and the barren zone is generally gradational. The contact zones in high grade (>1% U) deposits, on the other hand, are normally very sharp. Therefore, unless the density of the sampling is sufficiently high to delineate the actual limit of the orebody, grade interpolation or extrapolation based on rich intersection can be strongly biased. This problem is known as the "nugget effect".

One of the first step in ore reserve estimation is to calculate the volume of the mineralized rock. To convert it into weight the density of the mineralized rock has to be determined. Typical values of specific gravity are 1.6 to 2.5 for sandstone, 2.5 to 2.7 for granitic rocks and 2.7 for limestones and metasedimentary rocks. Special consideration is required for rocks containing extremely high-grade ore (>5% U). Unless appropriate correction is made, it may result in the underestimation of the overall ore grade of the deposit.

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5.1.1.2. Cut-off grade and thickness

One commonly quoted definition states that an orebody is a mineral deposit that can be worked at a profit under the existing economic condition. At the same time mineral deposits do not constitute discrete bodies with specific grade and tonnage, but more of statistical population of mineralized material varying in grade, thickness and configuration. The reserve is determined by applying sets of factors to this population. This determination is done mainly with reference to the cut-off grade and thickness, the cut-off grade being more of the determining factor. Cut-off grade, once a reserve estimate has been completed, is defined as the grade at which the value of the recovered ore equals the direct operating costs per tonne of ore. Selection of the appropriate cut-off grade is therefore determined by product price, operating cost (mining, haulage, processing and decommissioning) and metallurgical recovery. Changes in any of these factors will modify the value of the cut-off grade. Since cut-off grade determines the mineralized blocks that will be included or excluded in ore reserve estimation, it is obvious that such a change has marked influence on the calculated reserve and the average grade. Fig. 9 illustrates the effect of changing cut-off grades to the fraction of total uranium content in deposit and average grade for five different uranium deposits. The five deposits are varying in grade from high, No. 1, gradually to lower and lower grade deposits for the higher numbers (No. 5 being the lowest grade). Taking the high grade deposit (No. 1) as an example, if one applies a cut-off grade of 0.12% U about 90% of the total uranium content of the deposit will be included into the calculated reserve with an average grade of 0.42% U. If the cut-off is lowered to 0.04% U, 98% of the total will be included with an average grade of 0.29% U. In the case of the lower grade deposit, No. 5, only 70% of the total with an average grade of about 0.9% U is above the a cut-off grade of 0.04% U. Increasing the cut-off grade to 0.12% U will drastically reduced the amount of total uranium included to 32% at an average grade of 0.2% U.

As noted above, the calculated reserve change if the cut-off grade is changed. This in turn will also affect operating cost estimate, even mining methods might have to be modified. With this, the cut-off grade needs to be re-examined. The selection of the final cut-off grade used in pre-feasibility and feasibility studies, therefore, tends to be an iterative process.

Cut-off thickness is basically the minimum thickness of part of the orebody that can be practically mined. The ultimate mining methods selected to exploit the deposit is normally dictated by the shape and configuration of the deposit (continuous or discontinuous, as lenses, flat lying, gently or steeply dipping, etc.) and its depth from the surface. It is always the objective of every mine planning to minimize inclusion of the barren host rock. Such an inclusion will dilute the reserve. In general a cut-off thickness of less than 1.5 m is not recommended for ore reserve estimation.

5.1.1.3. Ore reserve estimation

It has been stressed in a number occasions that ore reserve is the main foundation of every mining venture. Inaccuracy in determining the ore reserve can have an adverse effect to the entire project. Ore reserve estimation is, therefore, constitute one of the most important step in any mineral deposit development. As noted above, the estimation of the ore reserve depends on a number of factors. Most of importance are the geological model used, the derived grades and dimension (volume) of the deposit. It goes without saying that the accuracy and precision of this estimate depend on the quality of the various factors used in this calculation. At the pre-feasibility stage, it is sufficient that this estimate produces what is known as in situ or geological reserve. However, as the project progresses into a more advance stage (feasibility study), the acceptable error has become very small and the economic factor will play a more important role. The resulting product is known as minable reserve.
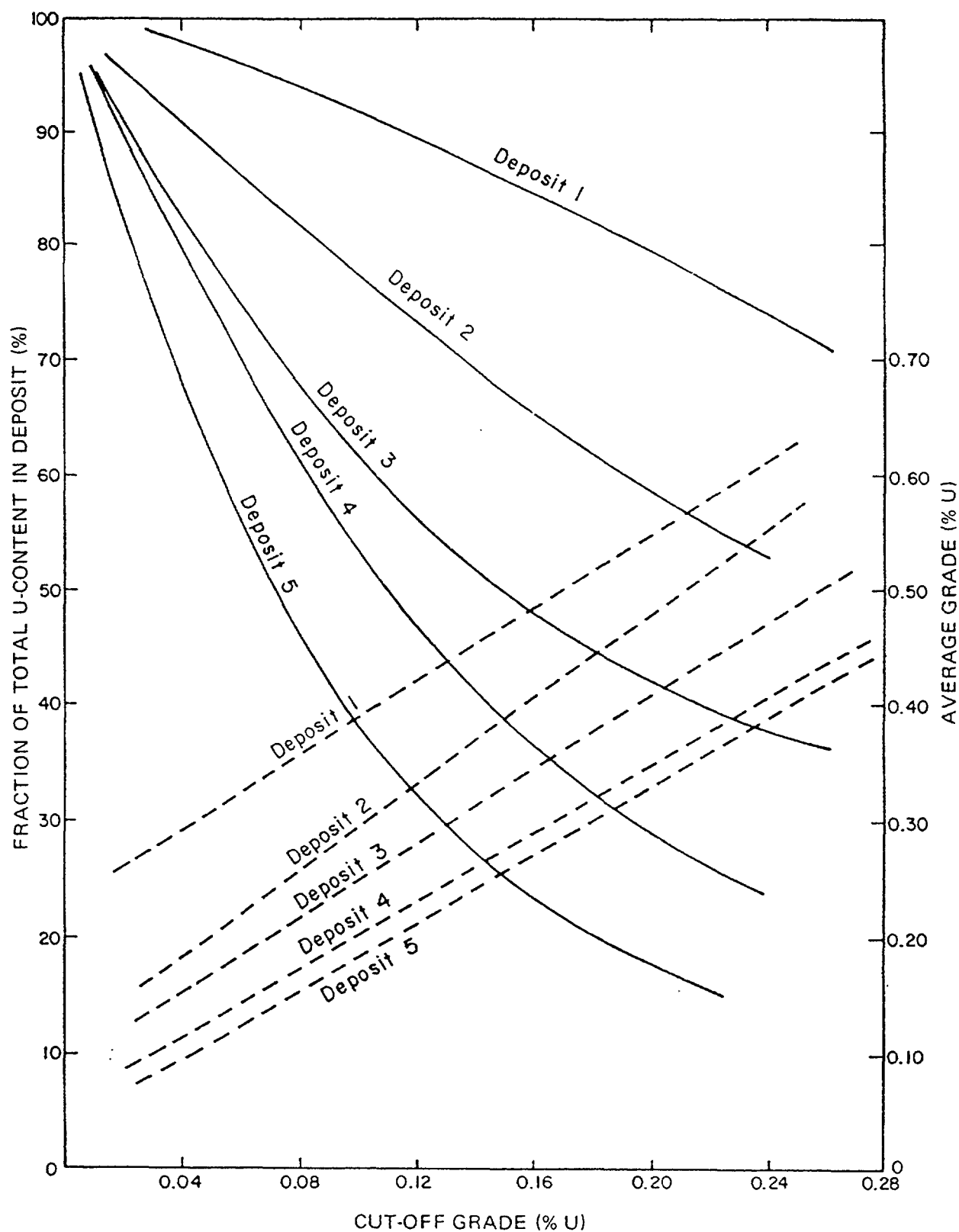
FIG. 9. The effects of changing cut-off grads in the case of five uranium deposits. Full line: fraction of total U-content; dashed line: average grade.

It is more common than not that an ore deposit consisted of more than one ore blocks. Ore estimation of the entire orebody is therefore the sum of the calculated uranium contained in these blocks. The amount of uranium contained in each block is basically the product of its average grade and the tonnage. The tonnage of the block is derived from multiplying its volume and the average density of the ore. It is obvious that the derived average grade, thickness of ore-bearing zone and density of the ore measured from the samples (hole intersections, channel samples, etc.) are expected to represent the entire blocks or orebody, something that is very hard to verify until the deposit is actually mined. Because of this, ore reserve estimation carries high element of uncertainties and risk.

There are two basic approaches in ore reserve estimation, the classical or conventional and the statistical and geostatistical methods. Much of the information noted here is taken from an IAEA publication "Methods for the Estimation of Uranium Reserves — An Instruction Manual" (1985). As noted at the beginning of this section, regardless of the method used, the outcome of the estimate depends primarily on the reliability of the data.

There are several methods of ore reserve estimation classified within the conventional approach. Each has its own advantages and disadvantages and general applicability to one but less so for other types of deposits. All of these methods are basically geometrical in character. It assigns a certain geometry perceived to be most appropriate for the type of information available for the calculation. Each assumed the area of influence of the sample (e.g. hole intersection) that will be included in this estimate.

The configuration of the uranium deposits further dictates what method of estimation can be used. For practical purposes, one can grouped the uranium deposits into either the flat-lying or the steeply dipping types. For the flat-lying uranium deposits, the following methods are known:

- uniform area of influence method;
- variable area of influence method;
- polygonal method;
- triangular method;
- cross-section method;
- isopach method; and
- general outline method.

The IAEA publication "Methods for the Estimation of Uranium Reserves — An Instruction Manual" (1985) noted earlier, describes the procedures, advantages and disadvantages of each method in considerable detail. Of all these methods, the isopach method is the least used. Using the same set of data, a comparative study on results obtained from using the different methods noted above, indicate that the triangular method, which assumes no ore extensions beyond the periphery of the constructed triangular block, gave the most conservative result. Results from the uniform and variable area of influence, polygonal and general outline methods are much closer to one another. A lower estimate is noted from the cross-section method. This underlines the degree of uncertainties involved in using any one of these methods.

The IAEA Manual also describes a method for steeply dipping uranium deposits such as in veins and some unconformity related, intrusive and volcanic types. Most of these deposits are complex in their geometry with greater variability in their uranium content and therefore require higher density of sampling. The complexity of this geometry and grade variation can be seen in the example shown in Figs 10-12. The cross-section procedure noted earlier is the common method used to estimate the reserve.

An additional procedure considers useful to reduce the uncertainty in ore reserve estimation is the inverse distance method. All the methods previously discussed assigns an area to each sample point. In the inverse distance method, the influence of sample points outside the areas or blocks, for which reserve are being estimated, can be considered. Influences from outside points are weighted in accordance to their distances from the centre of the area being evaluated. The value of the nearest sample has a heavier weight than those for samples farther away.
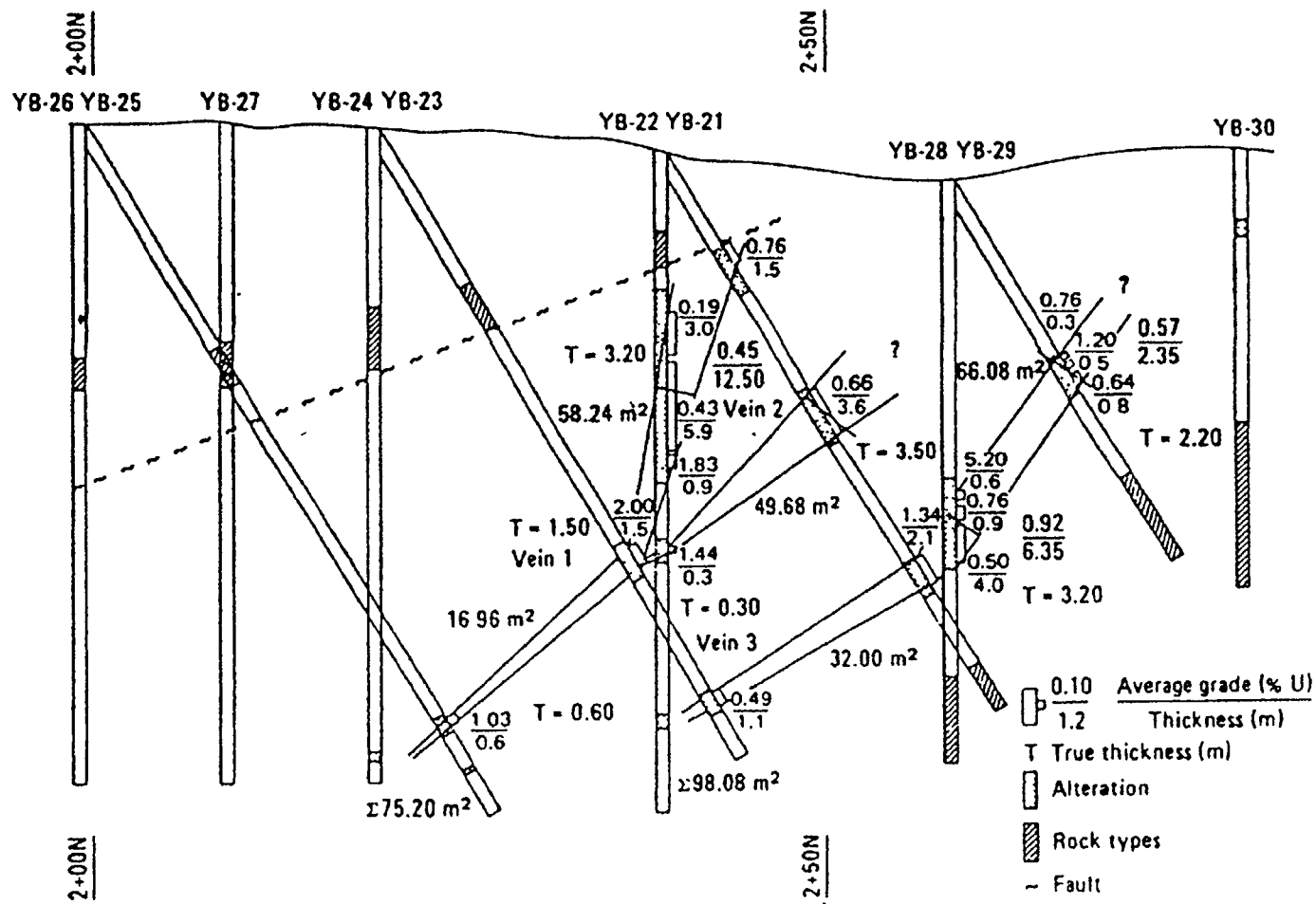
FIG. 10. A cross-section through a steeply dipping vein deposit at fence 1+00W.
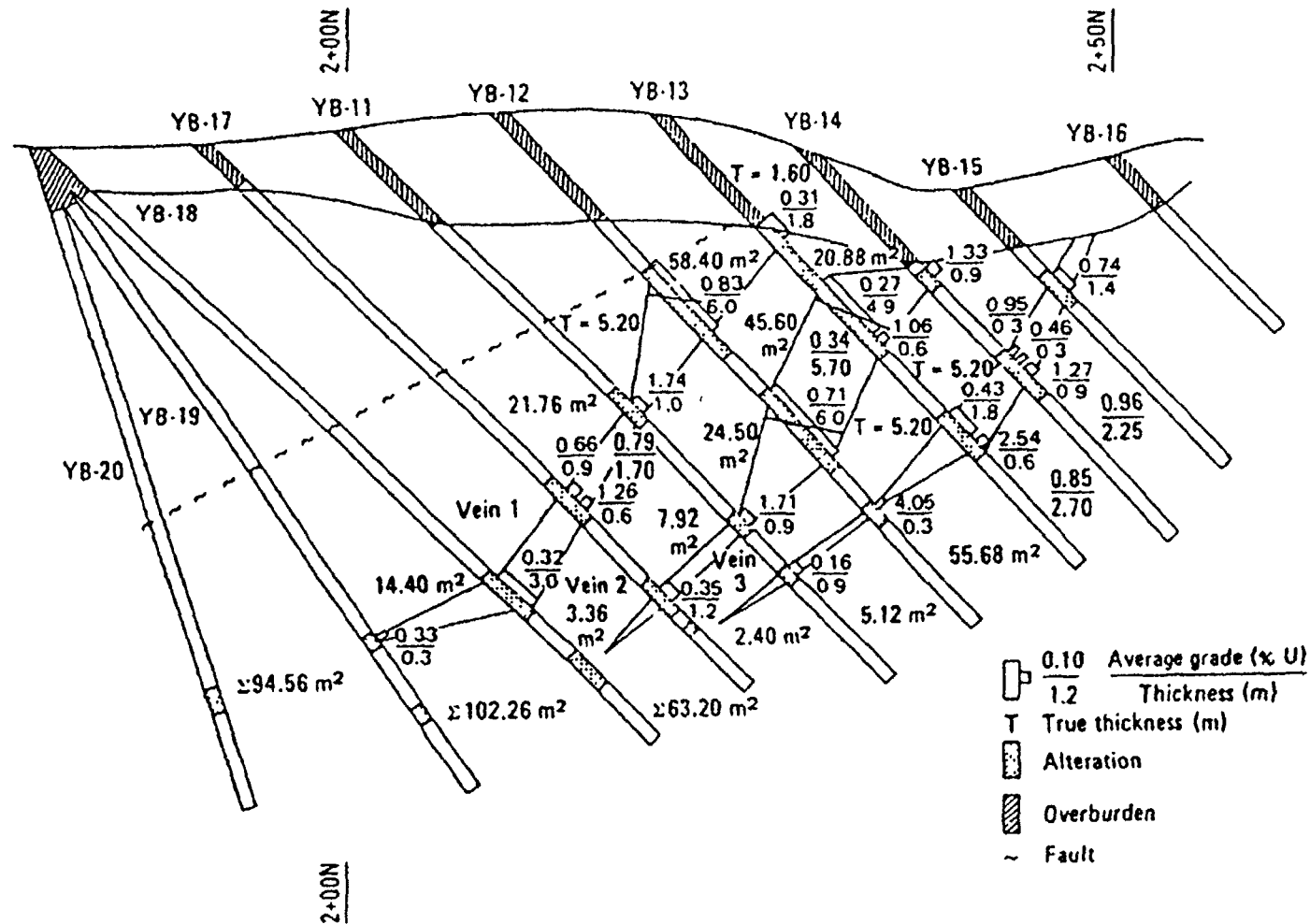
FIG. 11. A cross-section through a steeply dipping vein deposit at fence 2+00W.
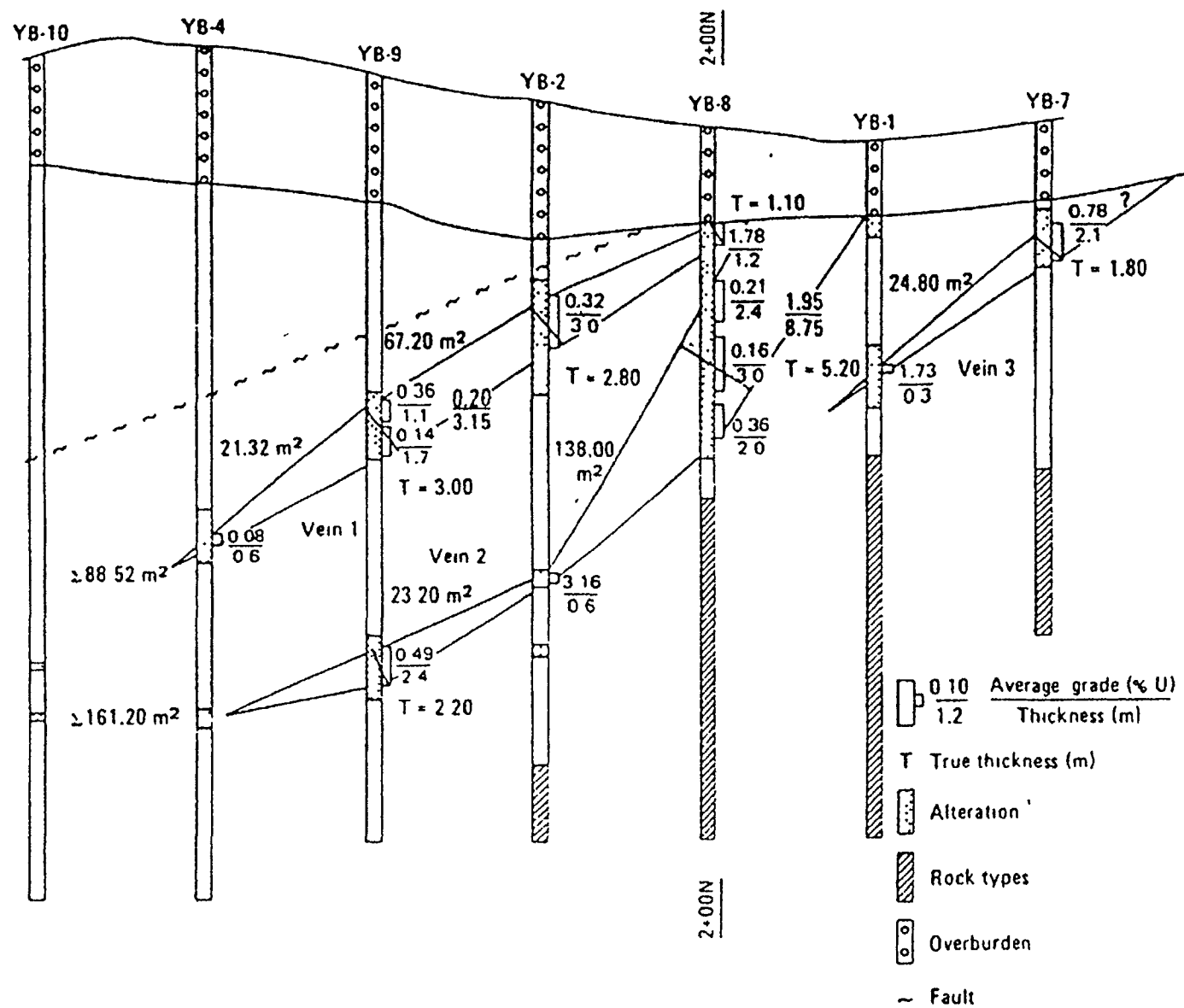
FIG. 12. A cross-section through a steeply dipping vein deposit at fence 3+00W.

Greater uncertainties inherent to almost all the methods described above and wider use of computers has resulted in the increasing used of statistical and geostatistical procedures of ore reserve estimation. Regardless, it should be emphasized that the conventional methods still represent very useful and simple tools to give a "quick and dirty" estimate.

It was noted in Section 5.1.1.2 that mineral deposits do not constitute discrete bodies with specific grade and tonnage, but more of statistical population of mineralized material varying in grade, thickness and configuration. Variations of the different parameters in the deposits can therefore be represented as histograms or curves. Analysis of this distribution, will allow selection of a series of cut-off grades and the calculation of the corresponding amounts of ore and uranium and the average ore grades for each case. Such a data manipulation will also provide information on the character of the deposit over wide range of conditions, including costs and prices.

The present day state of the art approach to ore reserve estimation is the geostatistical method. It is based on the theory of regionalized variables for geological applications. The method considers the position at which a variable is measured and information on the surrounding area are considered as well as the numerical value of the variable. The procedure of estimation can be divided into two parts. The first consists of an analysis of the variation of physical and statistical characteristics of the orebody. This is done by construction of variograms, which are graphs of the variations of values as a function of their separation. The second step of the procedure is the estimation itself, called "kriging" after D.G. Krige. In this step an unbiased statistical estimator of the average block value is determined by seeking the optimum combination of weighing values for all samples in the vicinity. The information from the variogram and kriging is used to determine the weighing factors which minimize the estimation variance of the blocks and the variance of the estimator.

In summary, given sufficient density of samples, conventional method of ore reserve estimation can provide adequate result. Geostatistics, however, can improve this estimate with the possibility of providing a quantitative measure of reliability of the calculated reserve. Furthermore, it facilitates rapid recalculation of the estimate that may be required due to new situation or information.

5.1.1.4. Recovery factors

Almost without exception, exploitation of a mineral deposits does not recover all the measured in-situ reserve. Because of the non-uniform configuration of the orebody, grade control, the mining plan and method used, as well as non-accessibility of the ore, a certain portion of the in-situ reserve has to be left behind. Mining losses in uranium mines may reach as high as 25 or 30%. Dilution caused by the necessary inclusion of barren host rocks, while increasing the tonnage, is reducing the overall grade of the mined materials. Additional losses is still to be expected in ore processing since not all the uranium contained in the mined ore can be recovered at reasonable cost. The percent of recovered uranium depend mainly on the mineralogy, chemical and physical property of the ore and the enclosing rocks, as well as the metallurgical process used. This losses may range between 5 and 30%. After all these losses are considered, the final product is the recoverable uranium resources of deposit.

5.1.2. Mining and geotechnical information

The required mining and geotechnical data are similar to those required during the pre-evaluation stage. The quality of the information will have to be better, more representative and reliable. Statistically, this means a larger number of tests and measurements including some in-situ measurements. In order to achieve this requirement, additional drilling will have to be done. The programme will be defined according to the needs of the geologists and the metallurgists. Core drilling of an appropriate diameter will provide sufficient samples in size and volume to perform the necessary geotechnical, geological and metallurgical tests, analyses and measurements.

As explained in the previous paragraph, the required geotechnical information will vary according to the nature of the ground and ore, the geological setting and the proposed mining method.

A. Rock mechanics

Generally when dealing with hard rock, uniaxial compression tests are sufficient. At this stage of the study, these tests will allow the study team:

- In open pits, to define the slope of the walls and to determine the preferred place of the haulage road. In open pits, tectonic is also important and must be looked at carefully, mainly at the limits of the pit.

- Underground, to determine the location of the shaft, drifts and other components. The tests also provide valuable information needed to more precisely define the mining method.

When studying a sedimentary deposit the characteristics of the ore and of the walls are *sine qua non* data without which it is not possible to determine the percentage of pillars to be left. In vein type deposits, the rock mechanic tests will help to determine 'the most appropriate mining method and determine the need for, and characteristics of back fill, bolting, and in some cases, timbering.

Clayey materials required more sophisticated tests such as triaxial compression tests. These tests must be carried on with the utmost care and the results used with caution. Sometimes the fact that a clay sample is freed from the in situ stresses changes the geotechnical characteristics of the material. In such a case in-situ measurements should be made.

B. Density and water content

These data are very important because a good estimation of the mine and mill capacity, capital and operating cost depends on them. Their precise determination is sometimes difficult, especially with clayey or unconsolidated material. The dry, in place density can vary inside a deposit, from 1.5 to 2 for instance. Many samples must be taken, and in extreme cases, geostatistical laws applied.

C. Water inflow

The miners would prefer a dry deposit, but mostly they have to deal with ground water inflow. Pumping tests and porosity and permeability measurements on drill cores combined with modelling studies of the deposit will usually give a good estimate of the future inflow. These data are very important and necessary for:

- Designing and estimating the capital cost of the sumps, pumping station, settling pond, and water treatment facilities, key points in a uranium mine.

- Estimating the operating cost of pumping and water treatment.

- The impact of the water on the stability of the open pit walls.

- Estimating the reduction in productivity due to poor working conditions.

- Calculating the impact of radon released by the water.

D. Radioactivity

Radiometric counters, very useful in the mining industry for grade control and analysis, do not give the actual grade of the uranium but only the content of the uranium daughter products. It is therefore necessary to develop the correlation between uranium and radioactivity in order to improve the estimation of the grade when using radiometric techniques. In younger deposits, for example the Tertiary sandstones, the disequilibrium factor can vary greatly from 0.1 to 10. The additional expenditures for γ spectrometry, generated by the necessity to determine the exact location of the uranium before mining must be considered. The correlation between uranium content and radioactivity is also important in determining if radiometric sorting is workable.

5.1.3. Metallurgical information

To embark on a meaningful pre-feasibility study of a potential mine requires completion of sufficient bench scale metallurgical testwork to determine a preliminary process flowsheet. The testwork must be performed on samples that are representative of the entire orebody. If wide fluctuations in ore characteristics are expected, samples representing the range must be obtained and tested.

Some important factors about samples are:

Sulphide ore is often subject to oxidation after exposure to air. This can be of minor importance in the actual mining operation but catastrophic as far as sampling and metallurgical testwork. Careful sampling, handling and storage (including refrigeration) are required with many ores. Obviously a metallurgist should be involved early in the sampling program.

Ore sorting and semi-autogenous grinding (SAG) are common, low-cost process options. Proper testing requires that the size distribution and competency of the sample is representative. Development muck is usually finer and more fractured than production ore and can therefore be misleading.

Samples can be contaminated by diesel fuel, lubricants, ANFO and other materials. In a classic incident, a geologist shipped a bulk sample of uranium ore for testing. The ore leached well but the phase disengagement during solvent extraction was very slow. Investigation showed that this was due to contamination of the samples with hydraulic fluid. The geologist had got a bargain on some old hydraulic oil drums to ship the sample.

The following metallurgical tests should be performed for the pre-feasibility study:

A. Complete analysis of ore

The ore must be analysed for U, Th, V, Mo, S, CO_2, Au, and scanned to determine the presence of other major elements. These analytical data are required to properly plan the test programme, to develop the flowsheet, to allow examination of possible by-products and to assist in environmental design.

B. Ore sorting tests

The applicability of sorting techniques to the ore should be determined. Initial tests are readily performed on +25 mm ore using a γ counter and a series of buckets to receive the different classes of rock.

C. Bond work index tests

The Bond work index is a simple test which indicates the amount of energy required to grind the ore to specified degrees of fineness. These data are required to allow the sizing of the grinding mills. Additional test work is required to allow full consideration of SAG (semi-autogenous grinding). However, these tests can be deferred until after the pre-feasibility study.

D. Preliminary acid and alkaline leach tests (grind, time, temperature and oxidant)

Representative samples of ore must be acid leached at various degrees of grind. If high CO_2 levels are indicated in the ore, it would be prudent to also perform alkaline leach tests. The objective of the tests is to obtain definition of the time-temperature-grind-oxidant-lixiviant relationship as it affects uranium recovery and reagent demand.

E. Preliminary liquid-solid separation tests (thickening and filtration)

As suitable leach conditions become evident, initial filtration and thickening tests should be performed. These should be done promptly since leached slurry can change as it ages or cools. The liquid-solid separation tests should include the evaluation of flocculants. Filtration tests should include an evaluation of the washing characteristics of the filter cake.

F. Preliminary IX and SX tests (isotherm, absorption tests)

The test programme should include preliminary measurement of the solvent extraction (SX) isotherm and the ion exchange (IX) absorption characteristics of the pregnant solution. Resin-in-pulp (RIP) is an attractive recovery option, especially for ores that are difficult to filter or thicken.

G. Effluent treatment experiments

The pregnant solutions must be carefully analyzed to determine any potential problem elements. Appropriate tests must be performed to allow proper design of a treatment plant.

H. Acid generation tests on tailings and waste rock

If analyses show that the ore or waste rock contains sulphides, then acid generation tests should be performed. In these tests, the acid-base balance of the ore is established. If the ore has the potential to produce more acid than it can neutralize, then additional tests are performed to establish the probability of acid generation.

Throughout all of the tests, the researchers must be alert to any peculiarities of the ore. The deportment of impurity/problem elements must be carefully followed and tests for their control made if necessary.

5.1.4. Infrastructure

To develop meaningful project costs, the infrastructure requirements must be carefully defined — even at the pre-feasibility stage. Important information that must be available to the design team includes:

- project location
- information on land ownership, royalty arrangements, etc.
- local mapping
- climatic data
- preliminary alignments for any new roads
- preliminary tailings dam site selection and quantities
- environmental requirements of area
- preliminary data concerning on-site office, shop and laboratory requirements
- project operating philosophy, airstrip and camp requirements

5.1.5. Waste management and environmental consideration

Quite properly, the environment is now receiving more attention from the operating companies. In many parts of the world, extremely strict government regulations await those who are still uncertain about the need to protect the ecosphere.

Compliance with the regulations requires the expenditure of both capital and operating costs. This is nothing new although the costs are rising and a relatively new class of environmental cost has appeared, decommissioning costs. The cost to properly close down a uranium mining operation is becoming very significant. As an example, perhaps an extreme one, the costs to decommission a particular group of three uranium mines in Canada is expected to exceed $200 million.

Effort should also be made to obtain or, if necessary, create a baseline information of the natural environment before actual mining and processing start. A good baseline information on the natural radioactivity level of the area as well complete geochemical information of the ecosystem will be needed at the rehabilitation stage.

Operations

To be acceptable as a means of raising development funds, the feasibility study must describe a project that is environmentally responsible. Indeed the feasibility study will only be acceptable to the lending institutes if the environmental application process is well advanced and the regulatory agencies have indicated that there are no problems in sight.

The operating concerns include the migration of solutions containing acid, heavy metals or radionuclides from waste rock and tailings storage areas. The escape of dust and gaseous contaminants must also be controlled. The environmental regulations are quite clear concerning the requirements of a mining project. Suitable testwork and designs will result in systems that will be acceptable to the authorities. These designs must be submitted to the regulatory agencies in certain prescribed ways.

Decommissioning

A few decades ago, people just walked away from a mine when the ore ran out. The unacceptability of this strategy is evident in the catalogue of old tailings dam failures, fatal falls into abandoned mine shafts, acid mine waste drainage and other problems.

Governments are now requiring that mining companies declare how they will close-out a property before an application for an operating license will be entertained. The regulations in many parts of North America require that all mine openings be made safe, that plants and structures be removed, and that tailings dams be stabilized. In these same ares, existing mines are now required to formulate decommissioning plans and lodge these with the regulatory agencies.

The new decommissioning regulations will increase the initial project costs. They will also cause a significant outlay of money at the end of the life of the project. Until recently, the latter cost was lightly dismissed with the assumption that it would be covered by the sale of used equipment. It is now apparent that this will generally not be the case. Prefeasibility and feasibility studies must include a decommissioning plan and an assessment of the decommissioning costs. It would also be prudent to examine the impact of different levels of decommissioning cost on project DCFROR.

The question of bonding is being reviewed by government and industry and will further add to the development costs for a project.

These are key factors in a uranium project and must be looked at as soon as possible. The following aspects must be taken into account:

—   The location of the tailings pond. A good solution consists of putting the tailings in a depleted open pit if there is one available. In this opinion, there is little or no risk for the future and the capital investment is reduced.

—   An hydrogeological study of the location. The tailings must be placed in an areas as close as possible to the plant, but where their influence on the local aquifer will be acceptable. In this regard, it is strongly recommended that all drill holes are carefully plugged.

—   Waste management. The quantity of waste removed from an underground mine is low and easily managed. It is the contrary for open pits where the volume of waste can be ten times or more greater than the volume of mined ore. The waste must be managed according to the best compromise between the trucking distance and the reclamation cost. Ore sorting generates coarse rejects which also require disposal.

The developer has little or no choice regarding the locating of the mine. However, the mine access, process facilities, repair shop, stores, camps, and stockpiles must be designed and located to minimize impact on the environment.

At the pre-feasibility stage of the project, the project team should install the necessary sensing equipment and commence the environmental base line measurements.

5.2.   GENERAL DESIGN

5.2.1.   Mining

5.2.1.1.   Open pit operations

Uranium open pits are no different from other metals mines except that, on average, the size is low or medium compared with the same kind of operations in coal, iron, copper and so on.

The profile of the walls of the pit is determined according to the geotechnical, and hydrogeological data.

An important issue is to define the location of the haulage road. The best solution will certainly be a compromise between cost and security, that is to say between:

—   The volume of waste to extract. It naturally depends on the size of the pit. In the uranium mining business, this volume typically falls in the range of 10 to 15% of the total mined waste.

—   The quality of the walls in the area where the road is established.

5.2.1.2. Underground operations

The comment on the relatively small size of open pit uranium mines can also be applied to underground mines. Thus, the infrastructure equipment, the surface and underground facilities will be of a moderate size. This is particularly the case for hoisting equipment.

Regarding mine access, there is the choice between two options: shaft or decline. With the development of trackless mining equipments it is suggested that the decline option should be selected except when the deposit is too deep. The economic balance between decline and shaft access costs falls in the range of 300 to 400 metres.

As the tonnage hauled in an underground uranium mine is usually low, from 100 000 to 800 000 tonnes per year, the use of trucks to transport the ore to the surface is a good solution. Reliable 50 tonnes capacity trucks are now available, that can each hoist at least 1 000 tonnes per day. Accordingly, trucking 800 000 tonnes per year through a decline is no more a problem, it requires four units, three in operation and one in maintenance.

There is also the possibility of hoisting the ore with conveyor belts, but the capital cost is higher than with trucks. To allow conveying, the ore must be crushed underground which implies costly equipment. With the trucking option, the decline is always available and can be used to transport large equipment like jumbo drills or loaders. As it is easy to drive these equipment to the surface for servicing, maintenance facilities underground can be limited.

When the deposit is deeper, it is necessary to sink a shaft. The dimensions of the shaft will be moderate due to the level of tonnage hoisted. Generally, the shaft will be equipped with a skip and counterweight. A cage for personnel, small equipment and consumables is located above the skip.

In some cases, especially with clayey and rich ore, it will be better to use cars for hoisting the ore. Due to the clay, there could be some risk of contaminating the shaft should the closing mechanism of the skip fail.

The inconvenience with a shaft of a small or medium size is that it is not very easy to transport the trackless equipment which is generally used now in the underground mines. Consequently, maintenance facilities have been established underground.

5.2.1.3. Service infrastructure

In order to satisfy safety requirements, a second exit in addition to the haulage decline or shaft must be established.

–   Depending on to the size of the mine in the decline option the second exit can be:

    ·   either a second smaller decline for the bigger mines, or
    ·   a raise equipped with loaders, platforms and safety equipment for the smaller mines. Raises are now easily, safely and economically driven with the raise boring machines.

–   In the shaft hoisting system used when the deposit is deep, a series of raises driven with raise boring machines is a good solution.

5.2.1.4. Drifting and mining equipment

In order to maintain the best level of working conditions, safety and productivity it is strongly suggested that the mine uses:

- hydraulic percussion drilling on trackless jumbos. This option provides a good working environment for the miners; no dust, low noise, safety and productivity. Air compressors and piping is not needed.

- The load-haul dump equipment

- Bolting and meshing will be used if necessary in place of timbering except in extreme cases of poor ground conditions.

5.2.1.5. Dewatering and pumping

One or more settling basins should be included in the design of the dewatering system in order to protect the main pumps from particulates and reduce the operating cost.

It is recommended, for safety and reduced operating cost, that the main dewatering pumps be fed from submersible pumps in the setting basin. Reliable and efficient mud pumps are available for pumping the slimes accumulated in the settling basins.

5.2.1.6. Ventilation

Ventilation is a very important issue in uranium mines due to the health effects of radon and radon daughter products. The ventilation circuit will be designed so that the air inflow is the maximum and that each stope receives fresh air from the primary circuit.

5.2.2. Process flowsheet

The test data and other information must be used in a brief series of 'mini-studies' to determine the preferred processing rate and the processing route. Following resolution of these issues, the project metallurgist must tabulate the design criteria to be used for the plant.

The criteria listing will start with the *production criteria*. These will include the number of tonnes processed per year, the normal and peak uranium grade, the operating days per year, the plant availability and so on. Extreme care must be taken in selecting these criteria. The planned processing rate and the possible mining rates must correspond. The reality of plant availability must be recognized. Sensible production rates that match market requirements should be selected.

Following the production criteria, the metallurgist must produce detailed design criteria for every other aspect of the production plant. Typical criteria are presented below.

TABLE V. TYPICAL PROCESS DESIGN CRITERIA
(Note filtration and all subsequent sections omitted for brevity)

PARAMETER	UNITS	DATA	NOTE	REV
PRODUCTION CRITERIA				
Required U production rate	t/a	1 000	A	A
Average ore grade	kg/t	12.5	B	A
Ore processing rate	t/a	80 000	C	A
Operating days per year	d/a	350	D	C
Average daily ore processing rate	t/d	228.6	C	A
Plant availability	%	90	D	A
Calculated daily design capacity	t/d	254	C	A
Calculated hourly design capacity	t/h	10.6	C	A
Peak ore grade	kg/t	18	B	B
Peak uranium flow	kg/h	190.8	C	B
MINING				
Days per week mined	d/week	5	B	A
Hours per day mined	h/d	16	B	A
Maximum ore lump size	mm	450	B	B
Ore delivered from mine	-	Headframe	B	A
Headframe ore bin capacity	t	50	B	A
SURFACE CRUSHING				
Location in circuit	-	In headframe	B	A
Required top size in product	mm	180	D	A
Maximum required crushing rate	t/h	50	B	A
Selected crusher size	mm × mm	800 × 600	D	A
SURFACE ORE STORAGE				
Storage time required	d	3	D	A
Storage capacity	t	760	C	A
Storage system	-	Silo	D	A
Feeder type	-	Plate	D	A
GRINDING CIRCUIT				
Circuit type	-	SAG-ball	D	A
Expected 80% passing size-feed	mm	150	D	A
Desired 80% passing size-product	μm	74	F	A
SAG mill work index	kWh/t	18	G	A
SAG discharge 80% passing size	μm	400	D	A
SAG classification	-	Screen	D	A
Ball mill work index	kWh/t	15	G	A
Final classification	-	Cyclone	D	A
THICKENING				
Expected feed solids	%	15	D	A
Desired underflow solids	%	55	F	A
Unit area required	$m^2/t/d$	1	F	A
Flocculant type	-	Non-ionic	F	A
Flocculant dosage	g/t	10	F	A
LEACHING				
Leach retention time	h	24	E	A
Leach temperature	°C	20	E	A
Acid type	-	H_2SO_4	E	A
Acid consumption	kg/t	20	E	A
Oxidant type	-	$NaClO_3$	E	A
Oxidant consumption	kg/t	1	E	A
Minimum number of tanks	-	5	D	A
Installed number of tanks	-	6	D	A
Agitation means	-	Mechanical	D	A
Agitation power	kW/m^3	0.2	D	A
Off-gas handling	-	Scrubbed	D	A

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NOTES:
A   Data provided by management
B   Data provided by geology/mining department
C   Calculated from more basic design criteria   REVISIONS:
D   Proposed by project metallurgist   A   Original issue for comments (91-06-18)
E   Obtained from Laboratory Test Report 1   B   Comments from Geology/Mining (91-06-25)
F   Obtained from Laboratory Test Report 2   C   Comments from Management (91-06-26)
G   Obtained from Laboratory Test Report 3
H   From vendor data

Data for the filtration, clarification, solvent extraction, yellow cake, reagents, tailings, effluent treatment, and water distribution sections of the project are not tabulated above. However they too would be prepared and to the same level of detail.

The columns in the table of design criteria headed "Notes" and "Revision" are essential for keeping track of the origin of data, changes made to the criteria, and the reasons for the changes.

It is important that agreement to the design criteria be obtained from all project team members before proceeding to the next step. This is best obtained by circulating the criteria to all involved parties and having them sign the document.

Following agreement to the design criteria, the project metallurgist can then prepare a series of process flowsheets. It is good practice to show every piece of equipment, including installed spares, and every flow line on the process flowsheet. This reduces the possibility of pieces of equipment becoming 'lost'. A typical, small tonnage, acid leach plant would probably require seven flowsheets. A possible drawing list is provided below and a typical study drawing is appended:

LIST OF FLOWSHEETS
URANIUM PRE-FEASIBILITY STUDY

1.   Grinding and thickening
2.   Leaching, filtration and tailings
3.   Solvent extraction
4.   Yellow cake precipitation and handling
5.   Tailings system and effluent treatment
6.   Reagents and utilities
7.   Water supply distribution

The flowsheets should clearly show the flow in all significant lines. A useful format is presented below:

STREAM NAME	
SOLIDS FLOW (dry tonnes/hour)	SOLUTION FLOW (cubic metres/hour)
SOLIDS IN STREAM (mass percentage)	SLURRY FLOW (cubic metres/hour)
SOLIDS RELATIVE DENSITY	STREAM RELATIVE DENSITY
URANIUM GRADE/FLOW (kg U/t and kg/h)	OPERATING TIME (hours/year)

The flowsheet should also show for each piece of equipment, the name, description, size, and materials of construction. A typical description alongside a surge tank might look as follows:

<div align="center">

FILTER FEED SURGE TANK

5000 x 5000

RUBBER LINED MILD
STEEL

COVERED AND VENTED

</div>

This type of format makes it very clear what the equipment will look like. Since the flowsheets will be used by many other people, they must convey as much information as clearly as possible. For example the cost estimating department must know if a piece of equipment is made from mild steel or titanium. The cost difference in this example would be enormous.

All electric motors should be noted on the process flowsheets, their power rating, speed and any special requirements such as `explosion-proof`.

5.2.3. Arrangement drawings

At the pre-feasibility stage, a series of simplified site and plant general arrangement drawings are required. These drawings will be used to verify that the process flowsheets are workable. They will also be used by the estimating department to develop costs for the site and the plant structures. A typical listing follows:

LIST OF ARRANGEMENT DRAWINGS
URANIUM PRE-FEASIBILITY STUDY

1   Location map (about 1:250 000 scale)
2   Overall project plan (about 1:2500 scale)
3   Site plan (about 1:500 scale)
4   Headframe/Adit area plan (about 1:100 scale)
5   Process plant plans (about 1:100 scale)
6   Process plant sections (about 1:100 scale)
7   Peripheral plant plans and sections, e.g. offices, labs, shops (about 1:200 scale)

The location map, Drawing 1 in the above listing, would show the project site with reference to local towns, roads, railway lines, rivers and airports. The overall project plan, Drawing 2, should show the tailings dam, water source, all project roads, and the actual site of the project facilities. Other drawings will show the project facilities in more detail. A typical layout drawing is appended.

The site drawings should show the peripheral buildings. This is important since the drawings are the main method of conveying information to the cost estimating section. Peripheral buildings might include an office, analytical laboratory, test laboratory, camp buildings, shops, airport building, and garages.

The process plant general arrangement drawings must not be simply left to the draughtsmen to prepare. The general arrangement drawings will probably form the basis for the construction of a processing plant that must be easy to operate, and operate successfully. Input should be obtained from all who can contribute, but especially from the project metallurgist and the plant operating staff. In this regard, it should be stressed that the operating team be assigned to the project at the earliest opportunity.

If the design team is inexperienced, it would be prudent to consider other options. These include an intensive series of visits to similar operating plants to determine what works and what does not. Another option is to employ an experienced engineering company to perform the work.

It is extremely useful if all of the project drawings are prepared using a computer assisted design and drafting program (CADD). This is particularly true of the general arrangement drawings. The project facilities need only be drawn once and then plotted at different scales to get the different levels of detail required. Furthermore, any changes required on the drawings to reflect new data can be quickly effected.

If the project proceeds to a feasibility study and on to detailed design, CADD becomes an increasingly efficient way of working. The general arrangement drawing files can be used by other design disciplines as backgrounds for their work. For example, the electrical and instrumentation cable layouts can be superimposed on the general arrangement drawing. Similarly, the P&ID (piping and instrumentation diagrams) can be superimposed onto the process flowsheets.

5.3. PROJECT SCHEDULE

An estimate of the time required to complete the project should be made during the pre-feasibility study. This schedule is required as part of the cash flow analysis. At the pre-feasibility stage, the analysis of the schedule will necessarily be limited. Key items on the schedule would include:

- commitment to proceed to feasibility stage
- completion of testwork and other studies
- feasibility study
- financing
- environmental permitting and licensing requirements
- commitment to construct
- mine development
- equipment procurement and delivery
- on-site construction
- seasonal constraints (shipping windows, winter works, rainy season, etc.)
- commissioning

5.3.1. General mining schedule

In many cases, the choice between open pit and underground mine is clear and does not require much discussion. There is just one type of mining either: open pit mining or underground.

But sometimes, especially with steeply dipping vein type deposits, the upper part of the orebody can be economically mined by open pit and the lower part by underground methods. The limit of the open pit corresponds to a depth where the cost of an additional tonne of ore becomes higher than the cost of the same tonne mined by an underground method. The logic and profitable plan is to mine the open pit first and, if the stability of the pit walls is good, to mine underground from the bottom of the open pit.

5.3.2. Open pit schedule

The schedule must take into account the cost of the stripping and the cost of the reclamation which itself depends on the regulations of the country, state or province where the deposit is located.

It is not possible to give unique and absolute advice because each case is different from the other, and because the commitment for the backfilling of open pits differ from one country to another.

Nevertheless, it ca be mentioned that it is often economically feasible to mine first one part of the pit and enlarge it later. Obviously the first pit will be the one with the best compromise between the strip ratio and the grade. The waste rock produced during the enlargement can be backfilled into the previous pit and so on. Such a schedule can save a lot of money when there is a commitment for backfilling.

5.3.3. Underground mining schedule

The underground development schedule depends on the mine access method. As noted before, many uranium mines are not very deep and because the tonnage hauled from the mine is generally not high and because of the availability of efficient trucking equipment, the decline, trackless system is preferred.

In the case of medium or large mines a decline for haulage and one for service is suggested. In order to get revenue as soon as possible it is recommended that the service decline be driven down to the top of the ore which can then be mined whilst the plant is being built. Ore is placed on a stockpile which will allow the plant to be started at full capacity while continuing to increase the capacity of the mine. Such a schedule is flexible and gives time to develop the mine and to hire and train the personnel.

5.4. CAPITAL AND OPERATING COSTS

5.4.1. Project capital cost

To evaluate the project, it must be subjected to capital cost estimating procedures. For a pre-feasibility study, the development of the costs will include the following components:

Acquisition cost

The cost to acquire and hold the required land must be determined. It should be realized that area will be required for tailing, waste rock dumps and all infrastructure items such as camps.

Mine development

The preliminary mine design must be studied and an estimate made of the amount of mine development work required. The quantity estimate must then be combined with unit rates to obtain development costs.

If a headframe is required, then an appropriate preliminary design must be developed and costed. A preliminary quotation for a suitable hoist should be obtained from a supplier.

Civil

Preliminary estimates of quantities are combined with assumed unit rates for the various construction tasks e.g. access and site road construction, clearing and grubbing, excavation, backfill, concrete, etc.

Building and structural steel

Costs for a pre-feasibility study would be based on estimated building volumes combined with historical building costs on a per cubic metre basis.

Equipment supply costs

are based on budget quotations from suppliers received in response to preliminary enquiries. Delivery costs should be included in the quotation or developed by the project team for remote projects.

Used process equipment is often suggested as a means of reducing capital costs. However experience shows that the direct cost savings are often eliminated by refurbishment costs and start-up delays. At the pre-feasibility stage, used equipment should not be included in the capital cost estimate unless it is already owned by the developer and is known to be in excellent condition and suitable.

Equipment installation costs

are developed from a combination of historical installation man-hour requirements and the appropriate project crew rate. The crew rate is developed by the estimating team and includes the direct labour cost, the cost of overtime, direct supervision, travel time, subsistence allowance, and contractors profit. The installation cost estimates must reflect labour productivity including the impact of project location.

Instrumentation

At the pre-feasibility level, the cost of instrumentation will be included as a lump sum allowance. The quantity will be selected to reflect local practice, and the complexity of the process plant.

Piping

For a pre-feasibility study, a factor related to the installed cost of the process equipment will generally be sufficient for the in-plant piping. Care must be exercised in plants requiring exotic piping e.g. titanium, or those operations requiring extensive pipe runs such as heap leach operations. In such these cases, the piping specification must be prepared, and the routing of all major lines identified and costed.

The cost for all piping outside the plant, such as water supply and tailings lines, should be estimated in some detail.

Electrical

The source of power, location of transmission lines, switchgear, and primary transformers should be identified where possible. Long power lines, or diesel generating plants should be estimated in some detail. At the pre-feasibility stage, it is sufficient to factor secondary electrical distribution costs based on installed mechanical equipment costs.

Infrastructure costs

A new development in a remote area will incur substantial costs for infrastructure. The obvious requirements are for various ancillary buildings such as warehouses, shops, assay laboratories and administration buildings. Even more remote projects will probably include a fly-in/fly-out program. Therefore the project will include costs for at least an airstrip, and a camp. The camp will probably include accommodation, a cookery, and recreation facilities.

Capital costs in this category are estimated using the techniques mentioned previously. For a pre-feasibility study, the infrastructure facilities must be designed and costed in sufficient detail for a ±20% estimate. Airstrips should be sized for the aircraft that would probably be used for the proposed operation. Camp accommodation should be evaluated for the proposed level of staffing. The camp philosophy (single or double rooms, etc.) must also be determined and used in the design and costing processes.

Indirect construction costs

include the costs for working under winter conditions, establishing and operating construction management offices, crane rental, bonds, permits, construction camp, fly-in/out operation and other similar costs. For the pre-feasibility study, these costs may be estimated as an percentage allowance based on the total direct project costs. The numerical value of the allowance will vary between 6 and 15% of direct costs depending on circumstances.

Engineering, procurement and construction management

These costs are estimated as a percentage allowance based on the total direct project costs. The numerical value of the allowance will vary between 6 and 15% of direct costs depending on circumstances.

Contingency

Money is set aside to cover the cost of items in the original scope, but either missed, or not estimated in detail, or unknown at the time of the estimate. At the feasibility stage this is often included as an allowance equal to 20% of the estimated direct cost of the project. Contingency is not intended to cover scope changes.

The project capital cost estimate will normally be presented in a report including support documentation from suppliers and contractors.

5.4.2. Project operating cost

Project operating costs are more readily determined than capital costs. The main components of the pre-feasibility operating cost estimate are :

Operating labour costs

A manning document is prepared for the mine, process, and administration aspects of the project. The document will include all supervision, operating and maintenance personnel and is based

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on required staffing levels for other, similar plants. Labour rates must be competitive. They may be determined from union contracts, published salary surveys, and discussion with other mines in the area.

Project supply costs

The quantities of explosives, reagents and other supplies are obtained from test data and relevant data from other, similar operations. Delivered costs for explosives, reagents and supplies are obtained from suppliers as budget quotations. In the case of remote projects, costs might be obtained for delivery to a marshalling point. The project team would then develop costs for transport to the project site.

Power costs

An estimate of mine, process, and general power consumption is obtained by an initial analysis of the electrical loads. The unit cost of power is usually obtained as a quotation from a supply utility. At remote sites, power will be generated in a hydro-electric plant or diesel-electric power plant. In either event, preliminary power costs should be developed.

Infrastructure costs

More remote projects will probably include a fly-in/fly-out program. The project costs must include costs for operation of the transportation program and the cost of operating the camp.

Administration costs

On-site administration costs are usually included through the manning document. However there are other costs which should be recognized through appropriate allowances. These include the cost of communications; reproduction; mail; travel to head office, conventions and the like; local taxes; insurance and security.

Head office costs

Various costs are incurred off-site and generally categorized as head office costs. These include the cost of administering the project and uranium marketing costs. These costs are real and must be assessed and included in the pre-feasibility study.

5.5. PRE-FEASIBILITY CASH FLOW AND DCFROR ANALYSIS

Investigation into the possible royalty and tax structure and financial regime of the host country including possible rules on amortization and depreciation, depletion allowances if any, etc., severance taxes, property taxes, provincial taxes and national taxes, and the available financing also needs to occur. An example of a typical, (more detailed) Pre-Feasibility stage cash flow including the items mentioned here is shown on the following pages:

REVENUES

less	mine operating costs (MINE)
	mill operating costs (MILL)
	all transport operating costs (TRANS)
	    mine to mill (MM)
	    mill to smelter (MS)
	    smelter to refinery (SR)
	    refinery to market (RM)
	any post processing operating costs (POST)
	    smelter operating cost (SOC)
	    refinery operating cost (ROC)
	infrastructure operating costs (INFRA)
	leach (if any) operating costs (LEACH)
	any overhead operating costs (OHEAD)
	interest payments (INT)
	depreciation/amortization (DEPR)
	royalty expenses (ROY)
	property tax assessments (PTAX)
	various exploration expenditures (EXPL)
	<u>various development expenditures (DEVL)</u>

equals	BEFORE TAX INCOME (BTINC)
less	depletion (DEPL) various severance taxes (SEVTAX) <u>various state/provincial income taxes (STAX)</u>
equals	FEDERAL/NATIONAL TAXABLE INCOME (FTINC)
less	various federal/national income taxes (FTAX)
plus	<u>any tax adjustment from previous years (ADJ)</u>
equals	NET INCOME
plus	depreciation/amortization (DEPR) depletion (DEPL) any deferred deductions (DD)
less	<u>equity investments (EQI)</u>
equals	NET UNDISCOUNTED CASH FLOW

Again, for further and more detailed explanations on these above mentioned items, please refer to Appendix II.

Compared to the analysis performed during the Pre-Evaluation stage, during this phase, more detail is available, so more in-depth level economic analysis is justified. The analysis often proceeds on two levels.

On one level more emphasis is placed upon geology, engineering and cost parameters so that various sensitivity analyses can be performed. On the second level, attention is placed on the outside effects and impacts that royalties, taxes, potential environmental regulations and financing schemes have upon the economics of the project.

Using the hypothetical base case information, but time with more detail in the areas as described above, the potential Pre-Feasibility stage analysis data are as follows:

- A potential mineral occurrence exists with an estimated 16 000 tonnes of minable Uranium (U_3O_8) which would translate to approximately 4 000 000 tonnes of ore, allowing for a possible life of 12 years, at 320 000 tonnes of ore per year.

- The grade is estimated to average 0.30% U_3O_8 (with estimated recoveries)

- Upon completion of further drilling, the geologists know that there are actually three deposits in one. They are confident that the first two deposits are shallow enough to be considered for open-pit mining. The strip ratio is estimated to be 8:1.

- Testing verified the density of the rock to be 2.5 t/cubic metre.

- The capital cost has been estimated to be $75 000 000 not including the possible cost of acquisition. The operating costs have been further detailed to approximately $65.00/t of ore, not including any production royalty or finance charges, eg., 100% equity.

- Potential financing was researched and found available (and guaranteed by the government and or the lending institution in question) at a number of commercial sources for approximately 50% of the equity necessary, at an average interest rate of 12% for a period of no more than 10 years following production of the first pound of U_3O_8.

- A government royalty of 5.0% of the gross will be levied.

- Taxes are levied on this project at the national level only at a rate of 55% based on a net profits configuration. There are no tax holiday or grace periods offered. Depletion and various types of depreciation/amortization are allowed as deductions to calculate taxable income. Tax loss carry is not used.

Given the above hypothetical, but more detailed information, another DCFROR analysis is performed as was in the Pre-Evaluation stage, and it was determined that a price of approximately $13/lb of U_3O_8 would be necessary to cover all costs and attain a pre-specified rate of return of 20%. This price was much lower than that determined during the Pre-Evaluation stage for a number of reasons.

Possibly one of the more important reasons is the change in the debt to equity ration from 100% equity financing (or, no borrowed funds) to 50% debt/50% equity. Further, a number of sensitivity analyses were performed to determine the impact various parameters have on the potential economics of the project. Figs 13-A, B, C, D and E show the sensitivity of these parameters.

Figures 13-A, B, and C show the impact of varying feed grade, capital and operating costs respectively on the uranium price needed to give a DCFROR of 20%. The required price is quite sensitive to feed grade and capital costs while for this particular hypothetical case, operating cost is less important. While varying the feed grade, the necessary price to attain a 20% DCFROR ranges from approximately $13.00 to $28/lb U_3O_8. Given the comparatively short life of this hypothetical operation, the increase of capital be approximately two thirds times, increases the necessary price by a little more than 80%.
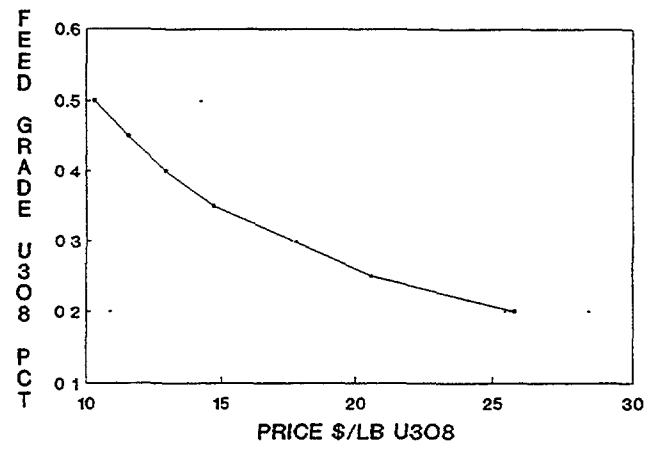
FIG. 13-A. Grade variations vs. price.
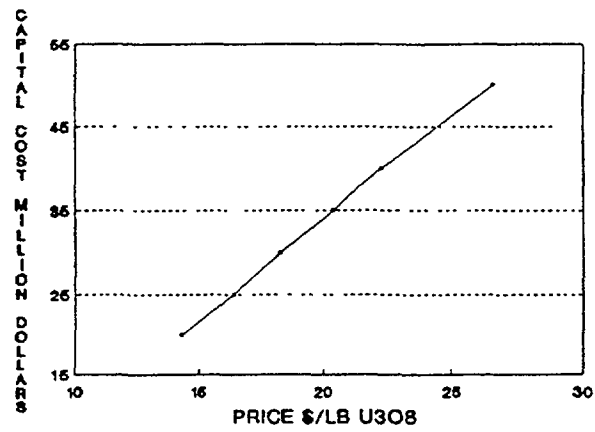
FIG. 13-B. Capital cost vs. price.
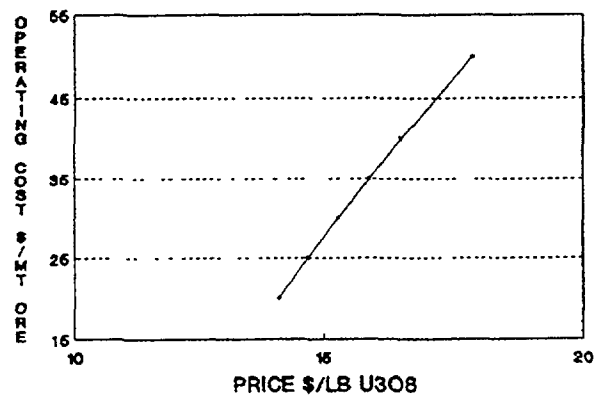
FIG. 13-C. Operating cost vs. price.
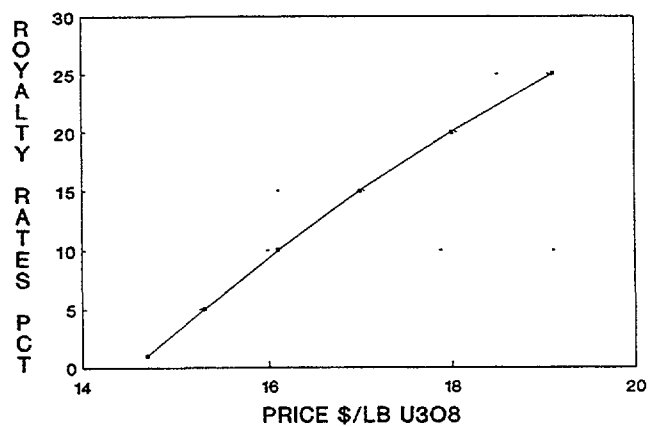
FIG. 13-D. Royalty rates vs price.
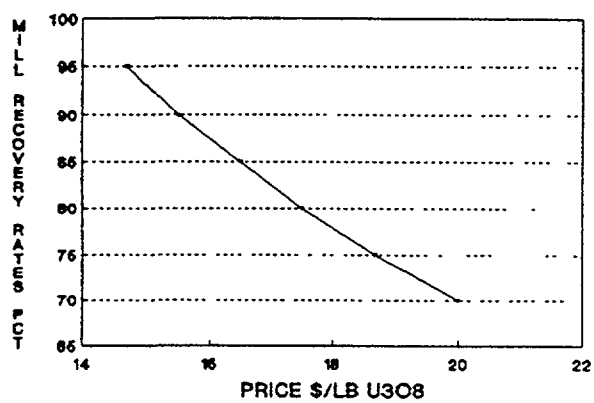
FIG. 13-E. Mill recovery vs. price.

In Fig. 13-D, the royalty rate is varied from the base of 5.0% to 25%, impacting the determined price by approximately 30%.

Fig. 13-E shows the impact of various prices upon the project potential DCFROR. While varying the price between $15 to $50/lb U_3O_8, the project DCFROR varies from 20 to 80%.

Since the determined price still falls within or below the current market price, further investigation and analysis is again, warranted. Therefore, for this hypothetical base case example, after completing the necessary steps within the Pre-Feasibility stage, it is still a "GO" decision. Again, though the numbers suggest "GO", is there a market for the product, and how would the proposed amount of tonnage being considered, effect the supply/demand relationships in the market, and therefore the price?

6. INTERMEDIATE ASSESSMENTS

The pre-feasibility study report must clearly identify the additional work required before the final feasibility can be started. The additional work might include further drilling, test mining, bulk sampling, pilot sorting tests, pilot processing tests and initial financial negotiations.

As each one of the intermediate assessments is completed, the financial analysis of the pre-feasibility study should be revised to confirm that the project is still viable. If, for example, an intermediate pilot plant assessment of the milling process provides revised reagent consumption, then the project operating costs should be revised and the cash flow rerun.

Some typical intermediate studies and assessments are presented below.

6.1. RESOURCE AND GEOLOGICAL INFORMATION

The outcome of the pre-feasibility will dictate whether additional information related to resource estimate, chemical and physical characteristics of the ore and the host rocks, and baseline information of the environment need improving. Additional information needed by the mining and ore processing engineers should receive particular consideration.

6.1.1. Reserves estimation

If the morphology of the deposit, the grade of the ore and the amount of the minable reserves are not known with sufficient degree certainty, it is necessary to improve the quality of these data at this stage. In general, improvement of the reserve estimate was done by increasing the density of sampling and expansion of the area under consideration. This is normally done with complementary drilling programme to fill gaps where denser or more adequate data are required. Any uncertainty in the parameters required for ore reserve estimation noted in 5.1.1 should be corrected. The reserve estimate should then be re-calculated using the newly acquired information.

6.1.2. General geology and hydrogeology information

At this stage the geologist may be asked to provide other geological information useful in the mining of the deposit. One of particular importance are information on the hydrology of the proposed site and the surrounding areas. Determination of ground water patterns and their characteristics are important information needed by the mining engineer. Information on faults and structural discontinuities of the area will also be needed in mine design and planning.

6.2. MINING

Many times, the data used to make the pre-feasibility study are not precise enough and need to be checked and better defined before starting the pre-feasibility study.

Moreover, mining tests and pilot tests which are generally costly have not been carried out on a appropriate scale and have to be done. This work is called: *intermediate assessment*. Among the complementary studies and tests to be done in most cases are noted below:

6.2.1. Hydrogeological conditions

As new information from the geological group becomes available, additional pumping tests, modelling and computer calculation can give a better idea of the data and that can facilitate a better approach to the dewatering and pumping problems related to the best definition of the slope of the walls in open pit operations. It is needed to have a better definition of the pump and pumping stations and the best mining method underground. In some cases, the estimated amount of water is so high that exotic techniques are to be used. For example: freezing the deposit as it had been tested underground at Cigar Lake or dredging in unconsolidated and clayey sandstone.

6.2.2. Geotechnical conditions

A good knowledge of the ground conditions mainly of the ore, hanging and footwall is necessary to calculate the slope of the open pits walls and the best appropriate methods in underground operations.

If this is not the case, additional tests can be done on the cores available from the drilling programme. When the waste or the ore are clayey, for instance, more sophisticated tests are necessary.

6.2.3. Ore dressing

As was noted in the Pre-evaluation Section, the best way to mine a deposit is very often a compromise between mining selectivity and treatment costs.

It is strongly recommended to check very carefully if the ore can not be enriched consequently by a cheap physical technique. If it is really the case, a serious pilot tests can result in a lot of money being saved and at the same time allowing the use of a less selective and consequently a less costly mining method for the same production by treating less tonnage in a smaller mill.

The various techniques to be considered for this purpose are:

– shedding,
– screening and cycloning,
– gravimetry,
– radiometric ore sorting,
– colorimetry.

For checking these techniques, it might be necessary to have a sufficient amount of ore (in the range of some tens or hundreds of tonnes of ore). This will necessitate the drilling of some high ore holes or carry out a small mining test.

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6.2.4. Mining tests

Generally when the hydrogeology, the geotechnical conditions and the characteristics of the ore and the waste are well known, it is not necessary to carry out a mining test which is a costly operation, mainly when the mining method is a classic one. The experience of other miners using the same method can provide valuable and sufficient information for the required job.

However, in some difficult cases, it will be necessary to make a mining test. The location of the test must be determined with the idea that it must be representative and that the needed infrastructures must be established so that they can be used later for the operation.

6.2.5. Tailing disposal

In some cases, when the ground characteristics fit with the requirement, an economic and ecologic way for the tailing disposal is to settle them in an open pit. In such a case, this problem must be looked at as soon as possible. More detailed geological and hydrogeological studies can be asked for by the administration and environmental agencies. It will certainly necessary to launch a programme plugging the drill holes before mining.

6.3. PROCESSING

It is usual for the pre-feasibility study to include recommendations for further testwork aimed at properly defining various design parameters. As an example, it is common to include a semi-autogenous grinding (SAG) circuit in the pre-feasibility study but unusual to have any tests done. Such tests would be recommended in the pre-feasibility study and executed as an intermediate assessment before preparing the final designs of the feasibility study. The SAG tests might be performed in a 0.46 m or 1.83 m diameter pilot mill. The resulting data should confirm the suitability of the SAG method for the ore, provide a refined estimate of power requirements and mill sizing, and identify the necessity to include a crusher in the SAG circuit.

A similar metallurgical pilot plant might be performed for other parts of the process. Radiometric sorting is often the subject of a pilot plant operation in which a large sample of coarse ore is processed through a sorter. The sorted products would then be sampled, assayed and subjected to metallurgical tests.

It is also common to undertake solvent extraction of ion exchange tests on leach liquor as an intermediate assessment. Unwanted impurities can accumulate on the solvent or ion exchange heads. Other permanent changes can occur and can only be detected by longer-term, cyclical testings.

6.4. INFRASTRUCTURE

The pre-feasibility study usually identifies various infrastructure items requiring additional study prior to the initiation of the feasibility study. These might include:

Roads

Will the government assist in the development of roads into the site?

Power

Can a power line be brought into site or will the project require a power plant? If the latter, is there any potential for hydro-electric power generation? Is diesel plant is required, what kind of units will be used?

Camp/town

Will the operation be supported by an existing local town or be operated from new town or using a camp and fly-in/fly-out programme? Will there be government support for a new town or operation camp?

6.5. FINANCIAL

At this stage of operation, the cash flow and DCFROR analysis noted under 5.5 is likely to be repeated, using parameters that are newly acquired in the preceding steps, to test whether the project is still viable. Concurrently, there is probably a sufficient justification to start serious consideration how the mining project might be financed. This may lead to initial financial negotiations.

7. FEASIBILITY STUDY

7.1. INTRODUCTION

In earlier sections of this guidebook it has been shown how the financial viability of a project is assessed in a series of stages. The final measurement of the project viability is assessed in the Feasibility Study. If the Feasibility Study shows that the project can profitably produce uranium at the current market price, the project will probably proceed to detailed design and construction.

In order to illustrate the components of a typical feasibility study, an example project is presented in this section. Further details are provided below but the project can be summarized as follows:

Location: Semi-arid area, grassland, within 100 km of town and 5 km from power lines.

Geology: The deposit consists of two adjacent vein-type orebodies containing approximately 15 000 tonnes of uranium (as U). The predominant uranium mineral is pitchblende with no other significant mineralization. In situ grade of the ore is greater than 0.3% U.

In the sections which follow, the key components of the feasibility study are illustrated by reference to the development of the hypothetical orebody mentioned above.

The differences between the pre-feasibility study and the feasibility study are predominantly due to the greater level of design detail and greater accuracy of the basic cost data. These differences result from the investigation and study work performed as part of the intermediate assessment process described in Section 6.

7.2. A TYPICAL FEASIBILITY STUDY

7.2.1. General description of the deposits

As noted above, the orebodies consist of two adjacent, structurally controlled, steeply dipping vein type deposits with a total minable reserve of 15 000 tonnes of U. Both are located near the surface with a dip varying from 70° to 80° to the southwest. The smaller deposit has a maximum length of 680 m, a depth of 50 m and thickness varying from 3 to 6 m. Its total minable reserve is 2000 tonnes U with an average grade of 0.3% U. The larger vein has a maximum length of 800 m, a depth extension to 340 m and varying thickness from 3.5 to a little over 20 m. The thicker sections of the orebody are found at the shallower part. Below 70 m, its thickness become more uniform which vary from 3.5 to 5 m. The estimated total minable reserve of the larger vein is 13 000 tonnes U grading from 0.3% U in the upper part to 0.4% U in the lower section.

The uranium mineralizations are found in fine to medium grained leucogranite associated with episyeinitization. Both deposits have a simple mineralogy with pitchblende being the primary uranium mineral. No significant amount of other elements are found associated with the uranium.

7.2.2. Mining

From the geological description of the deposit, the general mining scenario can be summarized below:

Orebody No. 1		mined only by open pit
–	*open pit No. 1*	
	reserves	2000 tonnes U
	grade	0.3% U
	strip ratio	10
Orebody No. 2		mined first by open pit and also by underground method in the lower part below the open pit
–	*open pit No. 2*	
	reserves	4000 tonnes U
	grade	0.3% U
	strip ratio	10
–	*underground*	
	reserves	9000 tonnes U
	grade	0.4% U

7.2.2.1. Optimization of the pits

The limits and depth of the pits have been determined using a specific computer program based on the following criteria:

– For each orebody, the computer program will give the optimum profile of the pit so that the average total cost of the uranium mined in the orebody is equal to the theoretical value estimated for the project.

Bench No 1
Bench No 2
Bench No 3
Bench No 4
Bench No 5
Trench No 6

Bench No 1
Bench No 2
Bench No 3
Bench No 4
Bench No 5
Trench No 6

A - A
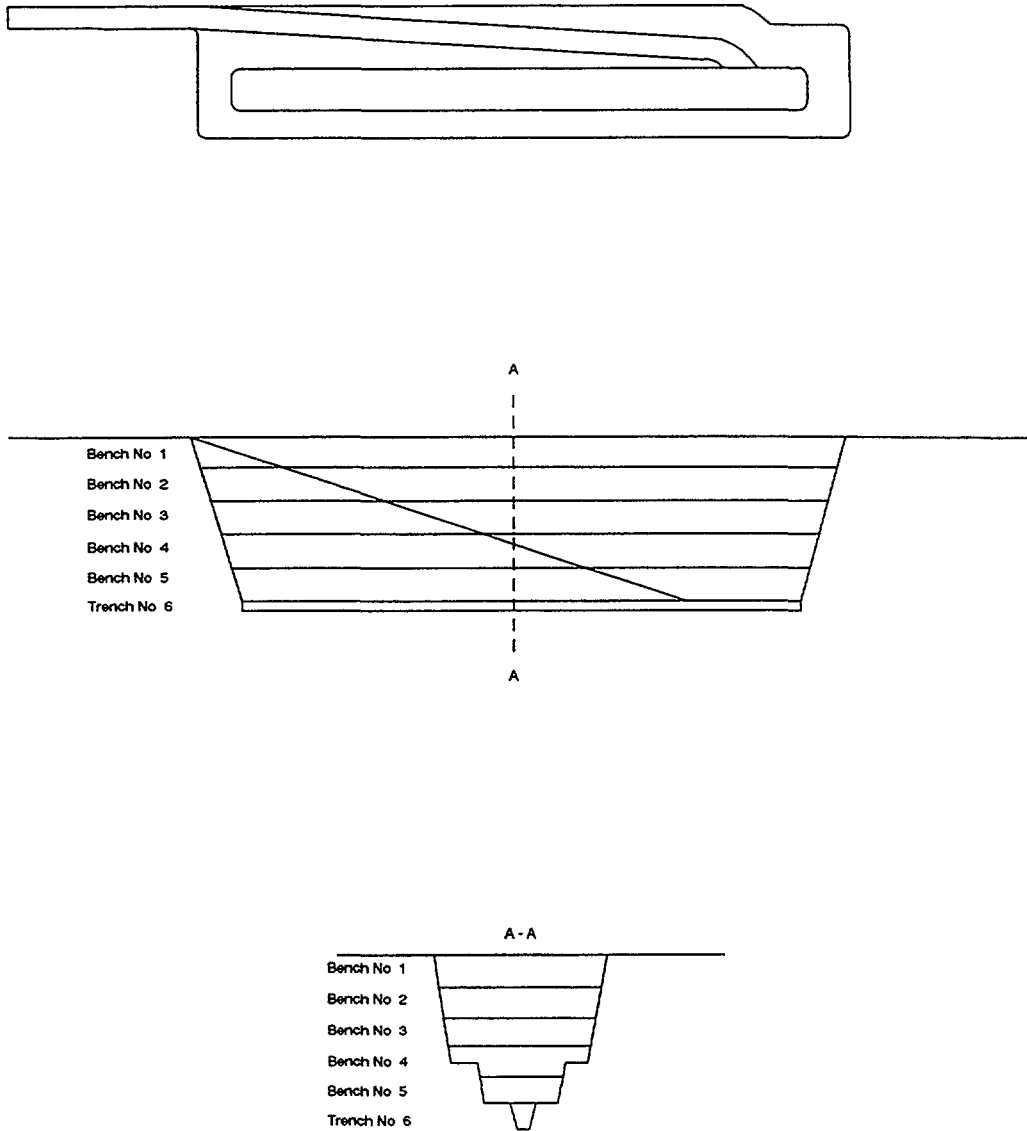
FIG. 14. Hypothetical open pit No. 1.

Bench No. 1
Bench No. 2
Bench No. 3
Bench No. 4
Bench No. 5
Bench No. 6
Bench No. 7

A

A

A - A

Bench No. 1
Bench No. 2
Bench No. 3
Bench No. 4   Ramp
Bench No. 5
Bench No. 6
Bench No. 7
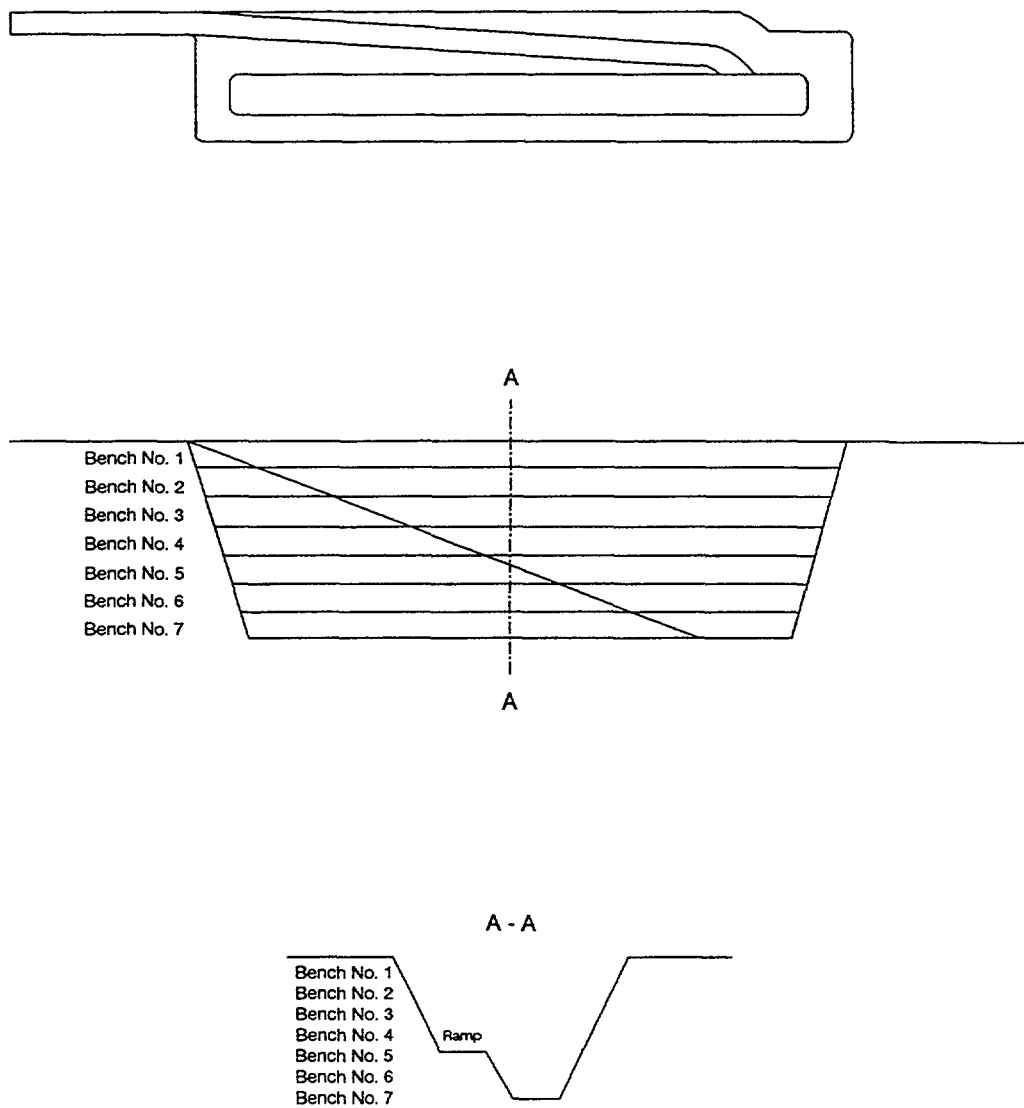
FIG. 15. Hypothetical open pit No. 2.

– When the orebody is also mined by underground methods as for orebody No. 2, the limit between open pit and underground mining is determined by comparing, at a certain depth, the cost of the additional tonne mined by open pit and by underground mining. When the open pit cost is higher than the underground cost, then you have to stop the open pit and begin to mine by underground method.

This study and for a total production cost of US$13/lb U_3O_8 (186 FF/kg U) indicates that:

–   the open pit No. 1 is 684 m long and 55 m deep (Fig. 14);
–   the open pit No. 2 is 800 m long and 70 m deep (Fig. 15);
–   the underground mining should be 800 m in length starting from the upper level at 70 m below the surface to the deepest level at 310 m below the surface.

7.2.2.2. Mining planning

Production and mine life duration

In the base case, it is assumed a production level of 3 million pounds of U_3O_8 per year (1154 tonnes U). Using a total average recovery of 0.925, the related feed mill is 1.25 tonnes of uranium per year. This will give a 12-year life for the mine.

Tailings disposal

The previous studies, pre-evaluation, pre-feasibility study and assessment have shown that the best way for the tailings disposal is to place them in the open pits and cover them with a 10 metre thick layer of waste before reclamation of the land.

Using this approach, it is necessary to begin mining with the open pit before starting the mill. Three different options may be considered:

–   The first is to mine open pit No. 1 two years before starting the mill. At the end of the second year, enough space would have been developed at the bottom of the pit to place the first year of the tailing from the mill. This can be achieved by building a dam at the limit of the mined part of the pit and place the tailing behind the dam. This is probably more economical, however, there may be a concern from the management to continue mining and blasting so close to the dam.

–   A second option would be to complete mining the first pit before starting the mill. This way, the management will not have to face the problem noted in the first option.

–   A third option is to build on the surface a preliminary tailing pond using part of the waste coming from the pit.

A complete analysis of the above mentioned possibilities suggests that the second option was considered most appropriate, thus avoiding potential problem with regulatory and safety agencies.

The volume actually available in open pit No. 1 is large enough to accommodate the tailings of the nine first years of the mine. The tailings coming from to the last three years of the operation will be easily disposed off at the bottom of open pit No. 2. They will be located adjacent to the portal of the decline and will then not require any change in the mining operations.

7.2.2.3. Production planning

Mining is planned so that at any time, a sufficient space will be available to dispose the tailing. Furthermore, the ore stockpile should be so managed so that, at any time, there is enough ore to feed the mill.

During operation of the mine, there are a number of occasions where the quantities of stockpiled ore are sometimes high, the reasons are:

— at the starting of the mill (beginning of first year), sufficient volume of space has to be created in pit No. 1 for the tailing disposal;

— at the beginning of the underground operation, in order to establish the slopes and increase progressively the underground production.

7.2.2.4. Radiometric sorting

The pre-feasibility study has shown that the ore had the characteristics fitting well with a radiometric sorting:

— the ore is strong and rocky;
— the percentage of small and fines in the raw ore is low: 15%;
— there is little or no mixed ore.

Lab test conducted during the pre-feasibility study have shown that, on the total, it should be possible to increase the grade of the ore by eliminating 25% of subore at a grade of 0.0035% uranium. A simplified flow sheet for the radiometric ore sorting and stockpiling is shown in Fig. 16.

A pilot test conducted in the pilot plant of the maker has confirmed these figures. The final results are:

RAW ORE

	OPEN PIT				UNDERGROUND		
	% Weight	Grade U	% U		% T	Grade U	% U
< 25 mm Fines	30%	0.340	34%	Fines	30%	0.440	33%
> 25 mm Stones	70%	0.283	66%	Stones	70%	0.329	67%
Total	**100%**	**0.300**	**100%**	**Total**	**100%**	**0.400**	**100%**
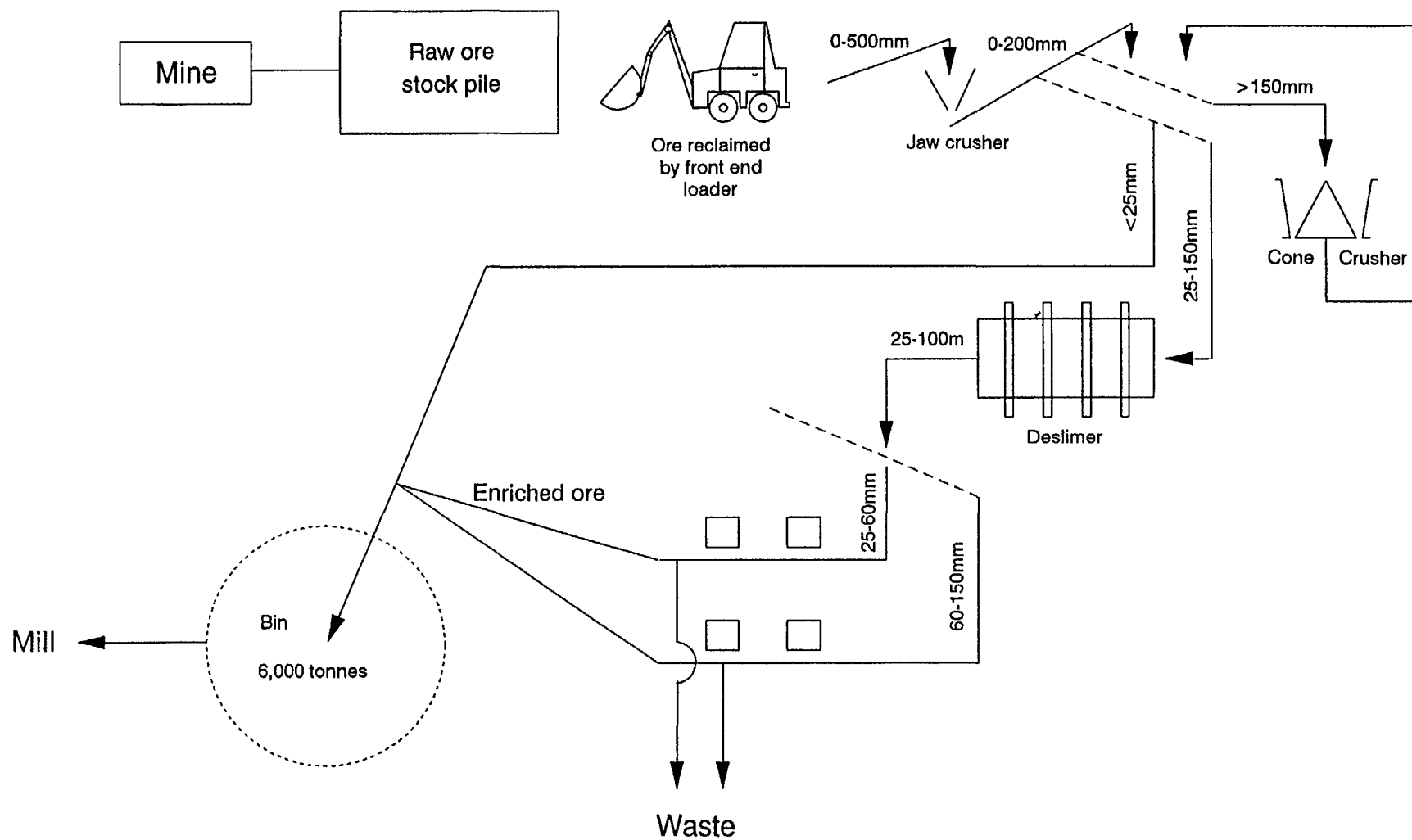
FIG. 16. A simplified flowsheet for radioactive ore sorting and stockpiling.

RADIOMETRIC SORTING — OPEN PIT

		% T on the stones	Grade U	% U	% T on the total raw ore	Grade U	% U
<25 mm	Raw	100	0.283	100	100	0.300	100%
	stones	35.71	0.035	4,42	25	0.0350	2.92%
	Sub ore	64.29	0.421	95,58	45	0.421	63.08%
	Enriched ore						
>25 mm	Fines	-	-	-	30	0.340	34.00%

		Uranium	Weight
Uranium recovery:	on the stones >25 mm	95.58%	64.29%
	on the total (stones and fines)	97.08%	75.00%

RADIOMETRIC SORTING — UNDERGROUND

		% T on the stones	Grade U	% U	% T on the total raw ore	Grade U	% U
<25 mm	Raw stones	100	0.383	100	100	0.400	100
	Subore	35.71	0.0350	3.800	25	0.035	2.91
	Enriched ore	64.29	0.576	96.200	45	0.576	64.81
>25 mm	Fines	-	-	-	30	0.440	33

		Uranium	Weight
Uranium recovery:	on the stones	96.201%	64.29%
	on the total	97.810%	75.00%

MILL FEED

		% T	Grade U %	% U	% T	Grade U %	% U
<25 mm	Fines	40	0.34	35	40	0.440	33.87
>25 mm		60	0.421	100	60	0.576	66.13
Total		**100**	**0.388**	**100**	**100**	**0.5196**	**100**

According to these results and due to the total recovery which varies from 92.23% for the open pits and 92.92% for the underground the production in the yellow cake being maintained constant at 3 million pounds per year of U_3O_8, the yearly production of the mine ROS-Mill are as shown in the following Tables VI and VII.

TABLE VI. MINE PRODUCTION

YEAR		-2	-1	0	1	2	3	4	5	6	7	8	to	12
OP No. 1														
Ore	MT	0.150	0.250	0.267	-	-	-	-	-	-	-	-		-
Grade	%U	0.3	0.3	0.3	-	-	-	-	-	-	-	-		-
Lbs U_3O_8	Mlbs	1.170	1.950	2.080	-	-	-	-	-	-	-	-		-
Lbs YTD	Mlbs	1.170	3.120	5.200	-	-	-	-	-	-	-	-		-
OP No. 2														
Ore	MT	-	-	-	0.300	0.300	0.300	0.300	0.133	-	-	-		-
Grade	%U	-	-	-	0.3	0.3	0.3	0.3	0.3	-	-	-		-
Lbs U_3O_8	Mlbs	-	-	-	2.340	2.340	2.340	2.340	1.040	-	-	-		-
Lbs YTD	Mlbs	-	-	-	2.340	4.680	7.020	9.369	10.400	-	-	-		-
U 6 No. 2														
Ore	MT	-	-	-	-	-	-	0.100	0.150	0.200	0.250	0.310		0.310
Grade	%U	-	-	-	-	-	-	0.4	0.4	0.4	0.4	0.4		0.4
Lbs U_3O_8	Mlbs	-	-	-	-	-	-	1.040	1.560	2.08	2.600	3.224		3.224
Lbs YTD	Mlbs	-	-	-	-	-	-	1.040	2.600	4.68	7.280	10.504		23.400
TOTAL														
Ore	MT	0.150	0.250	0.267	0.300	0.300	0.300	0.400	0.283	0.200	0.250	0.310		0.310
Grade	%U	0.3	0.3	0.3	0.3	0.3	0.3	0.325	0.353	0.4	0.4	0.4		0.4
Lbs U_3O_8	Mlbs	1.170	1.950	2.080	2.340	2.340	2.340	3.380	2.597	2.08	2.600	3.224		3.227
Lbs YTD	Mlbs	1.170	3.120	5.200	7.540	9.880	12.220	15.600	18.197	20.277	22.877	26.101		39.000

TABLE VII. ROS-MILL PRODUCTION STOCKPILE

YEAR		-2	-1	0	1	2	3	4	5	6	7	to	12
ROS													
Feed	MT	-	-	-	0.292	0.292	0.292	0.292	0.269	0.226	0.217		0.217
Grade	%U	-	-	-	0.283	0.283	0.283	0.283	0.309	0.368	0.383		0.383
Enriched	MT	-	-	-	0.1877	0.1877	0.1877	0.1877	0.173	0.145	0.1395		0.1395
Grade	%U	-	-	-	0.421	0.421	0.421	0.421	0.460	5.540	0.576		0.576
Lbs U_3O_8	M	-	-	-	2.055	2.055	2.055	2.055	2.070	2.090	2.090		2.090
Lbs U_3O_8	YTDM	-	-	-	2.055	4.110	6.165	8.220	10.290	12.380	14.470		22.830
FINES													
<25 mm	MT	-	-	-	0.125	0.125	0.125	0.125	0.1152	0.097	0.093		0.093
Grade	%U	-	-	-	0.34	0.34	0.340	0.340	0.364	0.425	0.44		0.44
Lbs U_3O_8	M	-	-	-	1.105	1.105	1.105	1.105	1.090	1.071	1.060		1.06
Lbs U_3O_8	YTDM	-	-	-	1.105	2.210	3.315	4.420	5.510	6.581	7.640		11.88
MILLFEED													
Ore	MT	-	-	-	0.3127	0.3127	0.3127	0.3127	0.288	0.242	0.233		0.235
Grade	%U	-	-	-	0.388	0.388	0.388	0.388	0.422	0.502	0.517		0.517
Lbs U_3O_8	M	-	-	-	3.160	3.160	3.160	3.160	3.160	3.160	3.160		3.160
Lbs U_3O_8	YTDM	-	-	-	3.160	7.320	9.480	12.640	15.800	18.960	22.120		37.920
PRODUCTION													
Recovery		-	-	-	0.95	0.95	0.95	0.95	0.95	0.95	0.95		0.95
Lbs U_3O_8	M	-	-	-	3.000	3.000	3.000	3.000	3.000	3.000	3.000		3.000
Lbs U_3O_8	YTDM	-	-	-	3.000	6.000	9.000	12.000	15.000	18.000	21.000		36.000
ORE STOCKPILE End of year													
Ore	MT	0.150	0.400	0.667	0.549	0.431	0.313	0.294	0.193	0.071	0.010		000
Grade	%U	0.3	0.3	0.3	0.3	0.3	0.3	0.335	0.376	0.400	0.400		-
Lbs U_3O_8	M	1.17	3.92	5.200	4.282	3.336	2.441	2.5607	1.887	0.738	0.104		000

7.2.2.5. Mining economic assessment

Open pits

It is assumed, as frequently recommended, that the open pit mining operations are done through contract arrangements, except for the geological control which is carried out by technician geologists of the mine. The topography team will measure the volumes of ore mined and waste excavated in order to control the bill sent at the end of every month.

The economic limit of the depth of open pit No. 1 is 50 metres but as the ore has still a good grade and thickness it is profitable to mine a trench as deep as possible in the bottom of the pit. It is assumed that it is possible to mine 5 metres below the 50 metres level.

Ore stockpile

This problem must be anticipated at the time of the operations at open pit No. 1 in order to create enough space for the tailing when the mill starts. Consequently, it is necessary to be prepared to manage a stockpile of up to 667.000 tonnes of ore.

Underground operations

The volume of the stockpile decreases from the moment when the mill starts, but it should be maintained at a certain level in order to increase progressively the underground production. By doing that, it is possible to delay the outline of the mine infrastructure and to better manage the produced waste for the backfilling.

Construction of the two main declines will be contracted as well as for all the ventilation shafts and raises. Similar arrangement should also be made for the shaft to backfill the waste, the shaft for the pumping pipes, etc.

All the other jobs will be done by mine personnel.

7.2.2.6. Geotechnic and general layout of the mines

-       Geotechnic conditions:

Ground: the ground conditions are rather good, a little less at the hanging wall than on the foot wall.

Water: the ground water is not a problem. There will be low inflows in the open pits and underground. The hydrogeologic model gives a value between 50 m^3/h at the beginning and 160 m^3/h at the end of the mine.

Morphology of the veins:
            the vein dips at an angle which can vary from 70 to 80°. This added to the good ground condition thus making the mining easy.

            The economic thickness varies in a large range from 4 to more than 20 metres in some places.

In the underground part, the vein is more regular, between 3.5 to 5 metres. 15% of the volume to be mined is non economic and considered as waste used for backfilling.

- Layout of the mine:

Open pits:

According to the geotechnic conditions, the open pits will be mined as follows:

– no bolting is required;

– the slopes of the wall will be mined at an angle of
  60% to the footwall
  50% to the hanging wall;

– the ramp has a depth of 10%. It has a width of
  20 metres for open pit No.1
  25 metres for open pit No. 2;

– in these conditions the striping ratio is 10;

– the waste will be mined by 10 metre benches and 5 metre benches for the ore;

– the first 20 metres can be ripped using a powerful bulldozer like Caterpillar D10;

– in the deeper part, it is necessary to blast. As the water inflow is low on the top ANFO can be used. Deeper blasting will be done with slimies;

– a concrete slab will be poured at the bottom of pit No. 2 in order to allow the underground miners to mine the ore below in good safety conditions.

Underground:

Due to the conditions and the level of the yearly planned production, the vein will be mined by the cut and fill method. The dimensions of the minable part of the vein which thickness varies from 3.5 to 5 metres with some 15% of non economical subore zone are:

length: 800 metres
depth: 270 metres.

The deposit is divided in 6 main levels, 45 metres high each. The levels are divided in five blocks with an average length of 160 metres. The average tonnage contained in a block is 75 000 tonnes of ore and 300 tonnes of U. The access to the stopes, 4.1 metres high and on an average 4 metres wide, is made from a decline, one decline for each block (see Fig. 17).

The infrastructure of the mine comprises:

– a double main decline drilled from the bottom of the open pit for hoisting the ore and the waste, when necessary also for the personnel, consumables, small equipment and miscellaneous;

– 6 main levels drilled parallel to the vein at a distance of 10 metres in the footwall where the conditions are better;

Primary and Final Portals *

Concrete Slab

Pit Bottom

Main Levels 1   Block No. 11

Main Declines

Block No. 33

Block No. 54

Main Levels

Ventilation Levels

Main Declines

Ventilation Shafts

Ore Vein
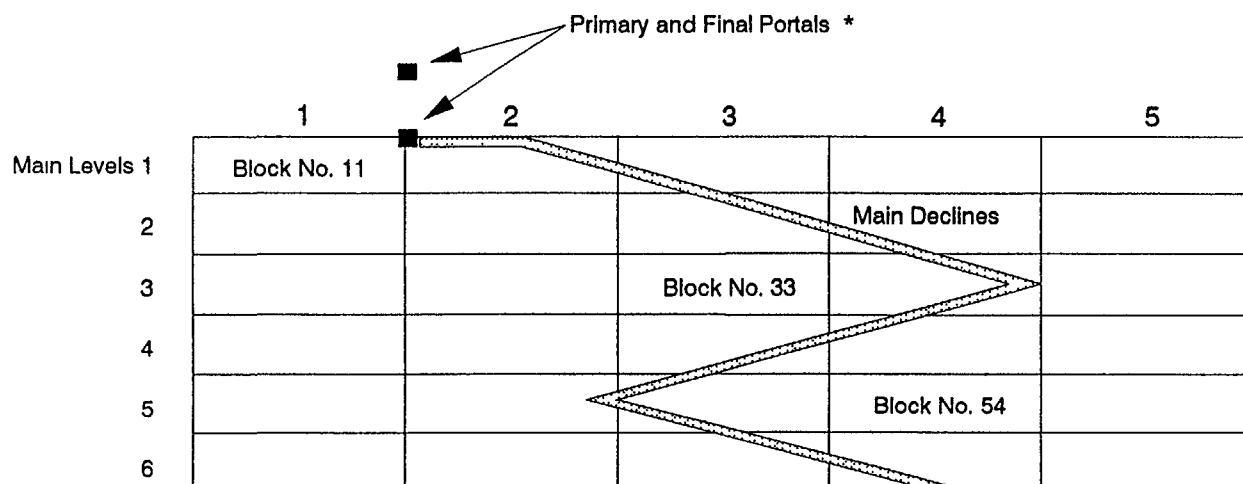
*   due to the fact that we must prepare the underground mine before termination of Pit No. 2, the first entry of the decline is located above the bottom of the pit. Later on, a new portal is established at the bottom of the pit. Horizontal junctions to the main declines are drilled.
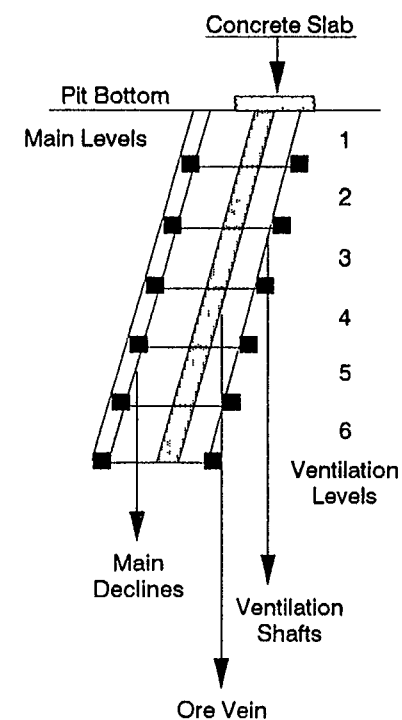
FIG. 17. Schematic diagram of the underground mine plan. General plan.

Concrete slab poured at MAIN LEVEL 4

Stopes 11 — MAIN LEVEL 4
10
9 Stope 549
8
7
6
5 Stope 545
4
3
2
1 — MAIN LEVEL 5

Concrete Slab — MAIN LEVEL 4

Stope Decline — Ore Vein in Place

Ventilation Shaft

Stope No. 546

Backfilling

MAIN LEVEL 5

Concrete Slab
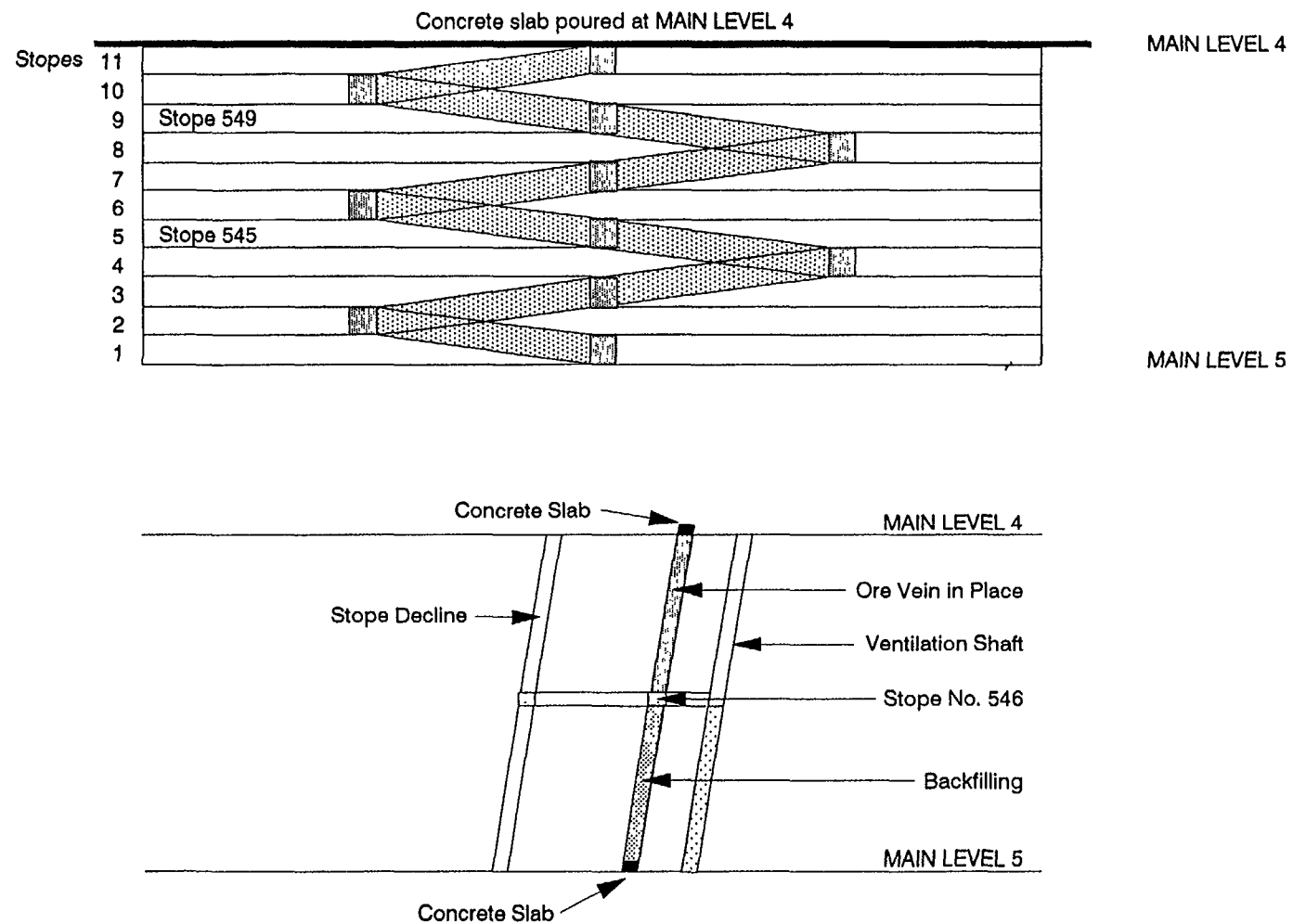
FIG. 18. Schematic diagram of the underground mine plan. Detail of level 4.

–   5 ventilation levels are drilled in hanging wall. These levels collect the exhaust and polluted air through five ventilation shafts.

With such a layout the fresh air coming from the main declines feeds the slopes and is directly exhausted through the ventilation shafts and ventilation levels. In these conditions each block is always fed with fresh air which is absolutely necessary due the radon problem.

Backfilling is made with the waste blasted in the non economic zones and the infrastructures. An effort should be made to try, as much as possible, to balance the production and the use of waste. It is easily achievable during the first half of the mine operation, but during the later part of the operation a considerable amount of waste has to be managed.

In the stopes, drilling is done with one boom hydraulic rig, the blasted products are then mucked with load haul dump (LHD) equipments with a bucket capacity of 6 cubic yards. All the equipment used is trackless. Backfilling in the stopes is carried out with the same equipment as used for mucking.

At each main level, when the first stope is terminated, a slab of concrete 1.5 metres high is poured on the floor as a protection for mining the stope below (the last stope of the next block) (Fig. 18).

7.2.2.7. Unit costs

Capital costs

Due to the good ground conditions it is assumed that the different costs will be as follows:

Main declines:	as the two declines are drilled together the cost is lowered compared with the cost of a single decline.	
	Section: 15 m² —	1500$/m
Main levels:	Section: 12 m² —	1200$/m
Ventilation levels:	Section: 10 m² —	1000$/m
Cross cuts:		900$/m
Ventilation shafts:		800$/m
Waste shaft:		1000$/m

Production costs	Underground
Slopes decline	1000$/m
Cross cuts	200$/m
Net ore tonne mined	32$/t (net tonne)
Hoisting to the surface	3$/t
Waste to the slopes	3$/t

	Open pit
Top soil	0.7$/t
Ripped waste	0.8$/t
Blasted waste	1.7$/t
Ripped ore	2.4$/t
Blasted ore	5.1$/t
Geological control	2$ t ore
Topo water	2$ t ore

OPEN PIT NO. 1

YEAR		-2	-1	0	TOTAL
Ripped ore	t	63.500	-	-	**63.500**
Blasted ore	t	86.500	250.000	267.000	**603.500**
TOTAL ORE	t	**150.000**	**250.000**	**267.000**	**667.000**
Top soil	t	206.500	-	-	**206.300**
Ripped waste	t	3,546.500	-	412.800	**3,546.500**
Blasted waste	t	713.000	1,393.200	811.000	**2,917.200**
TOTAL WASTE	t	**4,259.500**	**1,393.200**	**811.000**	**6,463.700**
Waste + top soil		4,452.800	1,393.200	811.000	**6,670.000**

OPEN PIT NO. 2

YEAR		-2	-1	0	1	2	3	4	5	TOTAL
Ripped ore	t			-	259.300				-	**259.300**
Blasted ore	t			-	40.700	300.000	300.000	300.000	133.000	**1,073.700**
TOTAL ORE	t			-	**300.000**	**300.000**	**300.000**	**300.000**	**133.000**	**1,333.000**
Top soil	t			363.000	-	-	-	-		**363.000**
Ripped waste	t			412.800	4,120.700	-	-	-		**4,533.500**
Blasted waste	t			-	379.300	2,490.000	1,992.700	2,157.200	1,417.600	**8,436.800**
TOTAL WASTE	t			**412.800**	**4,500.000**	**2,490.000**	**1,992.700**	**2,157.200**	**1,417.600**	**12,970.300**
Waste + top soil				775.800	4,500.000	2,490.000	1,992.700	2,157.200	1,417.600	**13,333.300**

UNDERGROUND NO. 2

YEAR		3	4	5	6	7	8	9	10	11	12
Main decline hoist (MDH)	m	350	350	350	350	350	280	-	-	-	-
Cross cuts (MDH-CC)	m	10	10	10	10	10	10	-	-	-	-
Main service decline (MSD)	m	350	350	350	350	350	280	-	-	-	-
Cross cuts (MSD-CC)	m	10	10	10	10	10	10	-	-	-	-
Main level (ML)	m	800	-	800	800	800	800	800	-	-	-
Cross cuts (ML-CC)	m	50	-	50	50	50	50	-	-	-	-
Ventilation level (VL)	m	-	800	-	800	800	800	-	800	-	-
Cross cuts (VL-CC)	m	25	-	25	25	25	25	-	25	-	-
Ventilation shafts (VS)		250	-	250	250	250	250	-	250	-	-
Cross cuts (VS-CC)	m	-	73	110	150	183	227	227	227	227	227
Stopes decline (SD)	m	-	420	630	840	1.050	1.300	1.300	1.300	1.300	1.300
Cross cuts (SD-CC)	m	-	146	220	300	366	453	453	453	453	453
Main pump											
Main pumping station											
Intermediate pumping stations											
Ore mined	KT	-	100	150	200	250	310	310	310	310	310
Ore hoisted	KT	-	100	150	200	250	310	310	310	310	310
Waste hoisted — up	KT	65.000	-	-	-	-	-	-	-	-	-
Waste hoisted — down	KT	-	-	-	-	15.000	46.000	96.000	96.000	122.000	122.000

CAPITAL COSTS

(1) UNDERGROUND

YEAR		3	4	5	6	7	8	9	10	11	12	TOTAL
Infrastructure	M$	2.296	1.868	2.296	3.096	3.396	2.886	0.960	1.025	-	-	**17.824**
Ventilation		0.020	0.040	0.100	0.200	0.200	-	-	-	-	-	**0.500**
Pumping		0.100	0.100	0.200	0.100	0.200	2.5	-	-	-	-	**3.200**
Equipment		1.930	2.090	2.625 1	0.930	0.357	1.930	2.090	2.625	-	-	**14.577**
Miscellaneous		0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	-	-	**1.600**
TOTAL		**4.546**	**4.298**	**5.421**	**4.526**	**4.353**	**7.516**	**3.250**	**3.851**	**-**	**-**	**37.761**

(2) OPEN PITS

YEAR	-2	-1	0	1	2	3	TOTAL
Geological control	0.500	0.500	-	-	-	-	1.000
Miscellaneous	0.300	0.300	0.200	-	-	-	0.800
TOTAL	**0.800**	**0.800**	**0.200**	**-**	**-**	**-**	**1.800**

PRODUCTION COSTS

(1) UNDERGROUND

YEAR		3	4	5	6	7	8	9	10	11	12	TOTAL
Stopes decline	M$	-	0.420	0.630	0.840	1.050	1.300	1.300	1.300	1.300	1.300	9.440
Cross cuts		-	0.215	0.383	0.491	0.580	0.680	0.612	0.635	0.612	0.612	4.820
Ore mined		-	3.200	4.800	6.400	8.000	9.920	9.920	9.920	9.920	9.920	72.000
Hoisting up		-	0.300	0.4501	0.600	0.750	0.930	0.930	0.930	0.930	0.930	6.750
Hoisting down		-	-	-	-	0.045	0.138	0.288	0.288	0.366	0.366	1.491
Miscellaneous	25%	-	1.034	1.566	2.083	2.606	3.242	3.263	3.288	3.282	3.282	23.646
TOTAL		-	5.169	7.829	10.414	13.031	16.210	16.313	16.361	16.410	16.410	118.147

(2) OPEN PITS

YEAR		-2 OP-2	-1 OP-1	0 OP-1	0 OP-2	1 OP-2	2 OP-2	3 OP-2	4 OP-2	5 OP-2	TOTAL OP1+OP2
Top soil		0.145	-	-	0.254	-	-	-	-	-	0.399
Ripped Waste		2.837	-	-	0.330	3.297	-	-	-	-	6.464
Blasted Waste		1.212	2.369	1.379	-	0.645	4.233	3.388	3.667	2.410	19.303
Ripped ore		0.135	-	-	-	0.622	-	-	-	-	0.757
Blasted ore		0.442	1.224	1.362	-	0.208	1.530	1.530	1.530	0.679	8.505
Geological control		0.300	0.500	0.534	-	0.600	0.600	0.600	0.600	0.266	4.000
Topography		0.300	0.500	0.534	-	0.600	0.600	0.600	0.600	0.266	4.000
Miscellaneous	10%	0.539	0.459	0.381	0.059	0.597	0.677	0.612	0.640	0.362	4.326
TOTAL		5.910	5.052	4.190	0.643	6.569	7.640	6.730	7.037	3.983	47.754

Mine Superintendant + Administration - 3 = 4

Mine Works	Geology - Topography	Mechanic - Electricity	Surface
2 Chief Foremen 10 Foremen 80 Miners	1 Geologist 7 Technicians	1 Foreman 6 Workers	1 Foreman 8 Workers

4 + 92 + 8 + 7 + 9 = 120
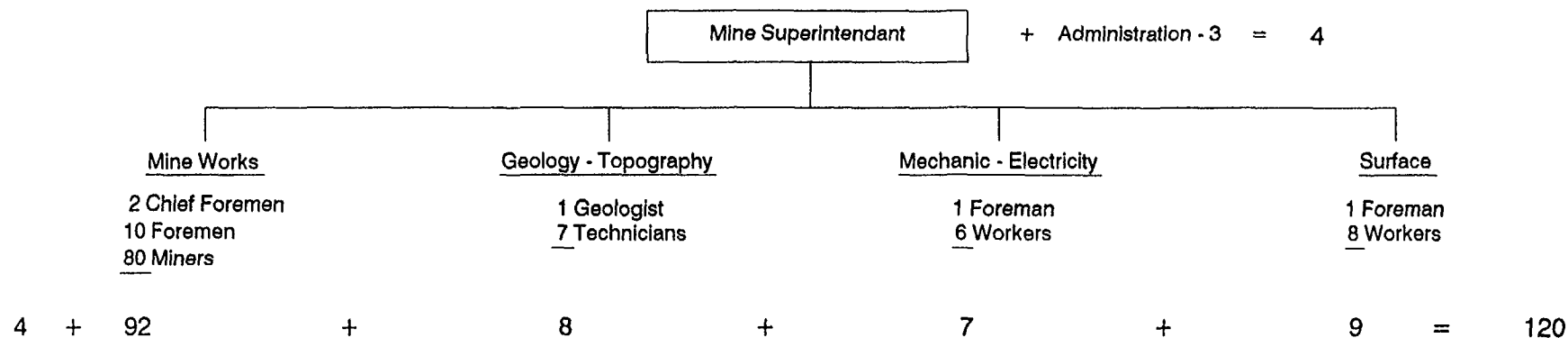
Work Force Distribution Across the Life of the Mine

Year	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Total Work Force	10	10	14	22	22	26	60	75	90	105	120	120	120	110	110	20	-
Engineers	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Chief Forman	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	-	-
Formen				2	2	5	6	8	9	10	12	12	12	12	12	2	-
Technicians	3	3	3	3	3	3	4	5	6	7	7	7	7	7	7	1	-
Employees	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	1	-
Workers	3	3	6	11	11	12	44	56	69	81	94	94	94	84	84	16	-

FIG. 19. Hypothetical worksheet and work force distribution for the mining part of the operation.

The hypothetical work sheet of the and its work force distribution during the entire life of the mine operation is shown in Fig. 19. It should be noted that these figures represent only the work force required for the mining operation part of the project from year to year.

7.2.3. Milling

7.2.3.1.General

Ore will be processed in mill facilities located at the project site. Ore from the open pit and underground mines will be stockpiled at the mill site. The mined-out open pit will be converted to a tailings disposal system.

The radiometric sorting plant and mill has been designed to produce 3 000 000 pounds of U_3O_8 per year in high quality uranium peroxide concentrate. The radiometric sorting plant has been designed to handle a total of 417 000 t of ore/year including 25 mm fines (292 000 t/a of +25 mm feed). The mill has been designed to process a peak of 312 700 t/a of sorted ore at a feed grade of 0.31% U, 95.0% U recovery, and an effective operating time of 330 days per year. All circuits will normally operate 24 hours per day for an average design mill throughput of 950 t/d.

The uranium recovery process will include:

- Radiometric ore sorting (ROS) of run-of-mine ore
- Grinding of mill feed
- Neutral thickening and pulp storage
- Two-stage acid leaching with intermediate thickening
- Countercurrent decantation (CCD) solid-liquid separation
- Solvent extraction of uranium from pregnant solution
- Gypsum precipitation to remove impurities from solvent extraction pregnant strip solution
- Uranium precipitation using hydrogen peroxide
- Uranium peroxide drying and packaging
- Tailing neutralization and disposal
- Water treatment.

7.2.3.2. Radiometric ore sorting

As noted in Section 7.2.2.5, ore received from the mines is sorted and stored based on uranium grade. This allows radiometric ore sorter (ROS) feed to be blended for a relatively constant uranium head grade. ROS feed is crushed in a jaw crusher, screened and washed free of fines. The +25 mm fractions (70% of the raw ore) is fed to the radiometric ore sorters. The accept fraction from the ore sorters recovers 64 weight percent of the feed material, with a uranium recovery exceeding 97%. The accept material is stockpiled as mill feed for further processing. The reject stream at a grade of 0.035% U is trucked to waste.

7.2.3.3. Processing

Grinding

During the pre-feasibility and intermediate assessment stages, the ore was shown to be amenable to SAG milling and all necessary sizing parameters were obtained from a pilot plant operation. In the plan, mill feed is reclaimed from stockpile by front-end-loader and fed into a surge hopper, from which it is withdrawn by a variable speed feeder and moved to the grinding circuit by conveyor belt.

A SAG mill/ball mill circuit is used to grind the mill feed. Final ground product is sized and separated by hydrocyclone.

Neutral thickening and pulp storage

Testwork and economic analyses performed during intermediate assessment confirmed that it is advantageous to thicken the ground ore before leaching. Therefore, ground ore slurry is directed to neutral thickener which dewaters the ore slurry to achieve the required density for leaching. Neutral thickener overflow is recycled for use as grinding water.

Thickened ore slurry is then pumped to air-agitated storage tanks which provide surge capacity, further blending, and radon removal. Pulp is metered from these tanks to the primary leach circuit.

Leaching

Leaching is achieved in mechanically agitated tanks. Studies show that a two-stage (primary and secondary) leach process could be used to reduce reagent requirements and to reduce the potential for organic solvent degradation. In the plant, pregnant leach solution is separated from the leached ore using a train of high capacity CCD thickeners. Overflow from the first CCD thickener, containing residual reagents, is used to dilute primary leach feed and provide the necessary leaching reagents for the primary stage leach (the weak acid stage). Slurry from the primary leach is pumped to the primary leach thickener. Pregnant aqueous overflow from the primary leach thickener is clarified using a clariflocculator and sand filters. Clarified pregnant aqueous is pumped to the solvent extraction circuit.

Primary leach thickener underflow and clariflocculator underflow is fed to the secondary leach circuit. Sodium chlorate and sulphuric acid are added to complete the leaching process. Slurry discharged from the secondary leach is pumped to the CCD circuit.

Solid-liquid separation

Leached solids are separated from the uraniferous solution and washed using a CCD circuit including high capacity thickeners. Overflow solution from the first CCD thickener is returned to the primary leach. Recycled raffinate solution and acidified process water are added to the last thickener as wash solution.

CCD circuit tailing from the last thickener, containing the washed leach residue solids, is pumped to the tailing neutralization circuit.

Solvent extraction

During the intermediate studies, a continuous solvent extraction mine-plant was operated for several days and confirmed that the system was stable. A total of 60 cycles were performed on the organic inventory. During this period, alternative mixer-settler designs are tested and final design data obtained. In the actual solvent extraction circuit, the pregnant aqueous solution (from the primary leach) is contacted countercurrent with a barren, continuously circulating organic solvent. Mixer-settler units are used. The organic solvent is a tertiary amine dissolved in kerosene. The organic extracts the uranium from the aqueous solution, giving a pregnant organic and an aqueous raffinate. The latter is partially recycled to CCD wash, and partially sent to the tailing neutralization circuit.

The loaded organic is contacted countercurrent with strong acid strip solution to remove the uranium from the solvent. The resulting barren organic is water-washed for recovery of sulphuric acid,

which is recycled to CCD wash or to strip solution make-up. Part of the barren organic is regenerated by contacting with a caustic sodium carbonate solution. The used regenerant is directed to tailing neutralization. The whole barren organic stream is then returned to the extraction mixer-settlers. The pregnant strip solution is pumped to the gypsum precipitation circuit.

Gypsum precipitation

In the gypsum precipitation process slaked lime slurry and, subsequently, magnesia slurry are added to precipitate impurities from solution and to reduce the sulphate concentration. The gypsum precipitate is separated from the solution by thickening and filtration. Test work showed that the gypsum contains some uranium. Therefore, the production plant was designed to allow the gypsum filter cake to be recycled to the first CCD thickener for dissolution and recovery of the entrained uranium. The solution separated from the gypsum is clarified in sand filters and constitutes the feed to uranium precipitation.

Uranium precipitation

In the uranium precipitation circuit, hydrogen peroxide is added to precipitate uranium peroxide. Magnesia slurry is added as required to control pH. The uranium peroxide is washed and separated from the barren strip solution by a thickener, then fed to a belt filter.

Drying and packaging

Uranium peroxide is filtered on a belt filter and washed countercurrent using both fresh water and scrubber solution as wash water. Filtrate is pumped to the uranium peroxide thickener as wash. Belt filter cake is conveyed to an indirectly heated paddle-type dryer, from which the dried product is discharged to the product surge bin. The uranium peroxide product is packaged in 210-L-capacity drums in a semi-automatic packaging capsule and either stored or loaded and trucked off site. Both the dryer and the packaging capsule are vented through high energy impact scrubbers which use fresh water for scrubbing.

7.2.3.4. Environmental controls

Tailing neutralization and disposal

The tailing neutralization circuit processes incoming waste streams for combined treatment. These streams include, for example, washed leach residue, raffinate bleed, spent solvent extraction regenerant, barren strip solution, etc. Slaked lime slurry and barium chloride solution are added to precipitate heavy metals and radium. The neutralized tailing slurry is then pumped to the in-pit disposal system. Decant and seepage water from the in-pit disposal system are pumped to the water treatment plant.

Water treatment

Contaminated water pumped from the mines and the in-pit disposal system is treated in stages by acidification and treatment with ferric sulphate followed by addition of barium chloride solution with the pH adjusted to 8 with slaked lime slurry. Solids formed in each stage are immediately separated from solution with lamella clarifiers and returned to the tailing neutralization circuit. Treated

water is filtered to remove suspended solids before the clarified treated water is discharged to holding ponds. Clarified treated water is used as process water in the mill. Excess water, satisfying all government requirements, is discharged to the environment.

7.2.3.5. Process economics

Plant capital cost

The plant design is subject to capital cost estimating procedures.For a feasibility study, the development of the costs will include the following components:

Civil

detailed estimates of quantities are combined with carefully developed unit rates for the various construction tasks, e.g. clearing and grubbing, excavation, backfill, concrete, etc.At the feasibility study level of detail, it is necessary that sources for backfill material be identified and unit costs for concrete be developed or quoted by reputable contractors.

Building and structural steel

costs are based on detailed steel quantity take-offs combined with quoted unit rates from supply and install contractors.

Equipment supply costs

are based on firm quotations from suppliers received in response to detailed technical specifications coupled with proposed commercial terms. Delivery costs must be included in the quotation or developed in detail by the cost engineers for remote projects. It is expected that the quotations used in the estimate will be suitable for the purchase of the required equipment.

Used process equipment is sometimes invoked as a means of reducing capital costs. Experience shows that the direct cost savings are often eradicated by refurbishment costs and startup delays. At the feasibility stage, the inclusion of purchased used equipment in the cost estimates must be supported by a letter of intent and a favourable mechanical inspection report. Used equipment already in the possession of the project owner must be carefully examined for the state of repair and the suitability.

Equipment installation costs

are developed from a combination of installation manhour requirements and the appropriate project crew rate. The crew rate is developed by the estimating team and includes the direct labour cost for a construction crew, the cost of overtime, direct supervision, travel time, subsistence allowance, and contractors profit.

Many will be surprised that the crew rate on most North American projects presently exceeds $50/manhour. Equally surprising is the time that contractors require for what appears to be a simple task. Installation of a small slurry pump, say one with a 50 mm suction line and weighing about 200 kg, requires one manweek of effort. It is tempting to look at the equipment and imagine installing it oneself in a day but the historical record must be relied upon.

Instrumentation

costs can be significant in modern plants ranging as high as $2 000 000 in a large and complex plant. However, in a plant producing 2000 t/a of U valued at about $44 million/a, an extra 1% recovery is worth $440 000 and could result from improved instrumentation. The costs for instrumentation must be estimated from detailed design documents.

Piping

for a detailed feasibility study, the piping specifications must be firm and the routing of all major lines identified and costed. This is particularly true for expensive materials of construction such as titanium and high alloy materials.

Electrical

the source of power, cost of transmission lines, switchgear, and primary transformers must all be identified and costed in detail.Costs for secondary transformers, motor control centres (MCC) will also be detailed in the feasibility study. Costs for final distribution will be based on average cable lengths and unit costs.

Indirect construction costs

includes the costs for working under difficult climatic conditions, establishing and operating construction management offices, crane rental, bonds, permits, construction camp, fly-in/out operation and other similar costs.For the feasibility study, these costs must be estimated in detail.

Engineering, procurement and construction management

these costs are estimated in detail at the feasibility stage. A significant proportion of the engineering costs have, of course, already been expended to arrive at the feasibility stage.

Contingency

money is set aside to cover the cost of items in the original scope, but either missed, or not estimated in detail, or unknown at the time of the estimate. At the feasibility stage this is often included as an allowance equal to 10% of the estimated direct cost of the project. Contingency is not intended to cover scope changes.

The capital cost estimate for the hypothetical plant is summarized below:

Process Plant Area	US$000	US$000
Site preparation		
roads, etc.		2,000
Building		
sorting plant	1,000	
process plant	7,000	
solvent extraction	1,000	
TOTAL	9,000	9,000

Equipment		
sorting	5,000	
grinding	4,300	
leaching	2,500	
CCD	5,200	
SX	2,900	
Gypsum precip	1,100	
Uranium precip	900	
product drying	1,400	
tailing/water treatment	1,000	
reagents	4,000	
laboratory	700	
services and instruments	2,200	
	31,200	31,200
Total direct cost — $1000s		42,200

Note that piping, electrical and indirect construction costs are included in the relevant process sections in the capital cost table. Other indirect costs are treated in subsequent sections.

Plant operating cost

Process operating costs must also be revised:

Operating labour costs

a revised manning document is prepared for the process aspects of the project. The document includes all supervision, operating and maintenance personnel and is based on required staffing levels for other, similar plants. Labour rates must be confirmed based on local union contracts, published salary surveys, and other sources.

Process supply costs

the requirement for reagents and other supplies must be reviewed based on the latest test data and relevant data from other, similar operations. Delivered costs for reagents are obtained from suppliers as firm quotations. In the case of remote projects, costs might be obtained for delivery to a marshalling point. The engineering company would then develop costs to transport to the project site.

Power costs

process power consumption is obtained by a detailed analysis of the electrical loads. The unit cost of power is usually obtained as a detailed quotation from a supply utility. At remote sites, power will be generated in hydro-electric plant or diesel-electric power plant. In either event costs must be developed in detail.

Other operating cost components must be developed in detail for the feasibility study. In all cost areas, the unit costs must be obtained as written quotations from reputable suppliers and in response to written enquiries.

Using the procedures outlined above, the following data were obtained for the hypothetical processing plant (including the radiometric sorter).

PROCESS OPERATING COSTS — $1000/a

ITEM	1	2	3	4	5	6	7	8	9	10	11	12
Mill labour cost	4.704	4.704	4.704	4.704	4.704	4.704	4.704	4.704	4.704	4.704	4.704	4.704
In-pit labour cost	906	906	906	906	868	680	680	680	680	680	680	680
Labour overhead	1.122	1.122	1.122	1.122	1.114	1.077	1.077	1.077	1.077	1.077	1.077	1.077
Milling supplies	15.386	15.386	15.386	15.386	13.821	11.062	11.062	11.062	11.062	11.062	11.062	11.062
Maintenance supplies	0	0	0	0	0	0	0	0	0	0	0	0
Power	0	0	0	0	0	0	0	0	0	0	0	0
Total operating costs $1000s/a	22.118	22.118	22.118	22.118	20.507	17.523	17.523	17.523	17.23	17.23	17.23	17.423
Unit operating cost ($/LB U_3O_8)	8.77	8.77	8.77	8.77	8.24	7.24	7.24	7.24	7.24	7.24	7.24	7.24

7.2.4. Services, general and administrative

Capital and operating costs are incurred for items that are not directly required for mining and processing ore. Examples of capital cost items include the administration offices, shops, warehouses, security facilities required for the operation.

Another series of costs are the indirect capital costs. These include the costs for the design of the plant, the indirect costs of constructing the facilities and the costs of performing environmental impact assessments. These costs can be significant and should be estimated carefully.

Operating costs are incurred for the operation of these facilities and to take care of project management, personnel officers, security staff and other support people.

For our hypothetical uranium operation, the following costs are estimated:

SERVICES, GENERAL AND ADMINISTRATION CAPITAL COSTS

	US$1000s
Engineering services	4,000
General and administration facilities	1,600
Environmental and permitting	5,000
Total — US$1000s	**10,600**

The ongoing operating costs required to support the project are estimated to be US$5 million per year of production. During the pre-production period, when the mines are being developed but the mill is not operational, it is estimated that costs in this category are US$2.5 million/a.

7.2.5. Contingency

It is normal to add a contingency allowance to the estimates of both capital and operating cost. The contingency allowances are intended to cover the costs of those items that are required for the construction and operation of the project but which have not been specifically estimated. At the feasibility stage, the contingency is often taken as 10%.

7.2.6. Project schedule

The pre-feasibility schedule must be critically reviewed and revised during the feasibility study. The development schedule and the evaluation results for a project are intimately tied together. If the evaluation method recognizes the time value of money, and most do, then the length of time before revenue begins is critical to the final value of the project. This is particularly true for delays in startup.

Schedule Analysis

All too often a feasibility study assumes a mine and mill will operate at 100% capacity from day one. This can give some very desirable results to the DCFROR and Net Present Value, but no one will accept this as a likely production forecast. Typically, the simpler uranium mills will reach full capacity in the first year. Mining operations, especially underground, often take longer to catch up to the mill. For simpler milling operations, it is common to assume 75% to 90% of full production in year one and 100% of full capacity thereafter. The operating costs during this period must be treated with care. Labour costs are usually fixed on a time basis and therefore higher on a per tonne basis

during the initial period. The costs for reagents and other tonnage dependent items might be higher than normal during the initial period as the plant operators become experienced.

Plants using very complex or new technology, and operations in very remote locations can take considerably longer to attain full throughput and recovery, and stable operating costs. The production and cost schedules must be critically developed.

By the time a project starts production, most of the capital has been expended and the project is experiencing its greatest exposure to debt, interest and uncertainty. If full production is delayed, the full cost of financing is extended. If the delay is prolonged (one to two years to reach full production) the impact can be several millions of dollars in present value and lost revenue. The cost to correct startup delays is usually worth it.

Seasonal Factors

It can be costly to attempt to undertake construction during extremely cold, or wet, or windy seasons. A common objective is to have the mill and other buildings closed in to permit continued construction of the process facilities during such periods. If work must take place during inclement weather because of logistical constraints, then appropriate cost penalties must be included in the capital cost estimates.

The preparation of a meaningful schedule, and its execution, for a remote project with seasonal transportation constraints offers special challenges. Narrow shipping windows must be met or a project delay of a year might result.

7.2.7. Financial analysis

CAPITAL COST	?	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	TOTAL
Mine		0 80	0 80	0 20	-	-	4 31	3 99	5 80	4 81	3 91	7 23	2 00	1 02	-	-	-	
Stock-crush		0 50	1 00	1 50	-	-	-	-	-	-	-	-	-	-	-	-	-	
ROS		-	-	6 00	-	-	-	-	-	-	-	-	-	-	-	-	-	
Mill		-	17 00	20 00	-	-	-	-	-	-	-	-	-	-	-	-	-	
Services		1 50	1 50	-	-	-	1 00	-	-	-	-	-	-	-	-	-	-	
G&A	0 60	1 00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
EIS	5 00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Previous	10 00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
SUBTOTAL	15 60	3 80	20 30	27 70	-	-	5 31	3 99	5 80	4 81	3 91	7 23	2 00	1 02	-	-	-	
Contingency	1 56	0 38	2 03	2 77	-	-	0 53	0 40	0 58	0 48	0 39	0 73	0 20	0 10	-	-	-	
TOTAL	17 16	4.18	22 33	30.47	-	-	5.84	4.39	6 38	5 29	4 30	7.96	2 20	1.12	-	-	-	

PRODUCT COST	?	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	TOTAL
Mine	-	5 40	4 22	3 90	5 60	6 08	5 71	9 64	10 00	7 71	9 68	11 86	12 02	12 02	10 72	10 72	-	
ROS-Mill	-	-	-	-	21 20	21 20	21 20	21 20	19 60	16 80	16 80	16 80	16 80	16 80	16 80	16 80	-	
Services	-	0 50	0 50	0 50	1 00	1 00	1 00	1 00	1 00	1 00	1 00	1 00	1 00	1 00	1 00	1 80	1 40	
G&A	-	2 00	2 00	2 00	4 00	4 00	4 00	4 00	4 00	4 00	4 00	4 00	4 00	4 00	4 00	4 00	1 00	
Taxes	-	0 20	0 20	0 20	0 40	0 40	0 40	0 40	0 40	0 40	0 40	0 40	0 40	0 40	0 40	0 40	-	
Royalty	-	-	-	-	0 60	0 60	0 60	0 60	0 60	0 60	0 60	0 60	0 60	0 60	0 60	0 60	-	
SUBTOTAL	-	8 10	6 92	6 60	32 80	33 28	32 91	35 84	35 60	30 51	32 48	34 66	34 82	34 82	33 52	33 52	2 40	
Contingency	-	0 81	0 69	0 66	3 28	3 33	3 30	3 69	3 56	3 05	3 25	3 47	3 48	3 48	3 55	3 35	0 24	
TOTAL	-	8.91	6.61	7.26	36.08	36.61	36.21	40.53	39.16	33.56	35.73	38.13	38.30	38.30	36 87	36.87	2.64	
Reclaim	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15 00	
Dismantle	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	10 00	
Salvage	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(10 00)	
TOTAL COST	17.16	13.09	28.94	37.73	36.08	36.61	42.05	44.92	45.54	38 85	40.03	46.09	40 50	39.42	36.87	36.87	17.64	
Work capacity	-	2 00	2 00	2 00	3 00	-	-	-	-	-	-	-	-	-	-	-	(9 00)	
TOTAL CASH	17.16	15.09	30.94	39.73	39.08	36.61	42 05	44.92	45.54	38 85	40.03	46.09	40.50	39.42	36.87	36 87	8.64	

7.3. FURTHER CASH FLOW AND SENSITIVITY ANALYSIS

The team evaluating this hypothetical deposit has researched every detail that could possibly impact the economics of this potential deposit. Reviewing the major steps that have been researched would serve very useful to insure that nothing has been forgotten by the team.

- The geometry and tonnage of the deposit has been completely drilled and fully delineated.

- The grade of the material is proven at 0.35 percent U_3O_8.

- The mine has been designed to its optimal tonnage and striping ratio.

- The initial capital and re-investments estimates/requirements have been fully estimated to within ±5%.

- All investment and leasing laws have been investigated, as to their potential impact.

- All the operating costs have been fully detailed through the direct and indirect components.

- All mine and mill recoveries have been fully defined.

- The terms for the acquisition of all the potentially disturbed acreage has been finalized and signed.

- All financing has been settled and agreed upon as to the terms for the length of the loan, the amount of the loan and the interest rate.

- All the rules and regulations pertaining to the regulatory regime of the host country have been researched and documented and all costs as a result of these rules and regulations have been incorporated into the engineering and economic analysis.

- The tax and financial regimes of the host country has been fully investigated and incorporated into the economics of the deposit. This included the tax structures and rates, and all the deductible items, such as, depletion, tax loss carry options, depreciation, interest expenses, royalties, etc.

- Potential markets for the product have been identified and appropriate contacts made.

- Numerous sensitivity analyses have been performed to ensure that all possible flaws or problems with this project have been accounted for.

Since numerous sensitivity analyses have been performed during the Pre-Evaluation and the Pre-Feasibility stages, one could reiterate that process in this stage, and to even more, excruciating detail. But for this guidebook, it was decided to perform a very simplistic mine on wheels type of analysis, where hypothetical deposits were used and move it between a few different tax regimes to illustrate the possible impact upon the economics of the deposit. The tax regimes will be actual (except for the hypothetical operation), but for example purposes, these actual tax regimes will be referred to as Countries A, B and C.

For the hypothetical base case with a royalty of one percent and a national tax levied at 40%, the price needed to attain a 20% DCFROR was determined to be approximately $15/lb U_3O_8. The tax regimes of the three other countries are shown below.

COUNTRY A

The tax structure for (Country A) is made up of the following information:

- *corporate tax*
  based on industrial profits resulting from the aggregate of business activities.

	REVENUES
less:	other taxes paid
	amortization/depreciation
	interest payments
	reserves
	losses
	rent
equals:	profit before tax
less:	tax at 35%
equals:	profit after tax

- *property tax*
  the rate is 2.5% of taxable value of physical plant (not mobile equipment) which, in this case, is based on 50% of the original value.

COUNTRY B

The tax structure for (Country B) is made up of the following information:

- *corporate tax*
  taxable income is based on net profits plus adjustments.

	REVENUES
less:	depletion
	interest payments
	depreciation
	royalties
	tax loss carry (5 years forward)
	certain taxes
equals:	profit before tax
less:	tax at 42%
equals:	income after tax
less:	tax on after tax income at 7.5%
equals:	profit after tax

- *depreciation*
  calculated on the straight-line and/or declining balance basis, depending on the type of asset in question. For purposes of this analysis, straight-line will be used for all plant and permanent type structures and declining balance for mobile type equipment.

- *depletion allowance*
  15% of the sales product, but limited by 50% of the profit before tax.

- *property tax*
  the rate is 3.0% of taxable value of physical plant (not mobile equipment) which, in this case, is based on 50% of the original value.

The tax structure for (Country C) is made up of the following information:

· *corporate tax*
taxable income is based on net profits plus adjustments.

	Revenues
less:	interest payments
	depreciation
	certain taxes
equals:	profit before tax
less:	tax at 30%
equals:	income after tax
less:	royalty of 2.5%
equals:	profit after tax and royalty

· *depreciation*
calculated on the straight-line basis.

· *royalty*
sometimes called a profit tax, based on 2.5% of the net value of sales of the mineral or metal.

Given these three country's different tax structures, it is illustrated in Fig. 20 how the price necessary to attain a 20% DCFROR would differ. Clearly, the tax regime in Country A is the most detrimental to the economics of this deposit. Total taxes paid under each of these taxes regimes was approximately $15 million for the hypothetical operation and $53 million, $32 million, and $39 million for Countries A, B, and C respectively. How these taxes are calculated in the cashflow for each country is the major determining factor on the necessary price determined.
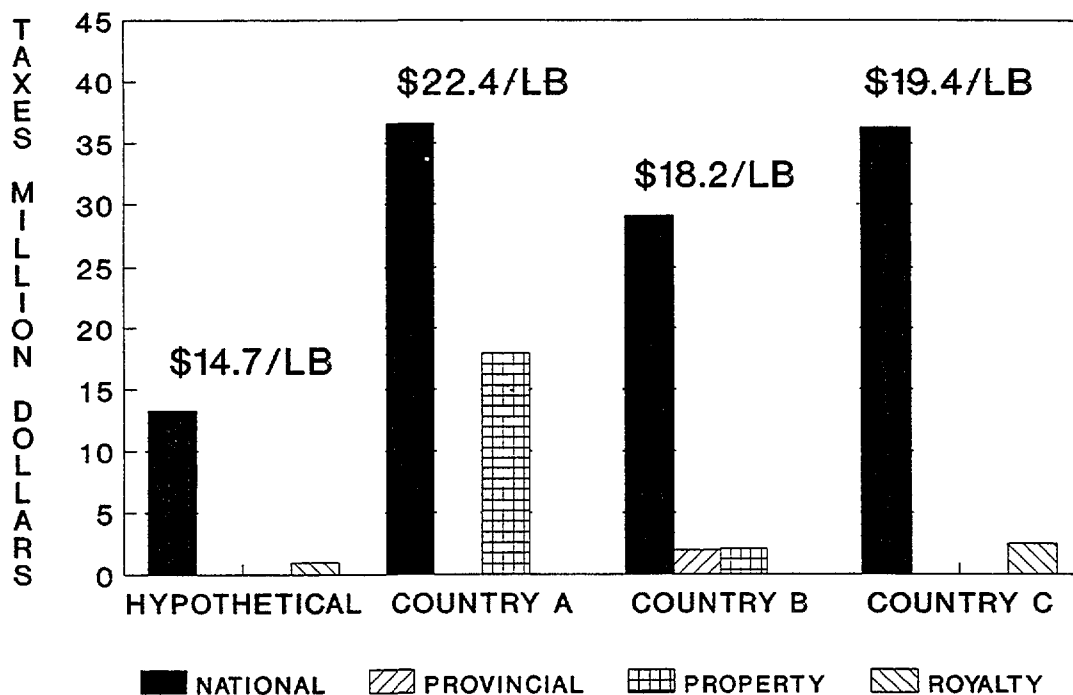

FIG. 20. Relation between Discounted-Cash-Flow-Rate-of-Return (DCFROR) and 3 different tax regimes.

Appendix I

OPEN PIT AND UNDERGROUND MINE CAPITAL COSTS

Open pit mine capital costs

Stripping ratios are characteristically high for open pit uranium mines. Ratios may vary from 10:1 to as high as 60:1 depending on the grade of the ore. Typical equipment requirements and costs for a scraper mining method at 1800 metric tonnes per day of ore with a stripping ratio of 20:1 are presented below.

TABLE A-I.I. TYPICAL MINE EQUIPMENT LIST AND COSTS FOR AN 1800 MTPD ORE PRODUCTION RATE WITH A STRIPPING RATIO OF 20:1

ITEM [1]	Quantity	Cost US$ [2]
Tractors-track, 370–770 hp	5	3 500 000
Motor Grader, 275 hp	2	680 000
Scrapers, 44 cu. yd.	9	6 160 000
Loaders, 7 cu. yd.	2	900 000
Trucks, 35 st	2	750 000
Fuel/lube Trucks	2	190 000
Water Trucks	2	580 000
Service Trucks	5	270 000
Pickup Trucks	9	150 000
Miscellaneous Equipment		200 000
Subtotal		13 380 000
Contingency @ 10%		1 340 000
TOTAL		**14 720 000**

[1]Item costs include labour, equipment, and supplies.
[2]Costs on average 1990 US dollars.

Surface facilities capital costs will vary considerably, primarily due to site location. Topography, vegetation, and climate all affect the costs. Infrastructure is a category that can only be adequately addressed on a site specific basis. An example of general mine plant facilities is presented in Table A-I.II.

TABLE A-I.II. TYPICAL MINE PLANT FACILITIES AND COSTS FOR AN OPEN PIT MINE AT 1800 MT PER DAY OF ORE

ITEM [1]	COST US$ [2]
Office Buildings/Change Room	3 120 000
Maintenance Shops/Warehouses	1 680 000
Access Roads/Parking	170 000
General Surface Facilities	880 000
Power Substation	640 000
Fire House/Ambulance	90 000
Powder Magazine/Security	110 000
Miscellaneous Facilities/Equipment	280 000
Subtotal	6 970 000
Contingency @ 10%	700 000
TOTAL	**7 670 000**

[1]Item costs include labor, equipment, and supplies.
[2]Costs on average 1990 US dollars.

Underground mine capital costs

Mine equipment costs will vary with type of mining method. Rubber tired and track equipment are both used for mucking and hauling ore, and slushers may also be used. Ventilation and mine dewatering are also variable depending on mine plan and underground conditions. These cost items will typically be 15 to 20% of the total capital cost. A summary of the major capital cost items is presented in Table A-I.III.

TABLE A-I.III. TYPICAL MINE CAPITAL COSTS FOR AN UNDERGROUND MINE 300 METRES DEEP WITH A PRODUCTION RATE OF 1000 MT PER DAY OF ORE

ITEM [1]	COST US$ [2]
Office Buildings/Change Room	2 280 000
Maintenance Shops/Warehouses	1 970 000
Shafts (2) and Headframe	7 740 000
Mine Equipment	5 400 000
Underground Development	5 900 000
Access Roads/Parking	160 000
General Surface Facilities	320 000
Power Substation	1 190 000
Fire House/Ambulance	90 000
Powder Magazine/Security	150 000
Ventilation/Heat/Miscellaneous	870 000
Subtotal	26 070 000
Contingency @ 10%	2 610 000
TOTAL	**28 680 000**

[1]Item costs include labor, equipment, and supplies
[2]Costs on average 1990 US dollars.

TABLE A-I.IV. MILL CAPITAL COST ESTIMATES

| ITEMS | Capacity (tonnes per day) | | | | | |
	200 (181.6)* US$	400 (361.1)U S$	500 (453.9) US$	1000 (907.8) US$	2000 (1 815.6) US$	%
Construction — Labor	1 456	2 257	2 129	3597	4 620	16.95
Construction — Supplies	2 291	3 551	3 350	5 660	7 270	26.67
Equipment — Purchase	3 391	5 257	4 959	8 378	10 762	39.48
Transportation	64	99	93	157	202	0.74
Total Labor and Materials	7 202	11 163	10 531	17 792	22 854	83.84
Miscellaneous Cost **	1 388	2 152	2 030	3 429	4 405	16.16
Total Capital Cost	8 590	13 315	12 561	21 221	27 259	100.00
High	11 932	19 257	22 468	36 270	58 551	
Low	5 975	9 727	11 380	18530	30 171	

*Numbers in parenthesis are metric tonne values.
**Miscellaneous cost includes cost items such as working capital, engineering and contractor's/consultant's fees, recruiting and startup.

TABLE A-I.V. MILL OPERATING COST ESTIMATES
Capital Cost in $000

| ITEMS | Production (tonnes per day) | | | | | |
	200 (181.6)* US$	400 (361.1) US$	500 (453.9) US$	1000 (907.8) US$	2000 (1,815.6) US$	%
Construction — Labor	1 456	2 257	2 129	3 597	4 620	16.95
Construction — Supplies	2 291	3 551	3 350	5 660	7270	26.67
Equipment — Purchase	3 391	5 257	4 959	8 378	10762	39.48
Transportation	64	99	93	157	202	0.74
Total Labor and Materials	7 202	11 163	10 531	17 792	22 854	83.84
Miscellaneous Cost**	1 388	2 152	2 030	3 429	4 405	16.16
Total Capital Cost	8 590	13 315	12 561	21 221	27 259	100.00
High	11 932	19 257	22 468	36 270	58 551	
Low	5 975	9 727	11 380	18 530	30 171	

*Numbers in parenthesis are metric tonne values.
**Miscellaneous cost includes cost items such as working capital, engineering and contractor's/consultant's fees, recruiting and startup.

TABLE A-I.VI. IN SITU LEACHING CAPITAL COST ESTIMATES
Capital Cost in $000

| ITEMS | Production (1000 lbs U$_3$O$_8$ per year) | | | |
	200 (90.7)* US$	500 (226.8) US$	750 (340.2) US$	%
Purchase — Process Equipment	1 035	1 888	5 647	7.77
Installation and Site Improvement	2 551	4 656	13 924	19.16
Plant Construction	804	1 468	4 389	6.04
Initial Wellfield Cost	2 557	4 666	13 953	19.20
Pilot Plant Cost	1 585	2 892	8 648	11.90
Restoration System Cost	3 453	6 302	18 844	25.93
Engin./Project Managements Cost	1 331	2 430	7 267	10.00
Total Fixed Capital Cost	13 316	24 302	72 673	100.00
Deferred Capital Cost	559	1 021	3 052	4.20**
Contingency Cost Cost	1 332	2 430	7 267	10.00**
Total Capital Cost	15 207	27 753	82 993	

*Numbers in parenthesis are kilogram values.
**Deferred capital cost   =   4.2% of total fixed capital cost.
**Contingency cost   =   10% of total fixed capital cost.

TABLE A-I.VI-A. IN-SITU LEACHING CAPITAL COST ESTIMATES
Initial Wellfield Cost Breakdown

ITEMS	% of Total Wellfield Cost	% of Total Capital Cost
Surface Preparation	5.90	1.13
Injection Wells	26.83	5.15
Production Wells	33.43	6.42
Monitor Wells	6.01	1.15
Injection Well Equipment	2.21	0.42
Production Well Equipment	18.87	3.62
Monitor Well Equipment	1.09	0.21
General Well Equipment	5.66	1.09
TOTAL	**100.00**	**19.20**

Note: Percentages may not add to totals shown, because of computer rounding

TABLE A-I. VII. IN SITU LEACHING CAPITAL COST ESTIMATES
Operating Cost in $/lb U_3O_8

ITEMS		Production (1000 lbs U_3O_8 per year)			
		200 (90.7)[*] US$	500 (226.8) US$	750 (340.2) US$	%
Wellfield Cost		8.76	7.45	7.00	41.32
Manpower Cost Improvement		3.09	2.63	2.47	14.56
Chemicals (reagents)		2.59	2.20	2.07	12.22
Utility Cost		3.03	2.57	2.42	14.26
Operation and Maintenance Supply		3.43	2.92	2.74	16.17
Makeup Water Cost		0.31	0.27	0.25	1.47
Total Direct Cost		21.19	18.03	16.94	100.00
G & A Cost		1.06	0.90	0.85	5.00[**]
Total Operating Cost	$/lb U_3O_8	22.25	18.93	17.79	
	$/kg U_3O_8	49.05	41.74	39.21	

[*]Numbers in parenthesis are kilogram values.
[**]G & A Cost = 5% of total direct cost.
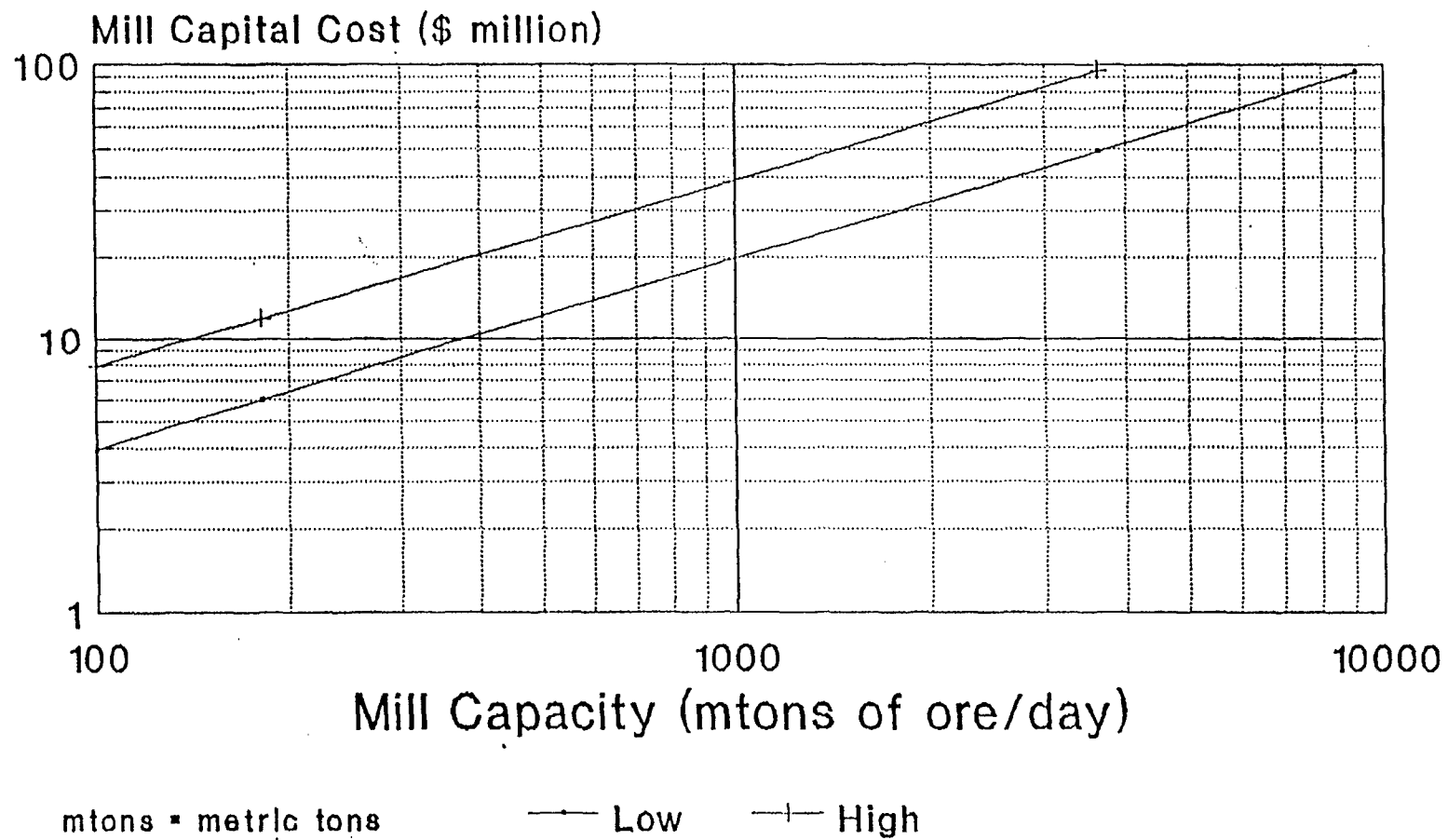
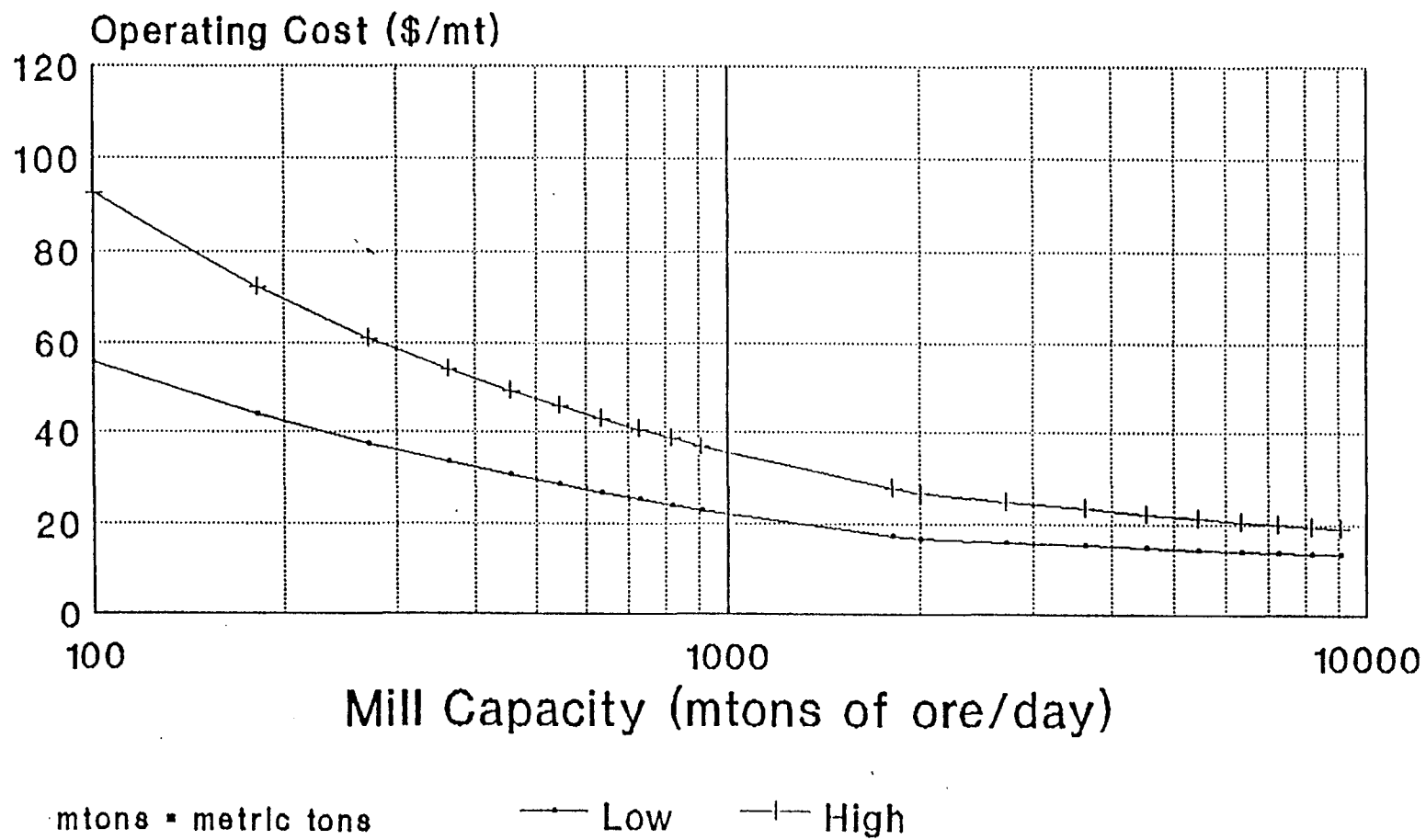Figure A-I.1. Uranium milling; mill capacity vs. mill capital cost (mill capital cost).

Operating Cost ($/mt)

Mill Capacity (mtons of ore/day)

mtons = metric tons   —— Low   —⊢— High

Figure A-I.2. Uranium milling; mill capacity vs. mill capital cost (operating cost).
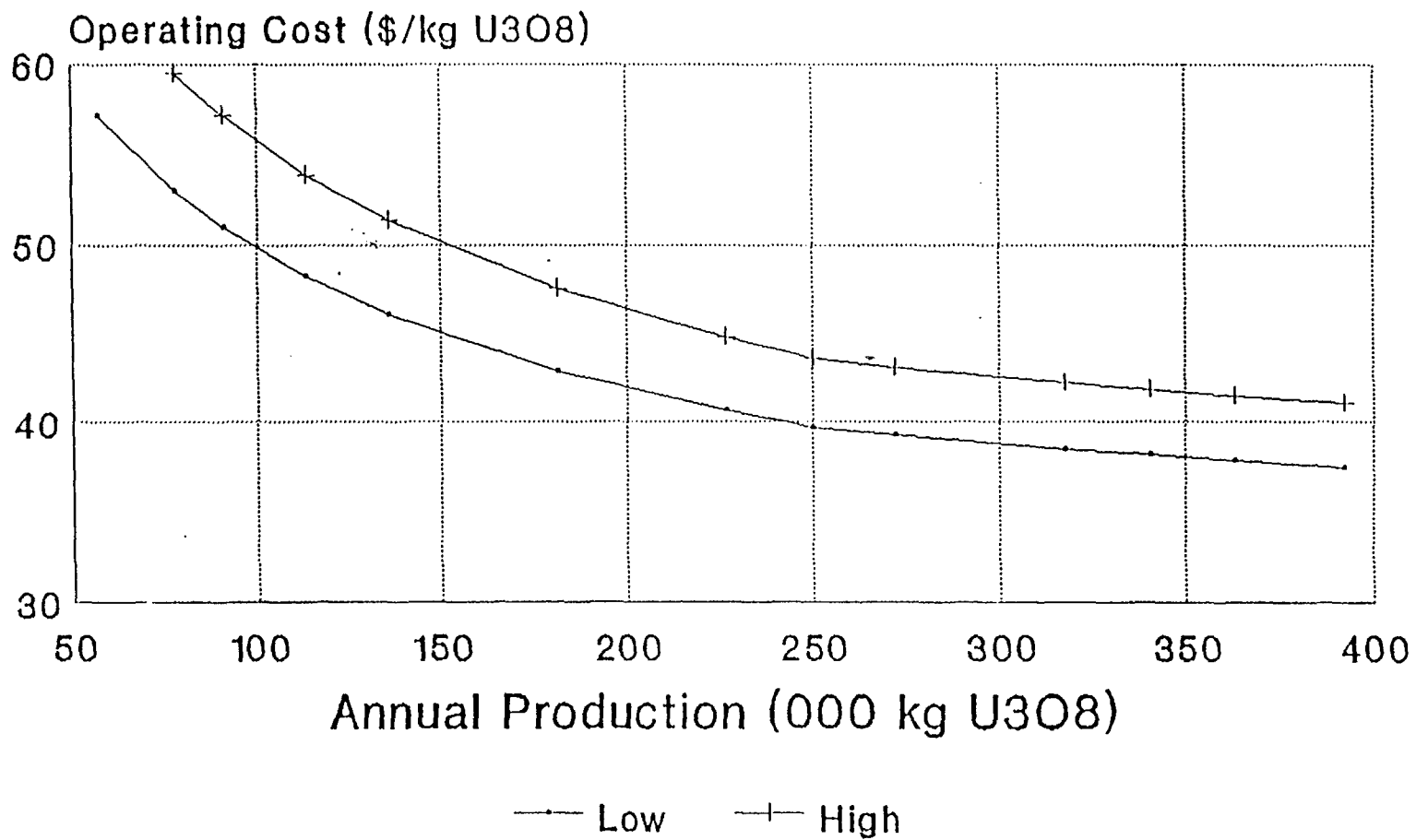
Figure A-I.3. Uranium in situ mining; annual production vs. operating cost.

DEFINITION OF VARIOUS CASH FLOW ITEMS

ADJ–   If losses from previous years can be brought forward or carried back (dependent upon the law of the country in question), sometimes the taxes that were calculated need to be adjusted.

CF –   The real remaining after-tax monies.

DEPL -   Depletion is another book item (like depreciation) that was designed and developed to recoup investment in exploration of minerals through tax savings. The owner of an economic interest in mineral deposits, oil and gas wells, or standing timber may recover his cost through federal tax deductions for depletion over the economic life of the property. A person has an economic interest if through investment he has: (1) acquired any interest in minerals in place or standing timber, and (2) received income by any form of legal relationship from extraction of the minerals for which one must look for a return of capital. There are two types of allowable depletion methods. They are cost depletion and percentage depletion:

Cost depletion = ((adjusted basis) × (mineral units removed or sold during the year)/(remaining mineral units at the beginning of the year)). (Adjusted basis is the cost basis reduced by the cumulative depletion paid).

Percentage depletion = ((a rate (based on specific commodities) × (the gross revenues received from a particular commodity, less royalty payments and post smelter charges)). This depletion is limited by 50% of before tax income (BTINC).

Cost depletion example

A mineral property with estimated reserves of 1 000 000 metric tonnes of ore is purchased for $500 000. Two hundred thousand metric tonnes of ore are mined each year with an annual operating cost of $60 000. Depreciation and amortization deductions total $150 000 this year. The ore is sold for $9.00 per tonne after royalties.

Income (200 000 MT) ($9.00)	=	$1 800 000
- op cost		- 60 000
- depreciation & amortization		- 150 000
Before tax income (BTINC)	=	$1 590 000
- Cost depletion = 500,000 × 200,000/1,000,000		- 100 000
Taxable income using cost depletion	=	$1 490 000
- 2nd year cost depletion:		
(500 000 - 100 000) × (200 000/800 000)	=	100 000

Percentage depletion example

A mining operation yields annual sales revenue of $1 500 000 from a mixed lead, zinc, and silver ore. One million dollars of the revenue is from the lead and zinc, and $500 000 from the silver. Operating costs are $700 000 and allowable depreciation is $100 000.

Sales revenue	$1 500 000
- op cost	- 700 000
- depreciation	- 100 000
Before tax income (BTINC)	700 000

	50% limit on BTINC for % depletion	350 000

Pb, Zn % depletion = 1 000 000 × (.22)   220 000
Ag % depletion = 500 000 × (.15)   75 000
Total depletion   295 000

Therefore, since 295 000 < the 350 000 limit
Taxable income = 700 000 - 295 000 = $ 405 000

DEPR–   This is what is considered a book item for tax purposes. It is a financial means to recoup investment through tax savings. A tax deduction allowed for a reasonable allowance for the exhaustion, wear and tear of property out of its use or employment in a business. Three of the most common methods of depreciation are:

1. *Straight line* — This method has uniform deductions over the life of the asset. It is calculated by dividing the investment by the number of years of depreciation allowed.

2. *Double declining balance* — This is an accelerated form of depreciation, where the investment is multiplied by the ratio of 2 over the life of the asset, and then continued each year by multiplying the ratio against a declining or remaining balance.

3. *Sum of the years digits* — A method where the investment is multiplied by the ratio of the life of the asset divided by the sum of the years.

An example problem will help to illustrate how each type of depreciation is calculated. Assume some chemical process equipment is used as auxiliary equipment in the processing of Uranium concentrates and that it has a cost of $200 000 with an estimated life of 5-years and zero salvage value (that is no resale value).

(a) straight line: $200 000/5 years = $40 000/yr.

(b) double declining balance:
1st year	— 2/5 ($200 000)	=	$80 000
2nd year	— 2/5 ($120 000)	=	$48 000
3rd year	— 2/5 ($ 72 000)	=	$28 800
4th year	— 2/5 ($ 43 200)	=	$17 280
5th year	— 2/5 ($ 25 920)	=	$10 368

(c) sum of the year digits:
1st year	— 5/15 ($200 000)	=	$66 666
2nd year	— 4/15 ($200 000)	=	$53 333
3rd year	— 3/15 ($200 000)	=	$40 000
4th year	— 2/15 ($200 000)	=	$26 667
5th year	— 1/15 ($200 000)	=	$13 333

DEVL–   For cash flow and tax purposes, the development expenditure can be accounted for in a number of ways.

EQI–   Equity investments are those amounts spent on capital outlays.

EXPL–   The expending of exploration can be accounted for in a number of ways for cash flow and tax purposes.

FTAX -   Those taxes charged at a federal or national level. Those taxes charged at a federal or national level. Usually the last taxes levied in the cash flow analysis. They can be structured as a Federal tax rate × gross revenues less various deductions.

INFRA -   The cost usually on a per tonne of ore basis to cover the operating of infrastructure facilities, such as ports, housing, medical facilities and airports. The operating cost for infrastructure is usually defined as ((the cost per tonne of ore to operate all the site infrastructure) × (the tonnes of ore mined)).

INT–   Annual interest payments that are from any loans that are usually directly related to the mineral operation. If funds are needed to be borrowed to develop and or operate a mineral operation, a loan must be acquired through various sources. Loans may be structured in numerous ways so as to enhance both the economics of the mineral operation and to minimize the risk (as much as possible) of the lender. Three common types of loans (with minor variations, of course, are:

1.   *Amortized payments* — where an amount is borrowed at a specific interest rate over a specified term (period of time). The principal and interest payments are uniform over the duration of the loan.

2.   *Fixed w/balloon* — another type of loan is where the lender will set up fixed interest only payments for a period of time and then have one final payment (balloon) at the end. No equity or principal is paid out except in the final balloon payment.

3.   *Special structure* — Structure a loan either by the lender and or the borrower (or hopefully in unison) specific to the needs of the operation. The loan can be tailored by the amounts of each of the payments and the number of payments.

Any one of these three types of loans can be structured with a front-end grace period. This allows the mineral operation to forego any payments for a pre-specified period of time and then the original principal amount borrowed is usually increased by the amount of interest foregone.

LEACH–   The cost on a per tonne basis of product, to leach various commodities. The cost to leach material is based on the product out. Therefore, ((the leach op cost) × (the product out)) is the equation used in this case.

MILL –   The cost on a per tonne basis to beneficiate the ore. The mill op cost is derived by multiplying the ((tonnes of ore) × (the cost per tonne to mill the ore)).

MINE –   The cost on a per tonne basis to mine usually just the ore but sometimes the ore and waste. The mine op cost is derived by ((tonnes of ore) × (the cost per tonne)).

OHEAD -   Those costs on a per tonne basis related to administrative things (corporate headquarters, etc.). Every mineral operation has some sort of administrative overhead. It is usually accounted for by multiplying the ((tonnes of ore mined) × (the overhead cost per tonne)).

POST–   The cost on a per tonne of input basis to smelt/refine the material. *Smelter operating cost (POST SOC)* — The cost to smelt a material is calculated as: ((tonnes of ore) × (feed grade) × (mill recovery) - (mill concentrate grade) × (the cost per tonne to smelt)). *Refinery operating cost (POST ROC)* — The cost to refine a material is:

((tonnes of ore) × (feed grade) × (mill recovery) × (smelter recovery) - (smelter concentrate grade) × (the cost per tonne to refine the material)).

PTAX–   Property taxes are assessed in various manners, and usually paid to local governments. Property taxes can be assessed against investments and/or against revenues. Like personal property, property taxes on mining investments can have both mill levy and assessment rates. Property taxes are usually assessed before any income taxes are calculated.

REVENUES –   Those monies received for the sale of the mineral product. This may be calculated dependent upon the stage at which the product is being marketed. That is as ore, concentrate, or refined product. Each being a higher grade product and therefore having a greater value. Revenues are calculated as: ((tonnes of ore) × (feed grade of material) × (all recoveries) × (a payfor, this is a percentage that is received) × (the price per unit of the commodity) - (any concentrate grade)) (if in fact the product is a concentrate).

ROY –   Royalties are those monies paid to private land and or minerals rights holders or to various levels of government. Royalties can be applied in various different methods. Royalty payments are usually handled as before tax deductions and they are also deducted from gross revenues before calculating the depletion allowance (to be discussed later). Royalties can be applied in various different methods. A list of some of those methods:

    1.   *Based on value after mining* — ((the royalty rate) × (gross revenues for a particular commodity - smelter and refiner operating costs - transportation from the mill - a portion of the total mill operating cost assigned to that commodity)).

    2.   *Net smelter return* — ((royalty rate) × (gross revenues for a particular commodity - smelter and refiner operating costs - the cost of transporting from the refiner)).

    3.   *Based on units of ore* — ((royalty rate) × (the units mined annually)).

    4.   *Based on units recovered* — ((royalty rate) × (units recovered annually)).

    5.   *Based on revenues less specified deductions* — ((A royalty rate) × (gross revenues less various commodity specific operating costs)).

    Many (if not all) of the rates and the bases that the rates are placed against can be set up in a sliding scale format.

SEVTAX -   Severance taxes are levied in various ways on the premise that minerals (for the most part) are non-renewal resources, so when they are severed from the earth, there is no replacing them. Some of the more common methods of severance tax calculations are:

    1.   *Based on value after mining* — ((the severance tax rate) × (gross revenues for a particular commodity - smelter and refiner operating costs - transportation from the mill - a portion of the total mill operating cost assigned to that commodity)).

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2.   *Net smelter return* — ((severance tax rate) × (gross revenues for a particular commodity - smelter and refiner operating costs - the cost of transporting to the refiner)).

3.   *Based on value after smelting* — ((severance tax rate) × (gross revenues of a particular commodity - refiner op cost - the transportation cost to the refiner)).

4.   *Based on value after refining* — ((severance tax rate) × (gross revenues of a particular commodity - the transportation cost to the refiner)).

5.   Based on units of ore — ((severance tax rate) × (the units mined annually)).

6.   *Based on units of concentrate* — ((severance tax rate) × (the units milled annually)).

7.   *Based on units recovered* — ((severance tax rate) × (units recovered annually)).

8.   *Based on revenues less specified deductions* — ((A severance tax royalty rate) × (gross revenues less various commodity specific operating costs)).

Many (if not all) of the rates and the bases that these severance tax rates are placed against can be set up in a sliding scale format.

STAX–   State or provincial taxes are usually levied right after most all the above costs have been deducted, but before federal or national taxes have been levied. They can be structured as a state tax rate × gross revenues less various deductions.

TRANS –   The cost usually on a per tonne basis to transport material through various stages of processing. The cost to transport ore from mine to mill is based on the ((tonnes of ore moved) times (the cost per tonne)). The cost to transport the concentrate from the mill to smelter is calculated as: ((tonnes of ore) × (the feed grade) × (the mill recovery) - (the mill concentrate grade) × (the cost per tonne to transport)). The cost to transport material from the smelter to the refinery is: ((tonnes of ore) × (feed grade) × (mill recovery) × (smelter recovery) - (smelter concentrate grade) × (the cost per tonne to transport)). The cost to transport material from the refinery to the market is: (tonnes of ore) × (feed grade) × (mill recovery) × (smelter recovery) × (refinery recovery) × (the cost per tonne to transport)).

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