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Potential uranium supply system based upon computer simulation of sequential exploration and decisions under risk

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POTENTIAL URANIUM SUPPLY SYSTEM BASED UPON COMPUTER SIMULATION OF SEQUENTIAL EXPLORATION AND DECISIONS UNDER RISK

by

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As members of the Final Examination Committee, we certify that we have read the dissertation prepared by Salvador Ortiz-Vertiz

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and recommend that it be accepted as fulfilling the dissertation requirement for the Degree of Doctor of Philosophy

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Final approval and acceptance of this dissertation is contingent upon the candidate's submission of the final copy of the dissertation to the Graduate College.

I hereby certify that I have read this dissertation prepared under my direction and recommend that it be accepted as fulfilling the dissertation requirement.

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SIGNED:

For my wife, Lucila,
unflagging companion,
and my children,
Mariana and Salvador.

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TABLE OF CONTENTS

TABLE OF CONTENTS--*Continued*

TABLE OF CONTENTS--Continued

TABLE OF CONTENTS--Continued

TABLE OF CONTENTS--Continued

LIST OF ILLUSTRATIONS

LIST OF ILLUSTRATIONS--Continued

LIST OF TABLES

LIST OF TABLES--Continued

ABSTRACT

One of the more elusive problems in the area of mineral resource assessment is the comprehensive description of potential supply from undiscovered deposits, especially for economic conditions outside of historical experience. This dissertation consists of the design of a Monte Carlo simulation system to estimate potential supply of roll-type deposits.

The system takes a given uranium endowment probability distribution and aims at two major and interrelated objectives. First, to design a system that estimates potential supply even when prices are much higher than previous or current prices. Second, to account fully for the cost of discovering and mining the individual mineral deposits contained in a given endowment.

Achievement of these objectives constitutes the major contribution of this study. To accomplish them, the system considers: cost of risk, return on investment, cost of failures during the search process, discovery depletion, and the effect of physical characteristics of the deposits on exploration and mining costs. It also considers that when economic conditions, such as product price, are outside historical experience, existing behavioral rules--exploration drilling density, stopping rules, minimum attractive deposit size and grade, and mining parameters--are irrelevant. Instead, those decision rules are determined internally by the system.

The system design task--particularly the exploration model--is greatly compounded by these objectives. As far as can be determined from models reported in published papers, the exploration model of this dissertation is unique in several ways. It is the most disaggregated sequential model designed to date. It is the first one that models geologic features that are used as exploration guides, and the only one that separates the physical elements of the search process from the economic elements and links the two to decide the optimal activity level at every exploration stage.

Although the system is designed for a specific mineral and mode of occurrence, the system architecture is general and can be used with an exploration model prepared specifically for other minerals.

CHAPTER 1

INTRODUCTION

This dissertation is based on the work performed by the author during 1979-81 on a research project for the University of Arizona, Mineral Economics Research, under contract to the Department of Energy of the United States Government and directed by Prof. DeVerle P. Harris. This study consists of the design of a system to provide potential uranium supply estimates given a probabilistic description of uranium endowment. It is particularly concerned with costing the transformation of endowment consisting of undiscovered uranium deposits to potential uranium supply. Endowment estimates are completely exogenous to the potential supply system of this dissertation. Specifically, the system is designed to accept as an input probability distributions of endowment provided by the Department of Energy (DOE). Thus, given an endowment description, the system simulates exploration for and production of that portion of the endowment that satisfies relevant economic conditions. In this way, the amount of uranium oxide (U_3O_8) that could be found and produced (i.e., potential supply) at various predetermined economic criteria levels (in this study price/cost[1] levels) is estimated.

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1. The combined term "price/cost" is used herein to emphasize that all costs incurred to create new supply, including return on investment, need to be accounted for. The terms "price/cost" and "price" are used interchangeably. Of all economic variables that enter in this system design, only price/cost is manipulated. Thus, potential supply estimates for prices much higher than, say 3 to 4 times previous or current prices, can be obtained. The rest of the economic conditions remain constant. It is assumed that markets are unlimited at any specified U_3O_8 price/cost level.

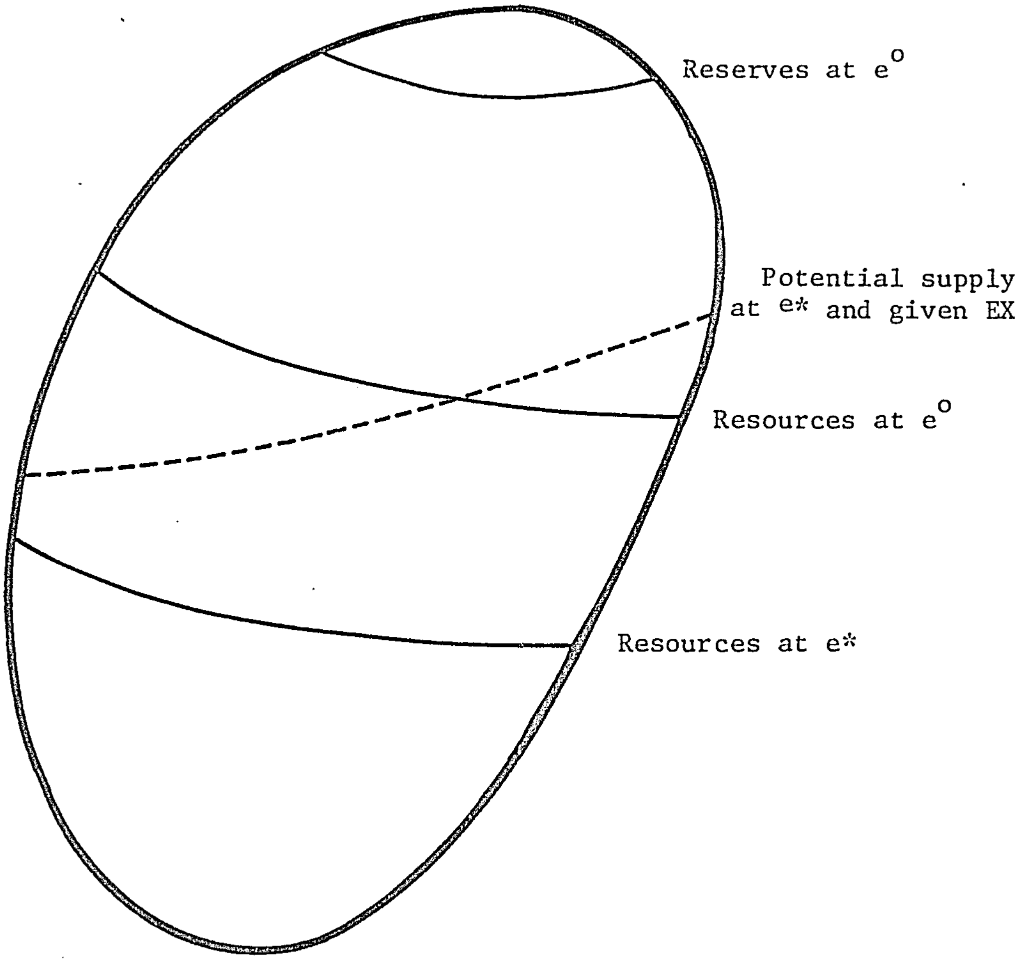
1.1 Resource Nomenclature Used in This Study

To put the potential supply system, the object of this dissertation, in proper perspective, it is necessary to present a clear notion of what this system attempts to estimate. For those not familiar with resource terminology, resource terms and conventions used in this study are defined in this section.

The resource terminology employed in this study is that developed by Harris (1977). Figure 1.1 graphically illustrates this terminology. Suppose that for resource analysis purposes, two economic conditions are stated: current economic conditions (e^o) and more favorable conditions (e^*). Economic conditions (e) could, for example, be different price/cost levels. The following terms can then be defined:

Resource base consists of the amount of an element, in this case uranium, in the totality of occurrences within the earth's crust (roll front, tabular, vein deposits, and even plain rock).

Mineral endowment is a more restrictive term than resource base. It includes those occurrences of U_3O_8 meeting predetermined criteria. These criteria constitute a selected subset of physical and/or chemical characteristics, such as minimum size and grade, maximum depth, modes of occurrence (roll front, tabular, and vein deposits), and chemical form (in this study U_3O_8). Hence, mineral endowment can be viewed as an absolute, a physical measure independent of the economics and technologies under consideration. In this study an occurrence is defined for endowment purposes as a discrete lens or cluster of lenses of uranium



e = economic conditions; e* better than e^o (e.g., improved price/cost level)

e^o = current economic conditions

EX = exploration effort.

Figure 1.1. Resource terminology of this study (after Harris, 1978, p. 8).

bearing rock containing at least 10 tons of U_3O_8 at a cutoff grade of 0.01% U_3O_8 at a maximum depth of 5000 ft. This is the occurrence definition used by DOE because this potential supply system is designed to accept DOE's endowment estimates as an input.

As defined, mineral endowment includes resources. This requires that the predetermined criteria are selected so that all occurrences that meet both the most favorable of the economic conditions considered in the resources analysis during contemplated and the currently feasible or near feasible production technologies are included in the endowment set.

Resources[1] refer to the amount of U_3O_8 in occurrences within the mineral endowment which, if they were already discovered, could be produced economically under specific price/cost conditions (e). Notice here that discovery costs are disregarded. The portion of the endowment that does not meet the economic condition stated is referred to as a "nonresource" with respect to "e."

Potential supply (at stated economic conditions "e") comprises the sum of the amounts of U_3O_8 in the occurrences within the resource

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1. The reader should be aware that the term "resources" has a dual meaning in resource appraisal literature:

a. "Resources," as formally defined above, has an economic connotation; i.e., that portion of the endowment economically producible under the stated price/cost regime.

b. "Resources," as used informally in the literature, is deprived of its economic meaning. In its informal use, the act of estimating endowment (which includes both resources and nonresources) falls in the category of "resource appraisal."

economically produced and discovered, given a specified exploration effort (EX). That is, the net present value of those occurrences should be greater than or equal to zero. In its purest form, the definition of potential supply specifies that the amount of exploration is optimal (EX*) with respect to "e." Note in Figure 1.1 that potential supply is a subset of resources (both, of course, with the same economic condition reference, e*).

Any of the resource categories referred to above are usually employed ad libitum, with a qualifier to reflect the current state of knowledge of the resource.

Reserves refers to the amount of U_3O_8 contained in those occurrences within the economic resources (at current economic conditions, e^o) that are known and economically producible at e^o.

1.2 Background Information on the Department of Energy Uranium Resource Appraisal

Uranium resource estimates by the U.S. government were made as early as 1948 by the former U.S. Atomic Energy Commission (Blanchfield, 1980). In 1974, a comprehensive national uranium resource evaluation (NURE) program to assess United States uranium resources was initiated (then under control of the Energy Research and Development Administration, ERDA) (DOE, 1980). This program subsequently became part of the U.S. Department of Energy. Since the early 1980s, the U.S. Geological Survey is responsible for the estimation of uranium resources. This section examines DOE's terms and conventions as well as its current methodology for estimating potential supply from undiscovered deposits.

1.2.1 DOE's Economic Criteria for Resource Analysis: The Forward Cost Concept

Both reserves and resources are assessed by DOE using forward cost as the economic reference (DOE, 1980).

The term forward cost refers to operating and capital costs not yet incurred when a resource estimate is made. Table 1.1 shows the components of both capital (part A) and operating costs (part B); part C of this table lists costs excluded from the forward cost definition of DOE.

For an operating mine, forward cost is equivalent to the cost of labor and operating expenses for producing the uranium under consideration. Thus, it is a marginal cost (exclusive of the return on investment made to bring that mine into production). It should be clear that forward cost is not intended to be taken as either "full cost" or as selling price. For a discussion of the history behind DOE's forward cost, see Patterson (1978).

1.2.2 DOE's Resource Terminology

DOE's terminology differs slightly from the basic terms stated above in two respects. First, the dichotomy between known and unknown mineralization is explicitly recognized. Furthermore, in the unknown case, gradation in the degree of confidence in the estimates is recognized. The second difference is in the definition of "resources" (DOE's

Table 1.1. Forward cost components used by DOE.

A. Capital costs (applied only on a forward basis)
   - A.1 Acquisition cost
   - A.2 Exploration cost
   - A.3 Development cost
   - A.4 Mine capital cost
   - A.5 Mill capital cost

B. Operating costs
   - B.1 Mining operations
   - B.2 Milling operations
   - B.3 Haulage of ore from mine site to mill
   - B.4 Royalty
   - B.5 Ad valorem tax
   - B.6 Severance tax

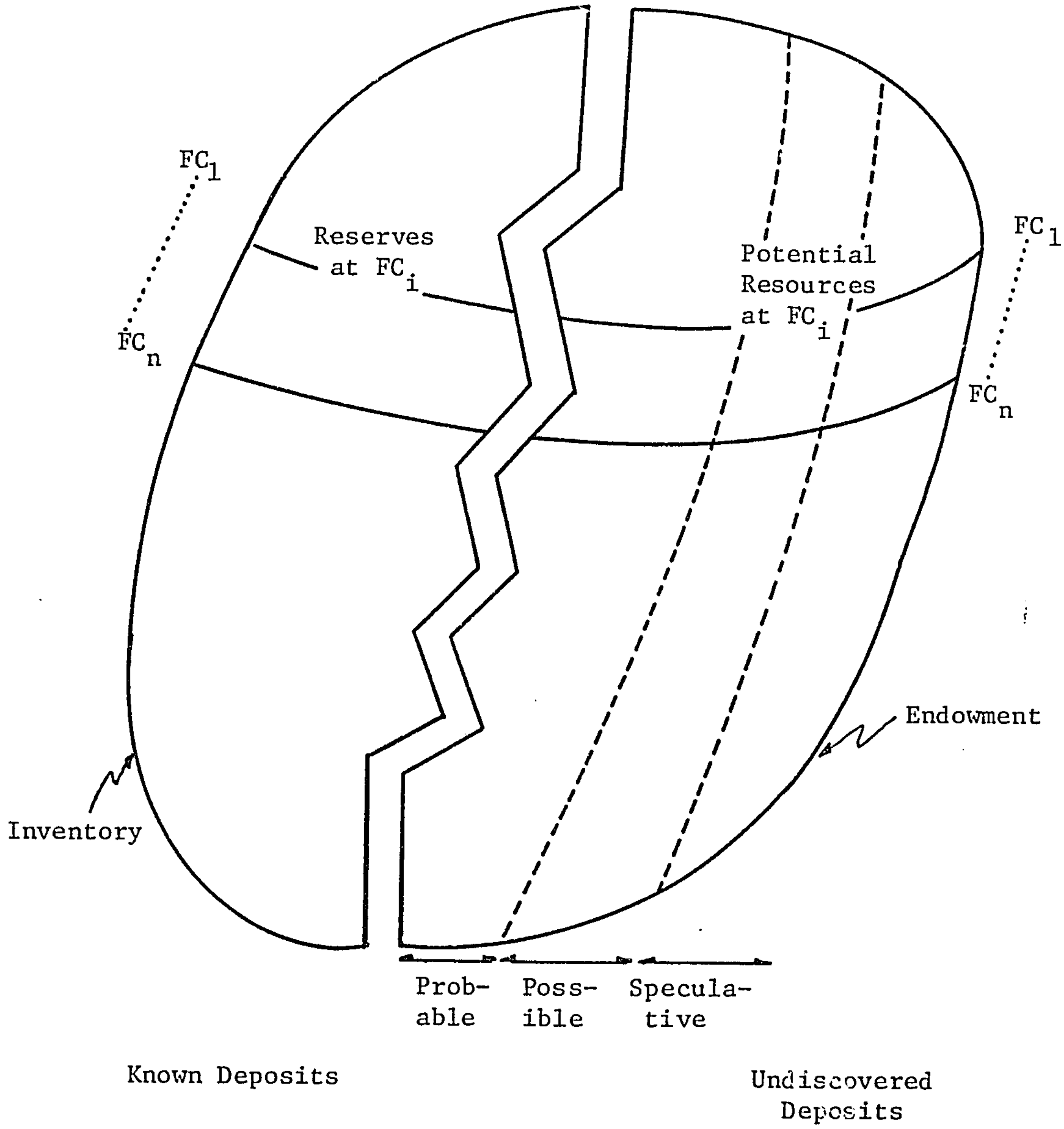
C. Cost components excluded from DOE costing scheme
   - C.1 Past capital costs (i.e., sunk costs)
   - C.2 Profits (return on invested capital)
   - C.3 Income taxes
   - C.4 Interest

"potential resources"): exploration costs are included in their costing framework used as a reference for economic conditions.

Figure 1.2 illustrates relationships implied by DOE's resource classification. Endowment, as used by DOE, is defined as the total amount of U_3O_8 existing in the undiscovered occurrences in a region. An occurrence is defined by DOE as a discrete lens or cluster of lenses of uranium-bearing rock containing at least 10 tons of U_3O_8 at a cutoff grade of 0.01% U_3O_8. These physical characteristics are also employed in the present study to define uranium endowment.

Potential resources refer to the amount of U_3O_8 contained in unknown deposits discoverable and producible at costs not exceeding the stated forward cost (FC_i). The assignment of potential resources to each forward cost category will be discussed later. Potential resources are further subdivided into probable, possible, and speculative classes to reflect the decreasing amount of geological information available and the increasing uncertainty of the estimate. It is important to note that for a region in which no uranium deposits are known, the term potential resources used by DOE corresponds to the potential supply definition in Section 1.1 with one exception: their forward cost understates the cost (price) of this potential supply because it neglects some significant costs, most notably return on invested capital and income taxes, which may be significant enough to make the net present value of those deposits negative.

Uranium inventory is defined by DOE as the amount of U_3O_8 known to exist in known deposits (based mainly on company drilling data) that



FC = forward cost

i = index to indicate forward cost category ($FC_{i+1} > FC_i$)

n = highest forward cost category

Figure 1.2. Uranium resource terms and relationships as used by the U.S. Department of Energy (DOE).

exceeds both specified minimum mining thickness and grades of 0.01% U_3O_8. With respect to resource terminology used in this study (Section 1.1 of this chapter), it corresponds to the portion of the endowment that is known.

The term reserves refers to the amount of U_3O_8 in the uranium inventory that is producible at or less than a specified forward cost with state-of-the-art mining and milling technologies.

In terms of the resource terminology used in this study, the concept of potential supply is equivalent to the addition of DOE's potential resources and DOE's reserves except for the fact that DOE underestimates the costs (Section 1.3 of this chapter).

It is important to emphasize the following relationships in DOE's terminology (Figure 1.2):

a. Forward cost levels are inclusive. That is, the FC_i class (either reserves or potential resources) includes all the lower cost classes.

b. Endowment is an inclusive term. It comprises all potential resources down to the highest forward cost considered as well as those "nonresources" that meet the endowment criteria.

c. Inventory comprises all reserves as well as the known, lower quality, mineralization (nonreserves) meeting the inventory limiting criteria.

1.2.3 Appraisal of Potential Resources by DOE

Since the present study deals primarily with appraisal of uranium resources in undiscovered deposits, only the analysis of DOE's potential resources appraisal methodology is discussed. Those interested in DOE's methodology for appraising uranium resources in identified deposits (i.e., estimates of reserves and uranium inventory) are referred to the DOE (1980) assessment report on uranium in the United States. Since the early 1980s, responsibility for the assessment of uranium resources was transferred from DOE to the U.S. Geological Survey.

DOE's appraisal of potential resources for a region can be divided into two major steps:

Step 1: Endowment estimation.

Step 2: Costing the endowment estimated in step 1 (i.e., determining that portion of the endowment that constitutes potential resources at various forward cost levels, FC_i).

1.2.3.1 Step 1: DOE Endowment Estimation

The method used for this estimation by DOE relies on geologic analogy and the subjective estimation of key factors entering the endowment equation (see equation below). The DOE procedure sequence is as follows (DOE, 1980):

a. Geologic evaluation for the occurrence of uranium in a region (usually a 2-degree National Topographic Quadrangle Map series).

b. Delineation of favorable areas within the region being assessed. This is done by identifying areas with key geologic characteristics similar to uranium deposit recognition criteria. This task was performed by an expert (the principal investigator of the region) and reviewed by DOE geologists.

c. Selection of a control area(s) for each of the favorable areas identified in (b). A control area is an area with known deposits that is geologically similar to the favorable area(s); endowment of a control area is (or is presumed to be) known from records of reserves and past production.

d. DOE estimated the endowment by the following equation:

$$\tilde{U} = A * (\tilde{F}|O) * (\tilde{T}|O) * (\tilde{G}|O) * \mathrm{Prob}(\tilde{O})$$

where:

U = uranium endowment

A = plan view area of the favorable area in square miles

F = fraction of A that is underlain by endowment

O = the event of "at least one deposit of at least 10 tons of U_3O_8 at a grade of at least 0.01% U_3O_8 occurring within the boundaries of the favorable area"

T = tons of endowed rock per square mile within A*F

G = average grade of endowment in decimal fraction form

Prob(O) = probability that the event "O" is true

~ signifies a random variable or event

Factor A is treated as a constant and represents the most likely extent of the favorable area. Factors F, T, and G are random variables estimated subjectively by the principal investigator by comparing geologic characteristics of the favorable area with those of the control area. The term Prob(O) was also elicited from the principal investigator.

Additional detail concerning endowment estimation methodology may be found in DOE (1980) and in the critique of this methodology by Harris (1980).

1.2.3.2 Step 2: Costing the Endowment

This second major step consists of determining that portion of the endowment that constitutes potential resources at various specified forward cost levels. The procedure used to allocate this uranium endowment for any stated forward cost level is predicated upon the following heuristics (based on Blanchfield, 1980):

a. Select the cutoff-average grade relationship (a simple restatement of the distribution of grades) to be applied to the entire estimated endowment in the favorable area. This relationship is usually derived from the control area (DOE, 1980) and is stated in the following form:

$$g = m\,(c - 0.01) + G$$

where:

g = average grade above cutoff

m = parameter (slope); determined from the control area

G = average grade of the endowment; determined subjectively from the favorable area

c = cutoff grade

b. Select a forward cost category, FC^o (e.g., \$30/lb. of U_3O_8).

c. Given depth of the endowment, forward cost category, and geographic location of the favorable area, compute total operating costs (OP) per ton of "ore" (part B of Table 1.1). It must be emphasized that mining operating costs as computed by DOE are a function of depth, forward cost, and regional location of the surmised uranium resources but are not a function of neither mine capacity nor deposit size and grade distribution within which these resources would occur.

d. Compute CC, the total capital costs remaining to be expended ($/ton of "ore") required to achieve production. In the case of undiscovered resources, these forward capital costs include all investment costs (except required return on invested capital) and exploration costs (Table 1.1, part A). Exploration costs are calculated as a function of the depth of the surmised resources and a pre-established number of drill holes per square mile. Mine capital costs are computed by DOE as a function of depth and forward cost category. However, they are not computed as a function of either mine capacity, or deposit grade and size in which these resources are hypothesized to occur. The capital cost of the mill (computed as a function of mill throughput) is either specified by the resource analyst or assumes a default value.

e. Compute the cutoff grade (CG^o) for the selected forward cost category (FC^o). CG^o is the maximum of either CG1 of CG2, which are defined below:

$$CG^o = \max [CG1, CG2]$$

1. Computation of CG1: CG1 is the grade at which the recoverable value of one ton of material of that grade (treating FC^o as the unit value) equals the total operating cost calculated in (c):

   $$FC^o * MR * (CG1/100) * 2000 = OP$$

   where:

   FC^o = selected forward cost category

   MR = mill recovery (decimal form)

   CG1 = cutoff grade "1" (% U_3O_8)

   OP = total operating costs per ton of "ore" (Table 1.1, part A)

   The actual computation is more complex than that shown here: mill recovery (MR) is itself a function of grade. This is solved by numerical methods.

   The "break even" cutoff grade CG1 would not be tolerable for large tonnages of ore. However, as most ore grades are usually well above a stated cutoff, CG1 in DOE's methodology constitutes a plausible candidate for the final cutoff grade (CG^o).

2. Computation of CG2: CG2 is the cutoff grade at which the recovered value of one ton of material of average grade above CG2 just covers total operating costs plus total capital costs:

   $$2000 * (AG2/100) * MR * FC^o = OP + CC$$

   where

$AG2$ = average grade of material above cutoff grade CG2

$CG2$ = cutoff grade (2) (% U_3O_8)

MR = mill recovery (decimal form)

FC^o= selected forward cost category

OP = operating costs (cost/ton of "ore")

CC = capital costs (cost/ton or "ore")

Substituting the cutoff-average grade relationship stated in (a) in the equation above:

$$2000 * \{[m(CG2 - 0.01) + G]/100\} * MR * FC^o = OP + CC$$

Solving for CG2:

$$CG2=0.01 + [(OC + CC) / (20 * MR * FC^o) - G]/m$$

f. Compute the average grade of material (AG^o) above cutoff grade CG^o by using the cutoff-average grade relationship from (a).

g. Using the cutoff-average grade relationship from (a), obtain the fraction of material above cutoff, $f(CG^o)$. Potential resources at a forward cost level of FC^o ($/lb U_3O_8) are then computed as follows:

$$U(FC^o) = TMR * f(CG^o) * (AG^o/100)$$

where

$U(FC^o)$ = tons of potential resources at forward cost FC^o

TMR = tons of mineralized rock constituting the endowment

$f(CG^o)$ = fraction of mineralized material above cutoff, CG^o

AG^o = average grade of mineralized material above cutoff, CG^o.

1.3 Statement of the Problem

Virtually any practical program of sizeable magnitude must compromise between meeting program objectives within the planned time frame and achieving these objectives using the best conceptual structure. DOE's national uranium resource evaluation program (NURE) is not an exception. In their quest for better resource estimates, DOE's resource analysts, well aware of this tradeoff, consistently revised and improved their resource estimation methodology. DOE also sponsored basic research projects to explore new methodologies for resource appraisal, including the one on which this dissertation is based.

This section identifies and discusses some problems in DOE's potential supply estimation methodology, not to criticize *per se*, but as a means of leading into the structure of the potential supply system of this study.

Provided that a potential supply estimate for a given region is to be a forecast of the quantity of uranium that will be discovered and produced under specified cost levels (referred to here as price/cost), the forward cost procedure employed by DOE for the costing of the endowment is inadequate. Given an endowment estimate, this costing methodology understates the cost at which uranium will be available.

The inadequacy of DOE's costing methodology arises not because of its "forward" spirit but because of both intrinsic problems in developing the costs and explicit omission of important cost components. These problems, which prevent a full costing of the endowment, can be grouped into three categories: (1) explicit omission of important cost components; (2) inadequate treatment of mining costs; and (3) inadequate treatment of exploration costs.

To emphasize the problems this study attempts to address and bring them into proper perspective, some comments are provided in the remainder of this section on the specifics of each of the problem categories above. Some of the factors that must be considered for proper costing of the endowment will necessarily be brought up in the context of this discussion; the philosophy behind these factors will be discussed in the conceptual framework chapter (Chapter 2).

1.3.1 Explicit Omission of Important Cost Components

The most notable costs omitted in the DOE's costing methodology are return on investment (cost of capital) and income taxes. The omission of cost of capital has far reaching implications: it ignores the time value of money, eliminates the significant cost of lead times and delays, and prevents adequate consideration of the cost of risk. The omission of federal and state income taxes implies greater future cash flows than would actually be received by firms in the industry, leading to overestimation of potential supply for a stated price/cost.

The overall implication of DOE's omission of these costs is that objectives and decisions of the firms in the industry are misrepresented, and the cost to industry of creating new supply is understated.

1.3.2 Inadequate Treatment of Mining Costs

Although in the DOE costing procedure the dollar-per-ton of mineralized rock figures for operating and capital costs are parameterized on depth of the endowment and forward cost category, other factors influencing mining cost, such as size of the deposit and its grade distribution, mine capacity, and life of the operation, are not explicitly considered. Further, for every economic deposit discovered, the mining cutoff grade, mine life, and mine capacity are, in real life, the result of optimizing mine planning decisions. These optimum decisions, of course, are predicated upon the physical characteristics of the deposit, expected economic conditions (i.e., price/cost level), and rate of return on investment required by firms in the industry.

Thus, it seems evident that the dollar-per-ton of mineralized rock obtained by DOE does not reflect well the entire chain of events and decisions that ultimately determine mining costs and quantity of uranium potential supply.

1.3.3 Inadequate Treatment of Exploration Costs

It must be stressed that modeling of exploration activities implied by DOE's forward cost procedure to estimate potential supply does not include the probability dimension of a real world exploration process. In other words, in the real world, valuable deposits are missed by exploration (i.e., the probability of discovering a deposit is less than unity) and hence do not contribute to the potential supply of an area; naturally, exploration costs incurred in the search for deposits must be charged against only those deposits discovered. Conversely, DOE treats exploration by a unit exploration cost ($/ton of mineralized rock) which enters into cutoff grade computation (see DOE's procedure, described in section 1.2.3). Since cutoff grade is higher when exploration cost is included, the ultimate effect in DOE's procedure is a reduction in the amount of uranium in material above cutoff. Presumably, this exploration cost per ton obtained by DOE from historical data reflects the omitted probability dimension and all of its consequences.

Concurrently with the successes and mistakes of exploration, proper exploration cost should reflect the magnitude and natural attributes of the endowment evaluated, namely: area of the favorable region, number, size, grade distribution, and depth of the deposits as well as the state of discovery depletion of that endowment. Additionally, exploration cost should also reflect sequential decisions characteristic of the exploration process and the impact on these decisions (e.g., drilling density) of both the stated price/cost level and risk perceived at every decision juncture by the firm conducting the exploration.

It shall be recalled that DOE's unit exploration cost is obtained as a function of depth of endowment and a predetermined number of drill holes per square mile. Naturally, such a cost fails to represent resolution of the factors referred to in the previous paragraph. The omission of those factors and of the complex dynamics of the exploration process may severely understate the marginal cost of potential supply.[1]

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1. Additionally, the fact that DOE's potential supply methodology does not employ a probabilistic dimension in their treatment of exploration and does not include cost of capital, precludes the incorporation in their approach of an adequate account of the economic cost of risk.

1.4 Objective

In an attempt to alleviate to some degree the problems inherent in the DOE costing methodology, this dissertation designs a comprehensive computer simulation system that estimates for different price/cost levels the potential supply from undiscovered uranium deposits in a region for which endowment estimates are available.

The objective of the system presented here is to provide a full costing procedure for a credible estimation of new uranium supply by modeling what industry does. The system simulates uranium exploration and production activities and the concurrent profit maximizing decisions under specified price/cost levels performed by a "typical firm"[1] in order to create new supply. Every effort was made to provide a structure that defines the marginal, rather than average cost, of potential supply.

To achieve full marginal costing, the exploration model explicitly treats uncertainty, sequential profit maximizing decisions, information gain, and discovery depletion. Modeling of the exploration process, basically a drilling model, relies heavily on subjective opinion inputs and utilizes a Bayesian approach to decision theory. Economics of risk in exploration decisions is simulated by employing an heuristic approach for investment decisions under risk adapted from Markowitz's portfolio analysis (Harris, 1971).

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1. In modeling potential supply, it is assumed that exploration and mining activities are conducted by only one firm--a "typical firm" that represents the uranium industry--so that exploration of the same ground is not repeated and the intricacies of the interplay of multiple firms is avoided.

The exploitation process is based upon a open pit mining cost model that utilizes cost equations developed by STRAAM Engineers (STRAAM Engineers Inc., 1979). The model imitates the profit maximizing decisions in the design of an open pit mine. It explicitly accounts for the deposit size and its grade distribution, mine capacity, mine life, and rates of removal of overburden and waste material.

1.5 Scope and Limitations of the Study

In the design of the system, great emphasis has been placed upon modeling the exploration process. Three important aspects of exploration activities are accounted for in this study. The first is the sequential nature of exploration activities. This allows incorporation of the dynamics of the information gained after a segment of the exploration program is performed. The second aspect springs from the first and utilizes the fact that exploration is predicated upon: (1) prior expectations of the exploration agent concerning the true but unknown state of the endowment (and ultimately the net present value implied by that endowment) and (2) subsequent revisions of those expectations as exploration unfolds. The third aspect is the explicit accounting of risk. This is accomplished by considering the exploration decision under risk within the financial framework of the firm assumed to be conducting the search activities and the stated price/cost levels.

The exploration model in this study is specific for one mode of uranium occurrence: the roll front-type deposit. Therefore, application of this potential supply system to other modes requires an exploration model specific for those modes of occurrence.

Because the objective of this study is to structure the most complete overall system design possible in terms of the full accounting of costs involved in creating new supply, a consultant[1] was employed for a pilot study. His description of exploration activities and phases and the relevant geologic factors influenced overall design. The subjective opinion of this expert is the basis for: (1) description of the morphology and size and grade distribution of roll front deposits and (2) description of his exploration performance throughout the exploration phases. These subjective estimates were used as an expedient to design.

The modeling of exploitation is limited to open pit mining (and subsequent milling). Thus, the potential supply system design in this study only applies to endowment within the open pit mining domain.

The scope of this dissertation is primarily limited to design of the potential supply system. Although most computer programs for the models comprising the system have been developed, additional testing and computer programming remains in order to integrate the models into the overall system design established here.

1.6 Related Work

1.6.1 Consad Model

A computer Monte Carlo simulation model of exploration for nonfuel minerals was constructed by Consad Research (1969) for the Bureau of Mines to assess the economic efficiency of alternative exploration programs.

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Emphasis is placed on indirect exploration methods, although drilling is considered. The user specifies an exploration program from a menu of techniques. The basis of the model is the generation of anomaly responses to most of those techniques from synthetic deposits and the determination of the error incurred in recognizing those anomalies.

The basic application of the model is the assessment of the economics (costs and benefits) of alternative exploration programs. This is accomplished by keeping track of the cost of using several techniques in an exploration program specified by the user, as well as the results of it in terms of the value of the deposits discovered. Furthermore, the contribution of each exploration technique to the benefits of the overall program can be examined.

The user needs to specify the following:

1. Economic data, which includes price of up to 13 nonfuel minerals (mostly metals) and discount rate.
2. A description of the geographic region consisting of the arrangement of up to 20 blocks; for each block, an index of topography and accessibility needs to be specified.
3. Parameters of predetermined probability distributions for the endowment components (deposit, size, average grade, depth, etc.).
4. A description of the exploration program: techniques to be applied, levels of their usage, and sequence to their use. Also, the critical value (cut-off value) for anomalies detected by most exploration techniques needs to be specified. There are 25 techniques to choose from, and they fall into four categories:

geological reconnaissance, geophysical, geochemical, and drilling.

The model "seeds" a hypothetical large region with deposits of different types of minerals (up to 13) based upon the probability distribution of endowment components. Their location within the synthetic region is based on a negative binomial distribution to mimic the actual "clustering" of deposits.

Once the seeding process is completed, a specified exploration program is simulated. For every technique employed, there is a cost function in the system that generates its total cost as a function of topography, accessibility, and area covered (or depth in the case of drilling).

Reconnaissance geological activities (geological mapping, photography, and photometry) are viewed by the model as a "front end"; i.e., preparatory to actual exploration. The model does not offer much in the way of evaluation of geologic techniques. No anomaly responses are generated, so no areas are rejected. It simply charges the necessary cost when any techniques of this group are employed.

The model generates responses from the seeded deposits to both geophysical and geochemical techniques. For the first group (magnetic, gravity, resistivity, electromagnetic, radioactivity, and induced polarization), the response is generated by incorporating theoretical work on anomaly response developed by geophysicists. Based upon the assumption of a spherical deposit, the response is a function of deposit size, depth, grade and nearby deposits. In the case of geochemical techniques, the response is generated by a simple negative exponential function; depth is the only independent variable. When an area contains no deposits and there is none nearby, the response function generates a background response.

Once a response is generated, either from a deposit or from an area void of mineral deposits (background), a random component is added by sampling a Beta distribution obtained from the discrimination constant[1] of the particular technique. This is done to approximate actual exploration "noise" and uncertainty; i.e., exploration information is imperfect. The final response so obtained is compared to the critical value (or cut-off value) specified by the user. If the final response is above that value, the area is retained on the prospect list, to be submitted to the next technique in the user specified sequence. Otherwise it is rejected. Note that false signals are generated when the random component is added to the "background" response. Both the critical value and discrimination constant of the technique can lead to an erroneous decision[2] regarding identification of a geophysical or geochemical anomaly; i.e.. retaining an area void of deposits or rejecting an area that contains deposits.

Response to drilling is deterministic. A stochastic character is added by the depth variable. If the input-specified drilling depth

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1. The discriminant constant is the ratio of the variance to the mean response. It provides a measure of the discriminating ability of an exploration method. If the ratio is small, then the method is a good discriminator (small variance).

2. Selection and rejection efficiencies of each technique are computed by the model.

is greater than depth to deposit generated by the model from random sampling of a rectangular depth distribution, drilling is successful and the result shows the deposit grade. Otherwise it is unsuccessful. In the case where no deposit exists, drilling meets with failure regardless of drilling depth.

The use of every exploration technique is costed. Upon completion of an exploration program (that is, after the drilling phase is completed), it is assumed that all valuable deposits discovered will be developed. A very simple value function (net of development, processing, and mining costs) is invoked and the model discounts the proceeds from the sale of the mined product on the assumption of a 10-year mining period.

Upon completion, the model computes overall benefits (discounted value of deposits discovered) and costs (sum of the total cost of each exploration technique employed), and the economic efficiency of the exploration program is evaluated.

The report concludes that the most efficient exploration program varies with the method of "seeding" deposits. Profitability of the exploration program could be increased using those techniques with the highest selection and rejection efficiencies.

Emphasis is placed in this detailed exploration model on indirect methods (geophysical and geochemical). Although the report recognizes that drilling is by far the most important exploration cost, this activity is not fully accounted for. Once an area has been selected for drilling, the only exploration mistake allowed by the model, if the area contains a deposit, is insufficient drilling depth, which is an input provided by the user who, of course, does not benefit from the knowledge of all the anomaly responses--generated inside the model--which led to the selection of the particular area. Thus, there may be overstatement of drilling errors due to depth. No drilling mistakes are allowed in the drilling location; the model assumes that once an area containing a deposit is selected, there is enough information to always correctly locate the drilling sites on top of the deposit. This may lead to an understatement of drilling mistakes due to the areal location of the holes. The model also lacks of any means to rationalize drilling density and when to cease drilling: a nonsequential drilling campaign is simply an input to the model, predetermined by the user, and no multistage drilling is allowed. No decision module is incorporated, so the user has to manipulate the selection of exploration techniques and critical values to arrive at the "best" exploration program under a specified set of prices.

1.6.2 Drew's Sequential Pattern Drilling Model

Rather than proposing a model to explain the exploration process, Drew (1979) provides a very illustrative example to demonstrate the important elements of a sequential drilling program: exploration architecture, revision of *a priori* probabilities of the state of nature in the light of new information, and translation of these probabilities for outcomes into financial terms.

The model is based upon a regular drilling pattern to search for elliptical-shaped deposits (targets). Detection probabilities are calculated using geometric probability concepts. The basis for their calculation is the unit cell of the uniform grid used for the regular drilling pattern. The detection probability is the ratio of the area of the target over the area of the unit cell.[1]

The exploration architecture includes several drilling stages. A relatively wide hole spacing is chosen for the initial drilling stage. At each subsequent stage the drill holes are placed at the midpoints between the holes drilled in the prior stage. This placement sequentially increases the density of drilling.

The model assumes that the existence of deposits is uncertain. Only one deposit is likely to occur.[2] The only information gained after a given drilling stage is the detection or no detection of the deposit.

Let g represent the perceived probability before drilling begins that a deposit of given size and shape occurs in the search area. Usually available geological, geophysical and geochemical data are used to estimate this a priori perception about the "state of nature."

Given conditional detection probabilities (conditional upon existence of the deposit) and that a stage of drilling ends in failure to detect the deposit believed to exist in the search area, the probability of occurrence, g, can be revised using a variation of Bayes' rule.

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1. This definition is valid when the size and shape of both the target and the drilling pattern are such that the target cannot be hit more than once. Otherwise, more complicated calculations have to be made.

2. In his example, Drew assumes 5 possible deposits. To simplify, we assume only one.

This revision is made according to the following equation, given that stage (i - 1) has been completed:

$$g_i = \frac{g_{i-1} \cdot [1 - P(s_{i-1}|f)]}{(1-g_{i-1}) + g_{i-1} \cdot [1 - P(s_{i-1}|f)]}$$

where

g_i = revised probability, at the end of stage i-1, that the deposit exists in the search area given that no deposit was detected yet; this probability will become the a priori probability once stage i is conducted

$P(s_{i-1}|f)$ = probability that the deposit would have been detected during the previous drilling stage given that the deposit (f) exists; geometric probability methods are used to compute this term using the spacing, s, specified for the drilling stage

g_{i-1} = the a priori probability, perceived at the start of the previous drilling stage (i-1)

As exploration unfolds and no deposit is detected, revised occurrence probabilities decline.

The perceived state of nature (occurrence probabilities) are translated into financial terms by computing the expected rewards at the end of every stage. The value of the deposit believed to exist in the search area is used:

$$GDR_i = V \cdot g_i \cdot P(s_i|f) - C(s_i)$$

where

GDR_i = Expected gross drilling return at the start of stage i

V = value of the deposit

g_i = perceived probability at the start of stage i that the deposit exists in the search area

$P(s_i|f)$ = probability of detection of the deposit during stage i given that the deposit exists

$C(s_i)$ = cost of drilling stage i

If no deposit is detected as more exploration stages are conducted, the expected reward (GDR) will tend to decrease as more exploration stages are performed, both because of the decline of the occurrence probability alluded to above, and the increase in drilling cost implied by high drilling density.

The study concludes that the decision, at any stage of drilling, should be made by weighing the expected reward (GDR) and the risk $[1 - g_i \cdot P(s_i|f)]$. The actual decision will depend on the risk preference of the exploration agent which, in theory, should be related to the size of his assets.

The model is limited inasmuch as the only information gained after each drilling stage is whether or not a deposit has been detected. In practice, geologic information acquired at the end of each stage should help to revise the occurrence probability more realistically than the revision performed by the Bayesian rule of the model. Thus, it is conceivable that geologic information could lead to an upward rather than downward revision of the occurrence probability at a given stage.

1.7 Merits of This Study

The major merit of this dissertation is the design of a potential supply system that aims at a comprehensive costing of the endowment estimates provided. The virtues of this system have been alluded to in previous comments:

- Parameterization of the system on key attributes of endowment.
- Modeling of exploration as a sequential decision making process.
- Explicit consideration of the price/cost level as an influential factor in both exploration and exploitation activities and decisions.
- Accounting for the time value of money and its implications for exploration and mining decisions.
- Consideration of the economic cost of risk.

Although the fundamental factors of the models constituting the potential supply system are discussed in the conceptual framework (Chapter 2), it is important to emphasize that for modeling purposes, exploration activity is separated into two different but complementary entities: the physical process of exploration (i.e., the "pure" search process); and the economics and decisions which exploration involves. This single fact constitutes the essence of the exploration model; at the same time, it greatly compounds the task of modeling exploration and has been the most important challenge of this system design.

The reason for the disaggregation of exploration activity is to make the system responsive to different price/cost levels so that, as levels are manipulated as required to "map" potential supply, exploration strategies and drilling "rules of thumb" change in ways dictated by profit maximizing criteria. The particulars of this disaggregated structure deserve some amplification. The relevant contention is that one of the most important elements in a realistic description of creation of new supply is the outcome (discoveries) resulting from exploration decisions and, further, that these decisions change as price/cost levels change. To illustrate this, consider a contrived example of a single-phase pattern drilling[1] program for a given area containing a set of m identical[2] undiscovered uranium deposits. This is a highly simplified situation but one which displays the essence of the exploration model and puts the disaggregation of physical exploration and exploration economics and decisions in proper perspective.

Let V(p) represent the value of each deposit (net of production costs) for a given price/cost level (p) and assume that V(p) is positive. The higher that "p" is, the higher the deposit value will be. Also, let C(n) represent the cost of drilling "n" holes and Prob(disc|D,n) represent the probability[3] of discovering a deposit with a drilling pattern of "n" holes given that the deposit "D" occurs. Assume that the only

1. Pattern drilling is used in this example for the purpose of illustration. This should not be interpreted as an endorsement of pattern drilling as an adequate representation of real-life drilling processes.

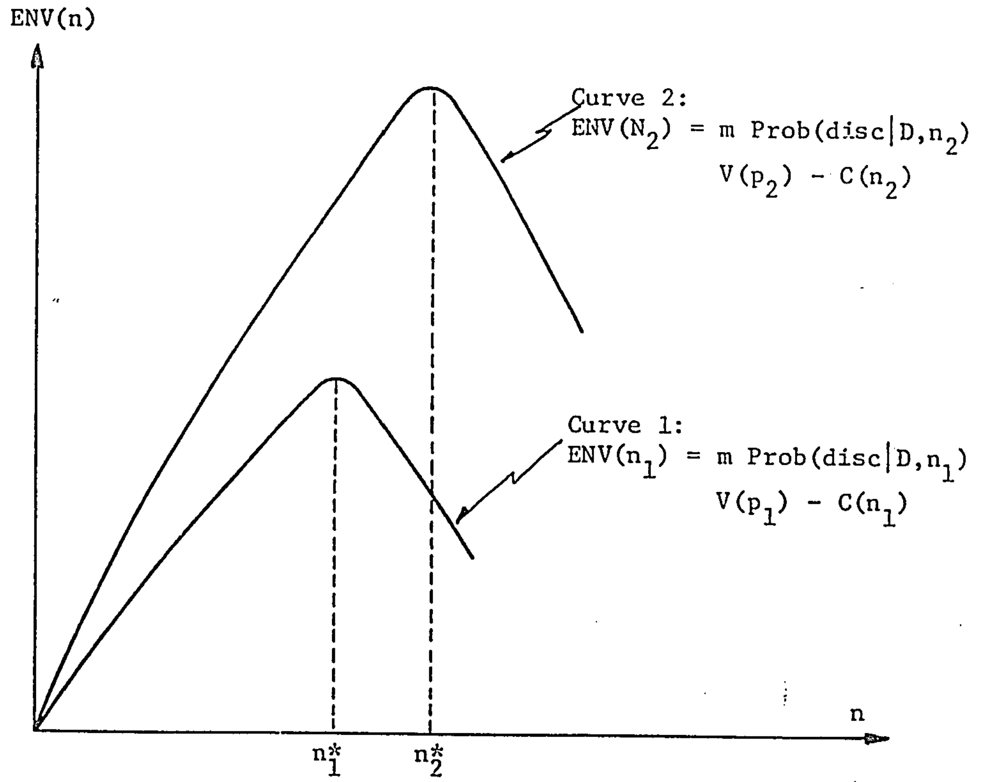
2. Deposits are assumed in this example to be identical in size, grade depth, shape, and orientation. In the remainder of this example, "D" is used to refer to this vector of characteristics.

3. Because drilling is assumed to be performed on a predetermined regular pattern (e.g., square, rectangular, or triangular grid), this term would be obtained using geometric probability methods, such as the ones detailed in Drew (1979).

exploration decision consists of choosing the exploration effort, in this example the number of drill holes (n^*) that maximizes the expected net value (net of production and discovery costs) of the exploration campaign, $E[m\ V(p)\ \cdot\ C(n)]$. Of course, the higher the number of holes, the smaller the spacing of the drilling pattern and the higher the probability of discovering a deposit.

Now suppose that the price/cost regime is p_1. Curve 1 in Figure 1.3 represents the expected net value curve for this price/cost; that is, the net value--on average--that a drilling pattern of "n" drill holes will yield. The optimum number of holes to drill when price/cost is p_1 is represented by n_1^*. Of course, some deposits will be missed with this optimum effort level, but the value of these deposits do not warrant more intensive exploration. Instead, if the price/cost regime is p_2 ($p_2 > p_1$), this situation completely changes. Curve 2 in Figure 1.3 represents the expected net value curve when price/cost is p_2. Because $V(p_2) > V(p_1)$, curve 2 lies above curve 1. When price/cost is p_2, n_2^* is the optimum decision. As the deposit value "$V(p_2)$" warrants more intensive exploration, n_2 is greater than n_1. Clearly more deposits will be discovered when price/cost is p_2, although some will still be missed.

In this simple example, the conditional discovery probability, $Prob(disc|D,n)$ represents the physical exploration process. The reason for the "physical" qualifier of the term is because no economic rationale is attached to it: it simply states the likelihood of discovery of a deposit D if one happens to drill "n" holes regardless of how one arrives at the choice of "n." The economics and decisions of exploration for the



Definition of variables:

p = price/cost; p_1 is level #1, p_2 is level #2; $p_2 > p_1$

n = number of holes drilled in a pattern drilling campaign (assumed continuous just for ease of illustration)

n_i = number of holes drilled in a pattern drilling campaign when price/cost is p_i $(i = 1,2)$

n_i^* = exploration intensity (number of holes in the pattern drilling campaign at which the expected net value $ENV(p_i)$ is maximum when price/cost is p_i $(i = 1,2)$

$ENV(n)$ = expected net value as a function of holes drilled

$Prob(disc|D,n)$ = probability of discovering a deposit D with a pattern drilling campaign of "n" holes

$V(p_i)$ = value of each deposit (net of production cost) for price/cost level p_i

$C(n_i)$ = cost of drilling n_i holes

m = number of undiscovered deposits (all identical) occurring in the exploration area

Figure 1.3. Diagrammatic representation of the economic and decision process of exploration in the simplified single-phase pattern drilling example.

two price/cost levels considered are represented by the graphs in Figure 1.3. The profit maximizing decisions are the graphical solutions n_1^* and n_2^*. The expected net value computation provides the link between physical exploration and economics.

Thus, it is clear that approaches to exploration modeling that do not treat physical exploration and exploration economics and decisions as two distinct processes can hardly be deemed appropriate to describe potential supply for different price/cost levels. Furthermore, this disaggregation and the subsequent integration within the model of these two processes by profit maximizing criteria, together with a comprehensive treatment of risk, is a necessary condition to obtaining the marginal cost of potential supply. If one were to ask an exploration expert for decision rules (say the drilling pattern and number of holes for a given exploration phase), these decision rules, which could be mistaken for "best practice," are more than likely a mixture of physical search and economics and cannot be used blindly as the optimum drilling decisions. At best, they represent the optimum drilling rules for the price/cost level and other economic and endowment conditions that the explorationist had in mind when those decision rules were elicited. Using those rules indiscriminately for any stated price/cost level is tantamount to bypassing the decision making process in real-life exploration.

By the same token, the approach to exploration modeling which consists of obtaining, via subjective opinion, the probability of discovery conditional upon deposit occurrence and deposit characteristics, as well as some exploration budget reference, and the subsequent use of the same discovery probability throughout simulation of the discovery process, is inappropriate for any price/cost level other than the one the explorationist had in mind when the response was elicited.

The exploration model procedure employed in this study is to describe the explorationist's behavior and performance in such a way that they are relatively free from current or past economic influences. This physical search process is fully coordinated with the entity describing the economics and decisions of exploration (referred to as the "executive model"). The physical process of exploration simulates the successes and failures of deposit search given a specified level of exploration effort, whereas the executive model decides and specifies the exploration effort levels for use by the physical search entity. This decision takes into consideration the risk of the exploration project as currently perceived in the model, the price/cost regime, and the financial position of the firm assumed to conduct exploration.

Dissimilarities in the treatment of exploration by DOE and by this study need to be emphasized. DOE models exploration by computing a unit exploration cost as a function of depth of endowment and a pre-established number of drill holes. The effect of improved price/cost is reflected in their model only by drilling for deeper endowment.

Conversely, the exploration model in this dissertation is designed to simulate what industry does. The consequence of increased price/cost in this system is not only to drill for deeper deposits but to also increase the intensity of exploration and to search for smaller deposits. Additionally, other factors are explicitly considered in this system such as density of packing and state of discovery depletion of the endowment.

This is the first exploration model explicitly featuring all the effects alluded to above; the design of this model constitutes the most significant contribution of this dissertation.

1.8 Plan of This Dissertation

This dissertation is divided into three parts. The first provides a formal statement of the philosophy that motivated design of the system (Chapter 2) and a general idea of the overall structure and flow of the system (Chapter 3).

The second part consists of background information required to understand the notions and conventions used in modeling roll front deposit morphology (Chapter 4 and Appendix A) as well as background information on the technical aspects of exploration activities (Chapter 5, and Appendices B and C). Chapter 5 concludes with a description of the representative exploration sequence (detailed in Appendix D) used in the exploration model.

The third part consists of development of the system design. The detailed computer design of the entire potential supply system is provided in Appendices G, L, and K. Appendix G shows the major flow of the system and the linkage with the executive model (detailed in Appendix L) and with the mining cost model (detailed in Appendix K). Discussion of the flow and logic of the potential supply system is provided in Chapters 6 to 10.

Chapter 6 provides the particulars regarding items required from DOE's endowment estimates and how these items are adapted for use in the potential supply system.

The process of "decomposing" the estimated endowment in terms of its natural appearance (i.e., "creation" of deposits) is detailed in Chapter 7, which uses the notions and conventions established in Chapter 4. Complements for Chapter 7 are Appendix E, which displays the questionnaire and responses for the necessary subjective opinion inputs, and Appendix G (part G.2), which describes the computer design for this task.

Chapter 8 covers design of the sequential exploration model, including linkage with the executive model (detailed in Chapter 10). Appendix F displays the questionnaire and responses for the necessary inputs on exploration performance. Appendix G (parts G.5 to G.10) complements Chapter 8 with the description of the computer instructions.

The mining cost model (open pit) is described in Chapter 9 and is complemented by Appendix K, which describes the design of the computer program.

In Chapter 10, design of the executive model and estimation of necessary items for one of its segments referred to as the decision module are discussed. It also includes linkages with the exploration and mining models. Details of the computer program design are described in Appendix L.

Chapter 11 states the conclusions, contributions, and other applications of this study.

CHAPTER 2

CONCEPTUAL FRAMEWORK

To provide credible potential supply estimates for several price/cost levels which conceivably include levels never experienced before, estimates must recognize the full cost to industry of discovering and exploiting the deposits upon which the potential supply is based. This has far-reaching implications for design of the potential supply system, as it is necessary to simulate exploration and exploitation activities and decisions and explicitly account for the factors that affect them.

A system designed to reflect the hypothesis stated above must account for the following factors or effects that ultimately influence a potential supply estimate:

1. effect of return on investment (cost of capital) and federal and state income taxes,[1]
2. effect of successes and mistakes of exploration,
3. effect of the economics of risk,
4. effect of key endowment attributes,
5. discovery depletion effect,

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1. Taxes levied on units of production or value of production (severance taxes, state land royalties, etc.) are also important. Since they are a cost item just as any other operating cost, they may affect the cutoff grade decision and, consequently, sterilize *in situ* resources. The system allows incorporation of those charges in the cash flow procedure imbedded in the mining cost model.

6. effect of the stated price/cost level.

This chapter discusses the concepts behind these factors.

2.1 Effect of Return on Investment and Income Taxes

Use of discounted cash flow by the majority of firms in industry is a fact of life. Both exploration (e.g., level of exploration expenditures) and mining (e.g., cutoff grade, capacity) decisions are made on this basis. If the cost of capital is ignored, these decisions are distorted because the cost of waiting for future earnings is not considered. If future revenues and cash flows count as much as present ones, uneconomic exploration and mining ventures contribute to potential supply. For deposits that would be economic if subjected to proper analysis, ignoring the time value of money results in mine lives that are too long and cutoff grades that are too low.

Moreover, if current lead times and delays ranging from 6 to 17 years from initial exploration to production and the expenditure pattern over this period (Harris, 1978) are considered, ignoring the cost of capital is analogous to ignoring the very significant economic cost of increasing lead times and delays and their effect on the firm's decisions. When the waiting period for future earnings is extended upwards to 17 years, requiring even a moderate return on investment imposes a heavy cost on a mineral deposit.

No inflation projections are considered in the system. The implicit assumption is that all the cash flow components involved in creation of potential supply are in constant dollars. Since potential supply is a "stock" measure, constant dollar analysis is an appropriate approach.

It is contended in this study that even though the rate of return on investment is a sensitive element of corporate economics, it is a cost component that must necessarily be considered. The system designed here accounts for this cost. In fact, the rate of return on investment is determined in the system. An approach to obtaining an estimate for the representative rate of return on investment for the industry is demonstrated.

The effect of federal and state income taxes in the reduction of future pre-tax cash flows of the firms in the industry hardly needs comment. As stated above, anything affecting the expected cash flows of a firm will affect its future decisions regarding mining and exploration activities. Since the effect of income tax on cash flow is generally substantial even after considering the damping effect of tax shields, it is a factor that must be included in the simulation cash flow analysis of the firm.

2.2 Effect of Successes and Mistakes of Exploration

Exploration cost is a reflection of how easy or difficult it is to find deposits. Anything affecting the likelihood of a successful exploration program will necessarily affect the potential supply estimate. Consider a simplified example of two regions with identical areas, identical undiscovered endowment quantities, and same endowment depths. Assume that for some reason it is easier to find the deposits in one of the regions; that is, the likelihood of success is different in the two regions. It is then evident that the potential supply estimate of the hard-to-find endowment region will tend to be smaller for any price/cost level specified. Alternatively, as exploration costs can be charged only against those deposits discovered, the exploration cost component is higher in the region containing hard-to-find endowment.

The system is designed to take explicit account of the successes and mistakes of an exploration program. A mistake is understood to include any error in selecting or rejecting an area for exploration drilling, early abandonment of a drilling area containing an economic deposit(s), and conducting excessive drilling in an area that is barren or contains uneconomic deposits. In short, an exploration mistake is any error due to imperfect knowledge of the true state of the endowment.

Additionally, the exploration model accounts for those causes affecting the probabilities of a successful program, such as size and depth of the deposits, state of discovery depletion, number of deposits, level of exploration effort, and handing of risk. All of these causes are discussed in subsequent sections of this chapter.

2.3 Effect of the Economics of Risk

The magnitude and cost of risk has a major influence on the activities required to create new supply. Specifically, consideration of risk affects the decisions made in exploration with respect to the size of the region to be explored, when to quite a search area, the type of exploration, and its intensity. Consequences of these decisions are reflected in the outcomes (i.e., the deposits discovered and those left undiscovered) and costs of exploration for present and future discoveries.

The exploration model captures the essence of the economic cost of risk by considering the major behavioral features of exploration:

1. The model is sequential; its design is based upon several interrelated stages or phases which reflect the architecture of exploration activities and decisions.

2. The model represents both the actual state of nature and the state perceived (i.e., state of endowment or geologic features associated with it) in a search area at any time during the search process. A key concept to achieving this purpose is that of prior and posterior distributions of the unknown state of nature. The prior distribution is a probabilistic description of the unknown state of nature at some point in the exploration sequence prior to some future search action. The posterior (or revised) distribution is the result of the revision of the prior distribution after the outcomes of search have been observed. This posterior distribution becomes the prior with respect to any future search activities.

   An important component of the exploration model is the exploration performance which specifies the success probabilities or some other measure of performance (e.g., a degree of achievement of the objective specified for a given exploration phase), conditional upon the true state of nature and exploration effort.

This component is provided by the subjective opinions of an expert.

3. Exploration decisions with respect to any future search action in the exploration sequence are predicated upon the prior distribution of the unknown state of nature at that point in the exploration sequence.

   This prior distribution is communicated to an "executive model," which performs a risk analysis and evaluates the next exploration decision (amount of effort which, if equal to zero, implies quitting the area) given the stated price/cost level.

   The risk analysis is based upon the probability distribution of net value of the exploration project derived from the prior distribution of the unknown state of nature. That is, the perceived state of nature is translated into dollar terms for making the optimal decision. The probability distribution of net present value reflects the discount rate (cost of capital of the firm), income taxes, and the uncertainties in the state of physical variables (e.g., deposit size and grade) and decision variables (e.g., number of holes to drill next).

The exploration decision problem (i.e., how much effort to allocate to an area of search at any point in the exploration sequence) implies the answer to when to quit searching. It is dealt with in two steps. First, assuming that the remaining exploration activities will be conducted, the optimal exploration effort in the remaining phases is determined. Optimal means that effort which maximizes the expected net present value (net of cost of that optimal exploration effort). Second, the impact of the risk of the project, which is reflected in both the probability distribution of net present value (its derivation was discussed above) and the cost of the optimal exploration effort, is determined. The latter is determined by relating the cost of capital to the risk of the firm and comparing the capitalized future wealth of the firm with and without the project. Thus, the decision is made whether to commit that optimal exploration expenditure or to quit exploring the area.

2.4 Effect of Key Endowment Attributes

The following attributes of a uranium endowment affect an estimate of potential supply:

1. magnitude of endowment,
2. density of packing of the endowment,
3. depth of the deposits,
4. deposit size and its grade distribution.

2.4.1 Magnitude of the Endowment

Ceteris paribus, it is clear that more abundant endowments imply larger amounts of undiscovered resources at any stated level of price/cost. Similarly, a more abundant endowment implies a higher likelihood of a successful exploration program. This increase in the probability of success is ultimately reflected in a lower exploration cost component in the cost of any potential supply.

2.4.2 Density of Packing of the Endowment

The impact of this endowment attribute is significant in terms of exploration efficiency. If a given amount of endowment is surmised to occur within a small region, the area of search is reduced and exploration expenditures will be lower than if the region is larger. This implies that the price/costs of the potential supply estimates will be lower in the small region.

Both the magnitude and density of packing of the endowment are considered in the potential supply system. The system is designed to accept both the probability distribution describing the magnitude of the true but unknown endowment and the outline of the favorable area in which this endowment is assumed to occur. The endowment is decomposed into uranium deposits, which are then located within the favorable area. Subsequently, exploration activities for those deposits are simulated.

2.4.3 Depth of the Deposits

The effect of the depth of the deposits that make up the endowment upon which a potential supply estimate is made, is twofold:

1. The first is obvious: an increase in drilling costs as well as capital and operating mining costs with depth.
2. The second influence is an increase in risk with depth. As depth increases, the ability to correctly discriminate between endowed and barren areas in locating drilling activities by geology or any other nondrilling method decreases; hence the chance for making errors increases. The degree of belief of the exploration agent about the true but unknown state of nature (endowment) in

a region becomes weak. In addition to the probability distribution of potential "rewards" becoming diffuse, the "cost of betting" is larger (i.e., increase in drilling costs with depth). That is, exploration becomes a more risky venture. Since risk bearing is a cost to business, increased risk implies a higher price-cost level on the margin of potential supply.

In the potential supply system presented here, both depth effects are introduced. The depth obtained from DOE's endowment estimate reports is information carried in the potential supply system; holes drilled in the simulated exploration activities are costed as a function of depth. Similarly, the mining cost model also considers deposit depth in the simulation of pit design and in the computation of the amount of overburden and haulage distances to the surface. The issue of increase in risk with depth is considered in the system by including the errors in exploration performance in terms of the probability of an erroneous drilling area selection or of erroneous rejection as a function of depth. This increased risk is incorporated in the exploration decisions simulated in the model.

2.4.4 Deposit Size and Grade Distribution

These endowment attributes have an impact on both mining and exploration activities. Both are explicitly modeled in this system.

With respect to mining, a large tonnage deposit supports the establishment of a mine of larger capacity, thus achieving economies of scale. Given two identical deposits mined with the same mining methods but under different capacities (tons of ore/day), the one with larger mine capacity will experience the lower capital and operating costs, both in terms of dollars per ton of ore and per pound of U_3O_8 as long as economies of scale are still present. Whereas in DOE methodology the cost per ton of mineralized rock is functionally related only to depth, in this system design it is predicated upon both mine capacity, which in turn is determined by the size of the deposit, and depth.

The effect of grade on mining costs can be illustrated in a similar manner. Given two deposits that differ only in their average grade, assuming they are being mined under identical circumstances, the one with the higher grade will have the lower mining cost per pound of U_3O_8 (both operating and capital costs).

For exploration activities, the relevant probability property is that the conditional probability of discovery of an object (conditional on size) is proportional to its size; larger deposits are easier to find. To illustrate this effect on potential supply, suppose that there are two identical endowments (in terms of magnitude, grades, area, depth), but one is in small deposits while the other is in large deposits. It will be easier and less costly to find the large deposits than the small ones. Thus, for a given level of price/cost and same technology, the potential supply estimates will necessarily be higher for that location with large deposits, even though both endowments are the same.

In reality, of course, deposits are nonuniform in size. Instead, there is a size distribution describing the relative frequency of each size category. As the area is explored, the larger deposits tend to be found first, decreasing the likelihood of success of subsequent exploration programs. Thus, the exploration cost becomes higher as exploration unfolds. The exploration performance component of the exploration model accounts for the effect of deposit size on exploration cost. This component specifies the success probabilities or other measures of performance (such as degree of achievement of the objective of a particular exploration phase), conditional upon the true state of nature and exploration effort. This component is based upon subjective opinion from an expert.

Deposit size and its grade distribution are information carried in the system as designed. This is done by "decomposing" the endowment (stated in terms of U_3O_8) into deposits which mimic natural appearance. The mining cost model imitates the mine planner's profit maximizing decision in the selection of cutoff grade and capacity of the mining operation. This decision considers deposit size as well as grade distribution, depth, and morphology.

Similarly, exploration outcomes are predicated upon the exploration performances, alluded to in previous comments, which are conditional upon some measure of deposit size. Additionally, the system keeps track of the identities of the deposits discovered and the deposits remaining undiscovered. Thus, the effect of size and the trend of decreasing size as exploration unfolds is explicitly considered.

2.5 Effect of Discovery Depletion

A feature of the real world search process is that it is a process of sampling without replacement. Once discovered, a deposit remains so forever and cannot be rediscovered. As deposits in the endowment are discovered, the likelihood of finding large, rich, near-surface deposits diminishes, giving rise to what is called discovery depletion. The unconditional probability of deposit discovery is proportional to the number of deposits remaining undiscovered. Additionally, as the conditional probability of discovery (conditional on size) is proportional to its size, the larger deposits tend to be discovered first. The consequence of the latter, added to the fact that the number of undiscovered deposits is reduced as more exploration programs are conducted, is that the likelihood of a successful exploration program decreases. In other words, it takes more failures and, hence, more costs to discover a deposit as the region is depleted by previous discoveries.

In this potential supply system, there is an explicit accounting of the discovery depletion of the region. This is accomplished by keeping track, in the computer, of the original number and characteristics of the deposits simulated in the region, the deposits discovered, and the remaining undiscovered deposits. Exploration activities are simulated on the undiscovered portion of the endowment.

In summary, the system designed in this study is parameterized on endowment. Furthermore, key attributes of the endowment are explicitly modeled, including the decomposition of the endowment into deposits of given depth, size, and grade distribution, as well as the state of discovery depletion of the endowment as result of exploration are simulated in the system. This explicit modeling of the endowment allows for the incorporation of its more significant effects on potential supply estimates.

2.6 Effect of the Level of Price/Cost

One aspect in the design of this potential supply system is that it is able to estimate the potential supply for any stated level of price/cost, up to 3 or 4 times previous or current prices. Therefore, special consideration must be given to the effect of the postulated level of price/cost on those decisions taken by firms in the industry in their pursuit of profit maxi-mization that will ultimately affect the creation of new supply. Specif-ically, the level of price/cost affects both mining and exploration decisions.

2.6.1 Effect of Price/Cost on Mining Decisions

The most evident effect of improved price/cost in mining is an increase in the tonnage that can be produced from a given deposit.[1]

In order to determine this effect, one must mimic what the firm --a "typical firm" representing the uranium industry--would do regarding the exploitation of a known, virgin deposit of given size, grade distribution, depth and morphology; that is, how much of the mineralized rock constituting that deposit will be economically translated into resources at that stated price/cost level.

---

1. Since this study is concerned primarily with the estimation of potential supply from unknown endowment, only new deposits are considered.

In this study it is assumed that mine planning decisions are predicated upon a profit maximization objective. By profit is meant the net present value of the mining project: the total of all discounted production cash flows minus the total of all discounted capital costs. It is also assumed that the most significant mine planning decisions are the mining cutoff grade decision and the mine capacity decision. The latter is important because both operating and capital costs depend, among other factors, on the capacity of the operation. Optimization with respect to these mine planning decision variables is considered explicitly in the mining model. No inflation is considered.

2.6.2 Effect of Price/Cost on Exploration Decisions

The foregoing comments on the economics of risk (section 2.3) imply an additional important fact: exploration activities and decisions are predicated upon the specified price/cost. Simply stated, an exploration project that is considered too risky to be conducted when the price/cost level is p_1, may not be so when the price/cost level is p_2 ($p_2 > p_1$). In general, since the rewards from exploration would be higher if price/cost were p_2, the exploration program would be more intensive, ceteris paribus, under price/cost p_2 than under p_1. Consequently, it is not sufficient to base the exploration model upon current exploration costs, efficiencies, and decision rules. For example, an exploration agent may currently state that in this phase of exploration one should drill on fences (relatively closely spaced drill holes on a linear pattern) "x" feet apart. Rather than being reflective of physical discovery alone, these spacing rules are a "mixed bag" of economics, current experiences, and physical influences. In other words, it is contended here that the "rules of thumb" of exploration are a function of economic and physical variables. On that basis, it is desirable to have an exploration model design in which the strategy variables (e.g., number of drill holes or fences) change in response to change in the economics.

To that end, in this study the physical and economic effects are separated. Then, as economic factors such as price/cost are manipulated, exploration strategies change in ways dictated by profit maximizing criteria.

The essence of this disaggregated structure has been discussed in Chapter 1 (section 1.7) above. Inasmuch as the modeling of exploration activities includes an optimization procedure (expected profit maximization) to decide on the optimal exploration effort, it is this feature of the system that defines potential supply in its purest form.

CHAPTER 3

SYSTEM OVERVIEW

This chapter presents a "macro-view" of the potential supply system. The intent is to introduce the reader to the general idea behind the system and its components and the ultimate result desired before he is confronted with the details provided in subsequent chapters. This involves providing highly generalized statements about the flow of the system, functions, and inputs of each of the models, and the interrelation among them.

The potential supply system of this study is conceived as a Monte Carlo simulation model. The system consists of five major components:

- Endowment model
- Creation submodel
- Exploration model
- Executive model
- Mining cost model

Figure 3.1 shows the interrelation of these components within the system. Operation of the system can be summarized in the following major steps:

a. Reading of input data, which include DOE's endowment probability distribution and the boundary of favorable area A_F, which is

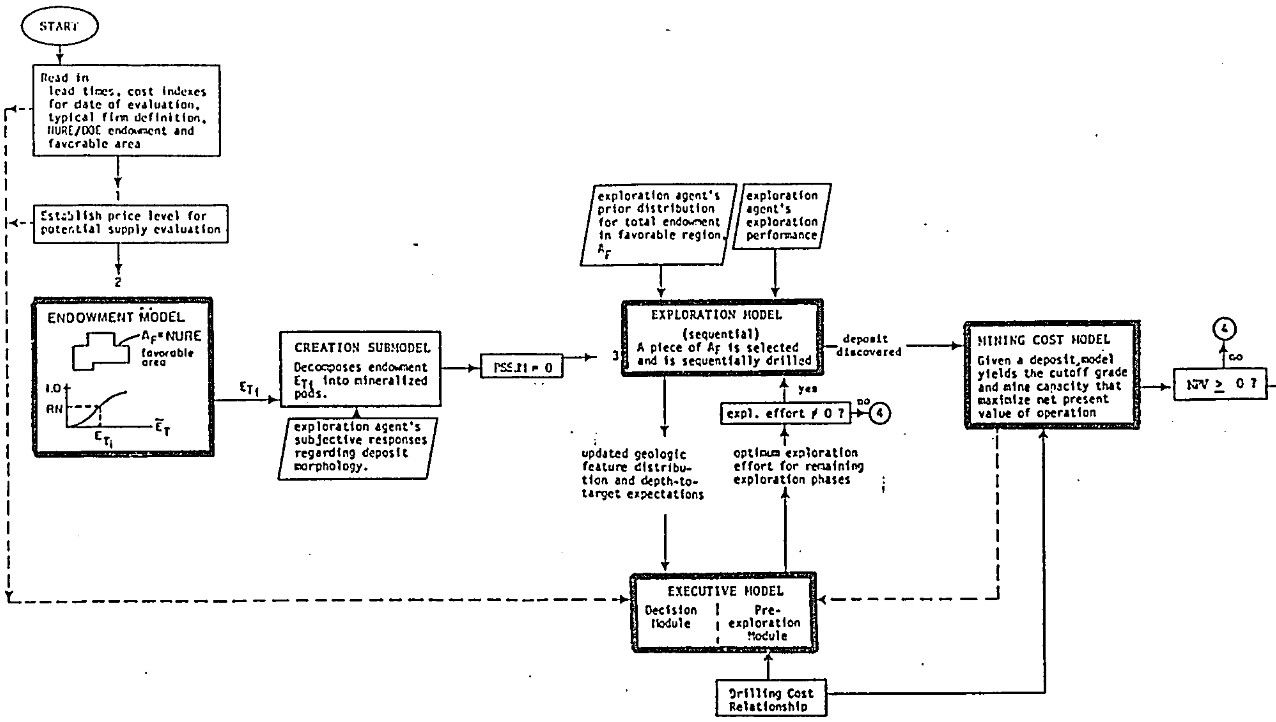
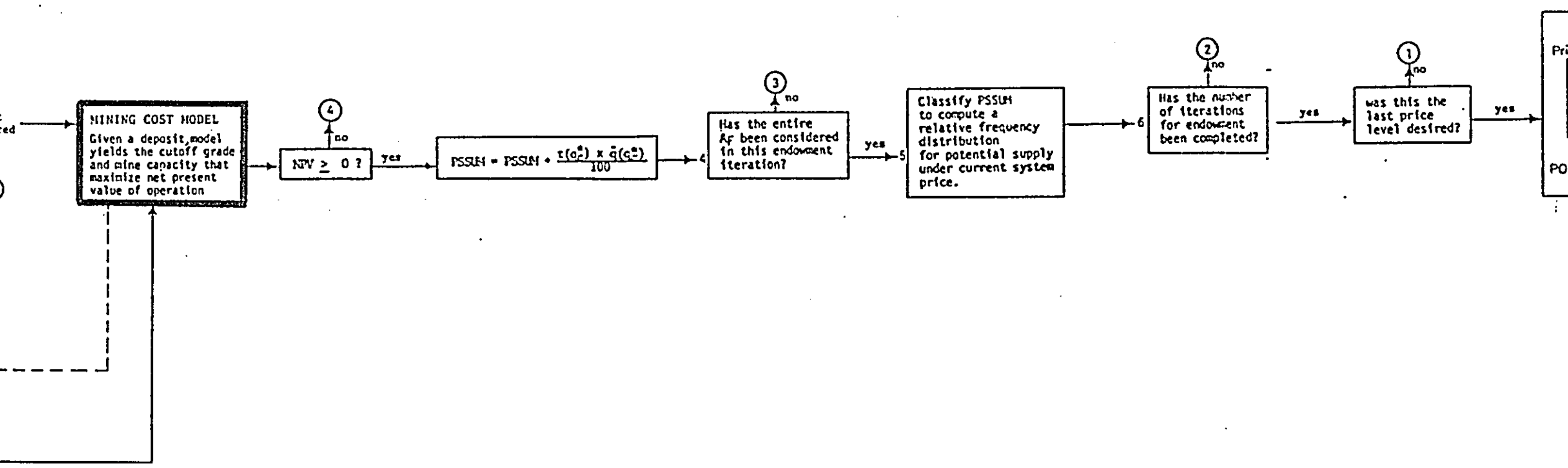
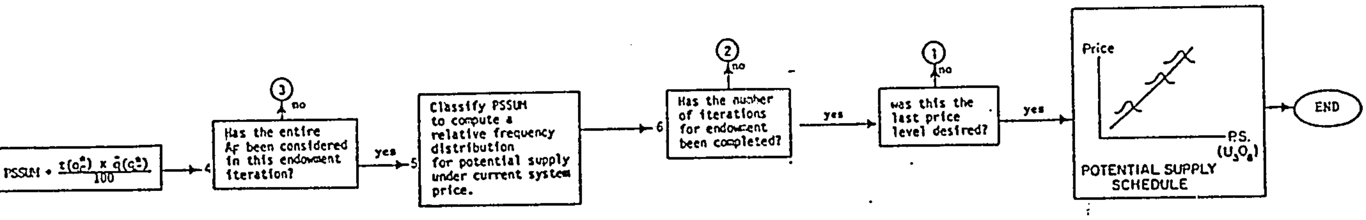


Figure 3.1. Structure of the uranium potential supply system.


MINING COST MODEL
Given a deposit, model yields the cutoff grade and mine capacity that maximize net present value of operation
NPV ≥ 0 ?
no
yes
4
PSSUM = PSSUM + T(c*) x q(c*) / 100
Has the entire Af been considered in this endowment iteration?
3
no
yes
5
Classify PSSUM to compute a relative frequency distribution for potential supply under current system price.
6
Has the number of iterations for endowment been completed?
2
no
yes
was this the last price level desired?
1
no
yes

stem.


PSSUM = t(c*) x q̄(c*) / 100
3
no
Has the entire Af been considered in this endowment iteration?
yes
4
5
Classify PSSUM to compute a relative frequency distribution for potential supply under current system price.
6
2
no
Has the number of iterations for endowment been completed?
yes
1
no
Was this the last price level desired?
yes
Price
P.S. (U3O8)
POTENTIAL SUPPLY SCHEDULE
END

subdivided into cells of a few square miles each, reflecting the typical size of the area selected for formal drilling.

b. Establish a price-cost level regime. This is the only economic variable that is manipulated. Note that the uranium price/cost constitutes the outermost iteration level. That is, the price scenario remains fixed throughout the rest of the system until enough inner iterations are completed. Only then is a new price drawn from a specified price vector.

c. Using Monte Carlo techniques, the endowment model samples the DOE endowment probability distribution and generates a specific endowment value, E_{Ti}. The value E_{Ti} becomes the "true" endowment upon which the rest of the models will act in order to derive the potential supply. A number of these "endowment iterations" will be performed for each price/cost level.[1]

d. The creation submodel decomposes the endowment value, E_{Ti} into mineralized pods and locates them within the subdivisions of the favorable area.

e. These mineralized pods are grouped into deposits. The exploration model selects a cell (which may or may not be barren) and simulates sequential drilling activities. Given the perceived

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1. As indicated in Figure 3.1, there are two major nested iteration levels in the potential supply system. The price/cost of U_3O_8 is the outer iteration level. The DOE endowment distribution sampling is the inner iteration level. When the "endowment iterations" are completed, a relative frequency distribution of potential supply is computed. A new "price/cost iteration" is established and the "endowment iterations" begin again. When all the desired price/cost levels have been examined, the potential supply evaluation of the favorable area (A_F) has been completed.

probability distribution at a given exploration phase about the (true but) unknown state of endowment in the location investigated by the exploration model, the executive model is invoked during every exploration phase to consult if search should continue in that locality. Additionally, it provides the optimal exploration expenditure under the stated price/cost level.

f. When a deposit is discovered, it is evaluated by the mining cost model, which simulates the optimizing mining decision. This model provides the resource tonnage from every discovered deposit that can be economically produced under the given price/cost regime.

g. The resource tonnage computed in (f) is included in the contribution to potential supply. When the totality of the favorable area has been considered for exploration, the potential supply value is classified by U_3O_8 class intervals to compute a relative frequency distribution under the prevailing price/cost. At this point, a new endowment iteration is initiated in (c).

h. When the desired number of endowment iterations is completed, a new price/cost level is established and a new set of endowment iterations is initiated.

The inputs referred to include results provided by DOE endowment reports (endowment probability distribution, favorable area, depths), economic inputs (cost indexes, lead times, definition of the "typical firm"), subjective responses regarding deposit morphology and exploration performance, and other inputs obtained from auxiliary external programs.

Table 3.1 provides a generalized list of the various inputs needed by each model. The inputs are listed under the model that requires them first. Those inputs that need to be provided to the system are indicated by an asterisk. The inputs with no asterisk are generated internally, usually by another model. Greater detail on the particulars of each of the inputs shall be provided throughout the discussion of each of the models in subsequent chapters.

3.1 Endowment Model

The endowment model consists of (1) DOE information regarding the delineation of the favorable area (A_F), (2) the probability distribution for the total undiscovered endowment in the favorable area ($\theta(E_T)$), and (3) the depths of the favorable host. In this model, a specific value for endowment (E_{Ti}) is generated from the distribution ($\theta(E_T)$) by Monte Carlo simulation. The endowment generated in any one interaction becomes the endowment value on which the rest of the models will act upon to derive the potential supply under the pre-established U_3O_8 price.

3.2 Creation Submodel

The creation submodel constitutes the link between the endowment model and the exploration and mining models. The design of the latter models requires the specification of the endowment in terms of physical occurrence of the mineralization provided by this model.

An important input required for the creation submodel consists of either (1) a number of subjective opinions elicited from a respondent

Table 3.1. Generalized list of the inputs required in each of the major components of the potential supply system.

ENDOWMENT MODEL

* probability distribution of endowment reported by DOE, $\theta(E_T)$.

CREATION SUBMODEL

_ A specific endowment value E_{T_i}. This is provided by the endowment model and results from Monte Carlo sampling of $\theta(E_T)$.

* Favorable area (A_F) in which endowment occurs (reported by DOE) subdivided into cells, (few square miles); the size of these subdivisions (called A_3's) performed by the user must reflect the typical size of a drilling area.

* Maximum and minimum depths for each A_F subdivision.

* Average thickness of the favorable lithology.

* Presence/absence of outcrops of the favorable lithology.

* Exploration agent's subjective responses regarding deposit morphology (Appendix E).

* Truncated probability distribution of redox lengths per A_3, $\delta(L|L > 0)$; this distribution is obtained from an auxiliary simulation program, A3GRID (Appendix H)

EXPLORATION MODEL

_ Subdivided favorable area with mineral pods redox length and minimum and maximum depth in each of the non barren A_3 cells; this input is provided by the creation submodel and constitutes the "true" state of nature on which exploration and exploitation activities will be simulated.

* Exploration agent's subjective responses regarding exploration performance (Appendix F).

* Prior distribution of the exploration agent about the true but unknown state of endowment in the favorable area, $\gamma(U_T)$.

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* = indicates an input external to the system.

Table 3.1--Continued

_* Joint distribution of endowment and redox length at the A_3 level, b(U,R); this input is obtained externally employing an auxiliary simulation program, A3SAMP, detailed in Appendix I.

_* Functions for minimum attractive criteria (for minimum deposit grade and width to search for); these functions mimic the explorationist's rules of thumb and are obtained externally employing an auxiliary program, MINATT detailed in section 8.7.4.

MINING COST MODEL

_ Physical description of the deposit to be evaluated (size, grade distribution, depth, morphology); these physical attributes are those generated in the creation submodel.

_ Current level of price/cost in the system.

_* Cost of capital (obtained from the definition of the typical firm and the risk-cost of capital relationships which are external inputs to the executive model).

_* Cost indexes for the desired date of evaluation.

_* Duration of pre-production activities and delays.

EXECUTIVE MODEL

_ Current price/cost level in the system.

_* Definition of the "typical firm" (in terms of risk and financial characteristics) assumed to be conducting the exploration and exploitation activities; this characteristics are obtained by statistical analysis of financial data on a sample of firms.

_* Risk-cost of capital relationships; this equation is obtained externally by statistical analysis of public financial data on a sample of firms.

_ Current exploration phase being conducted.

_ Achievement or lack of achievement of the objectives of the current exploration phase.

_ The probability distribution about the true but unknown relevant state of nature prior to the future search in the region being explored; this input is generated in the exploration model.

_ The exploration effort spent so far in the exploration activities as simulated by the exploration model.

_ Note: The executive model employs the creation submodel and both the exploration and mining models in order to "pre-explore" and analyze the exploration phases remaining to be conducted.

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regarding deposit morphology or (2) appropriate statistics.[1] The questionnaire for the subjective responses is designed to capture the probability distributions of key mineralization attributes (dimensions, tonnages, grades, etc.) for their subsequent Monte Carlo sampling to create mineralized pods. Details on this questionnaire and its use are discussed at length in a later section.

Additionally, the creation submodel receives as inputs the specific total endowment value E_{TI} provided by the endowment model as well as the boundary of the favorable area and its cell subdivisions (whose size reflect the typical size region selected for formal exploration drilling), including the maximum and minimum depth for each subdivision. The model proceeds to "decompose" E_{TI} into mineralized pods of given size, grade distribution, and depth, and locate them in a number of cells.[2] At the end of this model operation, the "true" state of nature in the favorable area has been simulated, and search for this endowment can now begin.

3.3 Exploration Model

The sequential exploration model takes the mineralized pods that constitute the endowment (E_{TI}) that were created by the creation submodel, and groups them into deposits. It proceeds to imitate the search process in coordination with the exploration agent's performance as described by

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1. Because of the unavailability of data from which to estimate the required statistics, this study employed the subjective responses of a consultant.

2. The cells are selected at random but using very special rules described in Chapter 7, sections 7.2.2 and 7.2.3.

the consultant. The executive model is invoked throughout the exploration sequence to control the intensity of search. The ultimate yield from the exploration model can be a discovery (economic or noneconomic) or an abandoned exploration region that may or may not contain a deposit. If the yield is a discovery, it is then passed to the mining cost model to be evaluated and to compute its contribution to the potential supply of the favorable area (A_F). As long as there is any remaining favorable area that has not been considered in the exploration model in the current endowment iteration, the system continues exploring and keeps adding economic discoveries to the potential supply. When all the favorable area has been considered, the total potential supply for that area is generated in the current endowment iteration (PSSUM). The system proceeds then to generate another endowment value ($E_{T l+1}$), and the process is repeated until a desired number of endowment iterations have been completed. When this occurs, the system sets a new price level, the endowment distribution is sampled, and the process described above is repeated to determine the relative frequency distribution of potential supply under this newly established price.

Three major inputs are required by the exploration model:

1. The exploration agent's endowment expectations for the A_F prior to the conduction of any exploration activity.
2. A measure of the performance of the exploration agent in each one of the exploration phases.
3. Mathematical relationships for the minimum attractive exploration criteria.

The prior distribution of endowment (input 1) is used by the system to initiate the sequential exploration process. This distribution is subjected to revision as exploration unfolds within the system. In Chapter 8 it is shown how this input is obtained from the respondent whose exploration activities are being modeled.

The performance measure, input 2 noted above, is a function of the state of nature generated by the creation submodel and the amount of exploration effort dictated by the executive model. The performance measures were obtained from subjective opinion of the respondent whose exploration sequence is being modeled. Full details on this questionnaire and its use in the exploration model will be given in Chapter 8. It is sufficient here to note that performance measures refer to such things as probability of detecting a given target or degree of achievement of a given objective in an exploration phase (e.g., percent error in evaluating an attribute of a geologic target) under given circumstances regarding target size and amount of exploration effort.

The minimum attractive exploration criteria (input 3) are minimum grade and minimum deposit width that the model should search for. They are postulated to be a function of price/cost and depth. The purpose of these relationships is to mimic the explorationist's "rules of thumb" regarding exploration targets, which necessarily depend on economic conditions and expected depths.

3.4 Executive Model

The executive model can be thought of as the financial brain of the system. This model is consulted throughout the course of exploration. It receives information on expectations about purely physical things (size of geologic features, depths to favorable host, etc.) in the geographic entity[1] currently being explored, and transforms it into a form that is adequate for decision making. Considering the economic environment prevailing in the potential supply system, the physical information is transformed (via the pre-exploration module) into the following financial parameters, which ultimately are the basis for decision making: (a) the optimum[2] level of exploration effort[3] for each of the remaining exploration phases and (b) the net present value (including costs of remaining exploration phases) probability distribution associated with the optimum effort in (a). Both the mining cost model and drilling cost relationships used in conjunction with current economic conditions in the system (uranium price, lead times, etc.) are instrumental in this operation of the executive model. Simply stated, the aforementioned financial parameters provide two important pieces of information for the exploration decision (gamble): the cost of gambling

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1. The geographic entity varies according to the current status in the exploration sequence. In early phases it refers to the pieces of favorable area in which drilling is conducted. In later phases it refers to a smaller zone in which a deposit is believed to occur.

2. Maximization of the expected net present value (including the cost of remaining exploration phases) given the current beliefs about the state of nature and current economics in the system.

3. Usually number of drill holes.

and the amount of the potential reward. Here the size of the reward is described as a probability distribution (probabilities for different net present value levels).

Given the net present value probability distribution, the executive model decides (via the decision module) whether or not to continue exploring the geographic entity currently under consideration. Thus, exploration of that particular location can be stopped at any point in the exploration sequence. When this happens, the system selects another geographic entity to explore. When the entire favorable area (A_F) has been considered for exploration, the current value for the running total of potential supply in the area (A_F) is classified in the appropriate U_3O_8 quantity frequency interval, and a new system iteration is initiated.

If the executive model decision is to continue exploring the geographic entity being considered, it passes this decision[1] to the exploration model along with its estimate of the optimum exploration effort for each of the remaining exploration phases. The subsequent exploration activity is performed by the exploration model, and an outcome is generated (in terms of degree of achievement of the objective typical of the exploration phase the system has just conducted). The probability distribution for the pertinent geologic feature[2] in the

1. A positive decision is signaled to the exploration model by non-zero total exploration effort.

2. Depending on the exploration phase, it can be the length of redox front system, the length of the mineralized target, or the quantity of U_3O_8.

geographic entity under consideration is revised and the executive model is consulted again.

It is important to note that in deciding whether or not to continue exploration, the decision module of the executive model takes into account not only the risk of the exploration venture (characterized by the probability distribution of net present values derived from the proposed venture) but also the financial and risk character of the hypothetical (typical) company that conducts the exploration and development. The company is characterized in this system by its current risk level, net worth, number of shares, and degree of diversification. To the extent that diversification here is used for the purpose of capturing the cost of capital and risk position of the typical firm, it should not be confused with the idea of exploration ventures as a separate activity for modeling exploration decisions. The specifics of the decision module are detailed in Chapter 10.

3.5 Mining Cost Model

The mining cost model[1] imitates the mining engineer's decisions in designing an open pit mine so as to maximize the net present value of the operation. Figure 3.1 portrays the interaction of this model with the rest of the components of the potential supply system. Figure 3.2 depicts the main function of the model and required inputs. Given the economic variables and a set of key physical characteristics of the geologic deposit, the model performs a simplified pit design, a cost

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1. In this study, the "mining cost model" also includes milling costs.

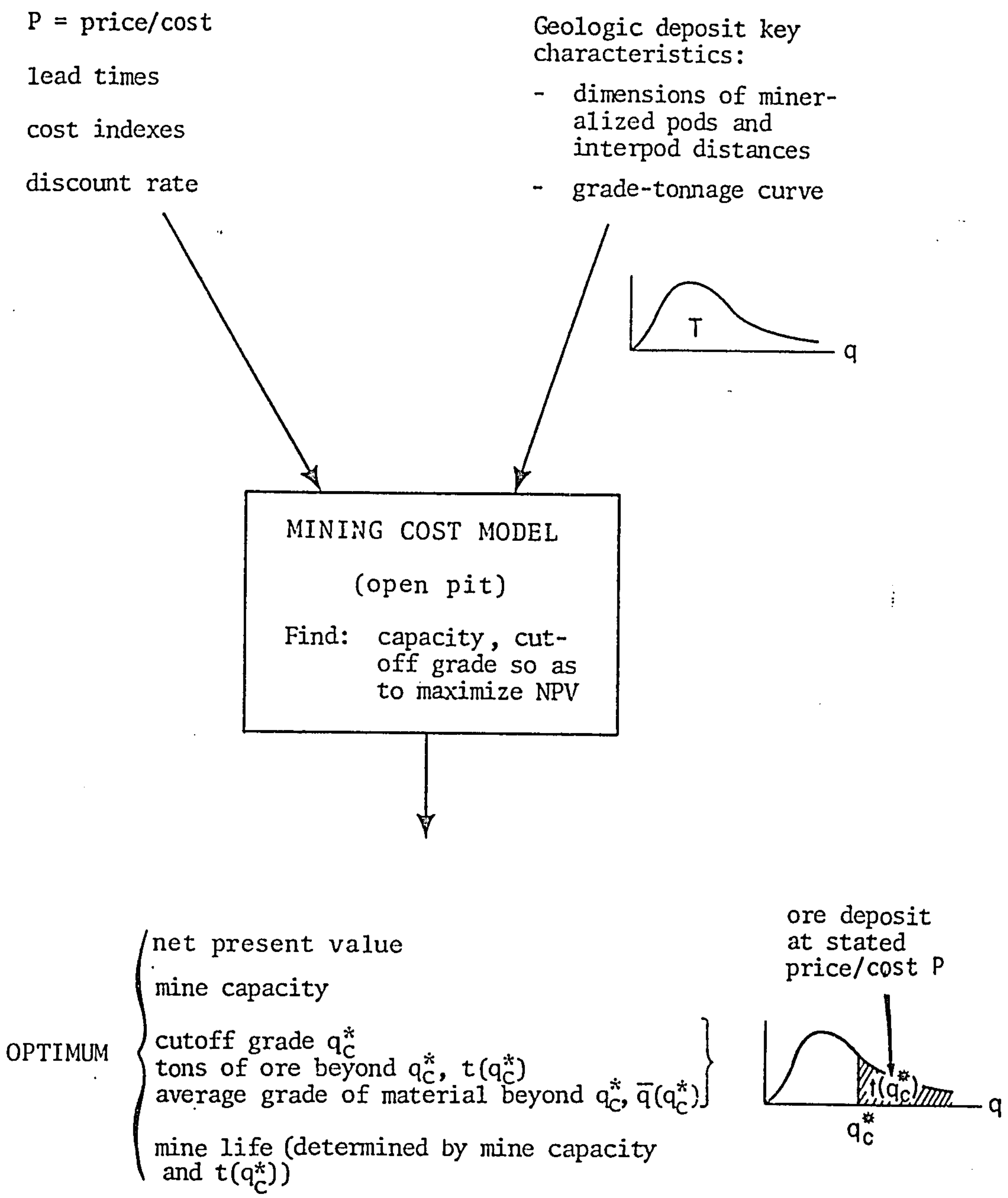

Figure 3.2. Schematic representation of major inputs and main function of the mining cost model.

analysis, an after-tax cash flow analysis, and the discounting of all cash flows and preproduction investment back to the present to compute the net present value of the operation. Simply stated, the model finds that cutoff grade and mine capacity that maximize the net present value. As a byproduct of this optimization, a variety of important measures are produced, including net present value, mine life and tonnage and average grade of material above the optimum cutoff grade.

The role of the mining cost model in the main flow of the potential supply system is to determine that portion of the mineralized material in the deposit just discovered that can be labeled as potential supply. That is:

$$\text{potential supply of deposit discovered} = t(q_c^*) \times \bar{q}\,(q_c^*)/100$$

where

q_c^* = optimum cutoff grade

$t(q_c^*)$ = tonnage of material beyond grades of at least q_c^*

$\bar{q}(q_c^*)$ = average grade of material beyond grades of at least q_c^*

Additionally, the mining cost model performs an auxiliary function for the executive model in the generation of net present value. Simply stated, the physical feature (e.g., length of redox, length of mineralized target, quantity of U_3O_8) expectations that the exploration model passes to the executive model are transformed into deposits. These deposits, in turn, are transformed into monetary expectations by computing their net present value via the mining cost model.

Although implicit in the definition of potential supply, it should be emphasized here that every time the system establishes a new price/cost, the system forgets what happened for any previous prices. The system explores and mines again as if no such activities had occurred at the prior price level. That is, at any new price, the system operates on a virgin endowment. In this regard, the quantity of uranium supplied is a "stock" measure--that is, the quantity of uranium that "could" be supplied if the price/cost were "p."

CHAPTER 4

CHARACTERIZATION OF ROLL FRONT DEPOSITS AND THEIR ENVIRONMENT FOR MODELING PURPOSES

The purpose of this chapter is to provide background information on the rather complex morphology of the roll front type uranium deposits and discuss the conventions used to model this mode of occurrence. One of the main characteristics is the occurrence of these deposits along a linear geologic feature called the oxidation-reduction interface.[1] The analogy has been drawn that roll front deposits occur like beads on a string, the string being the sinuous solution (redox) front marking the boundary between oxidized (altered) and unoxidized (or reduced) sediments. This solution front constitutes an important exploration guide.

4.1 "Stacked" or Multiple Redox Fronts

It is rather unusual in roll type deposit environments to find a single redox interface of tens of miles in length. Typically, the situation is one of multiple individual redox interfaces, usually hard to correlate, and often occurring in a stair-step manner with the uppermost redox interface extending the farthest into the unaltered zone.

For the purpose of potential supply modeling, the resolution needed for this geologic feature is at the aggregate level. That is, it is desirable to refer to the "braided" redox system rather than to the

1. Referred to as "redox front" or "redox interface."

individual redox fronts that constitute such a system. Thus, in this modeling effort, when reference is made to the "length of the redox front system,"[1] this length is that of the braided system.

4.2 Roll Front Deposits

4.2.1 Empirical Observations

Roll front deposits exhibit a very complex morphology. In cross section, the mineralization seldom assumes the ideal characteristics of the theoretical C-shaped roll front. Variations in lithology distort this ideal shape. Additionally, multiple or stacked rolls are common. Figure 4.1 is an idealized cross section of a roll front deposit showing the multiple roll feature and the ideal C-shape in each individual roll. In contrast, Figure 4.2 represents a cross section from a Powder River Basin ore body.

In plan view, the roll front deposits also exhibit a high degree of irregularity. Even in the case of an individual roll, its areal projection is often sinuous. When a set of stacked rolls constituting a multiple roll deposit is projected onto a horizontal plane, the complexity is compounded. Consider, for example, the highly tortuous pattern in Figure 4.3, representing the Grover uranium deposit in Weld County, Colorado. An additional factor that further compounds the irregularity of roll front deposits is the fact that uranium is deposited in a discontinuous series of elongated pods along the redox front system.

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1. Also referred to as "redox length."

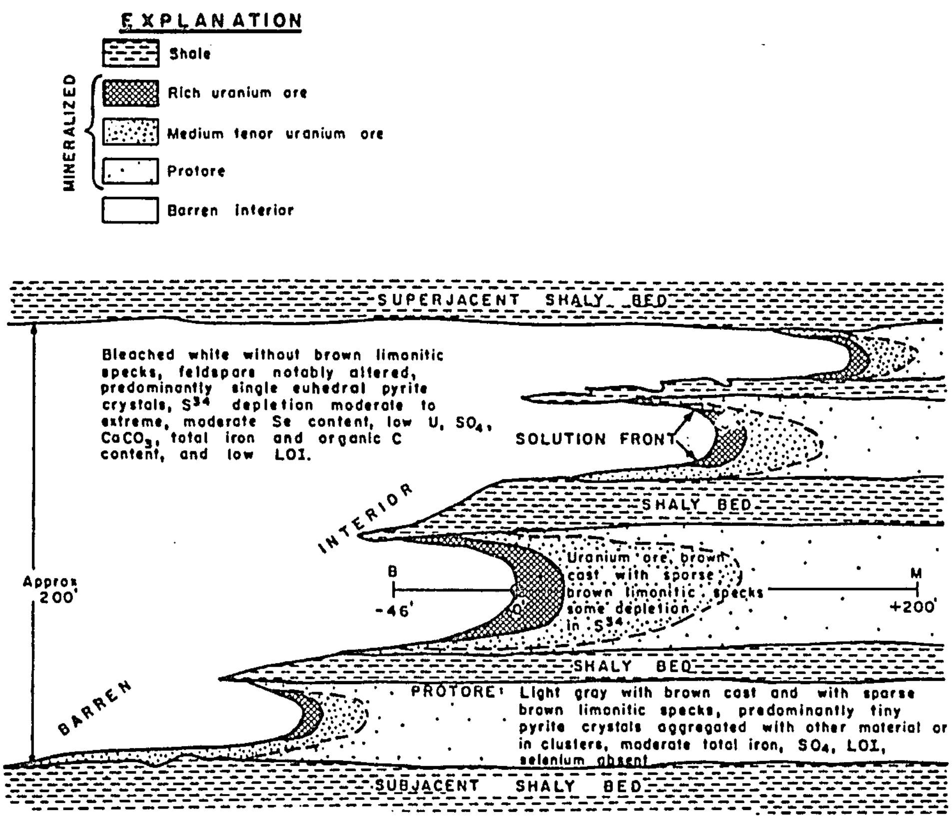

Figure 4.1. Idealized cross section of a multiple roll deposit (source: King and Austin, 1966, p. 74).

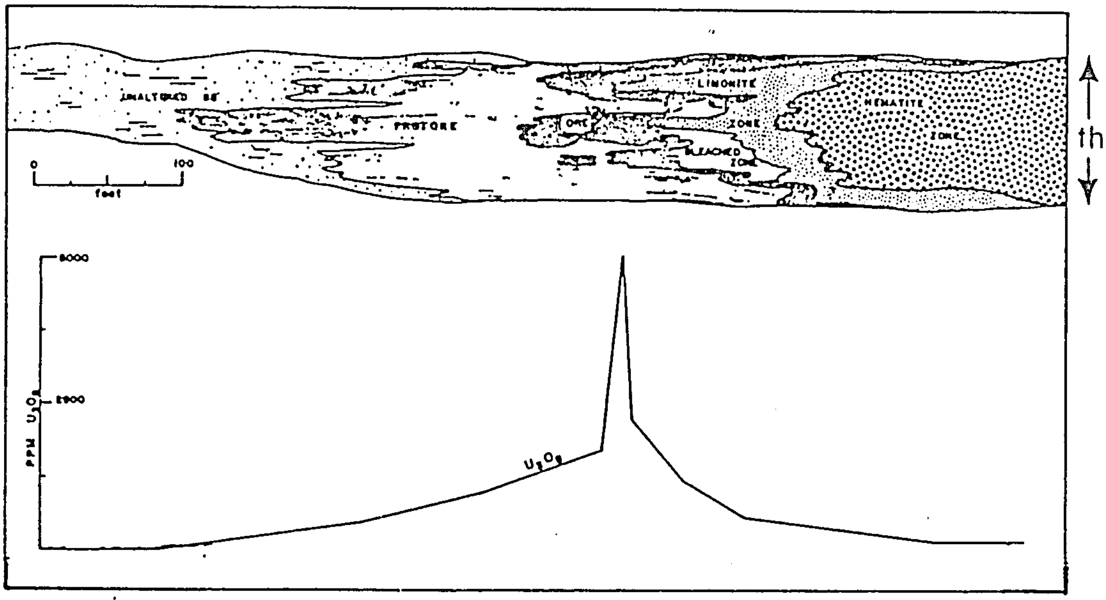

Figure 4.2. Cross section of a multiple roll deposit in Powder River Basin, Wyoming, and an example of the thickness measure (th) used in the model (after Davies, 1969, p. 139).

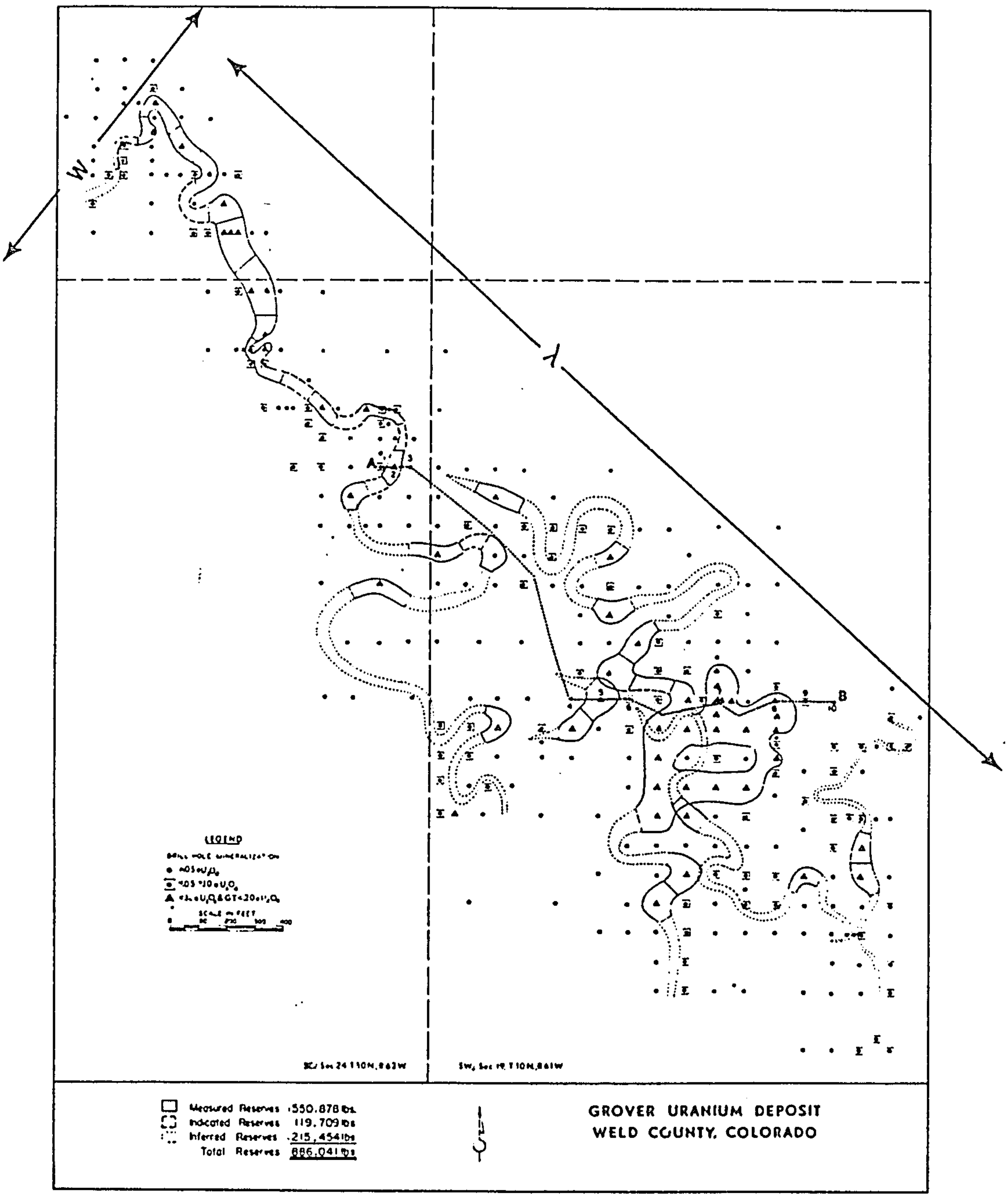

Figure 4.3. Map of the multiple roll Grover deposit in Weld County, Colorado, and an example of the length (λ) and width (w) used in the model (source: Reade, 1976, p. 30).

The above statements deserve further elaboration, for morphology and irregularity are not independent of cutoff grade.[1] It is rather intuitive that the spatial distribution of ore in a deposit, or any volume of mineralized ground, will show more irregularity and discontinuity for a high cutoff grade than it does for a lower grade contour. Unfortunately, published literature dealing with roll front deposit morphology seldom provides this kind of detail. What is sorely needed for exploration modeling is the completion of a study, at least for a sample of a few deposits, on morphology for cutoff grades down to 0.01% U_3O_8, the lower grade bound for DOE endowment definition. However, data necessary for such a study were not available at the time of this research.

Given the evidences in the "bits and pieces" available in published literature, personal communication with industry consultants, and DOE personnel, it can be concluded that roll type deposits indeed present a high degree of complexity at any grade above 0.01% U_3O_8.

4.2.2 The "Confining Prism" Notion: An Artifice to Model Roll Front Deposits

In order to model roll front occurrences (that is, to be able to refer to their dimensions in a simple way), a convenient artifice referred to as the "confining prism" has been adopted.

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1. Cutoff grade in this sense is that grade used to spatially separate mineralized material.

Consider a hypothetical roll front uranium occurrence for which we have available three-dimensional contours for all grades[1] (q) down to 0.01% U_3O_8. Imagine a parallelepiped ($\lambda \cdot w \cdot th$) which "confines" the contours of grade "q" and above.[2] That is, the contours of grade "q" touch the walls of that parallelepiped at least once. We call that parallelepiped the "q-confining prism" of dimensions $\lambda(q) \cdot w(q) \cdot th(q)$. These dimensions are illustrated in Figures 4.1 and 4.3. Note that the dimensions are for exemplification purposes only. The figures pertaining to different deposits and detail on the block grades are unavailable; that is why q was left unspecified.

Since uranium in roll front occurrences is deposited in a discontinuous series of elongated pods, it is convenient for modeling purposes to deal with pods, rather than deposits, as the basic mineralization unit (a deposit being simply a set of one or more pods).

4.2.3 Computation of the Dimensions of a Confining Prism at Different Grade Levels

A confining prism exists only for a grade reference. In general, as grade increases, the confining prism is reduced (at least up to a certain point). Figure 4.4 is a schematic portrayal of this notion. As will become evident in later chapters, the dimensions of the confining prism at a given grade is an important factor in exploration performances

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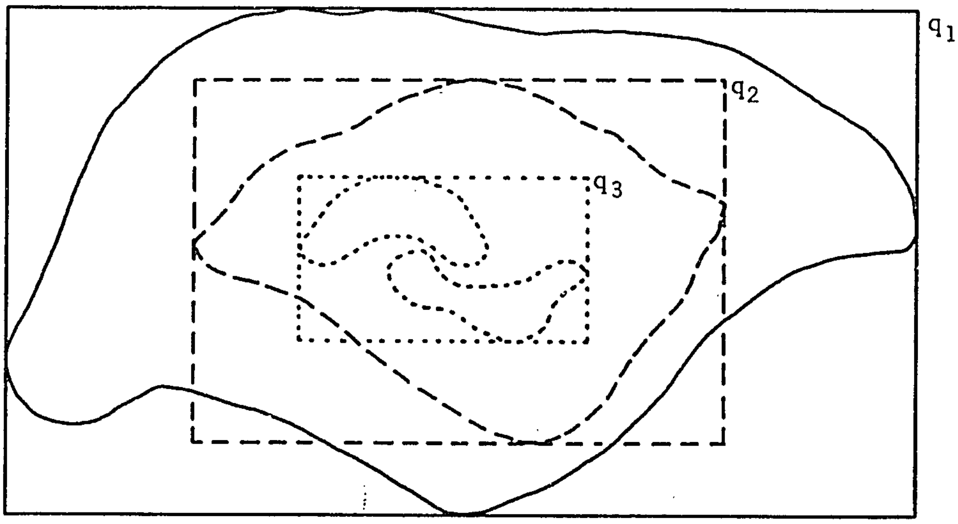
1. In this context, a grade has to be related with respect to a geometric support. In this case, a convenient support can be a block whose size is the smallest mining unit.

2. Length · width · thickness.



q = cutoff grade

$q_3 > q_2 > q_1$

Figure 4.4. Diagram of confining prisms at different cutoff grades (plan view).

and mining costs. Thus, it is necessary to have a procedure to compute the dimensions of the confining prism at any specified grade ($\geq 0.01\%$).

In order to model this changing confining prism simply, it is proposed that in any given pod, the fraction of the 0.01-confining prism volume is a decreasing function of grade:

$$\frac{V(q)}{V(.01)} = VR2(q) \tag{3-1}$$

where

q = any (cutoff) grade ≥ 0.01

$V(q)$ = volume of the q-confining prism

$= \lambda(q) \cdot w(q) \cdot th(q)$

$\lambda(q)$ = length of the q-confining prism

$w(q)$ = width of the q-confining prism

$th(q)$ = thickness of the q-confining prism

VR2 = "shrinkage" function

$VR2'(q) < 0$

Suppose that the following factors are known for a given mineralized pod:

a. the form of the shrinkage function, VR2,

b. length, width, and thickness of the 0.01-confining prism,

c. length, width, and thickness of the q*-confining prism[1] where $q^* > 0.01$.

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1. Note: $\lambda(q^*) \cdot w(q^*) \cdot th(q^*) = V(.01) \cdot VR2(q^*)$, where q^* is a reference cutoff grade higher than 0.01% U_3O_8. (q^* represents the grade used in the elicitation of the Deposit Morphology Questionnaire of Appendix F.)

It is assumed in this system that the dimensions of a confining prism for any cutoff grade $q > 0.01$ can be computed by a nonhomothetic expansion (if $q < q^*$) or contraction (if $q > q^*$) of the inner q^*-confining prism so as to meet the requirement:

$$\lambda(q) \cdot w(q) \cdot th(q) = V(.01) \cdot VR2(q)$$

The mathematical solution for this expansion/contraction procedure is provided in Appendix A.

4.3 The Length of Redox Front System--Endowment Relationship

The perception by the explorationist of a direct relationship between the length of the redox system and the endowment in an area is evident in the roll front exploration architectures described in a later section. Interestingly, in all roll front exploration designs, the redox interface is the primary target in the early drilling stages. The reason is simple. The explorationist searches for a major geologic feature first. Given the location of this feature, he searches for the mineral targets that may occur along that feature. Everything else being equal, the longer the redox front system, the more encouraged the explorationist will be (i.e., the better the chance for a high level of endowment).

An earlier description by DOE (1976, p. 16) of its previous methodology for appraising potential uranium resources further illustrates this dependency:

> Geologic studies of an area of certain Tertiary sedi-mentary rocks in Wyoming indicate that 35 linear miles (N) are potentially favorable for uranium in roll fronts. A comparison of the favorability criteria in a uranium-producing area (the control area) having a similar geologic setting indicates that the area

being appraised is only 40 percent (F) as favorable. It is determined that 70 percent (U) of the area being evaluated is unexplored. Production and reserves from the control area indicate a mineralization factor of 2,000 tons of U_3O_8 per linear mile (T) of roll front. Sub-stituting these factors in the aforementioned formula results in: Potential = 35 · .40 · .70 · 2,000; thus, the potential of the hypothetical area being evaluated would be 19,600 tons of U_3O_8.

The redox length-endowment dependency is incorporated in this potential supply system. This relationship plays a major role in the exploration model. Details on obtaining the degree of dependency between the two variables are reported in the creation submodel chapter (Chapter 7).

CHAPTER 5

EXPLORATION ARCHITECTURE

5.1 Exploration for Roll Fronts Contrasted

Exploration for roll type deposits is sufficiently different from exploration for other types of uranium deposits and deserves special treatment from the modeling point of view. This contrast arises from the difference in mode of occurrence, which has a great impact in the exploration guides. Roll front type deposits occur intermittently along a sinuous, but fairly continuous solution front, marking the boundary between oxidized (altered) and unoxidized (or reduced) sediments (Figure 5.1). Other types of deposits (including the so-called "tabular") tend to be discrete masses with no apparent "fronts" or "backs" that have been recognized by mineralogy, grade or shape, as in the case of rolls.

The main exploration guide for roll type deposits is the sinuous solution front, or system of fronts. It is characteristic in early phases of roll front exploration programs to establish the general location of the redox front along which significant uranium mineralization may or may not occur. Later exploration phases are characterized by intensive mapping and search for deposits along the redox front.

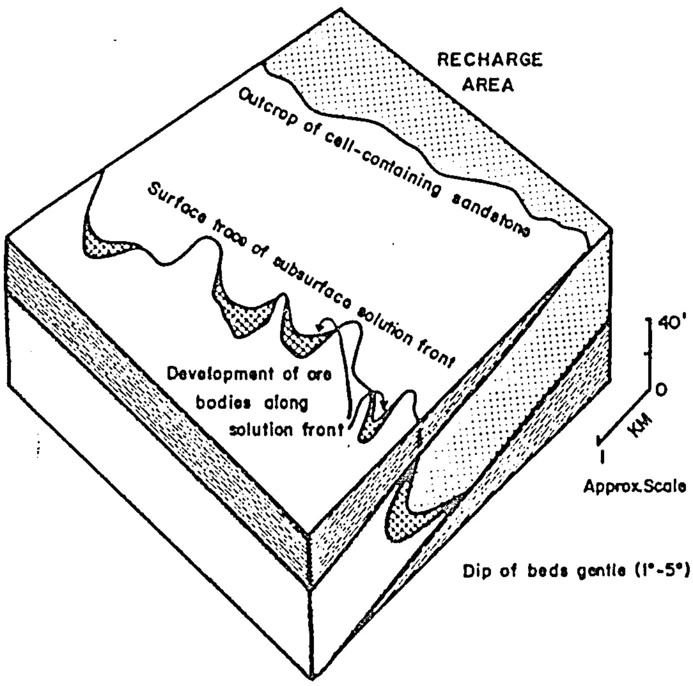

Figure 5.1. Schematic roll type occurrence (source: Campbell and Biddle, 1977).

5.2 Review of Relevant Published Exploration Architectures

5.2.1 OTA General Exploration Architecture for Sandstone Stratiform Uranium Deposits[1]

Exploration architecture refers to an identifiable structure in the sequence of activities and decisions followed in the search for ore deposits. Exploration architecture consists of "phases" or "stages." These stages reflect strategies employed by the firm to reduce the risk of gambler's ruin. Typically, these strategies begin with a larger area, hence reducing to a low level the probability that there is no deposit present, and successively reducing the area as more intensive higher-cost exploration takes place. The "exploration tools" employed converge to drilling, the ultimate and most costly exploration technology.

There is little published material on exploration architecture for uranium deposits. Even more scarce are case studies of a systematic exploration approach.

Although too generic to be used in this detailed modeling of roll front exploration, the work of OTA deserves mention. The OTA's report describes a full sequence of activities; that is, a sequence including exploration, development, and production. The OTA designed a questionnaire around the so-called "Bailly Table" for industries involved in exploration of a wide range of commodities and modes of occurrence, including sandstone and uranium deposits. Table 5.1 illustrates the

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1. Office of Technology Assessment, Congress of the United States. Management of Fuel and Non-fuel Minerals in Federal Land--Current Status and Issues. Washington, D.C., 1979. The exploration and exploitation related portions of the OTA report rely heavily on the work by Paul Bailly.

Table 5.1. Example of Bailly table for a uranium deposit in clastic formation in western United States.

Exploration Stages	Search for a New Uranium Deposit in Clastic Formation of Western U.S.A.
Regional Appraisal (Stage #1)	O-Geologic compilation* O-Photogeologic study (formation mapping, paleogeography), structure*
Detailed Reconnaissance (Stage #2	F-Airborne Radiometric Survey* F-Reconnaissance mapping of formations, facies* F-Drilling for stratigraphy and facies information
Detailed Surface Investigation of Target Area (Stage #3)	F-Detailed radiometric ground survey F-Detailed mapping of outcrops
Detailed Three Dimensional Physical Sampling of Target Area (Stage #4)	F-Drilling* - Logging* L-Mineralogical, Chemical analyses and physical tests on samples, cores and cuttings* F-Down-hole geophysical surveys L-Amenability tests on ore-grade mineralization O-Reserves computations* O-Preliminary Valuation* F-Investigation of water problems and water availability for plants* F-Investigation of suitability of ground for plant, tailings, dump and town sites F-Shaft sinking or tunneling to obtain bulk samples L-Ore dressing bulk tests

At the end of each stage all results are integrated and area of interest redefined.

Legend: O = Office study; F = Field investigation; L = Laboratory tests

* = activity of method which is indispensable

Source: Bailly, "Exploration Methods and Requirements," Table 2-1, in American Institute of Mining Engineers, Surface Mining, Ch. 2 (1968); Office of Technology Assessment, 1979, p. 375.

architecture and main activities recognized by OTA during the four stages of a successful uranium exploration venture. Table 5.2 displays the summary responses for a full sequence (exploration, development, and production) sandstone uranium project. It appears in this table that only one respondent was interviewed. The answer to the "percentage of each activity done by" of 100% medium sized companies[1] appears to be in error; it is unlikely that all these activities are done only by medium-size firms. And it is unlikely for multiple respondents to have concurred in the same error.

The OTA uranium questionnaire is deficient in terms of the specifics needed for detailed modeling of exploration activities and sequential decisions. Only two exploration drilling stages are reported, too few for the sequential drilling process that is needed. In summary, the OTA study provides a good overall picture of uranium exploration and production activity, but it is too aggregated to serve as a framework for our modeling purpose in two senses: (1) it does not differentiate exploration for roll versus other uranium deposit types and (2) it includes too few drilling stages.

5.2.2 Boberg's Article on Roll Front Exploration Architecture

In an article on exploration for uranium in Wyoming, Boberg (1975) describes what he calls a "generalized sequence of exploration." Figure 5.2 is a schematic representation of the drilling stages. A

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1. Medium sized companies as defined by OTA are those companies spending $250,000-$2,500,000 per year on mineral exploitation.

Table 5.2. Summary of mineral exploration and exploitation statistics for sandstone uranium deposits (source: OTA, 1979, p. 405).

	Main Activities and Methods, In Chronological Sequence	O, F, or L	Percentage of Each Activity Done by				Direct Cost ($000) of Each Activity or Method, Excluding Overhead				Area (Sq. Mi.) Being Investigated			Area (Acres) Under Claim, Option, Lease, Etc.			Duration (Months) of Each Stage									Additional Comments
																	Without Any Delays			Plus Non-Regulatory Delays			Plus Delays Due Solely to Regulation			
			I	S	M	L	Min	Avg	Max	Over-head	Min	Avg	Max	Min	Avg	Max	Min	Avg	Max	Min	Avg	Max	Min	Avg	Max	
Stage 1: Regional Appraisal	1 Select Basin	O			100		25	50	100	10		Western U S		0	0	0	3	6	9	0	0	0	0	0	0	
	2 Literature Exam	O			100		25	25	50	5		Western U S		0	0	0	2	4	6	0	0	0	0	0	0	
	3 Field check for proper host rock, alteration & possible mineralization	F			100		25	50	100	10	4K	6K	10K	0	0	0	3	6	12	1	2	3	0	0	0	
	Total						75	125	250	25							6	15	24	1	2	3				
Stage 2: Detailed Reconnaissance	1. Subsurface study of an area by use of gamma & electric logs, map major paleo drainage	O			100		25	25	50	5	2K	4K	6K	0	0	0	3	6	9	0	0	0	0	0	0	
	2 Conduct regional geo-physical programs	F			100		100	350	600	70	2K	4K	6K	0	0	0	6	12	24	1	2	3	1	2	3	
	3 Select best areas from above activities plus what land is available Best geological pros-pect not always available	O			100		25	25	50	5	1K	2K	3K	0	0	0	2	4	6	0	1	2	0	1	2	
	Total						150	400	700	80							9	15	3	1	3	5	1	3	5	
Stage 3: Detailed Surface Investigation of Target Area	1. Examination of area for surface showings of host rock, alteration & mineralization	F			100		25	50	100	10	4K	6K	10K	0	0	0	3	6	12	1	2	3	0	0	0	
	2 Check land for previous activity & possible staking by competitors	O F			100		25	25	50	5	500	1K	2K	0	0	0	1	2	3	0	1	2	0	0	0	
	3 Local geophysical program	F			100		25	50	100	10	200	400	600	0	0	0	3	6	9	0	1	2	0	1	2	
	4 Possible radiometrics geochemical or radon programs	F			100		100	350	600	70	2K	4K	6K	0	0	0	6	12	24	1	2	3	1	2	3	
	5 (Acquisition of land by staking or by negotiation)	F			100		(50)	(200)	(750)	(40)	8	30	150	5K	20K	100K	4	6	12	1	2	3	1	2	3	
	Total						175	475	850	95							6	12	24	1	3	5	1	2	3	

Table 5.2--Continued

	Main Activities and Methods, in Chronological Sequence	O, F, or L	Percentage of Each Activity Done by				Direct Cost ($000) of Each Activity or Method, Excluding Overhead				Area (Sq. Mi.) Being Investigated			Area (Acres) Under Claim, Option, Lease, Etc.			Duration (Months) of Each Stage: Without Any Delays			Plus Non-Regulatory Delays			Plus Delays Due Solely to Regulation			Additional Comments
			I	S	M	L	Min	Avg	Max	Over-head	Min	Avg	Max	Min	Avg	Max	Min	Avg	Max	Min	Avg	Max	Min	Avg	Max	
Stage 4: Detailed Three-Dimensional Physical Sampling of Target Area	1 Some broad-spaced drilling	F			100		50	250	500	50	8	30	150	5K	20K	100K	6	9	12	1	2	3	1	2	3	
	2 Grid Drilling on close-spaced pattern	F			100		500	1K	2.5K	200	2	4	8				24	36	48	2	4	6	1	2	4	
	3 Establish field camp, roads, etc	F			100		25	50	100	10							2	4	6	1	2	3	1	2	3	
	4. Calculation of reserves	O			100		25	50	100	10							2	4	6	1	2	3	0	0	0	
	5. Engineering studies	O F			100		50	100	200	20							6	12	24	2	4	6	1	2	3	
	6. Environmental studies & permits	O F			100		50	200	400	40							6	9	12	1	2	3	6	12	36	
	Total						700	2K	4.4K	330							30	45	72	3	6	12	10	20	50	
Stage 5: Development	1. Continuous drilling	F			100		1K	1.5K	2K	300	4	8	16	5K	20K	100K										
	2. Establish permanent camp				100		2K	4K	6K		2	4	8													
	3 Filing fed & state reports & permits				100		250	500	750											6	12	24	6	12	24	
	4 Mine development				100			32K									Any one of these stages may be delayed by 6 mos. - 2 yrs. by non-regulatory or regulatory reasons									
	5. Mine plant				100			2K																		
	6. Mill Construction				100			15K																		
	7. Transportation				100			5K																		
	Total							60K												6	12	24	6	12	24	
Stage 6: Production	1. Mining				100		12.5K				2	4	8	5K	20K	100K	Lifetime of property									
	2. Milling				100		35K																			
	3 Rehabilitation of Environment				100		5K																			
	Total						53K																			

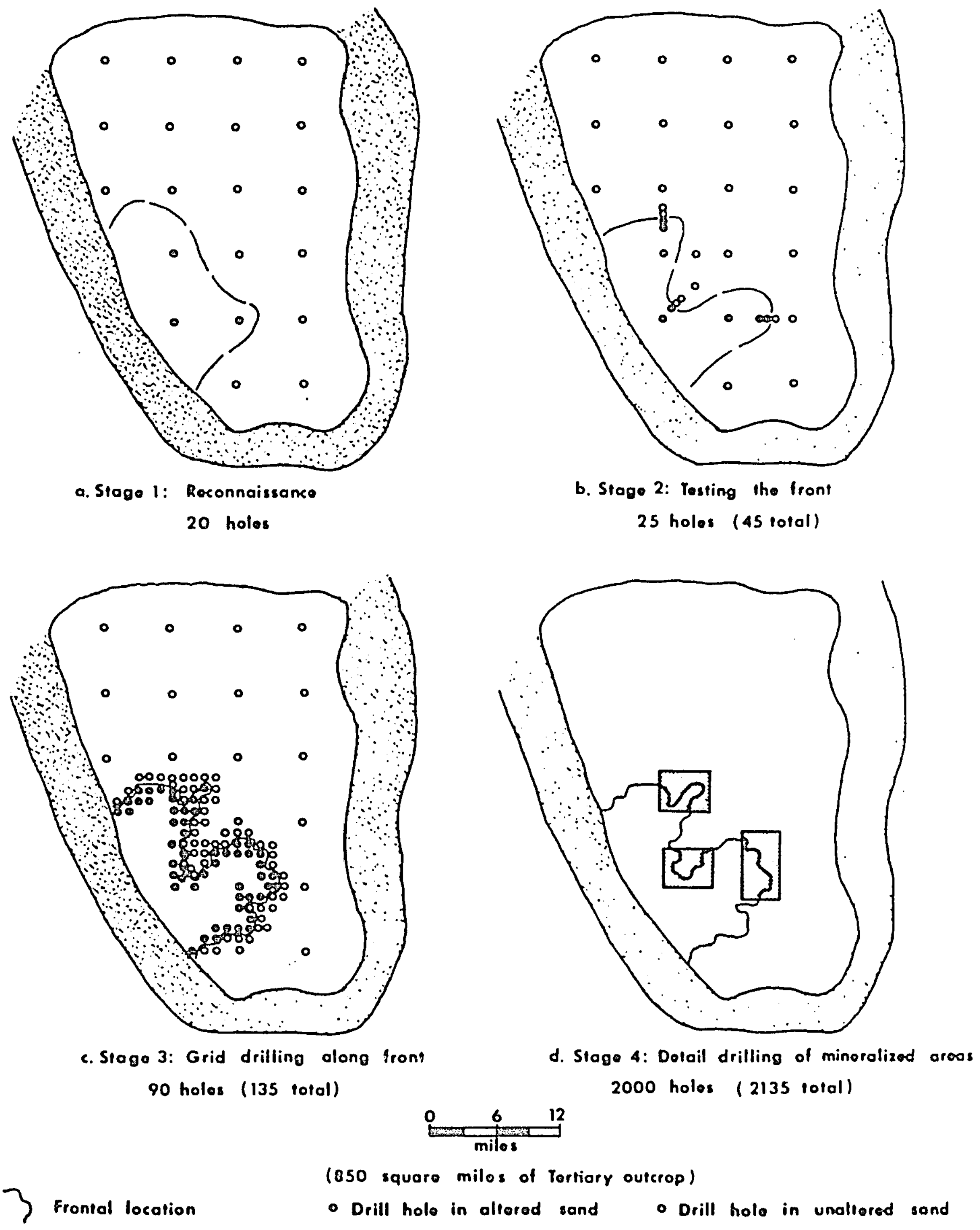

Figure 5.2. Generalized drilling sequence for uranium in a hypothetical tertiary basin in Wyoming (source: Boberg, 1975).

description of the key features, including what we term the "drilling rules" of his sequence, follow:

<u>Stage 0: Regional Appraisal</u>: A region is analyzed for criteria that may be regarded as favorable for potential uranium deposits. The main criteria include known occurrences and airborne or ground radiometric anomalies in stratigraphic units similar to those that are mineralized in other areas.

Techniques which may be used in this stage include:

a. field studies: mapping, geochemical sampling and radiometry.

b. remote sensing techniques: satellite multispectral imagery and color aerial photography

Given that the region appears interesting, a program of land acquisition is initiated upon completion of this stage.

<u>Stage 1: Reconnaissance Drilling</u>: The size of the area covered in this stage is on the order of 760 square miles. The objective is to gather information to allow the explorationist to establish the geologic framework required to properly evaluate the area's potential. Boberg's diagram on the drilling sequence (Figure 5.2) suggests that this "geologic framework" includes detection of the redox front during this stage. He provides "drilling rules" and suggests the initial drilling stage to be conducted on widespaced centers (6 miles or more) on a square grid pattern.

<u>Stage 2: Testing the Redox Front</u>: The purpose is to determine if potential for mineralization occurs along the redox front. To

accomplish this, "fences" (relatively close-spaced holes on a line) are drilled in a few selected areas of the frontal location.

Stage 3: Grid Drilling Along the Redox Front: The goal is to delineate the location of the redox front(s) and obtain data to infer which area or areas appear to have the best potential for ore-grade accumulations. From Figure 5.2, it can be seen that the size of the contiguous area(s) to be selected is on the order of 17-24 square miles.

The area involved in this stage is a "band" along the front. The "drilling rule" of this stage is to drill in a grid pattern on 0.5 to 1 mile centers.

Stage 4: Detail Drilling of Mineralized Areas. The ground covered in this exploration stage is one or more contiguous areas, each 17-24 square miles. The aim of this stage is to outline specific uranium deposits that may occur along the redox front. The drilling strategy is either to drill on a grid or perform "profile drilling" based on the principle of splitting the difference between holes with alteration and without alteration in the intercepted sands until the front location is established.

The next stage corresponds to development.

5.2.3 Bailey and Childer's Frontier Drilling Program for Roll Front Uranium Deposits

In their book, Bailly and Childers (1977) describe what appears to be a typical exploration architecture for uranium deposits, although details about the size of areas involved in the initial phases are not given. The author gives a detailed account, including existing "drilling rules," of the drilling phases in the exploration for roll type deposits.

A prerequisite they establish for a sound exploration program is development of a conceptual deposit model. Once this "concept" has taken form, literature pertinent to the concept is reviewed and exploration begins. The following is a summarized account of their exploration architecture:[1]

Phase 1, Reconnaissance:

A. Regional Studies. The goal of this portion of Phase 1 is to put together a regional picture and thereby formulate the setting for the uranium play. This includes mapping of major structural and stratigraphic features, identification of favorable formations, and the observation of evidence of competitors' activity.

Exploration tools used in these studies include:

- aerial photography
- regional subsurface studies (using existing oil well data)
- landsat multispectral data

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1. Numbering of the phases has been modified to suit the description of exploration sequence.

- airborne geological and strategic reconnaissance
- aeroradiometry (airborne gamma ray survey)

B. Preliminary Land Work. The major objectives of this part of the reconnaissance phase are:

1. To determine if surface evidence is favorable or unfavorable for the primary controls hypothesized (conceptualized) for the occurrence of uranium mineralization.
2. To determine the general potential for minable deposits. For the specific case of roll type deposits exploration, special attention is placed on stratigraphy, facies, unconformities, potential host sandstones, alteration, mineralization, and structures. Tools employed include geochemistry and geologic reconnaissance and mapping.

Phase 2, Detailed Geologic Work. The major purpose of this stage is to evaluate and improve the definition of drilling targets that were broadly or vaguely defined in the prior phase. Activities are aimed at:

1. Defining favorable areas for land acquisition and drilling.
2. Assisting in planning the initial drilling in the next phase.

In this phase the geological, geochemical, and geophysical studies started or carried out in the prior phase are combined. Techniques employed may include aeroradiometry, surface mapping, ground radiometric surveys, radon surveys, helium surveys, and soil geochemistry.

Phase 3, Initial Drilling. The main objective is to determine if the area under consideration has the basic qualities necessary to host minable uranium deposits. Initial drilling consists of "strat tests" designed primarily to locate and define sandstone trends, and to determine where alteration, if any, exists in the sandstone trends.

Once information suggests the location of sandstone trends, subsequent drill holes may be located in fences orthogonal to the projection of the trends. Suggested drill hole spacing is 2 km within fences, and from 5 to 7 km between fences, depending on the size of the target (i.e., the sandstone trend).

Detection of the redox front also is an important objective of this phase. However, the author warns about difficulty sometimes encountered in recognizing alteration. Because of this difficulty, recognition criteria should be established at the very beginning of the exploration program.

Phase 4A, Intermediate Drilling--Tightening of Existing Fences. The goal is to provide the control required to strengthen and confirm stratigraphic correlation within pre-existing fences. This control facilitates mapping of the redox front.

The drilling strategy is to locate additional holes within the pre-existing fences to give a spacing between holes of 150 to 30 m. Drilling is directed to those fences showing the altered/unaltered contrast.

<u>Phase 4B, Intermediate Drilling--Additional Fence Drilling</u>. The purpose is to yield data to confirm correlations between already drilled fences and to improve the reliability of redox front mapping.

Additional fences are drilled on trend with the projected redox front location. Spacing between holes in a fence varies from 150 m to as close as 30 m. Spacing between fences should be about 2 km. At this point, cross sections for each fence are prepared, as well as longitudinal sections tying the fences together. Favorable areas for the next stage are also selected.

<u>Phase 4C, Intermediate Drilling--Progressively Modified Grid</u>. The objective is to obtain a good estimate of the configuration of redox front(s), including determination of their degree of sinuosity, and detection of favorable areas for drilling in the next phase.

Holes are drilled in a "binary search" fashion in the neighborhood of the projected location of the redox interface (see Figure 5.3). As each hole is drilled, correlation and interpretation should be made and the redox interface re-projected. Detail information is recorded with regard to abundance of pyrite and organic carbonaceous material, apparent permeability and porosity in both sides of the front, and any mineralization (even if it is weak).

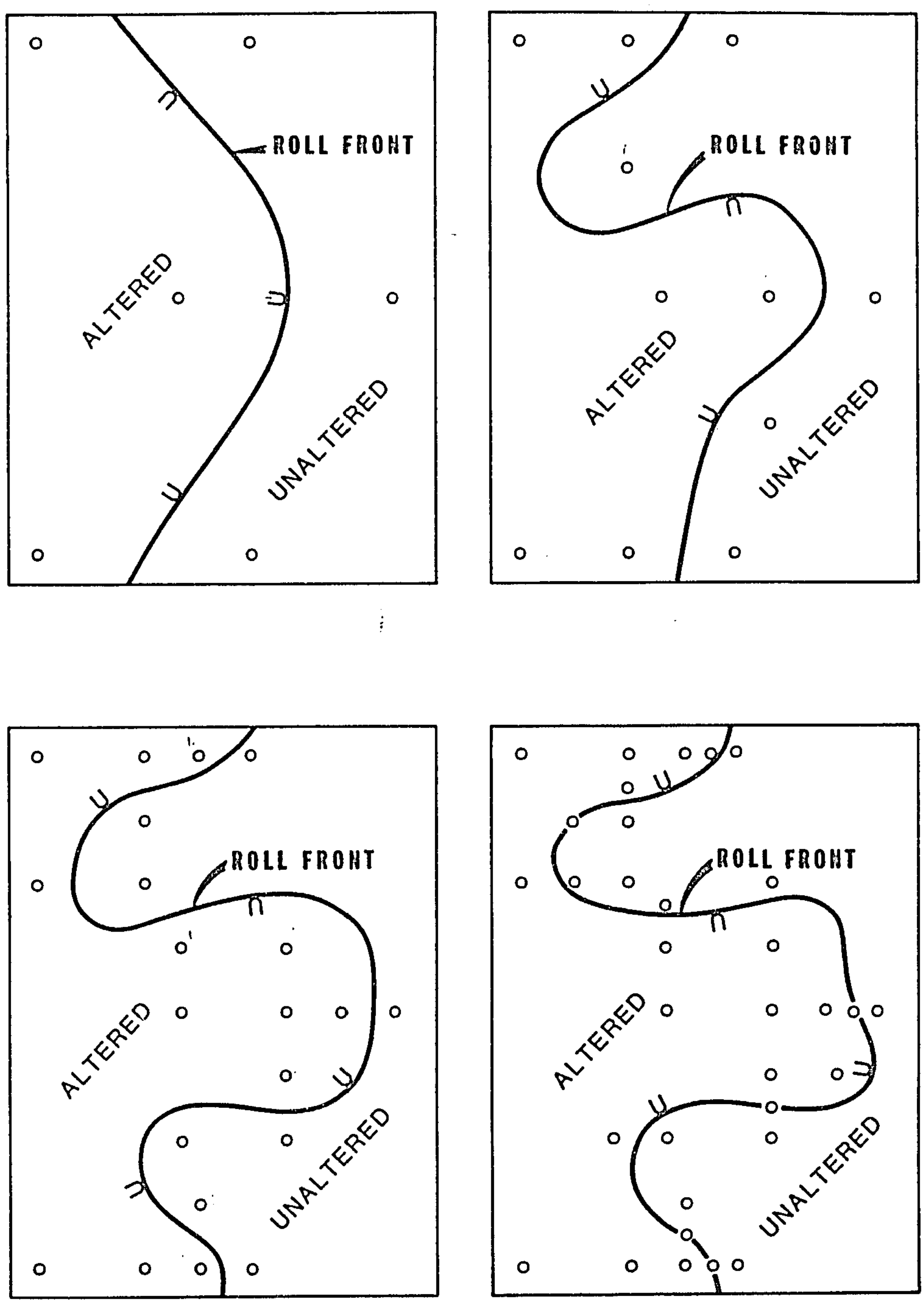

Figure 5.3. Schematic diagram showing the drilling sequence in the progressively modified grid phase (Phase 4C) (source: Bailey and Childers, 1977, p. 457).

<u>Phase 4D, Intermediate Drilling--Profiling</u>. The goal is to show details of the mineralization across the front.

The approach is to drill additional holes in pre-existing fences, particularly in the zone between contrasting holes bracketing the redox interface (see Figure 5.4). Recommended spacing for holes within a fence is approximately 10 m if the depth is less than 120 m. For deeper conditions, a spacing between 10 to 15 m is suggested.

<u>Phase 5, Predevelopment Drilling</u>. Preliminary reserve calculations are performed. It is recommended that sufficient coring be done to determine if a disequilibrium problem exists, and if so, its extent.

Drilling approach and spacing are largely determined by the complexity and sinuosity of the roll fronts. A simple morphology may require fences spaced 100-140 m apart; spacing within a fence may be as close as 8 m. A complex morphology may require closer-spaced drilling or even a dense grid drilling.

5.3 Reflections on Published Exploration Architectures with Respect to Exploration Modeling

The following points are relevant to modeling roll front exploration programs.

a. Drilling is employed, unusually early in the exploration sequence. Thus, an exploration model primarily must be a drilling model.

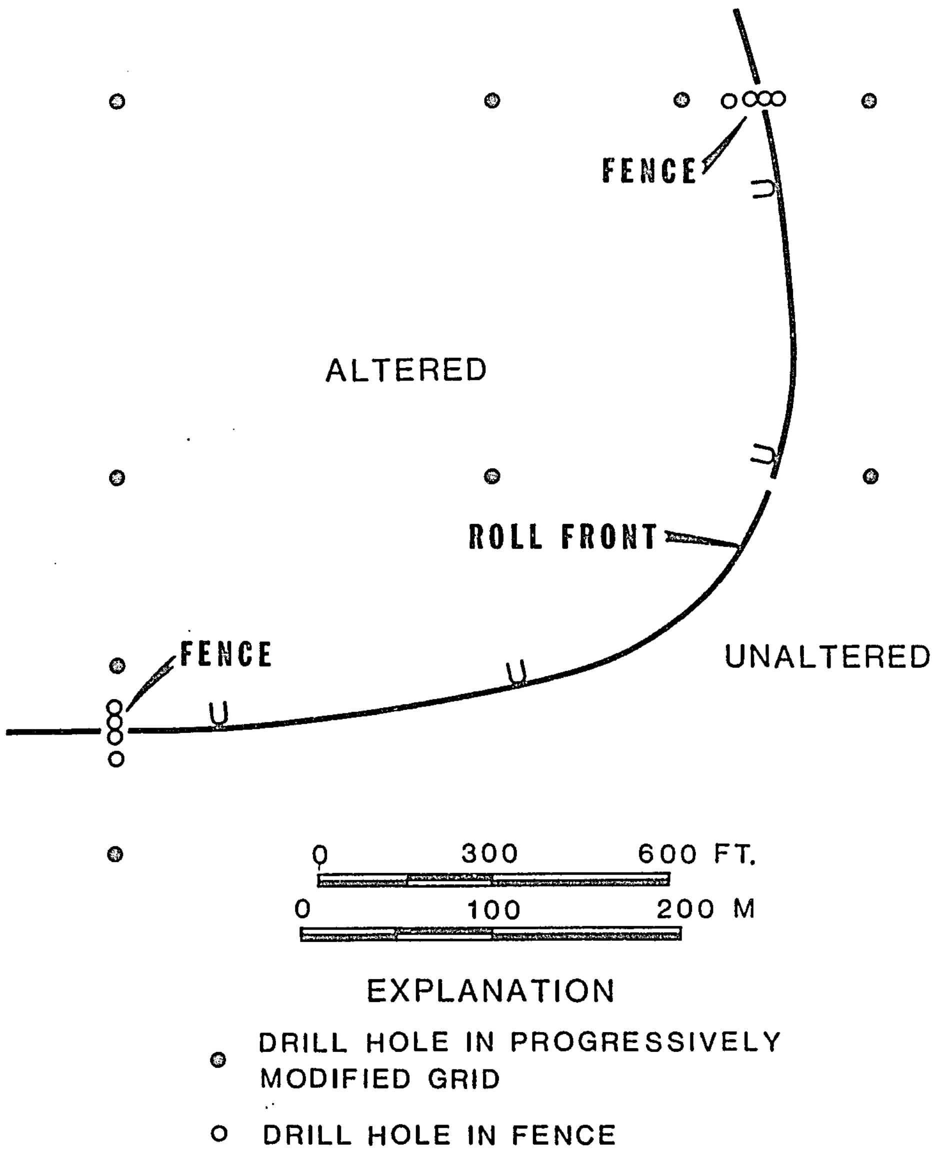

Figure 5.4. Schematic diagram showing drilling activities during the profiling phase (Phase 4D) (source: Bailey and Childers, 1977, p. 458).

b. Upon first examination, pattern drilling could be thought of as an attractive means to explain the exploration drilling process, especially in view of Boberg's description of the exploration sequence for uranium roll front deposits. In the extreme application of pattern drilling, exploration is represented as a process that completely disregards any geology, for holes are drilled in locations determined by the pattern regardless of circumstances or information obtained in the drilling process. Modeling such an approach would indeed be a very manageable task. However, three facts must be pointed out that should prevent us from using a pattern drilling model to explain the exploration process.

1. The regularly spaced drilling process described by Boberg actually results from sequential drilling, thus allowing geologic information to be incorporated into drilling location in the subsequent phase.
2. Not all the drill holes that the pattern calls for are drilled, especially in later phases. The decision whether or not to drill a pattern intersection has to be based on geology.
3. Boberg cautions against a blind pattern drilling procedure and suggests instead a combination of pattern drilling and geological guidance.

Although not very rich in details of drill spacing, guides, and drilling logic, Davies (1973) describes a roll type deposit exploration sequence along the same line as Boberg's, which emphasizes the synergistic character of drilling and geologic analysis.

c. Close examination of the exploration architecture, described by Bailey and Childers (1977), strengthens the case for a sequential exploration model, one that combines pattern drilling and geologic seeding.[1] Its details in terms of the breakdown of the exploration sequence, drill hole spacing and pattern, geologic guides to look for, and logic behind the drilling activities are appealing. Such a model seems to accommodate construction of a sequential exploration model in which not only uranium deposits are simulated, but also the redox front along which deposits occur, and the geologic guides associated with the deposits. One can argue that once deposits and geology are seeded in a digital computer, drilling actions can be simulated using the pattern, spacing, and drilling logic, suggested by Baily and Childers, or for that matter, obtained from subjective opinion from a sample of experienced explorationists.

Three problem areas which would arise in the construction of such a model must be recognized:

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1. The term "geologic seeding" deserves some clarification. Suppose we want to simulate exploration of a piece of the earth's crust. We can synthesize such an exploration region by "breaking up" the crust into a set of small blocks, and "seed" (or assign) in each one of them a set of quantified geologic attributes (redox front, abundance of pyrite and organic material, porosity, permeability, percent uranium mineralization, etc.). Block size needs to be small enough to allow satisfactory resolution.

1. Drill hole spacing is a function of economics; consequently, spacing described by an explorationist reflects the resolution of uranium price/cost (often unspecified), drilling cost, and production cost. Consider the following illustration: assuming that the endowment perception of the exploration agent in a region remained invariant, one would obtain a different hole spacing "rule" if the price of uranium is $30/lb. than if the price is $80/lb. Alternatively, think of this same constant endowment perception in two regions of different depth: shallow (100'), and deep (1000'). If uranium price remains constant, and assuming both regions are explored, spacing "rules" will call for closer holes in the shallow region. Simply stated, the better the potential reward, the more effort will be spent to discover it.

2. One way to overcome this problem with spacing "rules" could be to disregard drill hole spacing information, and instead, attempt to capture the "logic" behind the drilling process. Conceivably, if one knew the logic behind a drilling program (what geologic features to look for, what drilling pattern to use, etc.), one can "superimpose" in the computer system the economics that indicate the spacing of the pattern proposed. Explorationist expectations could be modeled and modified in the exploration sequence, as geologic features

are encountered during the simulation of his drilling actions.

The above approach relies heavily on modeling the relevant geologic guides. Even if the "drilling logic" of an exploration agent being modeled could be captured by an appropriately designed questionnaire, geologic seeding presents complex problems.

3. Seeding relevant geologic guides for this potential supply system is a very difficult task. For one thing, it would be necessary to seed relevant geologic guides for each of the NURE regions believed to contain roll front type deposits. The problem of associating relevant geologic guides with endowment (i.e., deposits) is very complex. Consider the following illustration. Assume that an area is being modeled in which one deposit (D) is being seeded; these types of deposits usually have a geologic feature (F) associated with them. If both variables were of the binary type, for this modeling illustration to be complete, it would be necessary to know:

   $\text{Prob}(F|D)$ = how often feature F is associated to an ore deposit

   $\text{Prob}(F|\bar{D})$ = how often feature F is a "solo" feature

These relations are conceptually acceptable, although the latter of these probabilities is rather difficult for geologists to estimate. Things, however, are not as simple as they appear.

Variables F and D come in a number of sizes (an infinite number, as a matter of fact) because both variables are continuous. Their size, especially their areal extent, has a substantial effect upon exploration performance. The smaller the areal extent, the less likely they will be encountered in the course of exploration. The fact that multiple geologic guides with interdependence among them is the rule rather than the exception further compounds difficulties in seeding the necessary information. Simply stated, a seeding model that permits the detailed imitation of the geologist's analysis and decisions in a comprehensive manner requires seeding the (1) geologic processes and their relations to the uranium deposits (endowment), (2) geologic features, and (3) appropriate spatial location of deposits with respect to processes and observable geologic features. Furthermore, all these relations would have to be modeled statistically; i.e, using conditional and marginal probability distributions.

The insufficiency of available data prevents construction of such a model.

d. A better approach appears to be design of an exploration model in which exploration performance (behavior) is modeled, leaving implicit in this performance and in the magnitude of estimated endowment for the region,[1] geoscience, geologic information, auxiliary geologic guides, and the link between drilling logic

1. Endowment is taken as given in this model; i.e., it is an input to the model.

and geologic information. This approach has been adopted in the exploration model of this potential supply system. More specifically, an exploration expert was used to obtain a detailed description of exploration architecture. This description included identification, for each stage of exploration, its goal and the activity employed to achieve it. As indicated earlier in this section, existing descriptions of exploration architecture were not considered to be sufficiently detailed. Additionally, the expert (respondent) provided for each stage of exploration performance criteria and actions to be taken in response to these criteria. In this study, both architecture of exploration and performance criteria were subjectively derived. Specifics of the model are discussed at length in Chapter 8.

5.4 Exploration Architecture for Exploration Modeling and The Role of Subjective Opinion

A respondent with experience in roll front deposits exploration was selected to cooperate in the design of the model questionnaire. Criteria for selecting the respondent included: experience in roll front type exploration, availability, and a desire to cooperate in a rather unconventional and mind-stretching exercise for an exploration geologist.

Survey of the explorationist consisted of two major phases:

I. Introduction and Exploration Architecture Questionnaires:

1. Introduction to the potential supply system, objectives, and philosophy.

2. Description by the respondent of his exploration architecture for a typical full sequence exploration program. Special emphasis was placed on the specific objectives of each exploration phase, size of areas involved, and information obtained upon completion of the phase (Appendices B and C).

3. Restatement of the exploration architecture for modeling suitability. This part required substantial interaction between the respondent and the author (Appendix D).

II. Formal Model Questionnaires:

1. Subjective Opinion on Roll Front Deposit Morphology (Appendix E);

2. Subjective Opinion on Exploration Performance (Appendix F).

The architecture of a typical full sequence exploration program as described by the respondent is reported in Appendix B. Appendix C shows, in schematic form, the area reduction process in early phases of his exploration program.

Restatement of his original exploration program is shown in Appendix D. This required a great deal of interaction between the author and the respondent to maintain adequate balance between the essence of his original exploration program and modeling needs. Major objectives were restated, and Phases 3 and 4 were each broken down into two parts. Part A corresponds to the <u>search</u> for the target (redox front in Phase 3, uranium deposit in Phase 4), and Part B pertains to <u>evaluation</u> of the target's physical characteristics (length in the case of the redox front, size and grade in the case of the deposit).

The restated exploration architecture in Appendix D constitutes the "final" exploration program and is the basis for the exploration model. The subjective Opinion on Exploration Performance questionnaire (Appendix F) is coordinated with this exploration program.

CHAPTER 6

ENDOWMENT MODEL: THE DOE--POTENTIAL SUPPLY SYSTEM INTERFACE

The potential supply system is designed to work on an individual basis for each DOE favorable area. While the overall system design applies to all modes of occurrence, the exploration model described by the explorationist is specific to roll front type deposits. The undiscovered U_3O_8 endowment of a favorable area has been described by DOE by a probability distribution, which is an input to this potential supply system. The potential supply system explores for this probabilistically described endowment and produces a probabilistic potential supply schedule.

Since the objective of the system is the evaluation of potential uranium supply using the DOE endowment estimates as a basis, it is reasonable to assume in the system that exploration starts in Phase 2 of the exploration model's architecture (see Appendices C and D) and the permissive area "F" delineated in this phase correspond to the favorable area delineated by NURE (A_F). That is, the design of the potential supply system was based upon the assumption that the exploration program starts in Phase 2 of the sequence and the perceived permissive area coincides with the area delineated by NURE as favorable for uranium mineralization (A_F).

There are six items provided by the DOE's National Uranium Resource Evaluation (NURE) quadrangle reports and assessments[1] that are important to this potential supply system:

a. The boundary of the favorable area (A_F) defined by DOE.

b. The probability distribution of the endowment[2] existing in the favorable area [$\theta(E_T)$].

c. Contour map of the favorable area (A_F).

d. Structural contours for the favorable host in the DOE area (A_F).

e. Average thickness (τ) of the favorable host in the DOE area (A_F).

f. Presence/absence of outcrops of the favorable lithology.

The six items enumerated above serve to derive the following inputs to the computer system:

1. The area delineated by DOE as favorable [A_F --item (a)] is subdivided into cells of size and shape equal to the typical drilling region, A_3, of Phase 3A (see Appendices C and D) according to the explorationist's responses.

   The subdivision can be made with the aid of a transparent overlay having a grid size of A_3. The overlay is then positioned to match, as close as possible, the favorable area boundary as defined by DOE. Figure 6.1 illustrates this input. In general, as the size of A_F increases relative to the size of the A_3

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1. See Amaral (1979) for an account of items a, c, d, e, and f.

2. Endowment is occurring in roll type deposits.

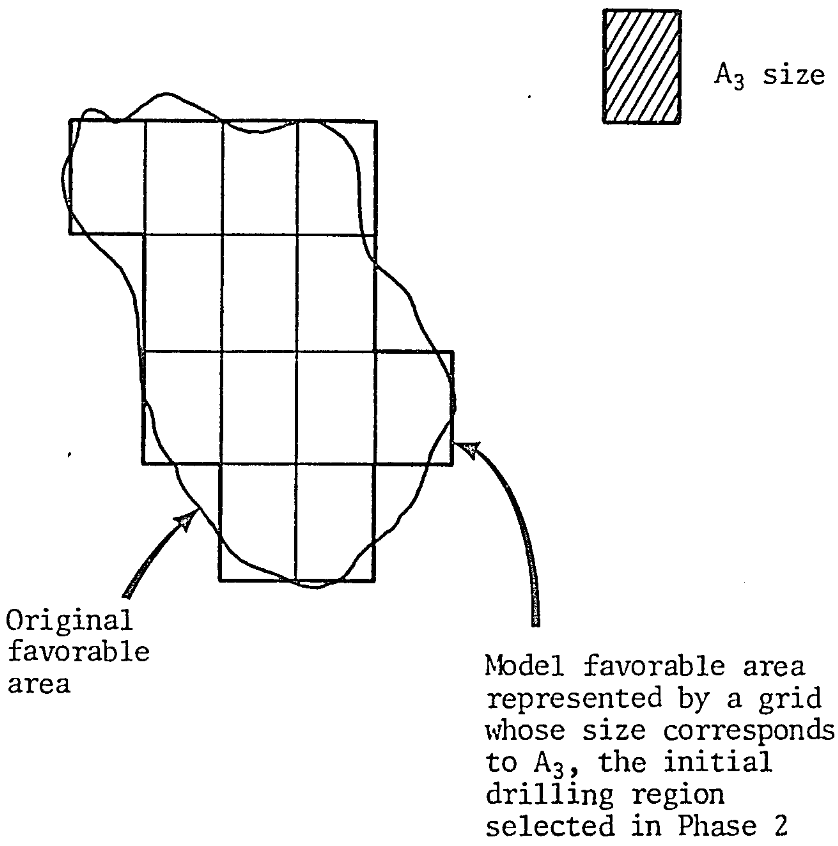

Figure 6.1. Deriving the model favorable area using a grid of size A_3.

forming the grid, the mismatch tends to cancel out, making the error due to this approximation negligible.

2. The contour map and the structural contours of the favorable host [items (c) and (d)] are coordinated to produce, for each cell defined by the A_3 grid, a minimum and maximum depth to the favorable host. These values become the parameters of a depth distribution (uniform) in the potential supply system. Drilling depths and deposit depths will be generated from this probability distribution.

3. Parameters or moments of the endowment probability distribution [$\theta(E_T)$] of the favorable area [item (b)].

4. Average thickness of the favorable host (τ) in the favorable area [item (e)].

5. Presence/absence of outcrops of the favorable lithology [item (f)].

The major operation occurring in the endowment model is shown in Figure G.1, Appendix G[1] and consists of drawing a specific value for

---

1. Appendix G shows the integration of all the models into the potential supply system. Computer operations of the creation and exploration models are detailed in that appendix, along with the linkages with the mining cost model and the executive model; however, the computer operations of these two models are detailed in separate appendices (J and K).

endowment (E_{Ti}) from the NURE/DOE endowment probability distribution[1] $\theta(E_T)$. This is accomplished by using Monte Carlo simulation techniques.

1. DOE has been using Pearson family distributions to describe the endowment probability distributions in the favorable areas. Routines developed by the Oak Ridge group should prove useful for the Monte Carlo sampling design.

CHAPTER 7

CREATION SUBMODEL

7.1 Overview

The functioning of the potential supply system implies the iteration on the endowment probability distribution reported by DOE, $\theta(E_T)$. In every endowment iteration, a specific value (E_{Ti}) is drawn from the probability distribution $\theta(E_T)$ via Monte Carlo simulation. The value E_{Ti} becomes the endowment value upon which the rest of the model will act in order to derive the potential supply in the favorable region, A_F.

The role of the creation submodel is to generate, from the specific endowment value E_{Ti} provided by the endowment model, a "state of nature" in the A_F on which exploration and mining activities can be simulated. The creation submodel performs three major operations (see Figure 7.1):

1. Sampling of redox length-endowment relationship. The redox front system along which deposits may occur plays an important role in exploration, as evidenced by the exploration architectures formerly described. Thus, it is necessary to generate this geologic feature in the model. Actually, length of the braided system is the only attribute required. The conditional distribution for redox system length, given the endowment E_{Ti}, is sampled by Monte Carlo simulation techniques. Thus, a specific value for redox system length (L_{Ti}) is generated.

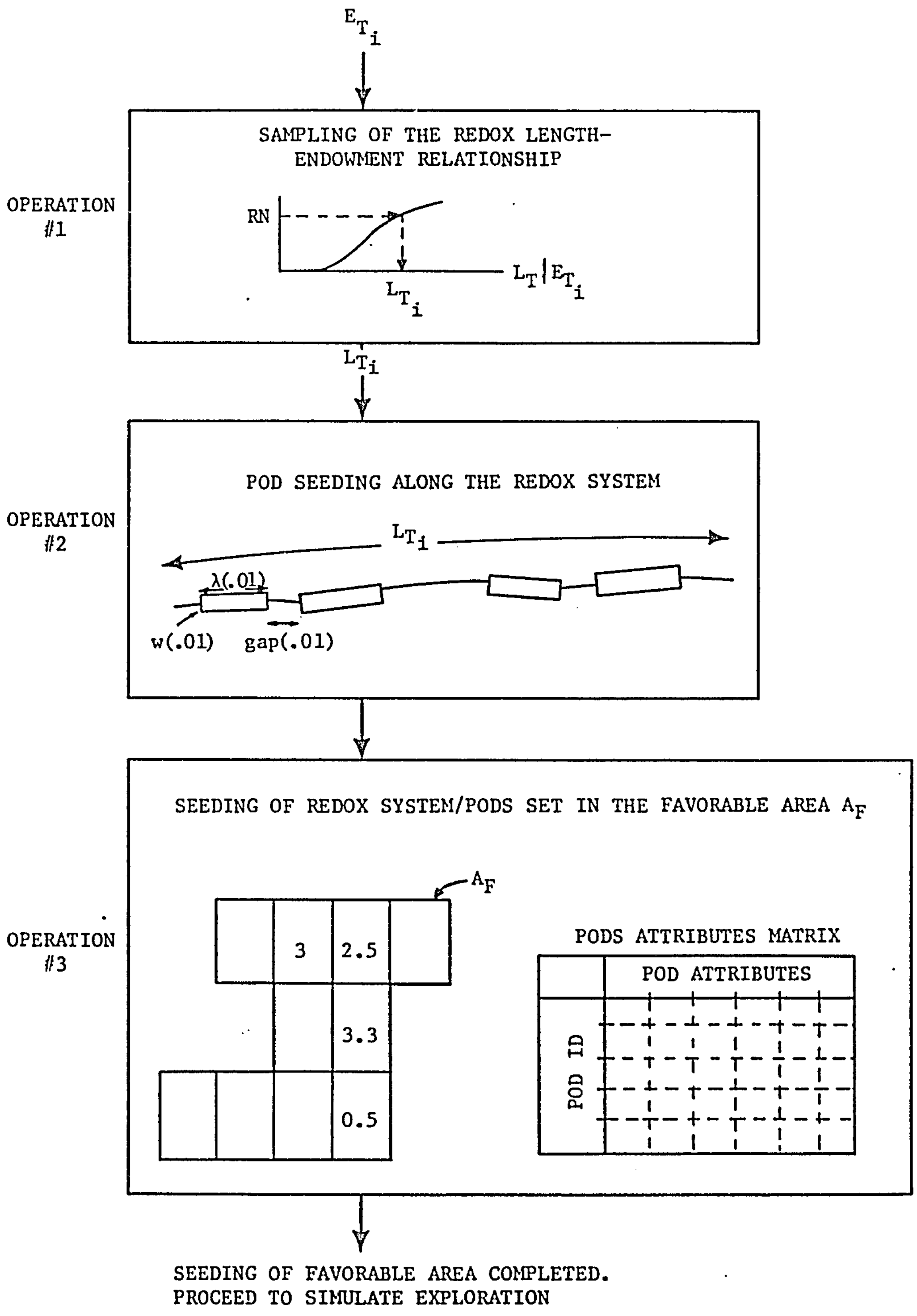

Figure 7.1. Schematic flow chart showing the three major operations of the creation submodel.

2. Pod seeding along the redox system.[1] Endowment is stated in terms of total U_3O_8 content. In order to simulate exploration and mining on the endowment E_{Ti}, it is necessary to "decompose" it into geologic pods defined by a 0.01% U_3O_8 cutoff grade (to be consistent with DOE's lower grade bound for endowment definition). A pod is defined by the following attributes:

- dimensions of the 0.01-confining prism.
- dimensions of the inner q*-confining prism (q* > .01 % U_3O_8).
- grade tonnage curve describing grades distribution in the pod.
- "shrinkage function" describing the ratio of the volume of the confining prism at any grade q > 0.01 to the volume of the 0.01-confining prism.
- depth to the top of the 0.01-confining prism.
- distance to the next 0.01 pod along the redox system (gap).

The necessary attributes to define a pod are generated until a sufficient number of pods have been created to exhaust the value of the total endowment E_{Ti} in the favorable area.

1. The term seeding in this context refers to the generation of the attributes of interest of the pods and the placing of these pods along the redox front system.

3. Seeding of the redox system/pods set in the favorable area.[1] The redox system may occur virtually anywhere in the favorable area (A_F) Its location is unknown but important because the area (A_F) is not uniform in depth to the favorable host.

The location of the redox system is treated as a random variable. Therefore, it is necessary to randomly locate the redox system and its accompanying "chain" of geologic pods in the favorable area (A_F).

Given the total length of redox system (L_{Tl}) along which uranium pods occur, segments of L_{Tl} are assigned to selected cells of A_3 size until the total length (L_{Tl}) is exhausted. Information on the preceding sequence of these selected A3's is stored. In Figure 7.1 (operation #3), the numbers inside the A_3 cells exemplify the redox length (L) in miles in each one of the cells.

Note that for each A_3, it is necessary to store only the redox system length in that particular A_3 and the attributes of the pods occurring along that redox system length. That is, the resolution of the system is at the A_3 level. The exact location of geologic objects inside the A_3's is not needed in this design.

7.2 Modeling Details

7.2.1 Operation 1: Sampling of the Redox Length-Endowment Relationship

"Hard data"-based studies would prove very useful in establishing this relationship. However, these studies require a massive effort

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1. Seeding in this context refers to locating a portion of the length of the redox system/pods "chain" in a specific A_3.

to assemble scattered drill hole data on mining properties and the availability of very sensitive data. These conditions precluded conduct of such a study. Instead, we adopted the alternative of developing this relationship via subjective opinion.

Figure G.2 (Appendix G) shows this operation in the potential supply system.

To obtain a statement of this relationship, question SM.8 of the Subjective Opinion on Deposit Morphology questionnaire (Appendix E) was posed to the respondent. Not surprisingly, his responses show a direct relationship between redox system length and endowment. That is, larger endowments imply higher probabilities of exceeding any stated redox system length. Had dependency been absent, his response would have been the same for any endowment level.

Three parameter lognormal distributions (μ, σ, d)[1] appear to represent reasonably well the data in question SM.8 (Appendix E) for every endowment level. Distributions of this type were fitted to the data using a numerical minimization routine that searches for those lognormal parameters[2] that minimized the following measure:

$$F = \sum_{i=1}^{4} w_i \quad 1 - \left(\frac{P(L \geq L_i)}{\hat{P}\ (L \geq L_i)} \right)^2$$

---

1. μ = logarithmic mean; $\sigma 2$ logarithmic variance; d = displacement.

2. The third parameter, d, was set equal to the "minimum length" required, stated in column A of question SM.8 (Appendix E).

where

L = redox length

L_i = the ith redox length level

$P(L \geq L_i)$ = probability of exceeding L_i (provided by the respondent)

$\hat{P}(L \geq L_i)$ = probability of exceeding L_i (implied by the 3-parameter lognormal)

w_i = weight assigned to the squared difference for the ith length level

The results are shown in Table 7.1.

The endowment levels for question SM.8 (Appendix E) were selected with two factors in mind: (1) to span the range of possible endowments *implied* by the probability distributions reported by NURE/DOE and (2) to state endowment levels with which the respondent feels familiar, or about which he at least feels comfortable in making inferences.

Table 7.1. Parameters of the lognormal distributions fitted to the data in question SM.8, Appendix E.

Endowment	Logarithmic Mean	Logarithmic Variance	Displacement
25×10^6 lbs	3.2925	0.7235	4.0 miles
75×10^6 lbs	4.3092	0.6867	15.0 miles
150×10^6 lbs	5.0323	0.5060	35.0 miles

For each endowment level, the respondent was asked to state the minimum redox length required to contain that endowment level and four additional redox system lengths and the probabilities that the actual length exceeds these lengths. Substantial interaction between the interviewer and the respondent was required for the completion of this task. The interviewer manipulated the weights w_i in Equation 7.1 and showed the respondent the results of the fit in terms of probabilities of exceeding specified lengths until the respondent was satisfied with the results.

It turns out that this is a difficult question to answer, for it implies substantial experience and requires considerable inference, extrapolation, and a high level of abstraction on the part of the respondent. Judging from the amount of effort spent, this was probably the most difficult of all questions the geologist had to deal with in the questionnaires. Specifically, endowment as defined[1] for this potential supply system is not a familiar concept for people in industry, mainly because of lack of experience with grades as low as 0.01% U_3O_8. Also, the respondent did not feel very comfortable with high endowment levels, especially the one at 150 million pounds of U_3O_8.

7.2.2 Operation 2: Seeding of Pods Along the Redox System

In order to generate pods and seed them along the redox front system, it is necessary to have probability distributions for most of the attributes that define a pod (see section 7.1).

1. DOE's definition was used.

Again, "hard" data-based studies should prove very useful in this regard, but the confidential character of deposit data prevented us from taking this route. Thus, we had to resort to subjective elicitation to obtain the probability distributions of key pod attributes.

First, the respondent was introduced to the notion of the confining prism (Chapter 4, section 4.2.2) as a means of describing the physical char-acteristics of a pod. Then, he was presented with the questionnaire on Roll Front Deposits Morphology. This questionnaire and the answers elicited from the respondent are reported in Appendix E (questions SM.1 to SM.7).

Figure G.3 (Appendix G) shows in a schematic form the use of the questionnaire and the design of the pod seeding operation.

Questions SM.1 to SM.7 of the questionnaire (Appendix E) consist of a series of tables designed to elicit the probability distributions for key pod attributes. The pod seeding operation consists, in general, of identifying the appropriate probability distribution and sampling from it to generate the specific attribute. The identification is simply a series of table "look-ups." However, due to the discrete nature of the tables, linear interpolation procedures are used in the programming of the pod seeding operation.

Pod seeding starts with the identification in Table SM.2[1] of the appropriate entry of the thickness of the 0.01-confining prism [th(.01)] for the average thickness (τ) of the favorable host as reported by DOE.

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1. All tables starting with "SM" are in Appendix E. Mention of the appendix will be omitted hereafter.

Next, Table SM.1-T is used to obtain the conditional distribution of the width dimension of the 0.01-confining prism. The original subjective opinions are shown in Table SM.1, which is a joint probability table for thickness and width of the 0.01-confining prism. Instead of asking for actual probabilities, a cardinal ranking was employed. This required the respondent to struggle only with relative scaling (weights). The entries in Table SM.1-T are derived using the following conditional probability relation:

$$\mathrm{Prob}(w^*|th^*) = \frac{N(w^*, th^*)}{\sum_{w} N(w, th^*)}$$

where

w = 0.01-confining prism width

th = 0.01-confining prism thickness

$*$ = a specific value

N = numerator operator; denotes entries in original table, SM.1

Question SM.3 is designed to obtain the subjective probability distributions for the length of the 0.01-confining prism [$\lambda(.01)$] of the current mineralized pod and the distance to the next pod [gap(.01)]. Both measures are along the strike of the redox system.

The probability distribution for gap distances [gap(.01)] is elicited in density form using scaling numerals [see Table SM.3, part (3)], which can be transformed to probabilities by dividing by their total. It is important to note that the gap(.01) probability distribution is conditional on the thickness of the aggregate geochemical cell. In the model this thickness is estimated by taking the average of the th(.01) dimension of the two pods separated by the gap(.01) distance. Because of the seeding operation design, it is necessary to know in advance the thickness th(.01) of the next pod; that is the reason that two th(.01) values are carried over in the computer operations design in Figure G.3, Appendix G (operation #2).

The width and thickness dimensions of the inner q*-confining prism are generated from the probability distribution obtained in Table SM.4 and SM.5. The third dimension of this inner prism has to be consistent with the volumetric relationships stated in Table SM.7. The volume of the q*-confining prism must be determined first; then, $\lambda(q^*)$ is solved for.

Question SM.7 deserves some comment here; its objective is threefold:

a. Provide the necessary information for the estimation of three of the parameters of the tonnage distribution[1] (μ, σ, and d) of U_3O_8 grades in a pod. This information is shown in column 1 of Table SM.7.

b. Provide information on the "shrinkage function" for the confining prism; that is, the description of the reduction of the 0.01-confining prism volume as cutoff grade is varied (details on this function in Chapter 4, section 4.2.3).

1. Assumed to be lognormal; μ = logarithmic mean; σ^2 = logarithmic variance; d = displacement.

c. Provide information on the amount of mineralized material above 0.01%, [t(.01)] occurring inside the pod. This information is conveyed by stating VR1, the ratio of volume of mineralized material above 0.01 to the volume of the 0.01-confining prism:

$$\text{VR1} = \frac{\text{t(.01)} \cdot \text{tonnage factor (ft}^3\text{/ton)}}{\text{volume of 0.01-confining prism}}$$

The first two parameters of the lognormal grade-tonnage distribution, μ and σ (here, d is always assumed to be 0.0), can be estimated by considering the data in column 1 of Table SM.7 to represent a truncated lognormal distribution (i.e., normalize the data by dividing the entire column 1 by 56 x 10^6), as shown in Table 7.2. A numerical search routine has been used to find the lognormal parameters (μ and σ) so as to minimize D, the measure described below:

$$\text{CMPR1} = 1 - F\left(\frac{\ln(0.01) - \mu}{\sigma}\right)$$

$$\text{CMPR2} = 1 - F\left(\frac{\ln(0.05) - \mu}{\sigma}\right)$$

$$\text{CMPR3} = 1 - F\left(\frac{\ln(0.10) - \mu}{\sigma}\right)$$

$$\text{CMPR4} = 1 - F\left(\frac{\ln(0.15) - \mu}{\sigma}\right)$$

$$D = \left(\frac{\frac{\text{CMPR2}}{\text{CMPR1}}}{0.25} - 1\right)^2 + \left(\frac{\frac{\text{CMPR3}}{\text{CMPR1}}}{0.0625} - 1\right)^2 + \left(\frac{\frac{\text{CMPR4}}{\text{CMPR1}}}{0.0125} - 1\right)^2$$

where

ln = natural logarithm

$F(z)$ = probability of $Z \le z$, where $Z \sim N(0,1)$

The results obtained are:

$\mu = -3.446$

$\sigma = 0.69300$

The shrinkage function of the confining prisms can be estimated by fitting a function to data in Table 7.3, derived from column 2 of Table SM.7.

Table 7.2. Truncated data derived from Table SM.7 for lognormal fit.

Grade	Column 1 Normalized
0.01	56/56 = 1.0
0.05	14/56 = 0.25
0.10	3.5/56 = 0.0625
0.15	0.7/56 = 0.0125

Table 7.3. Data to estimate the "shrinkage function."

Grade	$VR2(q) = \frac{V(q)}{V(.01)}$
0.01	$\frac{87.5}{87.5} = 1.0$
0.05	$\frac{52.8}{87.5} = 0.60343$
0.10	$\frac{40}{87.5} = 0.45714$
0.15	$\frac{20}{87.5} = 0.22857$

Figure 7.2 provides a plot of the data in Table 7.3. A modified exponential function was fitted to these data using nonlinear regression:

$$VR2(q) = \frac{V(q)}{V(.01)} = (1 - K2) + K2 \quad e^{ALPHA2(q - 0.01)}$$

where

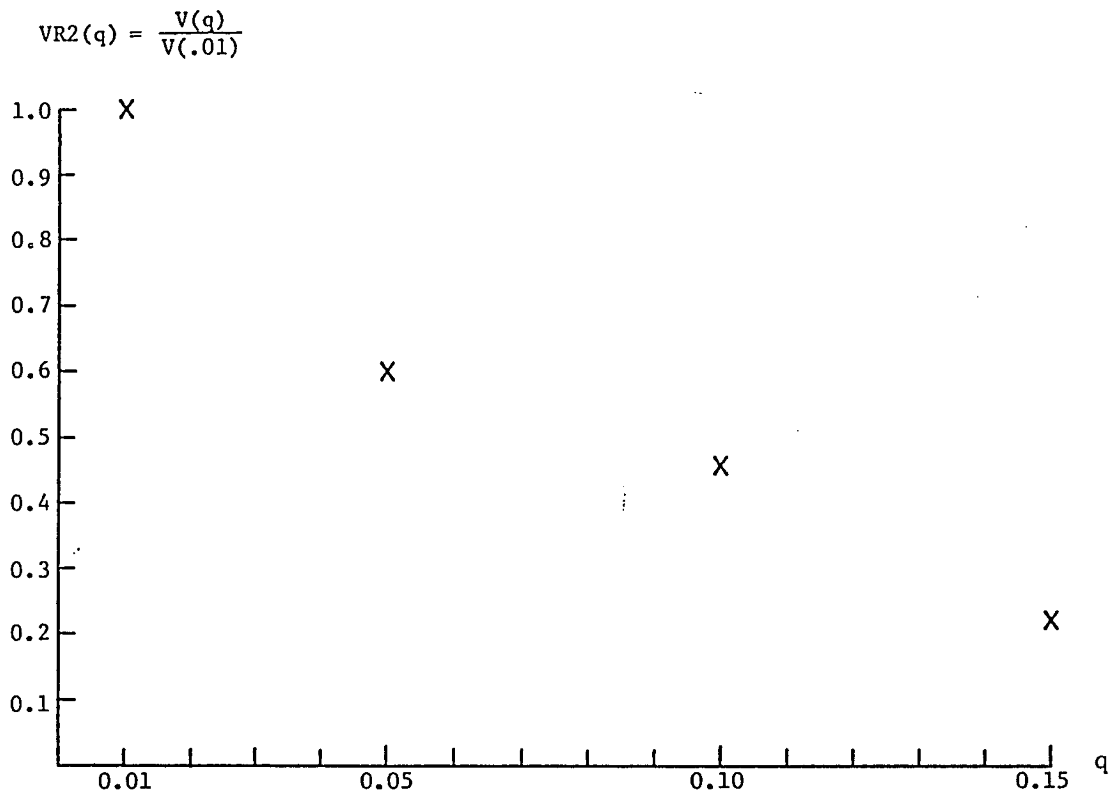
$K2 = 0.90652564$

$ALPHA2 = -12.222118$

The proportion, VR1, of 0.01-confining prism occupied by material above 0.01 can be derived simply:

$$VR1 = \frac{56 \times 10^6}{87 \times 10^6} = 0.64$$

Table SM.7 describes what the respondent considers to be a typical Wyoming roll front. It is clear that in reality μ, σ, VR1, and the parameters of the shrinkage function vary across pods; consequently, they are among the random variables that comprise the creation submodel. Again, if hard data were available, it would be possible to do a relative frequency study on these key attributes. Alternatively, subjective opinion from additional experts could be obtained.



q = cutoff grade

V(q) = volume of the q-confining prism

Figure 7.2. Plot of data on the "shrinkage function."

7.2.3 Operation 3: Seeding the Redox System and Its Accompanying Mineralized Pods in the Favorable Area (A_F)

This operation begins[1] by selecting an A_3 at random from all the A_3's in the favorable area. A specific value for the length of the redox system in that A_3 is drawn from the probability distribution for redox length per A_3 δ $(L|L > 0)$,[2] which is shown in Table 7.4. Another A_3 is selected at random from a restricted set of A_3's and $\delta(L|L > 0)$ is sampled again to "seed" the redox length in the A_3 selected. The set for A_3 selection is restricted as follows:

a. Only collateral A_3's are considered [see Figure 7.3, part (a)].

b. Only those A_3's that imply continuity of the redox length system constitute the set of possible A_3's. Aberrant A_3's are discarded from the set. [See Figure 7.3, part (b). Also, Appendix I provides further detail about these restrictions.]

This seeding process is continued until the total length of redox system to be seeded has been exhausted.

When this operation terminates, the "creation" of the state of nature has been completed. The final products of this creation are:

a. The A_3 grid making up the favorable area. For each A_3 in this grid, the length of the redox front, and the minimum and maximum depth to the favorable host are also available.

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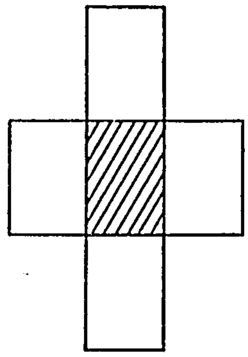
1. Prior to initiating this operation, the length of the redox system to be seeded is adjusted if necessary (see Figure G.3, Appendix G).

2. Details on obtaining this distribution via an auxiliary external simulation program called A3GRID is given in Appendix H.

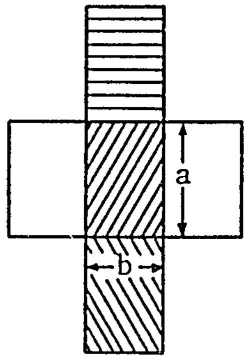
Table 7.4. Probability distribution for redox length in an A_3 (7 square miles) cell, $\delta(L|L > 0)$.[1]

Redox Length (per A_3) Class Interval (in miles)	Probability
0-0.5	.10154
0.5-1.0	.08441
1.0-1.5	.07093
1.5-2.0	.06945
2.0-2.5	.07119
2.5-3.0	.07274
3.0-3.5	.07260
3.5-4.0	.07022
4.0-4.5	.06572
4.5-5.0	.05954
5.0-5.5	.05232
5.5-6.0	.04470
6.0-6.5	.03722
6.5-7.0	.03027
7.0-7.5	.02410
7.5-8.0	.01882
8.0-8.5	.01444
8.5-9.0	.01092
9.0-9.5	.00813
9.5-10.0	.00598
10.0-10.5	.00434
10.5-11.0	.00312
11.0-11.5	.00223
11.5-12.0	.00157
12.0-12.5	.00110
12.5-13.0	.00077
13.0-13.5	.00053
13.5-14.0	.00037
14.0-14.5	.00025
14.5-15.0	.00017
15.5-16.0	.00012
16.0-16.5	.000076
16.5-17.0	.000053
¢17.0	.000037

1. Results obtained from the external A3GRID simulation program (Appendix H).



Part (a)



Part (b)

Symbology:

A_3 seeded last

A_3 seeded before last

A_3 not permitted to be seeded when the length in the last A_3 seeded is smaller than the "a" dimension

A_3 included in the set for selection

Figure 7.3. Schematic example of A_3 selection set restricted for redox front seeding.

b. A matrix that contains the identifications and attributes for each pod. Implicit in the data stored in this matrix is the sequence of the pods along the redox front.

c. A vector containing the order of the redox system, that is, the sequence in which the A_3's were seeded in operation No. 3.

Note that since both sequences [items (b) and (c) above] are known, the identity of the pods (and their attributes) in each one of the A_3's is readily available.

CHAPTER 8

THE EXPLORATION MODEL

8.1 General Performance Question Format

The heart of the exploration model is the exploration performance questionnaire. The questionnaire and the answers provided by the explorationist interviewed are reported in Appendix F.

Most of the questions proposed to the respondent in every exploration phase follow the general format illustrated in Figure 8.1. The questions are based upon a specified "true" state of nature, and objective, action, and performance of the exploration phase considered. More specifically, given the objective of an exploration phase and the action initiated to achieve that objective, the explorationist is presented with a statement about the state of nature and is requested to describe his performance in achieving the stated objective. If his objective in a specific exploration phase is detection of a certain geologic object, his performance is measured in terms of probability of detection. If instead his objective is evaluation of a certain geologic object, his performance is measured in units reflecting the degree of objective achievement (e.g., percent error in estimation). Table 8.1 lists the elements of the main performance questions for each of the exploration phases.

The general functioning of the exploration model can be related to the performance questions. The "true" state of nature (N) is generated by the creation submodel prior to beginning exploration. The

O = OBJECTIVE

a = ACTION

N = TRUE STATE OF NATURE

DOA = DEGREE OF OBJECTIVE ACHIEVEMENT

~ INDICATES RANDOM VARIABLE

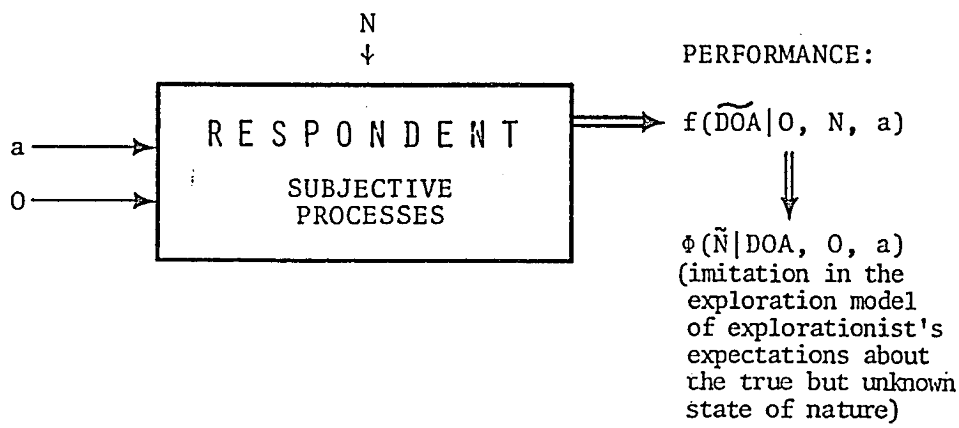


Figure 8.1. General format of the exploration performance question for each exploration phase.

Table 8.1. Elements of the general question for each exploration phase.

PHASE	O	a	N	DOA
Phase 2	Select a favorable A_3	Do geology and geochemistry	L = length of redox line contained in A_3	Do the right selection (if L > 0) or rejection (if L = 0) (Binary)
Phase 3.A	Find the redox	Drill n_c drill holes and geologic interpretation	L	Binary: Redox found/ not found
Phase 3.B	Estimate the general trend of the redox line, L	Drill n_R drill holes and geologic interpretation	L	% error incurred in redox line length estimate (target % error figure also asked)
Phase 4.A	Find the deposit. Deposit is defined as a concentration containing material $> q_{min}$.	Drill f_D fences and geologic interpretation	$\ell(q_{min})$ = length of target; L_D = search length	Binary: Deposit found or not found
Phase 4.B	Rough evaluation of tonnage and grade (U_3O_8) of deposit $\geq q_{min}$	Drill n_E drill holes and geologic interpretation	$\ell(q_{min})$ $w_D(q_{min})$	% error incurred (target % error figure also asked)
Phase 5	Continuing evaluation of tonnage and grade (U_3O_8 amt) of deposit $\geq q_{min}$	Drill n_E' drill holes and geologic interpretation	$\ell(q_{min})$ $w_D(q_{min})$	% error incurred (target % error figure also asked)

O = objective
a = action
N = state of nature
DOA = degree of objective achievement
q_{min} = minimum attractive grade
$\ell(q_{min})$ = length of the deposit at grade q_{min}
$w_D(q_{min})$ = width of the deposit at grade q_{min}

action taken by the explorationist in the system (e.g., drill "x" drill holes) is dictated by the executive model[1] after considering current expectations about the true but unknown state of nature. Subsequently, an outcome or result of the explorationist's actions is generated by the model (e.g., detection or no detection of the geologic object searched for). This outcome would be determined by identifying the appropriate entry in the exploration performance questionnaire according to the specific action level and state of nature. Expectations about the state of nature are updated to reflect the new outcome, and the executive model is consulted again for a decision on the next stage.

8.2 Prior Distribution Required to Drive the Exploration Model

8.2.1 Endowment Distribution

The DOE endowment distribution $[\theta(E_T)]$ is considered to describe nature. In other words, it is the distribution used in the creation submodel. This distribution is considered in the potential supply system as *the* endowment distribution in the favorable area for it controls the uranium that could be discovered and produced. Even so, the probability distribution reported by DOE for the total endowment in the favorable area does not necessarily have to reflect the expectations of the exploration agent about the endowment in the same area.

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1. Details on design of the executive model are discussed in Chapter 10. For now, consider the model to be a routine which, given the price of U_3O_8 and expectations about the state of nature, returns a decision on the optimum level of action to be conducted next (e.g., number of holes to be drilled).

The exploration agent's endowment distribution,[1] $[\gamma(U_T)]$, also called the "prior" endowment distribution, is the one that initially will drive the decisions to explore. This distribution is subject to revisions as exploration activities unfold.

Estimation of $\gamma(U_T)$ has not been explored in detail. Confronting the respondent with each DOE favorable area and asking him questions designed to capture his own endowment expectations would be impractical because of the great amount of effort and expertise required of the respondent. A more feasible approach seems to be to show the respondent DOE reports for familiar regions and ask him to state the degree of under- or overestimation of the endowment reported. If it were assumed that these patterns determined for familiar areas applied to all areas, $\gamma(U_T)$ could be determined by modifying the DOE endowment distribution $[\theta(E_T)]$ to account for this under- or overestimation that he deems inherent in DOE reports.

8.3 Endowment Distribution and the Endowment-Redox Length Relationship at the A_3 Level

Early phases of exploration (Phases 2, 3A, 3B) rely on the A_3 area as the exploration unit. Specifically, in Phase 2 the A_3 is the selection unit. In Phase 3A and 3B it is the drilling unit.

1. Both U and E refer to the same variable: endowment. U is used to indicate the explorationist's expectations about endowment. E is used to refer to DOE "actual" endowment.

Two probability distributions are used in the revision of expectations that occurs when exploration is conducted in the exploration model:

a. $\beta(R)$, the prior distribution[1] of redox system length,[2] R, in an A_3. This distribution is subject to revision in Phases 2, 3A, and 3B.

b. $a(U|R)$, the conditional distribution for endowment, U, in an A_3, given the redox length in an A_3. At the micro level (A_3), a $(U|R)$ is the equivalent of the redox length-endowment relationship (see section 4.3 in Chapter 4 and Table SM.8, Appendix E) at the macro level (favorable area, A_F). This conditional probability distribution is of key importance in the executive model (Chapter 10, section 10.3).

Both $\beta(R)$ and $a(U|2R)$ ultimately depend on the prior distribution of total endowment in the A_F, $\gamma(U_T)$. They can be obtained if the joint A_3-endowment and A_3-redox length probability distribution [b(U,R)] is known. The derivation of b(U,R) is done externally in a separate Monte Carlo simulation routine (A3SAMP) which includes $\gamma(U_T)$ as an input. Design of this routine is shown in Appendix I.

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1. Both R and L refer to the same variable: redox system length. R is used to refer to the explorationist's expectations of redox system length; L is used to refer to "actual" redox system length generated by the system.

2. Unless otherwise indicated, when dealing with redox length (for an A^3) probability distributions, the redox length has been discretized into half-mile class intervals throughout the system.

8.4 Conceptual Problems in Obtaining the Joint Probability Distribution Function, b(U,R) for Endowment and Redox Length at the A_3 Level

8.4.1 The Problem

As mentioned previously, the joint distribution of endowment and redox length at the micro level, A_3 (b(U,R)), is obtained using an external Monte Carlo simulation program, A3SAMP, detailed in Appendix I.

As stated in Chapter 7, section 7.2.1, the response to question SM.8 in the subjective opinion on deposit morphology questionnaire (Appendix E) shows a direct dependency between redox system length and endowment at the macro level; i.e., areas of the order of hundreds of square miles. Simply stated, this dependency implies that abundant endowments and long redox system lengths tend to be associated.

The positive dependence referred to above should be preserved at the micro level; i.e., in areas of the size of A_3, the basic selection and drilling unit. In other words, the joint frequency table representation of b(U,R) obtained from the external program A3SAMP must maintain this dependency. This dependency issue can be represented mathematically as follows. Let:

$$\text{Prob}(U \leq U'|R) = \frac{\sum_{i=1}^{u'} O_{RU_i}}{\sum_{j=1}^{n} O_{RU_j}} \tag{8-1}$$

and

$$\text{Prob}(R \leq R'|U) = \frac{\sum_{i=1}^{r'} O_{R_i U}}{\sum_{j=1}^{m} O_{R_j U}} \tag{8-2}$$

where:

U = random variable denoting the uranium endowment in an A3

R = random variable denoting the redox system length in an A3

U' = a particular realization of U

R' = a particular realization of R

i, j = indexes

m = total number of redox length class intervals

n = total number of endowment class intervals

r' = number of the class interval which includes R'

u' = number of the class interval which includes U'

$Prob(U \leq U' | R)$ = conditional probability of U_3O_8 endowment being no more than U', given that the redox system length is R, within an A3

$Prob(R \leq R' | U)$ = conditional probability of redox front length being no more than R', given that the U_3O_8 endowment is U, within an A3

O_{RU} = number of observations in the class interval whose midpoints are R and U in the joint frequency table provided by the program A3SAMP to present the point pdf, $b(U,R)$

A joint frequency table preserves the aforementioned dependency if it possesses the following properties[1] throughout:

---

1. If 8-3 and 8-4 are equal to zero, then there is independence.

$$\text{property 1: } \frac{\partial \text{Prob}(U<U'|1R)}{\delta R} < 0 \quad (8\text{-}3)$$

$$\text{property 2: } \frac{\partial \text{Prob}(R<R'|U)}{\delta U} < 0 \quad (8\text{-}4)$$

The first inequality implies that if the redox length is longer in a given A_3, the probability of exceeding a given endowment level should increase. This is illustrated in the first graph of Figure 8.2. Note in this figure that the conditional distribution function of endowment given a shorter length lies below that for a longer length.

The second inequality implies that if the endowment is more abundant, the tendency will be for longer redox lengths. The second graph of Figure 8.2 illustrates this point: the conditional distribution function for redox length (in a "greater than" form), when the endowment is scarce, has to lie below those for more abundant endowments.

8.4.2 The Solution

The external program A3SAMP was run using test data to generate the raw data representing the joint distribution b(U,R) and to illustrate with an example a solution to preserve the aforementioned dependency. The test date consists of a 175-square-mile area (i.e., a grid of 25 A_3's, 7 square miles each), containing an estimated total uranium endowment described by a lognormal distribution with the following parameters:

logarithmic mean = 14.383

logarithmic standard deviation = 1.76342

Results of the Monte Carlo simulation after 40000 iterations are shown in the Table 8.2. This is a joint frequency table containing the

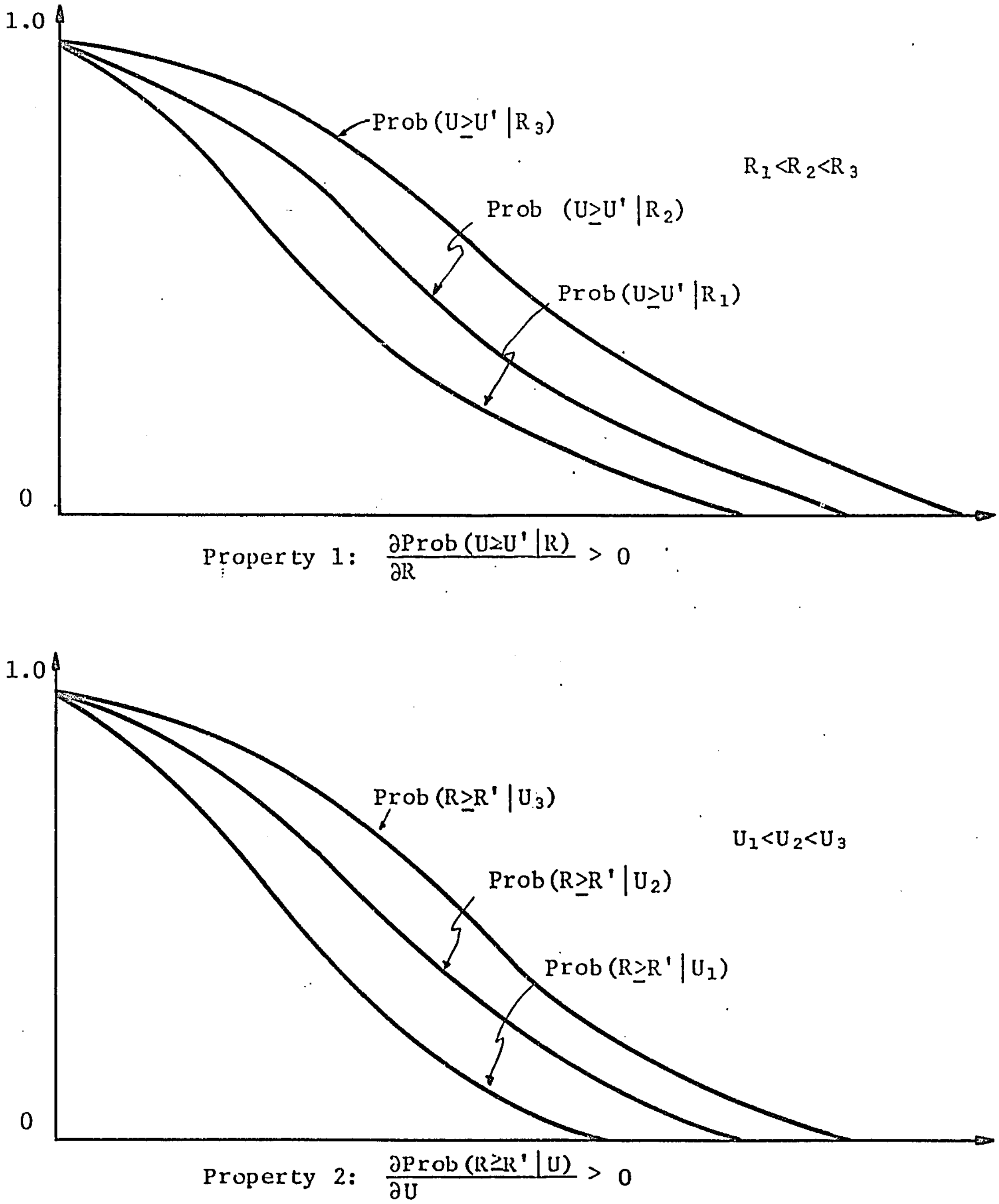

Figure 8.2. Diagrammatic representation of the monotone properties required to have a direct dependency between redox length and endowment in bivariate joint probability density function.

Table 8.2. Example of joint frequency distribution of length and endowment in an A_3 (b(R,U)).

R	U																				
	0	0-0.5	0.5-1	1-2	2-3	3-4	4-5	5-6	6-7	7-8	8-9	9-10	10-11	11-12	12-13	13-14	14-15	15-20	20-40	40-60	60-110
0	698462	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0-0.5	22684	4609	1086	725	304	155	60	55	45	33	24	21	18	7	9	12	10	18	17	3	0
0.5-1.0	18762	3298	1056	1032	448	259	168	108	69	58	41	29	17	24	9	11	13	37	35	8	1
1.0-1.5	16714	3056	1067	940	452	283	167	117	83	60	47	32	22	30	22	15	4	58	43	8	0
1.5-2.0	14443	2704	905	939	448	283	176	110	87	61	46	38	35	30	24	25	13	55	58	10	2
2.0-2.5	15041	2890	1031	1038	487	298	183	140	88	74	53	45	36	25	17	24	12	44	66	1	3
2.5-3.0	15257	3089	1094	1111	544	311	227	170	99	92	53	47	30	27	28	19	15	53	69	11	2
3.0-3.5	15137	3154	1147	1144	620	372	201	142	122	96	74	53	37	27	24	22	14	54	71	4	3
3.5-4.0	14299	3122	1114	1152	556	347	230	163	122	78	47	58	31	36	31	24	27	65	75	11	0
4.0-4.5	13191	3084	1150	1092	595	372	200	153	112	82	68	46	44	38	31	25	15	65	55	5	0
4.5-5.0	11470	2555	937	1037	497	292	214	134	77	81	51	46	36	15	23	17	24	73	54	8	2
5.0-5.5	10155	2319	914	874	474	259	220	156	102	67	56	41	33	26	17	14	16	41	52	5	1
5.5-6.0	8370	1928	775	727	423	288	159	112	91	62	58	39	31	16	19	24	8	39	53	4	2
6.0-6.5	6511	1632	604	617	338	199	141	98	70	57	43	31	30	26	15	15	11	38	46	4	1
6.5-7.0	5122	1332	482	512	251	174	124	101	65	59	33	29	26	12	15	7	13	32	42	5	0
7.0-7.5	4186	1042	356	412	251	120	119	60	53	37	29	25	30	9	12	14	13	26	30	8	1
7.5-8.0	3181	783	274	331	177	117	62	47	43	35	20	20	28	15	9	11	10	21	32	5	0
8.0-8.5	2532	676	245	259	140	82	80	48	41	28	28	17	13	7	5	8	7	18	18	2	2
8.5-9.0	1888	514	198	230	134	77	51	39	30	24	17	22	11	12	5	7	8	21	12	3	0
9.0-9.5	1242	343	129	145	77	44	37	28	23	19	12	15	6	10	7	5	4	9	5	2	0
9.5-10.0	1049	278	97	115	68	38	30	20	16	17	11	7	7	5	3	4	3	8	12	1	1
10.0-10.5	788	184	87	92	51	35	24	22	9	10	11	5	7	6	1	1	3	7	12	0	0
10.5-11.0	622	156	70	61	43	34	19	11	10	9	4	5	1	6	4	1	2	5	1	2	0
11.0-11.5	398	101	51	40	20	16	12	10	7	6	2	4	3	4	1	2	1	7	4	0	0
11.5-12.0	284	74	32	44	20	17	8	12	4	5	4	1	2	1	4	1	3	4	3	0	0
12.0-12.5	209	60	10	28	15	9	10	4	2	1	4	3	0	1	1	1	1	1	5	0	0
12.5-13.0	143	43	18	17	12	7	10	1	1	2	1	3	1	1	2	1	1	0	2	0	1
13.0-13.5	95	16	16	9	6	6	2	4	2	1	1	1	0	0	0	0	0	0	0	0	0
13.5-14.0	45	8	5	4	4	2	3	2	1	0	0	0	1	1	0	2	2	0	0	0	0
14.0-14.5	58	19	4	8	4	4	1	2	5	1	1	1	0	0	0	0	0	4	1	0	0
14.5-15.0	6	4	1	3	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
15.0-15.5	24	7	1	0	0	2	0	0	0	0	0	0	0	0	0	1	0	1	2	0	0
15.5-16.0	11	3	1	3	2	0	1	0	3	1	1	0	0	0	0	0	0	0	1	0	0
16.0-16.5	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0
16.5-17.0	2	2	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

R = class intervals for redox length (miles within an A_3 area).
U = class intervals for endowment (millions pounds of U_3O_8) within an A_3 area.

number of observations on each class interval. The division of each entry by the total 1,000,000 observations provides the absolute frequency distribution, b(U,R).

Following conventional Monte Carlo procedures to obtain a probability distribution, the joint bivariate Table 8.2 provided by the program A3SAMP is the result of random generation of observations and subsequent classification into predetermined class intervals. Because of irregularities inherent in any Monte Carlo sampling results (especially for those seldom-occurring events), there is no assurance that the resulting joint frequency distribution table will preserve the dependency between endowment and redox system length. In fact results shown in Table 8.2 do not preserve the direct dependency that exists at the macro level for endowment and redox length. This is due to the "noise" induced by the Monte Carlo sampling procedure.

A solution, demonstrated here, is to smooth the data[1] in Table 8.2 so the probabilities implied by the smoothed data preserve the dependency mentioned above. A plausible function is a two-dimensional polynomial of 6th degree fitted to the logarithms of the number of observations, endowment, and redox system length. The form of this frequency function is detailed in Table 8.3.

It is important to point out that the joint probabilities implied by the smoothed data (i.e., data resulting from substituting the midpoints of the class intervals in the polynomial and obtaining the

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1. Number of observations in the body of the table was adjusted to account for the unequal width of endowment class intervals.

Table 8.3. Fit of a polynomial to data in Table 8.2 to obtain a smoothed joint frequency table preserving the dependency between redox length and endowment at the A_3 level.

MODEL

$$\hat{O} = \exp\,(b_1 + b_2X + b_3Y + b_4X^2 + b_5XY + b_6Y^2$$
$$+ b_7X^3 + b_8X^2Y + b_9XY^2 + b_{10}X^3$$
$$+ b_{11}X^4 + b_{12}X^3Y + b_{13}X^2Y^2 + b_{14}XY^3 + b_{15}Y^4$$
$$+ b_{16}X^5 + b_{17}X^4Y + b_{18}X^3Y^2 + b_{19}X^2Y^3 + b_{20}XY^4 + b_{21}Y^5$$
$$+ b_{22}X^6 + b_{23}X^5Y + b_{24}X^4Y^2 + b_{25}X^3Y^3 + b_{26}X^2Y^4 + b_{27}XY^5 + b_{28}Y^6)$$

where:

$\hat{O}$ = estimated number of observations

$X = \ln\left(\frac{U}{1{,}000{,}000} + 1\right)$

$Y = \ln\left(\frac{R}{1{,}000}\right)$

U, R = midpoints of endowment (U) and redox length (R) class intervals in table 8.1.A

b_i = coefficients to be estimated by regression (i = 1, 28)

CONSTRAINTS

$b_5, b_8, b_9, b_{12}, b_{13}, b_{14}, b_{17}, b_{18}, b_{19}, b_{20}, b_{23}, b_{24}, b_{25}, b_{26}$ and $b_{27} \geq 0$

RESULTS:

b_1 = 9.8830453	b_{11} = -6.8816036	b_{21} = - 0.093486773
b_2 = -4.2843816	b_{12} = 0	b_{22} = - 0.35963895
b_3 = 0.681144	b_{13} = 0	b_{23} = 0
b_4 = -1.742634	b_{14} = 0.022042	b_{24} = 0
b_5 = 0	b_{15} = 0.426488	b_{25} = 0
b_6 = -0.33747956	b_{16} = 2.622776	b_{26} = 0
b_7 = 7.3213572	b_{17} = 0	b_{27} = 0
b_8 = 0	b_{18} = 0	b_{28} = 0.005160082
b_9 = 0.0000101707	b_{19} = 0	
b_{10} = -0.55271754	b_{20} = 0	

smoothed number of observations) still may not preserve the aforementioned dependency. To ensure that this dependency is maintained throughout the smoothed table, it is necessary to impose constraints in the regression coefficients of the cross terms of the polynomial so that they are non-negative. The proof of this sufficient condition shall be given below.

The nonlinear regression program provided by the SPSS computer package (Statistical Package for the Social Sciences) was used to fit the polynomial to the data in Table 8.2. The fit was performed using only those observations greater than zero and excluding the one observation for the zero endowment-zero length class interval.

Results of the regression analysis for the fit are shown in Table 8.3. Most of the coefficients for the cross terms were driven to zero by the SPSS program. The nonzero ones are positive, ensuring that the dependency between endowment and redox length is maintained.

8.4.3 Proof That Non-negativity of the Cross Terms in a Polynomial of Any Degree is a Sufficient Condition to Preserve Dependency in the Joint Probability Distribution It Represents

Let the following equation represent the frequency function used to smooth Table 8.2:

$$\hat{O}_{RU} = f(U,R) \tag{8-5}$$

where

$\hat{O}_{RU}$ = estimated frequency of observations

U,R = midpoints of endowment and redox length class intervals

f = function notation

Proof is provided for property 2. The proof for property 1 is analogous.

Property 2 can be written as follows by substituting (8-2) into (8-4):

$$\frac{\partial}{\partial U} \frac{\sum_{i=1}^{r'} \hat{O}_{R_i U}}{\sum_{j=1}^{m} \hat{O}_{R_j U}} < 0 \tag{8-6}$$

Taking the partial derivative of the quotient, we have:

$$\frac{\sum_{j=1}^{m} \hat{O}_{R_j U} \cdot \frac{\partial}{\partial U}\left[\sum_{i=1}^{r'} \hat{O}_{R_i U} \right] - \sum_{i=1}^{r'} \hat{O}_{R_i U} \cdot \frac{\partial}{\partial U}\left[\sum_{j=1}^{m} \hat{O}_{R_j U} \right]}{\left[\sum_{j=1}^{m} \hat{O}_{R_j U} \right]^2} < 0 \tag{8-7}$$

Because the denominator in (8-7) is positive, the sign is determined by the numerator. Performing the partial derivation indicated in the numerator of (8-7):

$$\sum_{j=1}^{m} \hat{O}_{R_j U} \cdot \sum_{i=1}^{r} \hat{O}_{R_i U} \cdot \frac{\partial f}{\partial U} (U, R_i) - \sum_{i=1}^{r} \hat{O}_{R_i U} \cdot \sum_{j=1}^{m} \hat{O}_{R_j U} \cdot \frac{\partial f}{\partial U} (U, R_j) < 0 \tag{8-8}$$

Factoring out, it is obtained:

$$\sum_{j=1}^{m} \sum_{i=1}^{r'} \hat{O}_{R_j U} \cdot \hat{O}_{R_i U} \cdot \frac{\partial}{\partial U} [f(R_i, U) - f(R_j, U)] < 0 \tag{8-9}$$

Write (8-9) in two parts:

$$\sum_{j=1}^{r'} \sum_{i=1}^{r'} \hat{O}_{R_jU} \cdot \hat{O}_{R_iU} \cdot \frac{\delta[f(R_i,U) - f(R_j,U)]}{\delta U} \quad + \tag{8-10}$$

$$\sum_{j=r'+1}^{m} \sum_{i=1}^{r'} \hat{O}_{R_jU} \cdot \hat{O}_{R_iU} \cdot \frac{\partial[f(R_i,U) - f(R_j,U)]}{\partial U} < 0$$

The first part of (8-10) is zero; for i and j equal, the differences are zero; for $i \neq j$, all the terms of the double summation sign will appear in pairs, but with opposite sign.

In the second part of (8-10), $j > i$; that is, $j = r' + 1, \ldots m$ and $i = 1, \ldots r'$. If $f(R,U)$ is a polynomial in $\ln(R)$ and $\ln(U)$ of second degree or higher, having non-negative coefficients in the cross product terms will be a sufficient condition for the following inequality to hold:

$$\frac{\partial f}{\partial U}(R_j,U) > \frac{\delta f}{\delta U}(R_i,U) \tag{8-11}$$

When (8-11) holds, the second part of (8-9) is negative; that is:

$$\sum_{j=r'+1}^{m} \sum_{i=1}^{r'} O_{R_jU} \; O_{R_iU} \cdot [f(R_i,U) - f(R_j,U)] < 0 \tag{8-12}$$

This completes the proof. Thus, having non-negative coefficients of the cross terms and at least one positive in the polynomial form of the frequency function used to smooth Table 8.2 is a sufficient condition to preserve dependency between R and U at the A_3 level.

8.5 Phase 2: Geologic Reconnaissance

The exploration model assumes that the area, F, delineated by the exploration agent as favorable after conducting Phase 2 (see Appendix C) is the same as the favorable area delineated by DOE.

Details on the operation of this phase are provided in Figure G.5, Appendix G. The favorable area, A_F, is represented by a grid of A_3 size. The system proceeds to select from this grid an A_3 upon which the activity of the next phase will be conducted. The system considers the shallowest A_3 on the list first. A list of available A_3's in order of increasing average depths is maintained by deleting from the initial collection the A_3's as they are considered for selection by the system.

The attributes of the A_3 are important here: the true redox system length contained (L) and the average depth of the favorable host in the A_3 area ($\overline{DEPTH}$). These two attributes define the relevant state of nature for the question P2.1 (Appendix F). This state of nature identifies the appropriate function[1] in Table 8.4, P21(Sel|L, DEPTH). Evaluation of this function provides the probability for selection (Sel), given the state of nature.

A uniformly distributed random number on the interval (0,100) is generated and compared to the probability for selection P21(Sel|L,DEPTH).

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1. Functions were fitted to data in questions P2.1 (Appendix F) using nonlinear regression for Part (1) and linear regression for Part (2).

Table 8.4. Functions fitted to answers to question P2.1, Appendix F.

For L > 0, Case I (favorable host crops out in the basin)

$$P21(Sel|L,\overline{DEPTH}) = 0.43 - 0.37131591 \cdot e^{-75.379906 \cdot \frac{L}{\overline{DEPTH}}}$$

For L = 0, Case I

$$P21(Sel|L=0,\overline{DEPTH}) = 0.90996 \cdot e^{-34.465526 \cdot \frac{1}{\overline{DEPTH}}}$$

For L > 0, Case II (no favorable host outcrops in the basin)

$$P21(Sel|L,\overline{DEPTH}) = 0.26188 - 0.24181518 \cdot e^{-90.108659 \cdot \frac{L}{\overline{DEPTH}}}$$

For L = 0, Case II

$$P21(Sel|L=0,\overline{DEPTH}) = 0.93842 \cdot e^{-29.506534 \cdot \frac{1}{\overline{DEPTH}}}$$

Sel = selection of the A_3 cell for subsequent drilling.
L = true length of redox front in the A_3 cell (miles).
$\overline{DEPTH}$ = average depth of the favorable formation in the A_3 cell (ft).
$P21(Sel|L,\overline{DEPTH})$ = probability of selection of the A_3 cell given L and $\overline{DEPTH}$.

Note: Case I or Case II functions in this table need to be used according to the NURE quadrangle reports regarding the presence/absence of outcrops.

If the A_3 is not selected, the system proceeds to the consideration of the next A_3 in the list.

When the A_3 is selected, the $\beta(R)$ distribution is revised. Note that the distribution of A_3-redox length [$\beta(R)$] is the same for all A_3's prior to conducting Phase 2. This makes sense, for the typical Phase 2 activities have not been conducted yet. All the A_3 cells in the favorable area are of equal endowment potential in the mind of the exploration agent being simulated. After conducting Phase 2 and making the A_3 selection, $\beta(R)$ has to be modified to account for the fact that geologic/geochemical investigation was undertaken in an effort to improve selection and rejection decisions for A_3 areas.

Revision of the $\beta(R)$ procedure is detailed in the schematic flow diagram of Figure G.5, Appendix G. Note that for A_3's with the same average depth, $\overline{DEPTH}$, the revised $f(R|Sel,\overline{DEPTH})$ probability distribution is the same.

At this time, the executive model is consulted. Current expectations about the state of nature (redox length, depth to favorable host) in the A_3 selected are communicated to this model which returns the number of holes to drill in Phase 3A, n_{3A} (zero holes imply that exploration of current A_3 must be halted).

A decision to quit exploring the current A_3 implies that under the current price in the system, if any of the remaining[1] A_3's to be considered is selected, the decision will also be to stop exploration.

---

1. Recall that A_3's have been arranged in a list in order of increasing average depths.

The reasons are: (a) the revised redox length distribution for and A_3 $f(R|Sel,\overline{DEPTH})$ does not depend on the actual length existing in the A_3 and (b) the distribution $f(R\ sel|\overline{DEPTH})$ will imply shorter redox length expectations which, in turn, imply worst monetary expectations in the executive model. This is so because an increase in depth implies poorer selectivity and increasing exploration and exploitation costs.

Because of the above, the remaining A_3's in the list are left unconsidered and a new "endowment iteration" is initiated.

When the decision is to continue exploring the A_3 (as signaled to the exploration model by a positive value for the number of holes to drill in Phase 3A), the exploration model proceeds to Phase 3A to initiate the drilling activities.

8.6 Phase 3A: Reconnaissance Drilling and Redox Front Search

8.6.1 Design

Figure G.6 of Appendix G shows in schematic form the design for modeling the exploration Phase 3A.

This phase is initiated by drilling n_{3A} holes. Each hole is costed[1] according to its depth, which is generated by random sampling of the distribution of depth to the host formation in the A_3 being explored.

1. The drilling cost relationship used here and in subsequent exploration phases is discussed in section 8.12 of this chapter.

Upon identification of the appropriate probability distribution of success (i.e., detection of the redox system) in Table[1] P3A.1 of Appendix F, an outcome "S" (success or failure) is generated. When the actual redox system length existing in the A_3 and number of holes drilled in this phase, n_c, do not coincide with headings in Table P3A.1, a simple two-dimensional linear interpolation is made to obtain the success probability.

After the outcome (success/failure) in Phase 3A has been generated by the exploration model, the probability distribution for redox system length (within the A_3 being investigated) that was derived at the end of Phase 2, $f(R|\mathrm{Sel},\overline{\mathrm{DEPTH}})$, becomes the prior distribution in this phase. This distribution is revised to produce the posterior distribution, $c(R|S,n_c)$.[2] The revision consists of a conventional Bayesian posterior computation. Details on the revision procedure are given in Figure G.6 of Appendix G.

When updated expectations about the length of the redox system are determined, the executive model is consulted to establish the number of drill holes the exploration model should drill next (zero holes imply that exploration of current A_3 must be terminated).

When Phase 3A drilling has been unsuccessful and the executive model dictates to drill additional 3A holes, these holes are added to the

1. The code P3A.1 refers to "Table #1 of exploration phase 3A." All table identifications starting with a "P" refer to tables on Appendix F, even if appendix identification is omitted in this chapter.

2. S is the success/failure indicator variable.

running total of Phase 3A holes, n_c, and the Phase 3A procedure, including the revision, is redone.

If Phase 3A drilling has been successful and the executive model determines that continuing drilling is worthwhile, Phase 3B is initiated by drilling the n_{3B} hole program dictated by the executive model.

8.6.2 Comments on Table P3A.2

As noted in Figure G.6 in Appendix G, one of the terms required in determining the revised redox length distribution $c(R|S, n_c)$ is the "experience" factor $P(n_c|R)$. This factor represents, in terms of relative frequency, the experience of the explorationist in drilling n_c holes in Phase 3A. The purpose of the questionnaire in Appendix F (see Question P3A.2 and table series P3A.2) is to obtain responses (estimates) for this term. It is important to note that size of the redox length intervals used to elicit these responses do not necessarily correspond to size of the intervals being used in the system to discretize the redox length distribution in an A_3. In general, the size of the former will be greater than the size of the latter.

To correctly identify the term $P(n_c|R)$ for the revision equation, it is necessary to subdivide the redox length intervals of Table P3A.2-T[1] into m subintervals.[2] m is a whole number[3] given by:

---

1. Table P3A.2-T is simply Table P3A.2 restated.

2. The procedure detailed here is external to the potential supply system. The table determined here has to be input in place of Table P3A.2-T.

3. This is ensured by making both interval sizes compatible to yield a whole number.

$$m = \frac{SI}{SS}$$

where

SI = size of the redox length interval in Table P3A.2-T

SS = size of the redox length interval used to discretize redox length in the potential supply system.

The entries of the new finer table (referred to hereafter as Table P3A.2-TN) are computed employing the procedure (completely external to the potential supply system) described next. Results are shown in Appendix F, Table P3A.2-TN.

Let:

i = index of redox length intervals used in Table P3A.2-T; i = 1, . . . , I (I = 4 in this table)

j = index of redox length subintervals within each interval above (j = 1, . . . , m_i). Note that a redox length subinterval is completely identified by "ij"

m_i = number of redox length subintervals in the i^{th} redox length interval

k = index of the number of Phase 3A holes (n_c) intervals used in Table P3A.2-T; k = 1, . . . , K (K = 3 in this table)

b_{ik} = quotient of the entries (fractional format) in Table P3A.2-T, f_{ik}/g_{ik}.

a_{ijk}= entries for the new (redox length) subintervals table[1] (Table P3A.2-TN) (note that it is this set of entries that we are after)

The "a" entries have to meet the following constraints:

$$\sum_{k=1}^{K} a_{ijk} = 1 \qquad \text{constraint 1}$$

$$\sum_{j=1}^{m_i} c_{ij} \cdot a_{ijk} = b_{ik} \qquad \text{constraint 2}$$

$$0 \leq a_{ijk} \leq 1 \qquad \text{constraint 3}$$

The c_{ij} in constraint 2 is a weight representing the relative frequency (relative with respect to the i^{th} redox length interval) with which the ij redox length subinterval occurs in nature:

$$c_{ij} = \frac{\text{probability of the ij redox length subinterval}}{\text{probability of the } i^{th} \text{ redox length interval}}$$

The probabilities for the c_{ij} are determined from $\delta(L|L > 0)$, the censored distribution of redox length per A_3 displayed in Table 7.4 (Chapter 7) and obtained form the external routine named A3GRID, described in Appendix I.

---

1. Note that when i = 1, the redox length "interval" corresponds to zero redox length in Table P3A.2-T. In this special case,

$$m_i = 1$$

and

$$a_{ijk} = b_{ik}$$

Determination of the "a" entries can be worked out independently for each of the redox length intervals indexed by "i." A reasonable measure for the "a" entries to meet is the "slope" between intervals:[1]

Let:

$$B_{ijk} = \frac{b_{ik} - b_{i-1,k}}{M_i - M_{i-1}} \quad \text{if } j < \frac{m_i}{2} + \frac{1}{2}, \text{ or } i = I$$

$$B_{ijk} = \frac{b_{i+1,k} - b_{ik}}{M_{i+1} - M_i} \quad \text{if } j < \frac{m_i}{2} + \frac{1}{2}, \text{ and } i < I$$

$$B_{ijk} = \left(\frac{b_{ik} - b_{i-1,k}}{M_i - M_{i-1}} + \frac{b_{i+1,k} - b_{ik}}{M_{i+1} - M_i}\right) \Big/ 2 \quad \text{if } j < \frac{m_i}{2} + \frac{1}{2}, \text{ and } i<I$$

where

B_{ijk} = the "slope" that the "a" entries should meet

M_x = midpoint of the x^{th} redox length interval

Ideally, the "slope" of the "a" entries should equal the slope of the "b" entries:

$$B_{ijk} = \frac{a_{ijk} - a_{i,j-1,k}}{M_j - M_{j-1}} \quad \text{if } j > 1$$

where

M_j = midpoint of the j^{th} redox length subinterval

Thus, the problem can be stated as follows:

For a given i, find a_{ijk} that meets constraints 1, 2, and 3 above, and minimizes the following measure:

---

1. Recall from discussion above that only cases in which i > 1 are relevant in this solution. Solution for i = 1 is trivial.

$$D = \sum_{k=1}^{K} \sum_{j=2}^{m_i} \left(B_{ijk} - \frac{a_{ijk} - a_{i,j-1,k}}{M_j - M_{j-1}} \right)^2$$

A numerical search routine is employed to solve this multivariate minimization problem for each of the I - 1 redox length intervals. Constraint number 3 can be met by imposing bounds on the search. Constraints numbers 1 and 2 can be met by eliminating $(m_i + K)$ variables, expressing them in terms of the remaining $[(M_i \cdot K) - (m_i + K)]$ variables and performing the appropriate substitution in the measure D above, the measure being minimized.

<u>Forming the New Table P3A.2-TN in Fractional Form</u>. Note that Table P3A.2-T (Appendix F) is expressed in fractional form. This is so because as new "experience" is gained by the exploration model, the numerator and denominator of the appropriate entries have to be updated (i.e., add a 1). The new Table P3A.2-TN has to be in the same format. This is, instead of having only the decimal entry a_{ijk}, it is necessary to have it expressed in fractional form:

$$a_{ijk} = \frac{u_{ijk}}{d_{ijk}}$$

where

a_{ijk} = entry for the new table as determined from the optimization procedure detailed above

u_{ijk} = numerator for the "ijk" entry (fractional format)

d_{ijk} = denominator for the "ijk" entry (fractional format)

The denominator can be calculated by:

$$d_{ijk} = c_{ij} \cdot g_{ik}$$

where

$$g_{ik} = \text{denominator of } b_{ik}$$

The numerator can be computed by:

$$u_{ijk} = d_{ijk} \cdot a_{ijk}$$

This conversion to fractional form takes place within the computer program.

Comments on Updating the New Table P3A.2-TN. Table P3A.2-T (Appendix F) is the representation of the experience of the explorationist. It is kept in fractional form to preserve absolute frequencies. As described above, Table P3A.2-T is used to derive a finer table, referred to as the "new" Table P3A.2-TN. It is this new table that is input to the system. As the exploration model conducts Phase 3A drilling programs, the exploration agent being modeled acquires more experience. Thus, to emulate this "learning" process of the exploration agent, the new Table P3A.2-TN is updated to account for new experiences.

Upon completion of either successful or not successful Phase 3A drilling, two important bits of information are recorded for subsequent use: number of holes drilled, n_c, and the actual length of redox front system,[1] L. These bits of information are used to create the new Table P3A.2-T by updating the appropriate n_c and L interval: a one is added to both numerator and denominator of the fraction in that element of the table. Also, a 1 is added to all denominators of the entries in the same row as that element. In this way, absolute frequency of that redox length interval is updated.

8.7 Phase 3B: Following the Redox Front System General Trend

8.7.1 Design

Figure G.7 (Appendix G) shows in schematic form the design for modeling this phase.

Phase 3B is initiated by drilling a n_{3B}-hole drilling program which was dictated by the executive model in the last consultation. Each hole is costed according to its depth, which is generated by random sampling of the (uniform) distribution of depth to the host formation in the A_3 being explored.

Using Tables P3B.2, the appropriate entry giving the parameters of a (triangular) distribution for redox system length in the current A_3, $k(R|L,n_c,n_R)$ is identified. Interpolation procedures for use of the P3B.2 tables are discussed in the next subsection.

---

1. It is assumed that the exploration agent shall have in later phases a pretty good idea of the actual length in the current A_3. The only exception is when Phase 3A has been completed without success (i.e., current A_3 is abandoned). The treatment for this incident is to classify the actual redox length as zero.

Assuming a diffuse prior state with regards to the (uncertain) knowledge of the redox length in the current A_3, $k(R|L,n_c,n_R)$ represents the revised (posterior) probability distribution for redox length.

The percent error in the estimates of redox length is computed from the $k(R|L,n_c,n_R)$ distribution and is compared with the allowed percentage of error (ERR^*_{3B}) in the estimate of redox length; this error limit is obtained by question P3B.1 of Appendix F. If ERR^*_{3B} has been met, the Phase 3B drilling program has been successful.

At this point, the exploration model consults the executive model and communicates, among other facts, updated expectations about the redox system length in the current A_3 and economic conditions prevailing in the system. The executive model decides how many holes (or fences, as pertinent) the exploration model shall drill in the next activity (again, zero holes or zero fences imply cessation of exploration of current A_3).

When Phase 3B drilling has been unsuccessful at this point and the executive model dictates additional Phase 3B holes, these are added to the running total of Phase 3B holes, n_R, and Phase 3B is repeated. On the other hand, if Phase 3B has been successful and the executive model determines that additional drilling is worthwhile, Phase 4A is initiated by drilling the f_{4A} fences program dictated by the executive model.

8.7.2 Comments on Interpolation of P3B.2 Tables

In general, system values for the actual redox length in the current A_3 (L), number of holes drilled in the prior exploration phase

(n_c), and number of holes drilled so far in Phase 3B (n_R) will not coincide with those values specified in the elicitation of Tables P3B.2. Thus, a three-dimensional linear interpolation is in order.

Rather than interpolating on actual parameters for the triangular redox length distribution, it is desirable[1] to interpolate on each of the following three measures, all three calculated from the triangular distribution parameters in the table:

$$\mu = \left| \frac{T_1 + T_2 + T_3}{3} - L \right|$$

$$\alpha = T_2 - T_1$$

$$\gamma = T_3 - T_2$$

where

T_1, T_2, T_3 = the triangular distribution parameters corresponding to the "low," "most likely," and "high" redox lengths

L = actual length existing in the A_3

μ = expected absolute deviation from the true value L

α = difference of most likely and low

γ = difference of high and most likely

---

1. To ensure consistency in the triangular distribution parameters; i.e., $T_1 \leq T_2 \leq T_3$, where T_1, T_2, and T_3 are the low, most likely, and high values. Also, to capture the trend in error improvement and improvement in unbiasedness (e.g., T_1, T_2, and T_3 approaching the actual length value as n_R increases).

Interpolation Procedure.

1. Determination of the enveloping labels: Generally, since the table is three-dimensional, we shall be given three entries E_1, E_2, and E_3, and will have to determine the enveloping labels from each set of dimension headings: $D1_i$, $D1_{i+1}$, $D2_j$, $D2_{j+1}$, $D3_k$, and $D3_{k+1}$. As we can deal with any single dimension separately, we shall simply refer generally to D_i and D_{i+1} as the appropriate enveloping labels, and to E as the entry corresponding to the particular dimension of D_i and D_{i+1}. We shall have two sets of parameters (T_{1i}, T_{2i}, T_{3i}) and (T_{1i+1}, T_{2i+1}, T_{3i+1}) corresponding to D_i and D_{i+1}. We shall also have ℓ_i and ℓ_{i+1} representing the redox length being estimated by the two sets of parameters, respectively. If $D \equiv D1$ (length), then $D_i = \ell_i$ and $D_{i+1} = \ell_{i+1}$; otherwise, $\ell_1 = \ell_{i+1}$. Step (1) is determination of enveloping labels D_i and D_{i+1}.

2. Determination of the "sign" variable δ: Determine the differences $d_i = (T_{1i} + T_{2i} + T_{3i})/3 - \ell_i$ and $d_{i+1} = T_{1i+1} + T_{2i+1} + T_{3i+1})/3 - \ell_{i+1}$. Then find δ as follows: if $d_i \leq 0$ and $d_{i+1} \leq 0$, and either $d_i \neq 0$ or $d_{i+1} \neq 0$, then $\delta = -1$; if $d_i \geq 0$ and $d_{i+1} \geq 0$, then $\delta = 1$; if $d_i < 0$ and $d_{i+1} > 0$, or $d_i > 0$ and $d_{i+1} < 0$, then there exists D, $D_i < D < D_{i+1}$, for which a linearly interpolated value d, falling between d_i and d_{i+1}, is exactly zero. Specifically, the interpolation equation:

$$d_i + \left(\frac{D - D_i}{D_{i+1} - D_i}\right)\left(d_{i+1} - d_i\right) = 0$$

yields the solution

$$D = d_i - \frac{(D_{i+1} - D_i)d_i}{(d_{i+1} - d_i)}$$

In this case, if $E < D$ and $d_i < 0$, or if $E \geq D$ and $d_{i+1} < 0$, then $\delta = -1$; if $E < D$ and $d_i > 0$, or if $E \geq D$ and $d_{i+1} > 0$, then $\delta = 1$. Step (2) is the determination of δ.

3. Determination of the interpolation measures for the two sets of parameters upon which the interpolation will be based: Using the appropriate parameter set, determine for both D_i and D_{i+1} the expected absolute deviation, and distances low to most likely and most likely to high: μ, α, and γ, respectively. These are computed (for D_i) as follows:

$$\mu_i = \frac{T_{1_i} + T_{2_i} + T_{3_i}}{3} - \ell_i$$

$$\alpha_i = T_{2_i} - T_{1_i}$$

$$\gamma_i = T_{3_i} - T_{2_i}$$

Likewise, compute μ_{i+1}, α_{i+1}, and γ_{i+1}.

4. Interpolation on the measures μ, α, and γ: Interpolate using the value of E on the computed values μ_i, μ_{i+1}, α_i, α_{i+1}, γ_i, and γ_{i+1} for the values C'_1, C'_2, and C'_3. Specifically, letting

$$F = \frac{E - D_i}{D_{i+1} - D_i}$$

we have

$$C'_1 = \mu_i + F \cdot (\mu_{i+1} - \mu_i)$$

$$C'1 = \alpha_i + F \cdot (\alpha_{i+1} - \alpha_i)$$

$$C'_3 = \gamma_i + F \cdot (\gamma_{i+1} - \gamma_i)$$

Now let

$$C_1 = \delta \cdot \max\{C'_1, 0\},$$

$$C_2 = \max\{C'_2, 0\}, \text{ and}$$

$$C_3 = \max\{C'_3, 0\}$$

5. <u>Solution of three simultaneous equations for the desired triangular distribution parameters</u>: Letting $L = \ell_i$ if $\ell_i = \ell_{i+1}$, or $L = E$ otherwise, solve the following system of equations for T_1, T_2, and T_3:

$$\frac{T_1 + T_2 + T_3}{3} - L = C_1$$

$$T_2 - T_1 = C_2$$

$$T_3 - T_2 = C_3$$

These yield solutions:

$$T_2 = L + C_1 + \frac{1}{3}(C_2 - C_3)$$

$$T_1 = T_2 - C_2$$

$$T_3 = T_2 + C_3$$

6. Since C_2 and C_3 are non-negative, it is easy to see that $T_1 \leq T_2 \leq T_3$ necessarily. It is still possible, however, to derive negative values for the parameters. Hence, we shall set:

$$T_j \leftarrow \max\{T_j, 0\} \qquad j \in \{1,2,3\}$$

<u>Numerical Example of the Interpolation Procedure in Tables P3B.2 of Appendix F</u>. Suppose it is desired to obtain from Tables P3B.2 the

redox length triangular distribution parameters T_1, T_2, and T_3 (low, most likely, and high) under the following conditions: redox length = 1.2 miles, n_R = 18 holes, and n_c = 29 holes. Consulting Table P3B.2, we determine the enveloping labels to be, for length, 1.0 and 5.0; for n_R, 10 and 20; and for n_c, 25 and 35. We commence with interpolation for the length, 1.2 miles. A table giving $d_i = (T_{1i} + T_{2i} + T_{3i})/3 - \ell_i$ for each combination of ℓ, n_R, and n_c above is given as follows:

	$n_c = 25$			$n_c = 35$	
	n_R			n_R	
ℓ	10	20	ℓ	10	20
1.0	-0.1667	-0.1333	1.0	0.1	0.0667
5.0	-1.1667	-0.6667	5.0	-0.6667	0.0

For the case n_n = 25, n_R = 10, since d_i is negative for both D_2 = 1.0 and D_3 = 5.0, step (2) of the interpolation procedure gives us $\delta = -1$. We can compute $\mu_2 = |(0.4 + 0.6 + 1.5)/3 - 1.0| = 0.1667$, $\alpha_2 = 0.6 - 0.4 = 0.2$, and $\gamma_2 = 1.5 - 0.6 = 0.9$, as well as $\mu_3 = |(2.0 + 3.5 + 6.0)/3 - 5.0| = 1.1667$, $\alpha_3 = 3.5 - 2.0 = 1.5$, and $\gamma_3 = 6.0 - 3.5 = 2.5$. In step (4), we compute F = (1.2 - 1.0)/(5.0 - 1.0) = 0.05, and:

$$C'_1 = 0.1667 + 0.05\ (1.1667 - 0.1667) = 0.2167$$

$$C'_2 = 0.2 + 0.05\ (1.5 - 0.2) = 0.265$$

$$C'_3 = 0.9 + 0.05\ (2.5 - 0.9) = 0.98$$

Hence C_1 = -0.2167, C_2 = 0.265, and C_3 = 0.98. We then obtain T_2 = 1.2 + (-0.2167) + 1/3(0.265 - 0.98) = 0.745, T_1 = 0.48, and T_3 = 1.725, which remain unchanged by step (6). (Note than these are the same numbers that would result from a straightforward linear interpolation in the table.)

Using the same methodology for the case $n_c = 25$, $n_R = 20$, we again obtain, by observation, $\delta = -1$. The remaining computations are as follows:

$\mu_2 = 0.1333$		$\mu_3 = 0.6667$
$\alpha_2 = 0.2$		$\alpha_3 = 1.0$
$\gamma_2 = 0.4$		$\gamma_3 = 2.0$
$C_1 = -0.16$	$C_2 = 0.24$	$C_3 = 0.48$
$T_2 = 0.96$	$T_1 = 0.72$	$T_3 = 1.44$

These again are the same parameters yielded by straightforward linear interpolation. In the next case, however, results will differ slightly.

Third case: $n_c = 35$, $n_R = 10$. Here we have $d_2 = 0.1$ and $d_3 = -0.6667$, of opposite signs; hence we must solve for D, that length entry which would give us $\ell = 0$:

$$D = 1.0 - (5.0 - 1.0)(0.1)/(-0.6667 - 0.1) = 1.5217$$

Since $1.2 < 1.5217$ and $0.1 > 0$, we have $\delta = 1$. Intuitively, this can be seen if we add the 1.5217 length to the table giving d_1 for $n_c = 35$ and $n_R = 10$:

	$n_c = 35$
ℓ	$n_R = 10$
1.0	0.1
1.5217	0.0
5.0	-0.6667

Since 1.2 falls between 1.0 and 1.5217, and entries for those are non-negative, δ is positive. We proceed as usual:

$\mu_2 = 0.1$		$\mu_3 = 0.6667$
$\alpha_2 = 0.8$		$a_3 = 1.0$
$\ell_2 = 0.2$		$\ell_3 = 2.0$
$C_1 = 0.1283$	$C_2 = 0.81$	$C_3 = 0.29$
$T_2 = 1.5016$	$T_1 = 0.6916$	$T_3 = 1.7916$

Straightforward linear interpolation would yield $T_1 = 0.625$, $T_2 = 1.435$, and $T_3 = 1.725$.

For the case $n_c = 35$, $n_R = 20$, we can observe d = 1 again. Computations yield:

$\mu_2 = 0.0667$		$\mu_3 = 0.0$
$\alpha_2 = 0.2$		$\alpha_3 = 1.0$
$\ell_2 = 0.1$		$\ell_3 = 1.0$
$C_1 = 0.0633$	$C_2 = 0.24$	$C_3 = 0.145$
$T_2 = 1.295$	$T_1 = 1.055$	$T_3 = 1.44$

We now have reduced sets of parameters for $\ell = 1.2$ miles:

$\ell = 1.2$

n_c	$n_R = 10$			$n_R = 20$		
	Low	Most Likely	High	Low	Most Likely	High
25	0.48	0.745	1.725	0.72	0.96	1.44
35	0.6916	1.5016	1.7916	1.055	1.295	1.44

The corresponding entries d_i will be:

$\ell = 1.2$

n_c	n_R 10	20
25	-0.2167	-0.16
35	0.1283	0.0633

Interpolating for $n_2 = 29$, we obtain, for $n_R = 10$, D = 31.28, and for $n_R = 20$, D = 32.16; in both cases E < D and $d_i < 0$, hence $\delta = -1$. For $n_R = 10$, $\mu_1 = 0.2167$, $\alpha_1 = 0.265$, and $\gamma_1 = 0.98$, while $\mu_2 = 0.1283$, $\alpha_2 = 0.81$, and $\gamma_2 = 0.29$; for $n_R = 20$, $\mu_1 = 0.16$, $\alpha_1 = 0.24$, $\gamma_1 = 0.48$, and $\mu_2 = 0.0633$, $\alpha_2 = 0.24$, $\gamma 2 = 0.145$. Here F = (29 - 25)/(35 - 25) = 0.4; so, for $n_R = 10$, $C_1 = -0.1813$, $C_2 = 0.483$, and $C_3 = 0.704$, while for $n_R = 20$, $C_1 = -0.1213$, $C_2 = 0.24$, and $C_3 = 0.346$. Solving for the parameters, we obtain the following table:

$\ell = 1.2$ and $n_c = 29$

	Low	Most Likely	High
10	0.4620	0.9450	1.649
20	0.8034	1.0434	1.3894

Next, we interpolate for $n_R = 18$. For step (2), $d_1 = -0.1813$ and $d_2 = -0.1213$, hence $\delta = -1$. We proceed:

$\mu_1 = 0.1813$ $\qquad$ $\mu_2 = 0.1213$

$\alpha_1 = 0.483$ $\qquad$ $\alpha_2 = 0.704$

$\gamma_1 = 0.24$ $\qquad$ $\gamma_2 = 0.346$

F = (18 - 10)/(20 - 10) = 0.8

$C_1 = -0.1333$ $\qquad$ $C_2 = 0.6598$ $\qquad$ $C_3 = 0.3248$

The interpolated parameters of the triangular distribution for $\ell = 1.2$, $n_c = 29$, and $n_R = 18$, then, are:

Low	Most Likely	High
0.5185	1.1783	1.5031

8.8 Phase 4A: Deposit Search

8.8.1 Design

Figure G.8 of Appendix G shows schematically the design for the modeling of exploration in Phase 4A.

Before Phase 4A drilling in the current A_3 begins, the exploration model performs the following operations to generate targets to be searched for:

- Identify the mineralized pods existing in the current A_3.
- Group the pods into individual deposits.
- Assign to each deposit its companion redox system length. This is done by simply halving the gap distance [gap(.01) value] between two deposits.

Grouping in its simplest form can be done by comparing the gap distance between pods with a threshold value and group if that value is not exceeded.

A more sophisticated grouping procedure is to group those pods that might be mined together, i.e., constitute one mine. This procedure would require the computation of the net present value of each arrangement (via the mining cost routine) and the selection of that arrangement which yields the maximum net present value.

These operations, then, produce a collection of geologic targets[1] in the current A_3. In general, each target[2] consists of two sets of objects:

- a set of mineralized pods (with their corresponding "in between" gaps).
- a redox length, L_D, referred hereafter as the "search length," along which the mineralized pods occur.

For each one of the targets generated, two minimum criteria are determined: the minimum attractive cutoff grade, q_{min}, and the minimum attractive average mineralization width,[3] Ω_{min}. These minimum criteria are a function of three main variables: the expected depth ($\overline{dpth}$), current economic conditions in the system (i.e., price/cost, assuming that the rest of the economic conditions remain invariant throughout the potential supply evaluation), and average thickness of the favorable formation (τ). These functions for minimum criteria are determined externally to the system by a design described in subsection 8.8.4.

---

1. Whereas in Phases 3A and 3B the exploration unit is the A_3 and the target is the redox system, in Phase 4A and subsequent phases, things are different: the exploration unit is the "band" along a portion of the redox length in the current A_3 and the target is the collection of mineralized pods constituting the deposit.

2. If the redox system existing in the A_3 is barren or if only one deposit (collection of pods) exists, then only one target will be generated. Thus, the target can be mineralized or not.

3. The average mineralization width, Ω, is not to be confused with the width of a confining prism, w. For any given pod, $\underline{w} \geq \overline{\Omega}$. Table SM.6 (Appendix E) provides the relationship between w and Ω.

The minimum attractive criteria determination emulates the exploration agent's rules of thumb for economic deposits. This operation reflects this study's effort to separate economics from the description of the physical exploration process. With such a scheme, exploration can be modeled under different economic conditions (e.g., price/cost levels) by superimposing, in the model operation, the desired economic scenarios.

Given the minimum attractive cutoff grade, q_{min}, determined above, the pertinent attributes of each target[1] can be generated:

- $\ell(q_{min})$, the actual length of the deposit at a cutoff grade equal to q_{min}.
- $\overline{\Omega}_D(q_{min})$, the actual average width of the mineralization above cutoff grade q_{min} in the deposit D.
- W(.01), the width of the .01-confining prism of the deposit.
- $w_D(q_{min})$, the width of the q_{min}-confining prism of the deposit.

These attributes of targets in the current A are important because they affect subsequent exploration outcomes when the target search is simulated in the model.

At this point, the system has "prepared" the A_3 so that Phase 4A drilling activities can be initiated. Phase 4A drilling is conducted for each one of the targets in the current A_3 on an individual basis. The search for another target is initiated only when a target is discovered and its contribution to potential supply determined (i.e., all subsequent exploration phases completed) or when exploration of a target is

1. If no deposit exists, these attributes are, of course, all zero.

abandoned. When no targets remain to be considered for exploration, drilling of the current A_3 is terminated and a new A_3 is considered for exploration.

The basic exploration tool in Phase 4A is the drilling of "fences" (rows of holes) rather than individual holes (see description of exploration architecture by the respondent in Appendix B). Number of fences, f_D, to be drilled initially in each target is determined as follows:

$$f_D = \frac{L_D}{L} f_{4A}$$

where

f_D = number of fences employed to initiate Phase 4A drilling program for this individual target (the d^{th} target)

f_{4A} = number of fences employed to initiate Phase 4A in the entire A_3 area. This number is determined by the executive model in Phase 3B (see Figure G.7 in Appendix G)

L = length of redox front system existing in the entire A_3

L_D = length of the redox front system assigned to this individual target (the d^{th} target) $L_D \leq L$.

Table P4A.3 is employed to obtain the probability of detecting mineralization above the minimum attractive cutoff grade when f_D fences are drilled; details on this table consultation are provided in the next subsection. Upon identification of the appropriate probability, an outcome (detection or non-detection) is generated by Monte Carlo techniques.

In this phase, expectations about the length of the mineralized target[1] are of primary importance in the inferences (by the executive model) about the amount of mineralization in the portion of the redox front system being explored. This posterior probability distribution of length of the mineralized target after drilling f_D fences (and under a set of conditions about actual target characteristics and minimum criteria, q_{min} and $\bar{\Omega}_{min}$, being searched for) is obtained directly[2] from Tables P4A.5 in Appendix F.

At this point the executive model is consulted by the exploration model, communicating, among other facts, current expectations about the length of the mineralized target and current economic conditions in the system. The executive model determines the number of holes (or additional fences if Phase 4A has been unsuccessful so far) the exploration model will drill in the next activity.

Costing of Phase 4A drilling is performed until Phase 4A has been totally completed (either successfully or unsuccessfully) for the target under consideration. Table P4A.1 is consulted[3] to obtain the parameters of the triangular distribution of average number of holes per fence. This distribution is sampled by Monte Carlo techniques to compute the

---

1. Recall that in previous phases redox length was the key to making inferences about the endowment.

2. That is, no revision of a prior distribution is needed.

3. A three-dimensional linear interpolation procedure on fD/L, W(.01), and Ω_{min} is needed for Table P4A.1.

total number of holes[1] drilled in Phase 4A for the current target under investigation. Given the total number of holes, each hole is costed according to its depth (employing the drilling cost relationships). Depth is generated by Monte Carlo sampling of a uniform depth distribution; parameters of this distribution are the minimum and maximum depths of the pods composing the deposit (if no deposit exists, these parameters are the minimum and maximum depths to the favorable host in the current A_3).

8.8.2 Details on Table P4A.3 Interpolation and Adjustment

In general, the actual conditions in the system (f_D, $\ell(q_{min})$, q_{min}, Ω_{min}, etc.) will seldom coincide with the conditions specified in the elicitation of responses to question P4A.3 (see Appendix F). Consequently, a three-dimensional linear interpolation and a subsequent adjustment are required.

Suppose it is desired to obtain the success probability $P4A3[S \mid f_D, L_D, \ell'(q'_{min}), \bar{\Omega}'(q_{min}), \bar{\Omega}'_{min}, q'_{min}]$. A three-dimensional interpolation on f_D/L, $\ell(q_{min})/L$ and q_{min} in Table P4A.3 gives the following unadjusted[2] probability value:

---

1. If Phase 4A has been successful, Table P4A.7 is employed to obtain the additional number of holes needed to verify the discovery.

2. Note that Table P4A.3 (see Appendix F) conditions used for elicitation stated that both the actual average mineralization width at a cutoff $q_{min}[\Omega(q_{min})]$ and the minimum attractive average mineralization width (Ω_{min}) are equal.
Also note that in the tables pertinent to Phase 4A (Appendix F), the L used to indicate redox length is equivalent to the "search length" notion, L_D, used in this section.

$$P4A3[S \mid f'_D,\ L'_D,\ \ell'(q'_{min}),\ q'_{min}, \overline{\Omega}(q'_{min}) = \overline{\Omega}_{min}]$$

This probability must be adjusted to account for the fact that, in general, $\overline{\Omega}'_{min} \neq \overline{\Omega}(q'_{min})$. Table series P4A.4 is needed to perform this adjustment.

Let P4A4 $[\alpha \mid \overline{\Omega}'(q'_{min}), \overline{\Omega}'_{min}, \ell'(q'_{min}), L'_D, f'_D, q'_{min}]$ represent the percent adjustment (α) required to account for the fact that $\overline{\Omega}'(q'_{min}) \neq \overline{\Omega}'_{min}$. The value of this adjustment, α, is obtained by a four-dimensional linear interpolation on $\overline{\Omega}(q_{min})/\overline{\Omega}_{min}, f_D/L_D, q_{min}$, and $\ell(q_{min})/L_D$.

Thus the final[1] (adjusted) value for probability of detection is obtained by:

$$P4A3[S \mid f'_D,\ L'_D,\ \ell'(q'_{min}),\ \overline{\Omega}'(q'_{min}),\ \overline{\Omega}'_{min},\ q'_{min}] =$$

$$(1 + \frac{\alpha}{100}) \cdot P4A3[S \mid f'_D,\ L'_D,\ \ell'(q'_{min}),\ q'_{min},\ \overline{\Omega}(q'_{min}) = \Omega_{min}]$$

8.8.3 <u>Details on Table P4A.5 Interpolation and Adjustment</u>

The problem in the consultation of this table series is analogous to that of Table P4A.3; that is, actual conditions in the system [f_D, $\ell(q_{min})$, q_{min}, etc.] in any one simulation of a target search will seldom coincide with conditions specified to elicit the responses in Table P4A.5. The proposed solution is similar: a three-dimensional interpolation and

---

1. If the adjustment implies a probability value <0% or >100%, the final value is made equal to 0 or 100 as pertinent.

Also note that before the interpolation, Tables P4A.4 have to be expressed in terms of percent difference with respect to the central column.

a subsequent adjustment. This procedure is done for each fractile of Table series P4A.5 on an individual basis.

Suppose it is desired to obtain the probability distribution for target length after Phase 4A drilling activities have been conducted. This probability distribution is described in Table series P4A.5 in terms of five fractiles.

Let:

$\ell_x(q_{min}) \mid S', f'_D, Th', L'_D, q'_{min}, \Omega'_{min}$ = the x% fractile of the desired probability distribution for target length

where

x = 10%, 25%, 50%, 75%, 90% (see table series P4A.5, Appendix F)

S = indicator variable for success/failure in Phase 4A

f_D = number of fences drilled so far in Phase 4A

Th = thickness of the composite geochemical cell in the neighborhood being explored

L_D = length of the redox front system being explored for this mineralized target (i.e., the "search length") (in table series P4A.5 it is noted as "L")

q_{min} = minimum attractive cutoff grade being searched for

$\overline{\Omega}_{min}$ = minimum attractive average mineralization width being searched for

′ denotes a specific value for the variable (not necessarily corresponding to any of the values used in the conditions stated to elicit responses)

A two-dimensional interpolation on q_{min} and f_D /L in the appropriate Table P4A.5 (according to the value of the success/failure indicator variable) yields the following unadjusted fractile value for target length (this value is for the reference redox length, L, used in the elicitation of question P4A.5 and P4A.6 of Appendix F):

$$\ell^\circ_x(q_{min}) \mid S', f'_D, Th, L, q'_{min}, \bar{\Omega}_{min}$$

Note that in this unadjusted value, Th is the reference composite geochemical cell thickness value used in the success and non-success tables (100'). Similarly, the interpolation on q_{min} determines $\bar{\Omega}_{min}$, the reference minimum attractive width.

Table series P4A.6 (Appendix F) is needed to adjust the fractiles for the fact that in general $\bar{\Omega}'_{min} \neq \Omega_{min}$ and Th' $\neq$ Th = 100'.

Let:

$$\gamma_x = \frac{\ell_x(q_{min}) \mid S', f'_D, Th', L, q'_{min}, \bar{\Omega}'_{min}}{\ell^\circ(q_{min}) \mid S', f'_D, Th, L, q'_{min}, \bar{\Omega}_{min}}$$

where

γ_x = adjustment factor for the x% fractile

Table series P4A.6 can be submitted to regression analysis to obtain a function relating the fraction γ_x with minimum average mineralization width, $\bar{\Omega}_{min}$, thickness of the composite geochemical cell, Th, and cutoff grade, q_{min}. A plausible function is one of the form:

$$\gamma_x = e^{b_1 \cdot \overline{\Omega}_{min} + \beta \cdot Th + b_3 \cdot \overline{\Omega}_{min} \cdot Th + b_4 \cdot q_{min}}$$

where

b_1, b_2, b_3, b_4 = coefficients to be estimated by regression analysis

Thus, the final value for the x% fractile of the target length distribution is obtained by:

$$\ell_x(q_{min}) \mid S', f'_D, Th', L'_D, q'_{min}, \overline{\Omega}'_{min} =$$

$$\gamma_x \cdot [\ell^\circ{}_x(q_{min}) \mid S', f_D, Th, L, q'_{min}, \ell_{min}] \cdot \frac{L'_D}{L}$$

8.8.4 <u>Design of an External Routine to Generate Data for the Two Minimum Criteria Functions Employed in Phase 4A</u>

In the design of the exploration model, the following two minimum criteria functions are proposed:

$$q_{min} = q(\overline{dpth}, P, \tau)$$

$$\overline{\Omega}_{min} = \Omega(\overline{dpth}, P, \tau)$$

where

q_{min} = minimum attractive cutoff grade

$\overline{\Omega}_{min}$ = minimum attractive average mineralization width

$\overline{dpth}$ = expected depth of the target

P = uranium price/cost (\$/lb U_3O_8)

τ = average thickness of the favorable formation in the region (A_F)

q, Ω = functions

To estimate the functions suggested above, it is necessary to generate data on the pertinent variables, propose a functional form, and submit these to regression analysis. MINATT is the data generation routine; its design is shown schematically in Figure 8.3. Basically, this routine generates a sequence of mineralized pods and gaps using the probability distributions of key pod attributes obtained in the subjective morphology questionnaire (Appendix E). These pods are then grouped into individual deposits. Each deposit is passed through the mining cost model[1] (open pit) which determines the optimum cutoff grade under that particular price (\$/lb U_3O_8) and depth conditions. A number of different deposits are generated, and their optimum cutoff grade and average mineralization width at that cutoff are compared with those of the rest of the deposits. Minimum cutoff grade and average mineralization width is always retained for each combination of price, depth, and thickness

1. Since only open pit is considered in this potential supply system design, it is not claimed here that data generated is complete. However, it provides a satisfactory representation for those deposits in the domain of adequate depths for open pit.

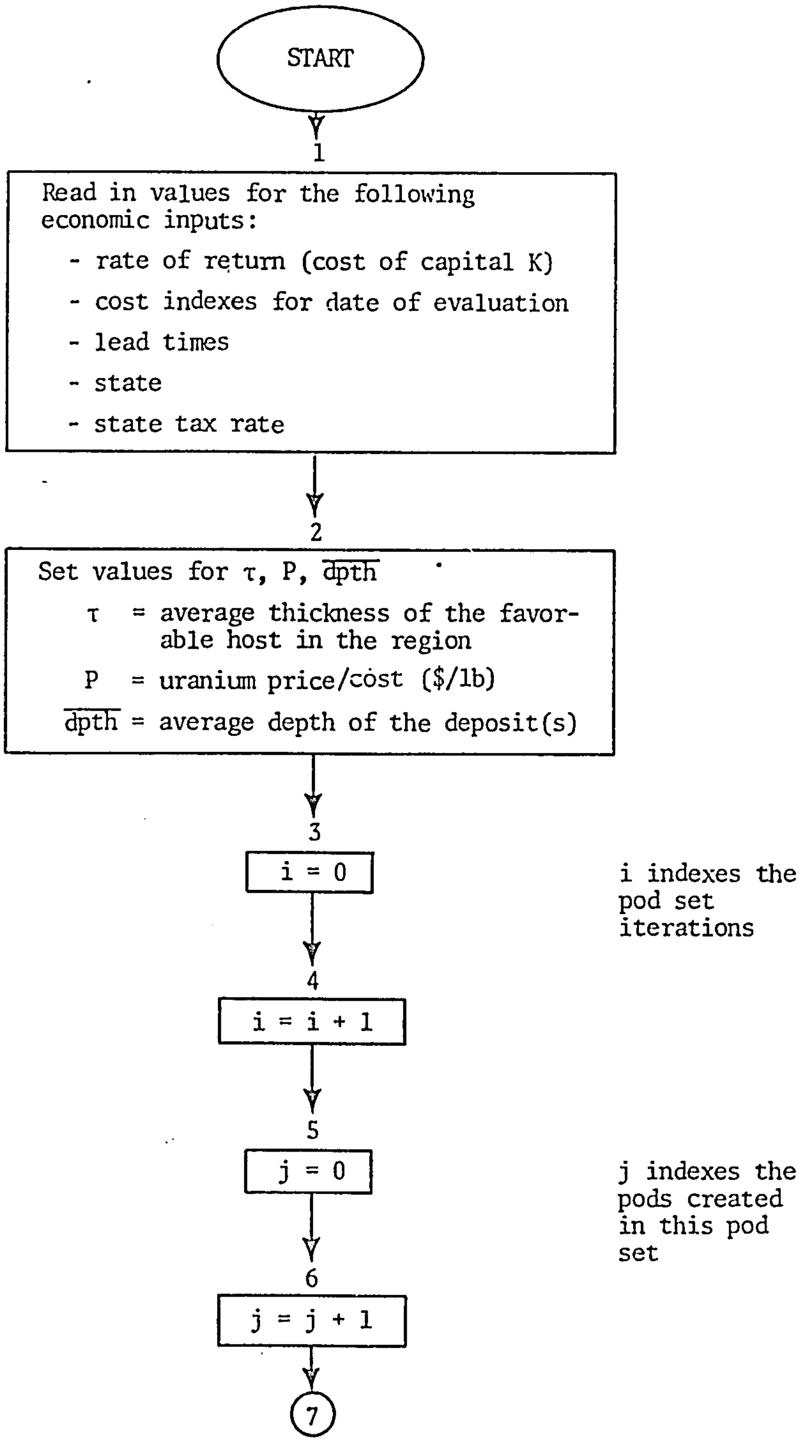

Figure 8.3. Schematic flow diagram of the MINATT routine.

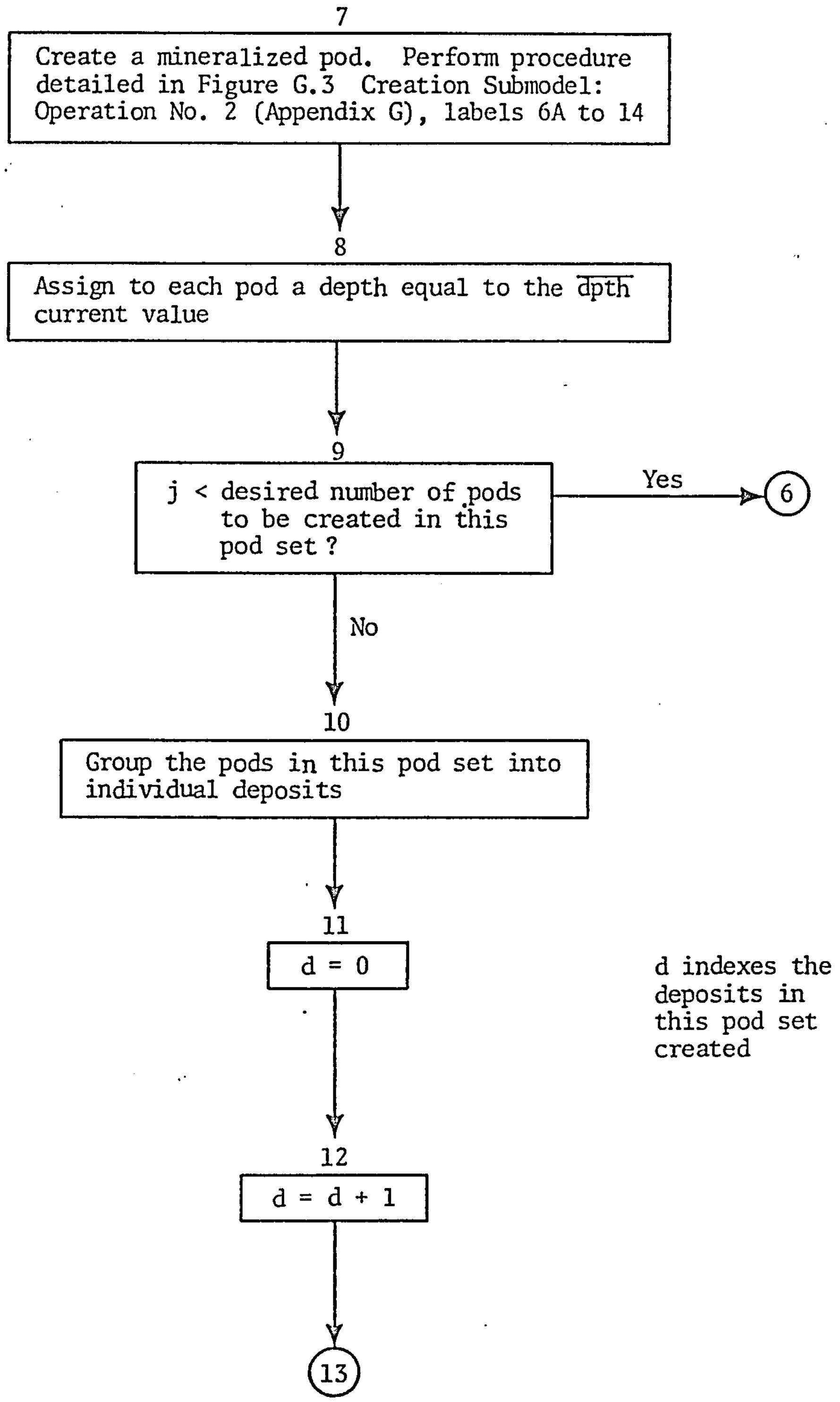

Figure 8.3.--Continued

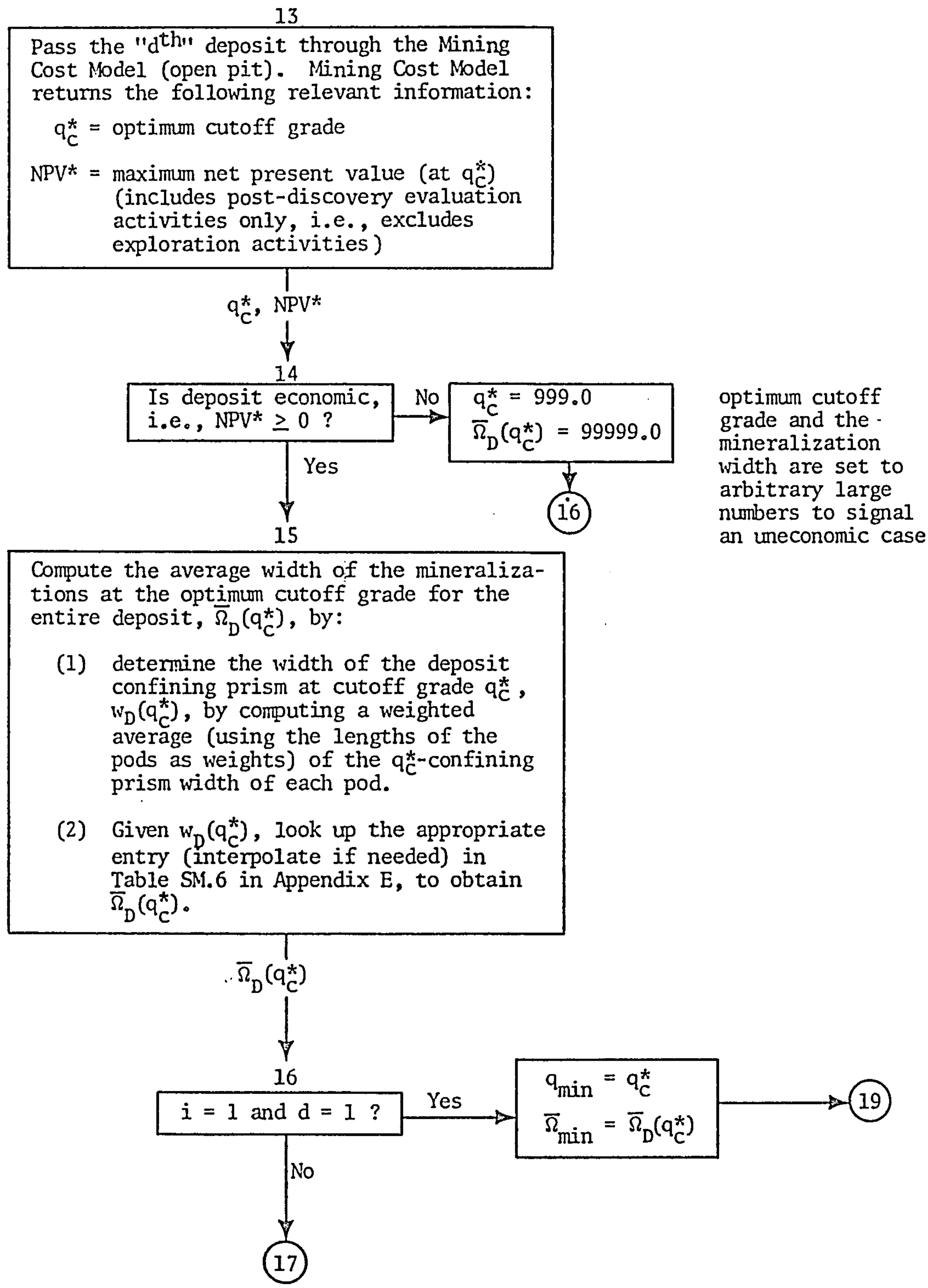

Figure 8.3.--Continued

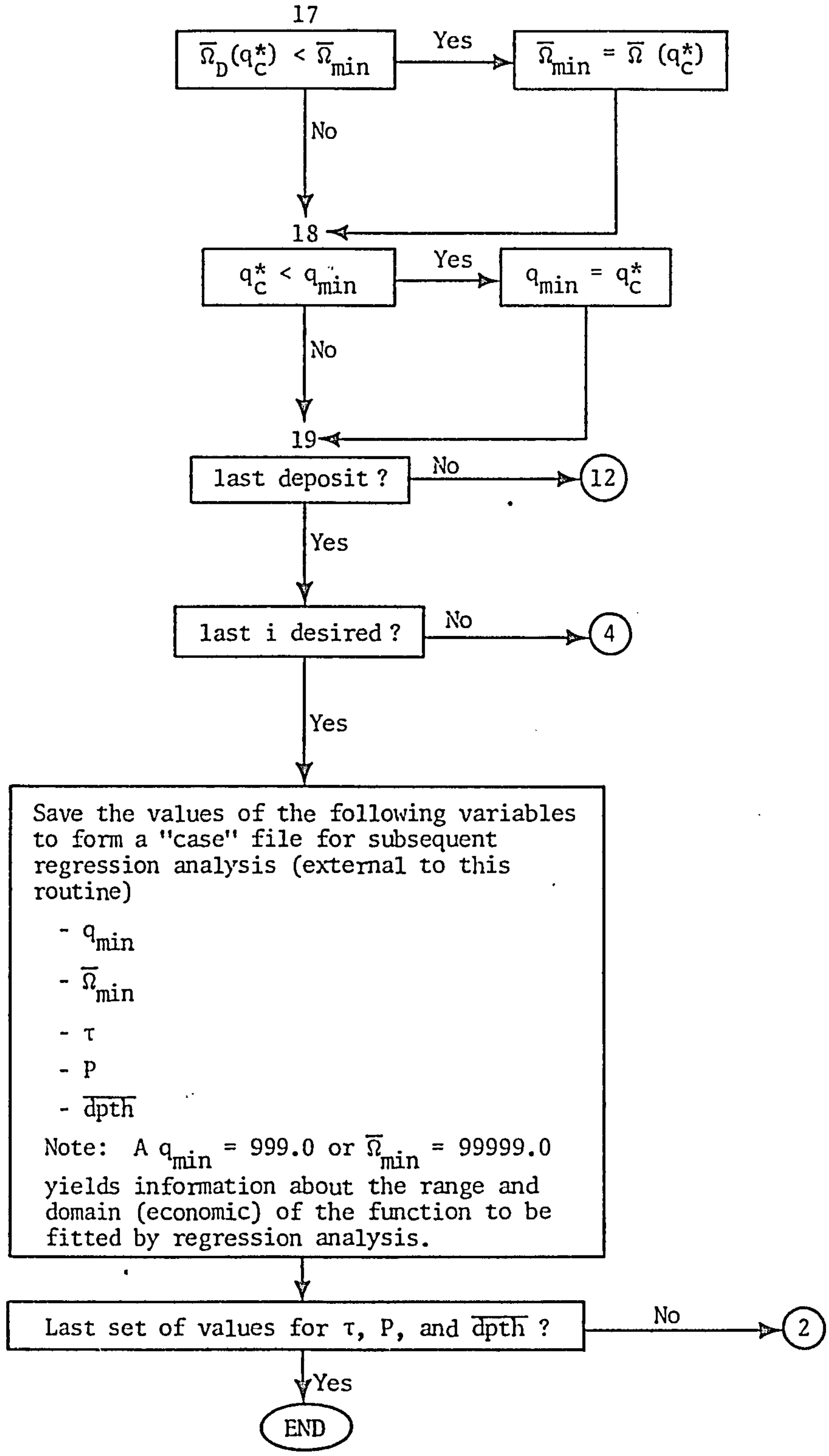

Figure 8.3.--Continued

of favorable formation to form a data file for subsequent regression analysis.

It is important to mention that the minimum criteria generated by this design reflects the probability distribution information of the mineralized pods' key attributes to allow us to obtain a satisfactory estimate of the exploration agent's minimum criteria rules that commonly would be obtained by "experience" on known deposits.

8.9 Phase 4B: Discovery Evaluation

Both Phases 4B and 5 are postdiscovery phases. The objective of these phases is evaluation of the discovery at increasing levels of accuracy. Figure G.9 in Appendix G shows the design for modeling of Phase 4B. Holes drilled in Phase 4B are aimed toward achieving a ±50% error at 80% confidence (see response to question P4B5.1 in Appendix F). The initial number of holes to be drilled in Phase 4B n_{4B}, are determined in the last consultation of Phase 4A to the executive model for this deposit being investigated.

Phase 4B is initiated by drilling n_{4B} holes. Each hole is costed according to its depth, which is generated by Monte Carlo sampling of the (uniform) depth-to-deposit distribution (parameters of this distribution are simply the minimum and maximum depths of pods comprising the deposit). The exploration model proceeds to identify,[1] employing Table P4B5.3, the appropriate percent error probability distribution parameters (triangular)

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1. The same interpolation procedure used to interpolate the parameters of the triangular distributions in Table P3B.2 (see section 8.7.2) is required here).

when n_{4B} holes have been drilled under the conditions of the actual deposit being investigated (length and width of the deposit confining prism at the minimum attractive cutoff grade under current economic conditions, q_{min}). This probability distribution is sampled[1] using Monte Carlo techniques to obtain a specific value for the percent error in the estimation of global reserves (ERR_{4B}). This error figure is compared with the percent error objective, and if the latter has been met, Phase 4B drilling has been successful.

Assuming that the estimator for global reserves is unbiased and normally distributed [i.e., ~ $N(\mu_u, \sigma_u)$], the actual amount of U_3O_8 contained in mineralization above q_{min}, $m_D(q_{min})$, is used as the mean μ_u. The variance σ^2_u is obtained from the error figure ERR_{4B} by solving the following equation for σ_u:

$$-1.28 = \frac{a - \mu_u}{\sigma_u}$$

where

-1.28 = Z value in the standard normal table for (100-80/2)% probability

$$a = \mu_u(1 - \frac{ERR_{4B}}{100})$$

1. For any given deposit being evaluated, the distribution is sampled only the first time through this probability (see section 8.10.2 of this chapter for details).

At this point, the exploration model consults the executive model, communicating, among other facts, the current expectations of global reserves [$\sim N(\mu_u \sigma_u)$] at cutoff grade q_{min}. The executive model determines how many holes the exploration model shall drill in the next drilling program (again, zero holes imply termination of exploration of this deposit).

When Phase 4B drilling has been unsuccessful (i.e., % error objective not met yet) and the executive model dictates to drill additional 4B holes, these are added to the running total of Phase 4B and Phase 5 holes, n_E, and Phase 4B is repeated. A procedure to avoid inconsistencies due to repeated consultation of Table P4B5.3 in Phase 4B or Phase 5 for the same deposit is described in section 8.10.2 in this chapter.

If Phase 4B has been successful and the executive model determines that continued drilling is worthwhile, Phase 5 drilling activities are initiated.

8.10 Phase 5: Continuing Evaluation

8.10.1 Design

The objective of drilling activities during Phase 5 is to decrease the percent error in evaluating the deposit discovered in Phase 4A. Target error for the estimation of global reserves is expressed in question P4B5.2. Figure G.10 in Appendix G shows the design for modeling of Phase 5, which is similar to that of Phase 4B.

Phase 5 is initiated by drilling n_5 holes. Each hole is costed according to its depth, which is generated by Monte Carlo sampling of the (uniform) depth-to-deposit probability distribution.

Table P4B5.3 in Appendix F is employed here again to "conduct" Phase 5 drilling and produce an outcome in terms of the percent error in the estimate of global reserves. Number of holes to be drilled in the initial Phase 5 drilling program, n_5, is added to the total of holes drilled in Phase 4B and stored in n_E, the variable name for the running total of Phase 4B and Phase 5 holes. The appropriate parameters for the triangular probability distribution for percent error when n_E holes have been drilled are identified in this table. This probability distribution is not sampled; instead, the specific outcome for percent error in the previous Phase 4B is employed with the parameters of the triangular distribution for Phases 4B and 5 to compute an error for Phase 5. This procedure is described in section 8.10.2. The percent error figure obtained (ERR_5), is compared to the desired error figure; if it has been met, Phase 5 drilling has been successful.

The probability distribution of global reserves [$\sim N(\mu_u, \sigma_u)$] is obtained in a manner similar to the procedure described in Phase 4B. The exploration model consults the executive model, communicating, among other facts, current probability distribution for global reserves.

If Phase 5 drilling has been unsuccessful (i.e., % error objective not met yet), the executive model determines if more Phase 5 holes shall be drilled to improve the accuracy figure. If so, these additional holes are added to the running total of Phase 4B and Phase 5 holes, n_E, and Phase 5 is repeated.

If Phase 5 has been successful, the executive model consultation is bypassed and the deposit is passed through the mining cost model. If the deposit is economic, its production is added to the potential supply of the region.

8.10.2 Comments on Avoiding Inconsistent Results Due to Repeated Consultation of Table P4B5.3 in Exploration Phases 4B and 5

A new consultation to Table P4B5.3 and the subsequent Monte Carlo sampling of the newly identified error probability distribution (triangular) for the increased number of holes (n_E) can yield inconsistent results. Specifically, the newly sampled value for % error in reserve estimation (ERR_{4B} or ERR_5) can be lower than the previously sampled percent error value, a value associated with fewer holes. To avoid this problem, the following procedure is suggested:

1. Save the first sampled value (for the current deposit being evaluated) for percent error ERR_{4B}, as well as the parameters of the triangular percent error distribution identified in the first consultation of Table P4B5.3.

2. In subsequent consultations of Table P4B5.3 (with increased number of holes, n_E), obtain a measure of improvement in the accuracy of the global reserve estimate. A plausible way of achieving this measure is to look at the trend of the expected error figure and compute a percent improvement.

T_1, T_2, T_3 = the (low, most likely, and high) parameters of the triangular percent error distribution identified in the <u>first</u> consultation to Table P4B5.3 (n_E holes drilled) for the deposit being evaluated.

ERR_{4B} = specific value resulting from Monte Carlo sampling of the triangular distribution T_1, T_2, T_3

T_1', T_2', T_3' = parameters of the triangular percent error distribution identified in any subsequent consultation to Table P4B5.3

n_E = holes drilled in Phase 4B initially (the first time)

n_E' = total holes drilled so far (in Phase 4B and Phase 5); $n_E' > n_E$

E(ERR) = mean percent error of the triangular distribution described by T_1, T_2, T_3 $E(ERR) = (T_1 + T_2 + T_3)/3$

E(ERR') = mean percent error of the triangular distribution described by T_1, T_2, T_3 $E(ERR') = (T_1 + T_2 + T_3)/3$

The percent improvement in the mean error when $n_E' - n_E$ additional Phase 4B or Phase 5 holes are drilled is:

$$\alpha = \frac{E(ERR') - E(ERR)}{E(ERR)} \times 100$$

Thus, the new specific error figure (ERR') when n_E total holes have been drilled can be computed by:

$$ERR' = ERR_{4B}(1 + \frac{\alpha}{100})$$

8.11 Contribution to Potential Supply

When a deposit is passed through the minimum cost model, the optimum cutoff grade q^*_c, the net present value, and the tonnage and average grade of mineralized material above that cutoff are determined (Figure G.11, Appendix G). If the deposit is economic (i.e., non-negative net present value), the amount of U_3O_8 contained in the mineralized material above the optimum cutoff grade q^*_c is added to the potential supply of the region for the current uranium price in the system. Then, the exploration model proceeds to simulate exploration of other targets (Figure G.12 in Appendix G).

When all the A_F region has been considered in the exploration model, the running total of potential supply contributions is classified in the appropriate quantity interval; these are histogram intervals which are constructed for the purpose of describing a relative frequency distribution of potential supply. Subsequent to this classification (frequency update for the appropriate quantity interval), a new endowment iteration is initiated, and the entire procedure is repeated.

8.12 The Drilling Cost Relationship

As mentioned in the model's description, each hole drilled is costed according to its depth. Such costing requires an equation relating drilling cost per foot (rotary drilling) with depth of the drill hole. An equation provided by DOE meets this general requirement:

$$DPF_t = \frac{WPI_t}{217.2} \cdot (2.015\ e^{.000578\ (Depth)} + 1.75)$$

where

where

DPF_t = average cost per foot (across all regions) in time t

WPI_t = WPI industrial commodity index in time t

217.2 = WPI industrial commodity index in 1979

Depth = drill hole depth in feet

1.75 = cost of geophysical logging ($/ft)

This drilling cost relationship is used by the exploration model, the executive model, and the mining cost model (for development drilling costing).

While the DOE cost relation meets the general requirements of the model, it is a general equation and represents average costs across regions. Obviously, cost relations specific to the various regions would be preferred.

CHAPTER 9

THE OPEN PIT MINING COST MODEL

9.1 Role of the Mining Cost Model

The open pit mining cost model (referred to as the "mining cost model") operates on a deposit-by-deposit basis. The mining cost model imitates the engineer's decision by selecting that mine design from a group of possible mining operations that yields the maximum net present value.

The mining cost model takes as inputs the economic conditions under which the evaluation is desired and the physical description of the geologic deposit (characterized by key attributes of the mineralized pods forming the deposit). Various mining cutoff grades and capacities are tested.[1] For each trial value, a pit configuration is determined, a full cash flow analysis is made, and a net present value is computed. This is repeated for a set of cutoff grades until that grade giving the maximum net present value under the prevailing price regime is found (see Figure 3 in Chapter 3). No attempt was made to incorporate inflation trends. Instead, a constant dollar analysis is employed. Since potential supply is a "stock" measure, it is an adequate approach.

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1. Only cutoff grades are manipulated. Mine capacity is a function of the model's cutoff grade, for it is determined by size of the deposit (tonnage above cutoff), which in turn depends on cutoff grade. Details are given in a later section.

9.2 Overview

The main body of the mining cost model can be perceived as a non-mathematically tractable "function" relating net present value (NPV) to cutoff grade (q_c) under a given set of economic conditions. A numerical optimization routine[1] is employed to find the cutoff grade value that maximizes the net present value of the operation. The numerical optimization routine works in an iterative fashion by examining a series of trial cutoff grade values, one at a time, and invoking the NPV-q_c "function" each time to evaluate deposit NPV at that trial cutoff. Choice of the specific trial cutoff grade is made by examining the gradient of the "function." Using this gradient as a guide to selecting cutoff grades, this procedure is repeated until that cutoff grade giving the maximum net present value is identified. The remainder of Chapter 9 deals with design and description of the components of this NPV-q_c "function." The schematic flow diagram of the mining cost model design is shown in a series of figures in Appendix J. The reader is encouraged to refer to this appendix throughout the remainder of this section.

The major steps in operating the mining cost model are:

1. Setting a trial cutoff grade, q_c.
2. Computing the "ore" deposit[2] geometry (i.e., dimensions, depths, and other deposits characteristics at the trial cutoff grade q_c).

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1. A routine at the University of Arizona Computer Center Library called MINSER is employed for this task.

2. The term "ore" is placed within quotes to indicate a mineral deposit at grades above the trial cutoff grade, q_c, which may or may not be economic. In any particular trial, q_c is the grade employed to distinguish between "ore" and "waste."

3. Establishing the pit geometry needed to extract the "ore" deposit (i.e., extracting the entire q_c-confining prism of each mineralized pod comprising the deposit).
4. Determining rates of materials (overburden, ore, and internal waste) removal and life of the mine.
5. Selecting equipment types to be used in the materials removal operation.
6. Determining operating costs. This mining cost model employs cost relationships provided in the STRAAM handbook (STRAAM Engineers, Inc., for the U.S. Department of the Interior, Bureau of Mines, revised June 1979), unless otherwise stated. Most of these costs are related to rates of materials handled.
7. Determining capital costs. As with operating costs, capital costs in this model are based upon relations reported in the STRAAM handbook (unless otherwise stated). Most of these costs also are related to rates of material removal.
8. Determining the preproduction investment and activities time schedule (i.e., beginning and finishing dates).
9. Determining the equipment replacement schedule to meet the life of the operation.
10. Cash flow analysis. Includes the year-by-year generation of the production cash flow stream. The cash flow analysis has provisions for royalties, depreciation, depletion, state taxes, federal taxes, operating losses carryover, and investment tax credits.

11. Net present value computation (uniform flow, continuous compounding assumed). This procedure determines the discounted values of preproduction investments and of the production cash flow stream. Subtracting the former from the latter yields the net present value.

12. The entire procedure (starting at step 1) is repeated until the maximum net present value is found.

9.3 Mining Cost Model Design

9.3.1 Inputs

Figure J.1 in Appendix J lists inputs required by the mining cost model. Most are self explanatory. Ones that deserve some clarification are detailed here.

Cost indexes: A total of twelve cost indexes for the desired date of evaluation are required. Most cost data come from the STRAAM handbook, which is based on July 1975 costs. Cost indexes are used to adjust these costs for the evaluation date. Indexes required are listed in Table 9.1. Beyond that date, inflation trends are ignored (constant dollars assumption).

The corresponding 12 escalation factors are constructed in the model by:

$$SC_i = \frac{CIXX_i}{CI75_i}$$

Table 9.1. Cost indexes required (source: STRAAM Engineers Inc., 1979).

No.	Item	Cost Index	Value (July '75)
1.	Mine & Plant Labor	Mining Labor	\$5.89/hr
2.	Construction Labor	Skilled Labor (BC)	1996.7
3.	Equipment & Repair Parts	Equipment	184.9
4.	Bits & Related Steel	Iron & Steel	197.3
5.	Timber & Lumber	Lumber	196.8
6.	Fuel	Petroleum	258.8
7.	Powder & Blasting Agents	Explosives	177.2
8.	Tires	Tire	158.8
9.	Construction Materials	Materials (Denver)	185.0
10.	Industrial Materials	Industrial Commodities	171.2
11.	Transportation	Rail - Metallic Ore	185.7
12.	Electrical Power		\$0.025/kw-hr*

*Date of cost not reported in STRAAM. \$0.025/kw-hr is the cost used throughout STRAAM's handbook.

where

SC_i = escalator for the i^{th} item

$CIXX_i$ = cost index for the i^{th} item in the desired date of evaluation

$CI75_i$ = cost index for the i^{th} item in July 1975

State Income Tax Rate. When the federal income tax is deductible from state income tax, an equivalent rate has to be calculated as follows:

$$STRTE' = STRTE * (1 - 0.48)$$

where

STRTE' = equivalent rate

STRTE = state tax rate

0.48 = federal income tax rate

This equivalent rate provides an approximation to the federal income tax deduction and is the one to be the input to the model.

9.3.2 Computation of Ore Deposit Geometry at the Trial Cutoff Grade

Figure J.2, in Appendix J, shows the design of this operation. The mining cost model establishes a trial cutoff grade[1] (q_c), which is assumed to be the same for all mineralized pods that form the deposit.

1. Trial cutoff grades are established by a numerical optimization routine that is also part of the mining cost model. Search for the optimum cutoff has to be constrained by a lower bound at 0.01% U_3O_8, for 0.01 is the lower grade limit in the endowment definition in this potential supply system.

Given the trial cutoff grade q_c, the model computes the following attributes for each one of the pods (indexed by "i"):

$\overline{q}_i(q_c)$ = average grade of mineralized material above q_c

$t_i(q_c)$ = tons of mineralized material above q_c

$\lambda_i(q_c)$ = length of the q_c-confining prism

$w_i(q_c)$ = width of the q_c-confining prism

$th_i(q_c)$ = thickness of the q_c-confining prism

DTQC = depth to the top of the q_c-confining prism

$gap_i(q_c)$ = gap distance to the q_c-confining prism of next pod[1] (for every pod, except the last one)

The total tonnage ($t(q_c)$) and average grade ($\overline{q}(q_c)$) of mineralized material above q_c in all pods is also compute.

9.3.3 Pit Geometry

Figure J.3 in Appendix J shows the design of the operation that determines the pit geometry and volumes and tonnages of materials needed to be removed.

The pit required for extracting the deposit at a mining cutoff grade q_c is approximated by centering a truncated inverted cone of elliptical cross section to simulate excavation of the pit around the elongated ore body. It is assumed that extraction of the deposit at a mining cutoff grade q_c requires extraction of all q_c-confining prisms

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1. Mineralized pods are assumed to be aligned in a straight line along their length axis (λ).

(i.e., the ore and waste material inside each prism) of each mineralized pod as well as the waste between the pods.

Two distinct pits are required to extract a deposit in the model:

a. The preproduction pit. This is the initial pit excavated to reach the top of the deposit. This pit is excavated in the mine development stage.

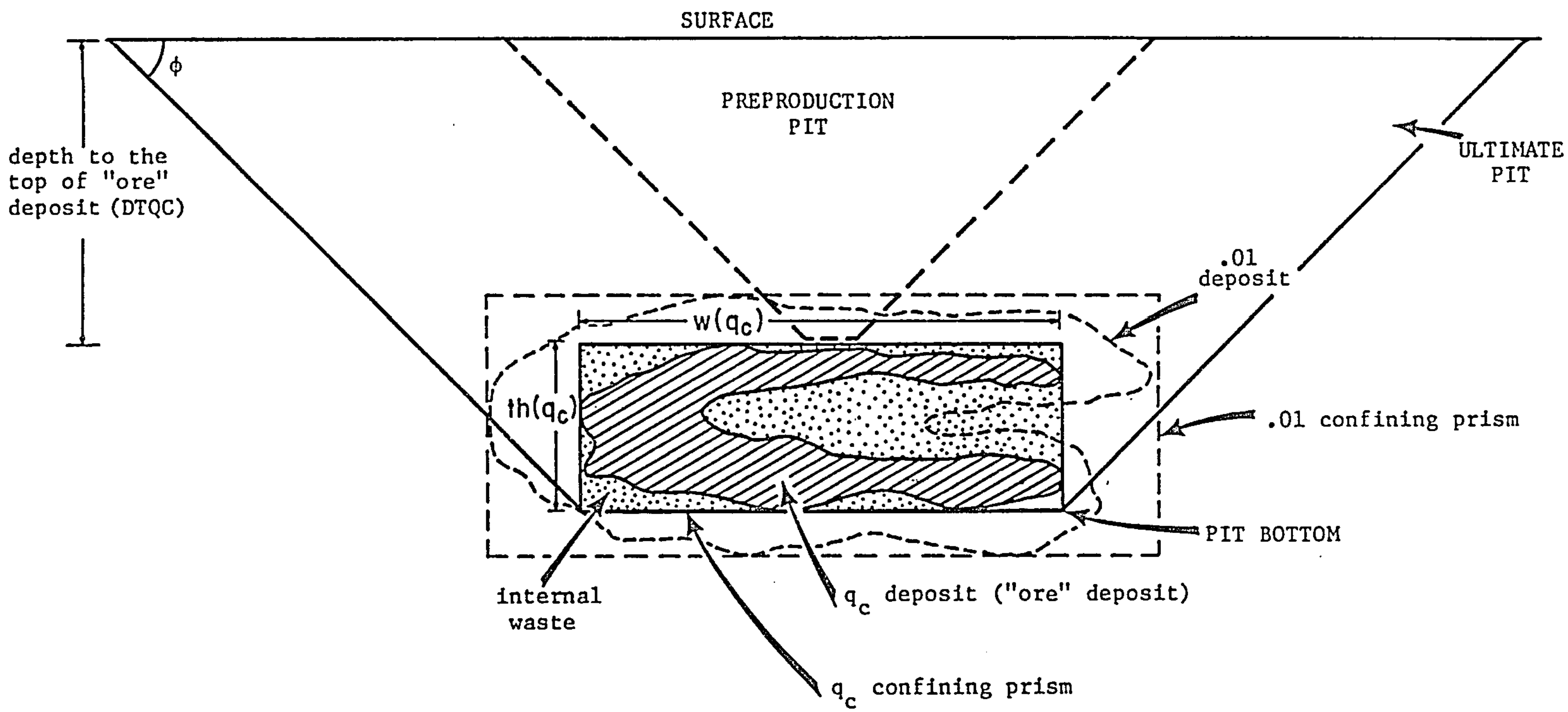
b. The ultimate pit. This is the pit excavated around the ore deposit. This pit is excavated during mine production.

Both pits are simulated by inverted cones of elliptical cross section. Figure 9.1 shows, in a schematic form, a cross section of the open pit design.

Ultimate pit. The semiaxes of the ellipse that simulates the bottom of the inverted ultimate cone are determined so that the area of this ellipse equals the plan view area of the pod-gap sequence (at a cutoff grade q_c) that characterizes the ore deposit. Semiaxes of the ellipse simulating the top of the inverted cone are determined by the depth to the bottom of the "ore" deposit and the pit slope angle (see Figure J.3, Appendix J).

Depth to ore deposit. Every pod comprising the deposit has a different depth. To simplify volume computation by reducing them to simple geometrical shapes (i.e., cones of elliptical section), it is necessary to determine a single depth that averages out the variation of individual pod depths.

Depth to top of the deposit is determined by a weighted average of individual pod depths using the areal projections as weights. A



q_c = mining cutoff grade.
th(q_c) = thickness of the q_c-confining prism.
w(q_c) = width of the q_c-confining prism.

Figure 9.1. Cross section of a stylized open pit mine.

similar procedure is used to compute depth to the bottom of the deposit. These procedures are shown in Figure J.3 in Appendix J.

Preproduction pit. The semiaxes of the ellipse that simulates the bottom of the inverted preproduction cone are determined so that the area of the ellipse meets a minimum pit floor area (α_{min}) and exhibits the same shape as the bottom ellipse of the ultimate cone. Semiaxes of the ellipse simulating the top of the inverted cone are determined by the depth to the top of the "ore" deposit and the pit slope angle.

Volumes and tonnages of the following materials are computed:

- preproduction stripping material (S),
- stripping material removed during production (U),
- internal waste (W) (i.e., material below the mining cutoff grade, q_c, which is inside the q_c-confining prism of every pod).

9.3.4 Material Removal Rates

Figure J.4 in Appendix J shows the design of rate computation operations.

Ore. The mine capacity is determined by the size of the deposit according to the following relationship:

$$TOPD = 507.58378 \left(\frac{t(q_c)}{1 \times 10^6} \right)^{0.68298945} \qquad (R^2 = 0.92)$$

where

TOPD = mine capacity (metric tons of ore per day)

$t(q_c)$ = size of the deposit (metric tons of material above cutoff grade)

This relationship was determined by regression analysis of data on capacities and deposit sizes for open pit projects. These data are listed in Table 9.2.

Use of this size-capacity relationship in the mining cost model implies that there is a "best practice" regarding selection of mine capacity. This best practice reflects physical and economic conditions not included in the potential supply system that constrain or influence selection of capacity or rate of production. Once the capacity is established, mine life (Y) is determined.

Other material removal rates. Rates of removal of other materials are obtained by dividing tonnage of the material by time available for removal (in days).

For hauling distance computation, it is assumed in the model that two piles of material will be formed: a waste pile, and an ore pile. An 8% maximum grade for roads also is assumed. These computations are shown in Figure J.4 in Appendix J.

9.3.5 Equipment Selection

Figure J.5 in Appendix J shows this operation.

For costing and equipment selection purposes, the mining cost model assumes three distinct material removal operations:

1. Preproduction stripping material removal (S), at a daily rate of TSPD.
2. Overburden material removal during production (U), at a daily rate of TUPD. This material includes the waste in gaps between mineralized pods.

Table 9.2. Data on size and capacity of open pit uranium projects.

Project	Size (10^6 s. tons)	Capacity (s. tons/day)
Bear Creek (Converse County, Wyo.)[1]	3.0	1400
Sherwood (Spokane Indian Reservation)[2]	7.95	2100
Pitch (Saquache County, Col.)[3]	2.1	600
Highland (Converse County, Wyo.)[4]	9.1	2800
Sweetwater (Sweetwater County, Wyo.)[5]	16.0	3000
Hansen (Fremont County, Col.)[6]	17.8	3750
Morton Ranch (Converse County, Wyo.)[7]	2.1	1000
Jackpile-Paguate Mine (Grants, N.M.)[8]	25.0	6000

SOURCES: [1]Dames and Moore; U.S. Nuclear Regulatory Commission, U.S.G.S., and USDA Forest Service, 1977

[2]U.S. Department of the Interior, Bur. of Indian Affairs, 1976

[3]U.S.D.A. Forest Service - Rocky Mtn. Region, 1979

[4]Humble Oil & Refining Company, 1971; U.S. Atomic Energy Commission, 1973

[5]U.S. Nuclear Regulatory Commission, 1978

[6]Cyprus Mines Corp./Wyoming Mineral Corp., 1979

[7]Tennessee Valley Authority, 1975

[8]Capacity: Graves, J.A., 1974; Size: confidential estimate

3. Joint removal of internal waste (W) and ore (O), at a joint daily rate of TOWPD.

Selection of the type of equipment to be used in each of the material removal operations depends on the daily rate of each operation. For equipment selection and operating cost determination purposes, the equipment is broken down into two categories,

1. Drilling and blasting equipment
2. Excavating, loading and hauling equipment

Details on the equipment selection procedure are given in Figure J.5, Appendix J.

9.3.6 Operating Costs Determination

Figure J.6 (Appendix J) schematically shows the procedure for determining operating costs. Unless stated otherwise, costs are determined using relations and procedures set out in the STRAAM handbook. The following subsections detail the determination of individual operating costs.

9.3.6.1 Direct Cost of Ore and Internal Waste Removal

Drilling and blasting. Table 9.3 shows the basic functions for costs of labor (Y_L), equipment (Y_E), and supplies (Y_S) for drilling and blasting, according to the type of equipment selected.

The mining cost model selects the appropriate set of cost functions and substitutes in the daily ore and waste rate, TOWPD, to obtain the costs. Each of the cost components is multiplied by a composite

Table 9.3. Drilling and blasting direct cost functions (ORE) (source: STRAAM Engineers Inc., 1979).

Crawler Type Percussion Drills	Rotary Drills
$Y_L = 1.798(TXPD)^{0.821}$	$Y_L = 0.096(TXPD)^{.861}$
$Y_S = 1.149(TXPD)^{.779}$	$Y_S = 0.146(TXPD)^{.917}$
$Y_E = 0.608(TXPD)^{.813}$	$Y_E = 0.073(TXPD)^{.811}$
$1000 \leq TXPD \leq 10{,}000$	$8000 \leq TXPD \leq 300{,}000$

Y = dollars per day

TXPD = metric tons per day

NOTE: This symbol is employed here and in subsequent cost functions to indicate in a general way the daily rate of material "X" handled per day. In the model TXPD is substituted by the actual rate of material handled (e.g. TOWPD for tons of ore and waste per day), as appropriate.

escalation factor, which is a linear combination of the twelve escalation factors derived from the cost indexes listed in Table 9.1:

$$E = \sum_{i=1}^{12} \alpha_i \cdot SC_i$$

where

E = composite escalation factor

α_i = weight or proportion of the i^{th} item (see Table 9.1) in the composite escalation factor

SC_i = escalation factor for the i^{th} item (see Table 9.1 for list of items)

The weights that determine the composite escalation factor for each of the cost components (i.e., Y_L, Y_S, Y_E) are given in the STRAAM handbook. This procedure is typical of the use of all STRAAM cost functions. For the specific values of the weights (α_i) of every composite escalation factor in this and subsequent use of STRAAM cost functions, the reader is referred to the STRAAM handbook.

The final drilling and blasting cost (DB_{OW}) desired is the sum of individual supply, labor, and equipment components after they have been adjusted for cost escalation.

Excavation, load and haul. Table 9.4 shows the basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of excavation, load, and haul cost according to type of equipment selected.

The mining cost model selects the appropriate set of cost functions and substitutes in the daily ore and waste rate, TOWPD, depth to

Table 9.4. Excavation, load, and haul direct cost functions (source: STRAAM Engineers Inc., 1979).

Front End Loaders and Trucks	Electric Shovels and Trucks
$Y_L = 23.60(TXPD)^{0.503} \cdot 0.155(d)^{.030}(H_x)^{.263}$	$Y_L = 0.430(TXPD)^{.889} \cdot 0.1173(d)^{.030}(H_x)^{.263}$
$Y_E = 8.40(TXPD)^{0.599} \cdot 0.080(d)^{.047}(H_x)^{.353}$	$Y_E = 0.225(TXPD)^{.986} \cdot 0.0546(d)^{.047}(H_x)^{.353}$
$Y_S = 0$	$Y_S = 0.027(TXPD)^{.967}$
$1000 \leq TXPD \leq 10{,}000$	$8000 \leq TXPD \leq 300{,}000$

TXPD = metric tons per day

d = depth in meters

H_x = length of haul in meters

Y = dollars per day

the bottom of the deposit, d_B, and the length of haul, H_{OW}, to obtain initial costs. After costs are adjusted by the appropriate composite escalation factors, they are added to give ELH_{OW}, the cost of excavation, loading, and hauling.

Total costs. The final direct cost of ore and waste removal (DCOWPD) is given by

$$DCOWPD = DW_{OW} + ELH_{OW}$$

where

DB_{OW} = drilling and blasting cost (for ore and waste)

ELH_{OW} = excavation, load, and haul cost (for ore and waste)

9.3.6.2 Direct Cost of Overburden Removal During Production

Table 9.5 shows the basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of drilling and blasting cost. Except for this table, the procedure for the direct cost computation is the same as direct cost computation for ore and waste detailed in section 9.3.6.1 of this chapter and shall not be repeated here.

9.3.6.3 Other Direct Costs

Clearing costs. The basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of clearing cost are:

$$Y_L = 839.33(CRPD)^{.779}$$

$$Y_S = 90.63(CRPD)^{.992}$$

$$Y_E = 282.56(CRPD)^{.889}$$

Table 9.5. Drilling and blasting direct cost functions (overburden and waste) (source: STRAAM Engineers Inc., 1979).

Crawler Type Percussion Drills	Rotary Drills
$Y_L = 1.798(TXPD)^{.821}$	$Y_L = 0.096(TXPD)^{.861}$
$Y_S = 1.149(TXPD)^{.779}$	$Y_S = 0.146(TXPD)^{.917}$
$Y_E = 0.608(TXPD)^{.813}$	$Y_E = 0.073(TXPD)^{.811}$
$1000 \leq TXPD \leq 10{,}000$	$8000 \leq TXPD \leq 300{,}000$

TXPD = metric tons per day of overburden (TUPD)

Y = dollars per day

where

CRPD = clearing rate per day (ha/day)

Y = dollars per day

Assuming a clearing rate of 0.2 hectares per day, the mining cost model computes the cost components. After cost components are adjusted by the appropriate composite escalation factor, they are added up to give the clearing cost, CCLEPD.

General items. The basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of the general cost items (i.e., general operations customarily required in surface mining operations, such as communications, sanitation, fire protection, etc.) are:

$Y_L = 5.152(\text{TXPD})^{.530}$

$Y_S = 0.133(\text{TXPD})^{.734}$

$Y_E = 0.124(\text{TXPD})^{.660}$

$1000 \le \text{TXPD} \le 400{,}000$

TXPD = total quantity of ore, waste, and overburden handled per day (TOWPD + TUPD)

Y = dollars per day

The sum of the three cost components, after being adjusted by the appropriate composite escalation factor, yields the general items cost, GICPD.

Water supply system. The basic function for the cost of the water supply system is the following:

$$\text{CWASPD} = 0.228(\text{TXPD})^{0.648}$$

where

CWASPD = water supply system cost ($/day)

TXPD = total quantity (m. tons) ore, waste and overburden handled per day (TOWPD + TUPD)

The mining cost model substitutes the daily rate of overburden removal (TUPD) and ore and waste handling (TOWPD) into this function to obtain the basic cost. This cost is then adjusted by the appropriate composite escalation factor.

The "other" direct costs are given by:

$$\text{DCHPD} = \text{CCLEPD} + \text{GICPD} + \text{CWASPD}$$

where

DCHPD = "other" direct costs ($/day)

CCLEPD = clearing cost ($/day)

GICPD = general items cost ($/day)

CWASPD = water supply system cost ($/day)

9.3.6.4 Administrative Cost

The basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of the administrative costs are given in Table 9.6.

The mining cost model substitutes the daily rate of overburden removal (TUPD) and ore and waste handling (TOWPD) into these functions to obtain the cost components. After each cost component is adjusted by the appropriate composite escalation factor, they are added up to give the administrative cost (ACPD).

Table 9.6. Administrative cost components (source: STRAAM Engineers Inc., 1979).

Item	Cost Function
Administrative salaries and wages	$Y_L = 0.344(\text{TXPD})^{.837} * 1.25$
Administrative purchases	$Y_S = 0.025(\text{TXPD})^{.926}$
Administrative equipment operation	$Y_E = 0.022(\text{TXPD})^{.822}$

Y = dollars per day

TXPD = total quantity(metric tons) of ore, waste and overburden handled per day (TOWPD + TUPD) ($1000 \leq \text{TXPD} \leq 400{,}000$)

9.3.6.5 Milling Operating and Administrative Costs

The following function, provided by DOE, is used to obtain the operating and administrative costs of milling:

$$\text{OAMLPT} = 13.7467 \left(\frac{\text{TOPD}}{1000}\right)^{-0.5778}$$

where

OAMLPT = operating and administrative milling cost in \$/s ton (costs as of 1/1/79)

TOPD = mill throughput (s. tons)

The mining cost model uses the mine daily capacity (TOPD) as the mill throughput to compute the milling operating and administrative cost. This cost then is adjusted by the WPI industrial commodities index (index #10 in Table 9.1).

9.3.7 Capital Costs Determination

Figure J.7 (Appendix J) shows the determination of individual capital costs items. Unless stated otherwise, capital costs are determined using the STRAAM handbook.

9.3.7.1 Development Drilling Cost

Assuming a 50′ x 50′ square grid to be drilled at every intersection over the plan view area of the ore deposit (at grade q_c), the mining cost model determines an approximation in the number of development drilling holes required. The model then uses the drilling cost relationship (described in Chapter 8, section 8.12) to obtain the cost of the drill holes, assuming that all of them are drilled to the bottom of the geologic deposit. Computational details are given in Figure J.7 in Appendix J.

9.3.7.2 Preproduction Stripping Cost

It is assumed that preproduction stripping is subcontracted and done in three shifts per day. Individual items are determined as follows.

Clearing. The labor (Y_L), equipment (Y_E), and supply (Y_S) components of clearing cost are given by the following functions:

$$Y_L = 1129.0 \cdot (A_{CL})^{-.107} \cdot 1.5$$

$$Y_S = 101.6 \cdot (A_{CL})^{-.038} \cdot 1.5$$

$$Y_E = 325.3 \cdot (A_{CL})^{-.049} \cdot 1.2 \cdot 1.5$$

where

A_{CL} = area in hectares $\quad 1 \leq A_{CL} \leq 500$

Y = dollars per hectare

Assuming that the area to be cleared (A_{CL}) is twice the area of the top elliptical section of the inverted cone that simulates the preproduction stripping pit, individual cost components are computed and adjusted by the appropriate components escalation factor. The cost of clearing (TCCL) is:

$$TCCL = (Y_L + Y_S + Y_E) \cdot A_{CL}$$

Drilling and blasting. Table 9.7 shows the basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of the drilling and blasting cost. These are basically the same functions as the ones in Table 9.5, restated to yield unit costs ($/ton), and including additional factors accounting for subcontracting and the three shifts worked per day.

The mining cost model selects the appropriate set of cost functions and substitutes the daily overburden removal rate (TSPD) into the function to obtain cost components. Total cost of drilling and blasting (TCDB) is obtained (after adjusting cost components with the appropriate composite escalation factor):

$$TCDB = (Y_L + Y_S + Y_E) \cdot S$$

where

S = total tons of overburden to be removed during the preproduction stripping operation (metric tons).

Table 9.7. Drilling and blasting cost functions (for preproduction stripping) (source: STRAAM Engineers Inc., 1979).

Crawler Type Percussion Drills	Rotary Drills
$Y_L = 1.798(TXPD)^{-.179} \cdot 1.50$	$Y_L = 0.096(TXPD)^{-.139} \cdot 1.50$
$Y_S = 1.149(TXPD)^{-.221} \cdot 1.20$	$Y_S = 0.146(TXPD)^{-.083} \cdot 1.20$
$Y_E = 0.608(TXPD)^{-.187} \cdot 1.20 \cdot 1.70$	$Y_E = 0.073(TXPD)^{-.189} \cdot 1.20 \cdot 2.47$
$1000 \leq TXPD \leq 10{,}000$	$8000 \leq TXPD \leq 300{,}000$

TXPD = metric tons of overburden handled per day (TSPD)

Y = dollars per metric ton

Costs of excavation, loading and hauling. Table 9.8 shows the basic functions for the labor (Y_L), equipment (Y_E), and supply (Y_S) components of this cost. These are basically the same functions as the ones in Table 9.4, restated to yield unit costs ($/ton), and including additional factors to account for subcontracting and three shifts worked per day.

The mining cost model selects the appropriate set of cost functions and substitutes in them the daily overburden removal rate (TSPD), depth to the top of the deposit (d_T), and length of haul (H_U) to compute the component costs. Total cost (TCELH) of excavation, loading, and hauling is obtained (after adjusting cost components with the appropriate composite escalation factor) by:

$$TCELH = (Y_L + Y_E + Y_S) \cdot S$$

where

S = total tons of overburden to be removed during the preproduction stripping operation (metric tons).

Preproduction stripping total cost. Preproduction stripping total cost (TCSTRP) is computed by

$$TCSTRP = TCCL + TCDB + TCELH$$

Table 9.8. Excavation, load, and haul cost functions (for pre-production stripping) (source: STRAAM Engineers Inc., 1979).

Front End Loaders and Trucks

$$Y_L = 23.6(\text{TXPD})^{-.497} \cdot 0.155(d)^{.030} \cdot (H_x)^{.263} \cdot 1.50$$

$$Y_E = 8.40(\text{TXPD})^{-.401} \cdot 0.08(d)^{.047} \cdot (H_x)^{.353} \cdot 1.20 \cdot 1.70$$

$$Y_S = 0$$

$$1000 \leq \text{TXPD} \leq 10{,}000$$

Electric Shovels and Trucks

$$Y_L = 0.430(\text{TXPD})^{-.111} \cdot 0.1173(d)^{.030}(H_x)^{.263} \cdot 1.5$$

$$Y_E = 0.225(\text{TXPD})^{-.014} \cdot 0.0546(d)^{.047}(H_x)^{.353} \cdot 1.2 \cdot 2.03$$

$$Y_S = 0.027(\text{TXPD})^{-.033} \cdot 1.2$$

$$8000 \leq \text{TXPD} \leq 300{,}000$$

TXPD = metric tons per day of overburden removal (TSPD)
d = depth in meters
H_x = length of haul in meters
Y = dollars per metric ton

9.3.7.3 Cost of Mine Plant and Buildings

Table 9.9 shows the basic functions for all the individual items comprising this cost. In these functions, the mining cost model substitutes the total tons of material handled per day to obtain individual costs (in dollars). Total cost of mine plant and buildings (TCMPB) is obtained by summing all the costs of individual items after each has been adjusted by the appropriate composite escalation factor.

9.3.7.4 Mill Capital Cost

The following function provided by DOE is used to obtain the mill capital cost:

$$TCMILL = 1.642 \times 10^7 \left(\frac{TOPD}{1000}\right)^{.7081}$$

where

TCMILL = total cost of the mill (dollars) (cost as of 1/1/79)

TOPC = mill throughput (s. tons)

The mining cost model uses the mine daily capacity (TOPD) as the mill throughput to compute the mill capital cost, which is then adjusted by the WPI industrial commodities index (index #10 in Table 9.1).

9.3.7.5 Equipment Cost

Table 9.10 shows the basic functions for equipment capital cost according to the type of equipment selected.

The mining cost model consults equipment cost functions twice; the first time is for the mining equipment. The model selects the

Table 9.9. Mine plant and buildings cost functions (source: STRAAM Engineers Inc., 1979).

Item	Function
Water system	$TCWSY = 15.54(TXPD)^{0.738}$ $1000 \le TXPD \le 400{,}000$
Communication system	$TCCSY = 247.1(TXPD)^{.495}$ $1000 \le TXPD \le 400{,}000$
Fueling system	$TCFSY = 36.61(TXPD)^{.765}$ $1000 \le TXPD \le 400{,}000$
Electrical system - non-electric shovel mines	$TCESY = 78.16(TXPD)^{.602}$ $1000 \le TXPD \le 20{,}000$
- electric shovel mines	$TCESY = 35.05(TXPD)^{.846}$ $8000 \le TXPD \le 400{,}000$
Repair shops and warehouses	$TCRSW = 3080(TXPD)^{.576}$ $1000 \le TXPD \le 400{,}000$
Office and laboratories	$TCPFLA = 964.9(TXPD)^{.485}$ $1000 \le TXPD \le 400{,}000$
Surface buildings	$TCSB = 2272.0(TXPD)^{.375}$ $1000 \le TXPD \le 400{,}000$

TXPD = total quantity (metric tons) of ore, waste and overburden handled per day (TOWPD + TUPD)

Table 9.10. Equipment capital cost (source: STRAAM Engineers Inc., 1979).

Loader and Truck Operation

$$EQK = (0.58 + 0.42 \cdot 0.80(d)^{.047}(H_x)^{.353})24215(\frac{3}{s} \cdot TXPD)^{.498}$$

$$1000 \leq TXPD \leq 20{,}000$$

Electric Shovels and Truck Operation

$$EQK = (0.43 + 0.57 \cdot 0.0546(d)^{.047}(H_x)^{.353})\ 743.8(\frac{3}{s} \cdot TXPD)^{.895}$$

$$8000 \leq TXPD \leq 400{,}000$$

TXPD = material handling rate (m. tons/day)

d = depth in meters

H_x = haul length in meters

s = number of shifts per day during mine production appropriate function and uses the joint ore and internal waste removal rate (TOWPD) to compute the mining equipment cost. The second time it looks again for the appropriate function and uses the overburden removal rate (TUPD) to compute stripping equipment cost.

The total equipment cost (TOTEQK) is the sum of both mining and stripping equipment costs, adjusted by the appropriate escalation factor.

9.3.7.6 Engineering and Construction Management Fees

The function to obtain this cost is the following:

$$TCECMF = 0.294(NCONC)^{0.922}$$

where

TCECMF = engineering construction and management fee (dollars)

NCONC = net construction cost

= TCSTRP + TCMPB + TCREST + TOTEQK

TCSTRP = total preproduction stripping cost

TCMPB = total cost of mine plant and buildings

TCREST = total cost of restoration during construction

TOTEQK = total equipment cost

9.3.8 Preproduction Activities Calendar

The mining cost model assumes that the preproduction activities occur in the following order:

1. Feasibility study and property consolidation.
2. Permits and licenses.
3. Land acquisition.

4. Development drilling.

5. Mine development (33% overlap with development drilling assumed).

6. Production.

Except as noted, one activity does not start until the previous one has been finished.

Determination of beginning and finishing dates for the preproduction activities calendar is shown in Figure J.8 in Appendix J. Computation of the preproduction activities schedule is based upon the order assumed and upon the duration read in for each activity.

Clearly, this activity schedule is important because of the effect of time on the net present value computation. In the mining cost model, all investments and production cash flows are discounted to time zero; i.e. the beginning date of the feasibility study and property consolidation activity. Total cost of the activity is assumed to flow uniformly throughout the duration of the activity.

9.3.9 Capital Replacement

The mining cost model assumes that all the assets except equipment will last for the entire life of the operation and have a salvage value of zero.

For the purpose of equipment replacement and depreciation, mining and stripping equipment is divided into three categories. Table 9.11 lists these categories, along with the percent of the equipment capital cost corresponding to each category according to the equipment type.

Table 9.11. Equipment cost breakdown (source: STRAAM Engineers Inc., 1979).

Category	Front End Loaders and Trucks	Electric Shovels and Trucks
Excavating and loading equipment	32%	36%
Hauling equipment	42%	57%
Drilling and blasting equipment	26%	7%

Before starting the cash flow analysis, the mining cost model computes the number of times that each equipment category will be purchased and timing of this equipment replacement (see also Figure J.9, Appendix J). This is done by assuming a physical life for each equipment category according to type of equipment used. Typical lives that can be assumed are:

Front end loaders	10 years
Electric Shovels	15 years
Trucks	6 years
Drilling and blasting equipment	6 years

In the number of purchases computation, useful life of the last purchase is adjusted within some margin (approximately ± 20%) to meet the required mine life.

The mining cost model assumes that the equipment will be salvaged when it is replaced (in which case the salvage value will be equal to the "scrap" value), or when life of the mine terminates. In the latter case, salvage value is assumed to be the undepreciated value. The depreciation method used is straight line depreciation over the physical life of the equipment. When equipment is used for only part of its physical life, this assumption costs the operation for only that portion of the equipment value that has been used.[1,2]

9.3.10 Cash Flow Analysis and Net Present Value Computation

Once the magnitudes and schedule of capital investments have been developed and operating cost determined for this trial mine, production cash flows can be generated for the subsequent net present value computation.

Figure J.10 in Appendix J shows the detail of the generation, year by year, of cash flows during the productive life of the mine. Table 9.12 is a summary of the cash flow computation and complements the description of the cash flow analysis illustrated in that figure.

Once cash flows for all the production years have been generated, the net present value computation is initiated. Design of this operation is detailed in Figure J.11 of Appendix J. Using the discount rate read in (K, the cost of capital of the firm modeled), and assuming continuous

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1. Assuming that the depreciation method truly reflects the decrease in equipment value.

2. Except for cost due to the time value of money.

compounding and uniform flow, present value of preproduction investments and present value of production cash flows are discounted at time zero (the beginning of the feasibility study and property consolidation activity). The net present value is computed by subtracting the present value of preproduction investment from the present value of production cash flows.

The numerical search routine in the mining cost model proceeds to set another trial value for cutoff grade, and the entire procedure is repeated until the maximum net present value is obtained.

Table 9.12. Annual cash flow computation summary.

Revenue	TRVY
Royalty payments	- RYY
Gross income from mining	= GIFMY
Mining operating cost	- TOCY
Milling operating cost	- OAMLY
Annual gross profit	= GPY
Depreciation	- DPRY
Net income before taxes and depletion	= NIBTDY
Depletion charge	- DPLY
Severance tax	- SVTXY
State income tax	- STITY
Net federal income tax[1]	- NFITY
Net profit	= PRFNET
Depreciation	+ DRPY
Depletion	+ DPLY
Replacement capital (equipment)	- INK
Equipment salvage value	+ TOTSVY
Tax refunds[2]	+ TXRF1 + TXRF2
CASH FLOW	= CF

[1]After adjustments for carry forwards and investment tax credits when applicable

[2]From carrybacks of operating losses (TXRF1) and/or investment tax credits (TXRF2)

CHAPTER 10

THE EXECUTIVE MODEL

10.1 The Role of the Executive Model

The exploration model *per se* simulates exploration as a physical process. That is, it explains costs incurred and successes or failures experienced in the search for endowment, given a specific exploration effort. The model does not decide that exploration should be conducted nor does it determine the magnitude of the effort. These decisions are made by the executive model. It is only with the executive model that exploration becomes an economic activity. That is, the executive model coordinates the pure physical process of exploration with the economic conditions specified in the system for complete potential supply evaluation. Moreover, the executive model explicitly incorporates treatment of risk in the potential supply system.

Exploration involves highly uncertain outcomes. The endowment in a specific geographic region being investigated is not known with certainty. Furthermore, successful discovery of existing endowment is not assured. As exploration activities are conducted, more is known about this uncertain endowment (usually via improved expectations about the size of geologic features directly associated with the endowment). The executive model considers this improved information in conjunction with economic conditions prevailing in the system and expectations on the depths of the target to perform its two major duties:

1. Determining if exploration of the piece of ground being investigated should continue or not, given the current perceived expectations about the size of the reward (i.e., present value of the deposit(s) in the geographic entity net of exploitation and exploration costs).

2. Determining, given that further exploration is warranted, the optimum level of effort[1] to be allocated to the next exploration activity in the geographic entity being investigated. This effort is communicated to the exploration model (usually number of holes to be drilled).

It must be emphasized that the decisions noted above are influenced strongly by depths of the targets and economic conditions specified in the system. Simply stated, both factors affect the distribution of "prize" sizes that the exploration agent searches for. Clearly, this has an influence on the decisions by a given firm about whether and how much to gamble in uranium exploration.

Another factor influencing these decisions is the kind of firm assumed to be conducting the exploration. In other words, what may be "too risky" for one firm does not need to be so for another firm because no two firms are identical in their financial structure and diversifi-

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1. The optimum level of effort in this system is that which maximizes the expected value of the geographic entity under investigation, net of exploitation, and remaining exploration costs. An optimum level of effort exists regardless of the decision determined in item 1 above. However, this optimum is irrelevant if the decision is negative; in such a case, optimum level of effort is set to zero. A zero level of effort signals the exploration model to stop exploration of the pertinent geographic entity.

cation. This influence shall be clarified in description of the Decision Module. Details on (1) the financial and risk parameters required to characterize a firm, and (2) the means by which we describe a "typical" firm will be given in the description of the Decision Module.

10.2 Components of the Executive Model

The executive model is composed of two separate modules:

1. The Pre-Exploration Module
2. The Decision Module

Figure 10.1 shows, in simplified form, the components of the executive model, their major functions, and their interaction with the exploration model. Simply stated, every time the exploration model consults the executive model, it makes two consecutive calls: the first to the Pre-exploration Module; and the second to the Decision Module (also see Appendix K, Figure K.1).

When the exploration model consults the executive model, the pre-exploration component takes control of the operations. Given the current status of the exploration model in the exploration sequence (i.e., what phase is being conducted[1], value of the success/failure indicator variable for that phase), and the current (revised) expectations about the size

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1. An "end of phase" convention has been adopted. By this convention, Phase "i" has not been finished (i.e., the status of the exploration model is reported as being in Phase "i") until drilling activities pertaining to Phase i + 1 are initiated.

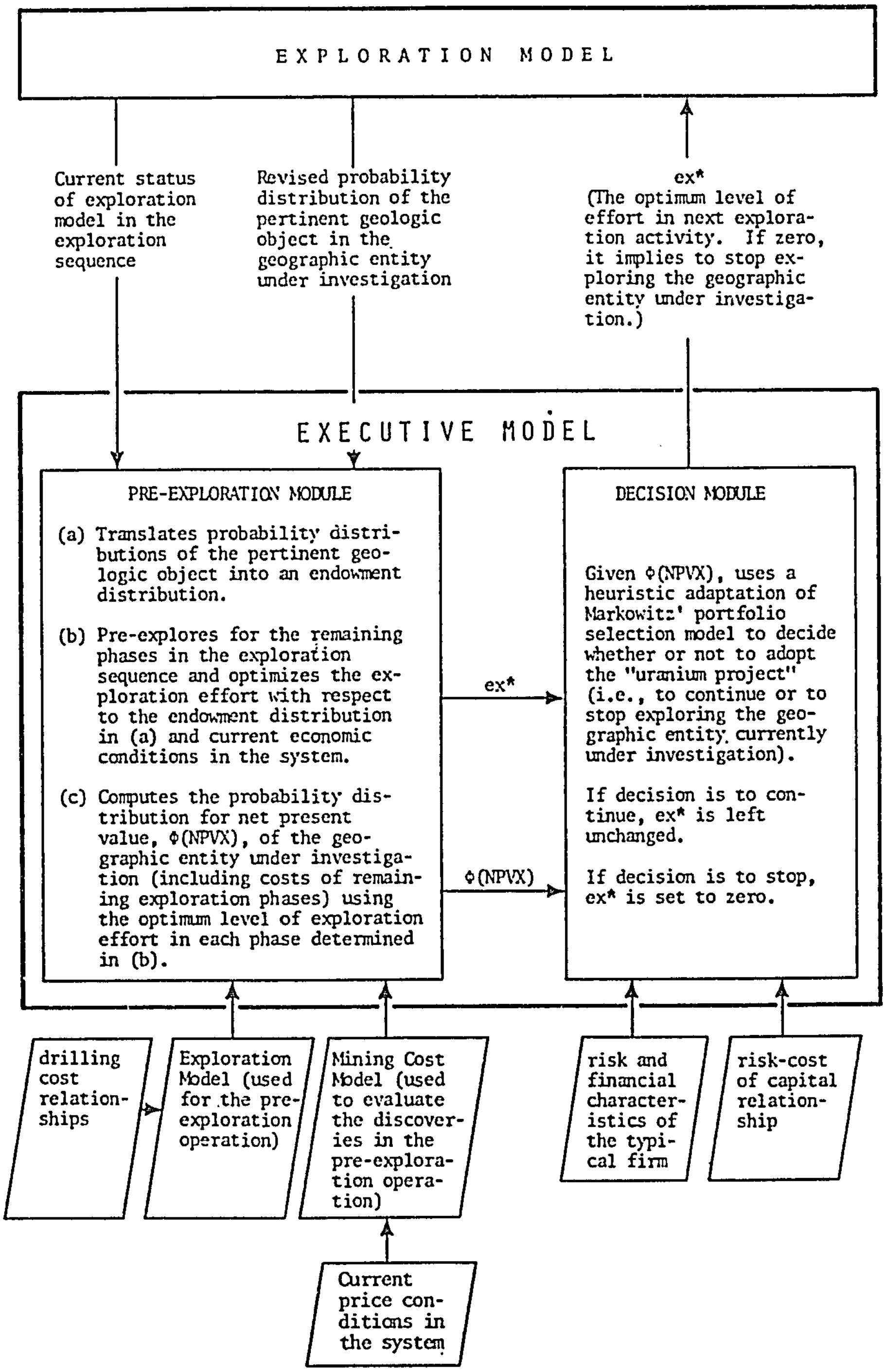

Figure 10.1. Schematic representation of components of the executive model and its interaction with the exploration model.

of the pertinent "geologic feature,"[1] the pre-exploration module considers the inferred endowment[2] (via geologic feature size expectations) in conjunction with the depth-of-target expectations and price/cost regime prevailing within the system to jointly determine the following:

- ex*, the optimum level of effort (e.g., number of holes) to be allocated to the next exploration activity.[3]
- Φ(NPVX), the distribution of net present value (including costs of remaining exploration phases) of the geographic entity under investigation using the optimum level of exploration effort in each one of the remaining phases.

Next, the Decision Module takes control of the operation. Given the net present value distribution, Φ(NPVX), it decides whether or not to continue exploring (using a heuristic adaptation of Markowitz' portfolio selection model, detailed in section 10.4 of this chapter) and

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1. Identity of the "geologic feature" depends on the current status of the exploration model in the exploration sequence. Table 10.1 details this geologic feature.

2. In general, size of the geologic feature is directly related to endowment. The geologic feature is used in the model to make inferences about the endowment.

3. Actually, the optimum level of effort for <u>each</u> of the remaining phases is determined (i.e., a vector with the optimum effort levels). However, the only relevant level of effort needed by the exploration model is ex*, the one corresponding to the next immediate drilling activity (e.g., holes to drill in the next exploration phase or additional holes to drill to complete the current phase). This is so because in the exploration model, when effort ex* is spent, the executive model shall be consulted again and a new optimum effort level vector will be determined in light of revised endowment expectations.

communicates this decision to the exploration model. At this point, control is returned to the exploration model.

If the executive model's decision were to continue, the exploration model would conduct the next exploration activity (i.e., spend ex* effort units) in the geographic entity under investigation, revise the size distribution of the pertinent geologic feature, and consult again the executive model.

On the other hand, if the executive model's decision is to stop exploration, the system quits investigating the current geographic entity and a new one is considered for exploration.

10.3 The Pre-Exploration Module

Design of most of this module is conceived as a Monte Carlo simulation procedure.

The major operations take place in the pre-exploration module: endowment inference, pre-exploration optimization of effort, and derivation of the net present value distribution. Figure K.2 in Appendix K[1] provides an adequate representation of details of these operations and is used as the basis for this discussion. The reader is encouraged to refer to this figure in the course of this description of the major operations.

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1. There are three slightly different cases to consider in design of the pre-exploration module; these correspond to Figures K.2, K.3, and K.4 in Appendix K. These cases (summarized in Table 10.1) correspond to the states of the exploration model in the exploration sequence at the time that the executive model is consulted.

Endowment inference. The revised distribution of the size of the pertinent geologic feature [in Figure K.2, the redox length distribution, $\epsilon(R)$] passed by the exploration model is converted to an endowment distribution (amount of U_3O_8 in mineralized material above 0.01% U_3O_8). In Case 1 of the pre-exploration module design (Figure K.2, Appendix K), this conversion is effected[1] by the following operations:

1. Monte Carlo sampling of the redox length distribution, $\epsilon(K)$, to generate a specific value of redox length R_k.
2. Identification of the endowment distribution conditional on length $a(U|R_k)$ and the subsequent Monte Carlo sampling to generate a specific value for endowment, U_k.
3. Creation of the mineralized pods making up the endowment $\dot{U}_k$.

Simply stated, this operation imitates the inference that the explorationist makes from perceived geologic feature dimensions to amount of mineralization associated with that geologic feature.

Pre-exploration and optimization of exploration effort. Given endowment expectations determined in operation 1 above, the current status of the exploration model in the exploration sequence, and general economic conditions prevailing in the system, the module "pre-explores" with different trial values of exploration effort in each of the remaining phases until it finds optimum exploration effort levels (i.e., that effort which maximizes the present value of the geographic entity being

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1. It is the repeated sampling of the redox length distribution $\epsilon(R)$ and of endowment distribution conditional on length, $a(U|R)$, that effects the redox length-endowment translation.

investigated, net of exploration, and exploitation costs). It is important that it be understood that all pre-exploration outcomes are determined by perceived endowment, not the actual endowment in the geographic entity under investigation.

Employing the endowment distribution determined in operation 1, and using the aforementioned trial values of the decision variables (exploration effort in each remaining phase), the pre-exploration module imitates exploration activities in the remaining phases (by recasting the exploration model and taking over the remaining phases) and passes the discovery(s), if any, through the mining cost model to compute the contribution to the net present value (accounting for exploration costs) of the geographic entity being investigated.

This operation mimics the budgeting and design of the "best" exploration program given perceived endowment and drilling depth expectations of the exploration agent. Conceivably, it emulates the heuristic of the decision maker in dealing with uncertainty as he trades off expected reward and risk.

It is important to note that the major portion of the pre-exploration module constitutes a "function" relating expected net present value (including remaining exploration costs) with exploration effort (number of holes or fences in each of the remaining exploration phases). This function is not mathematically tractable; thus, Monte Carlo simulation techniques are employed.

The design of the pre-exploration module, shown in Figures K.2, K.3, and K.4 of Appendix K, includes a simplistic trial and error solution to find the optimum exploration effort. This trial and error solution is included only to pose the optimization problem. Other more efficient procedures to solve this optimization problem need to be investigated.

An analogy should prove useful here to recapitulate the main points of operations 1 and 2 of the pre-exploration module. Consider the following simplified example: one target size and one exploration stage.

Suppose it is desired to compute the optimum amount of exploration effort, x^*(assume x is a continuous variable ≥ 0), in searching for a target of value, V, so as to maximize the expected net value. Let:

V = value of the target

p = probability that the target exists in region A

$d(x)$ = function describing the probability of detecting the target with an exploration effort, x, given that the target exists. Suppose this is a saturation-type function of the form 1 $e^{-\alpha x}$, where $\alpha > 0$.

$C(x)$ = cost of spending "x" units of exploration effort. Suppose this is linear function: $C(x) = ax$, where $a > 0$.

The expected net value is given by the equation:

$$\begin{aligned} ENV &= p \cdot d(x) \cdot V - C(x) \\ &= p \cdot V(1 - e^{-\alpha x}) - ax \end{aligned}$$

The exploration effort x^* that maximizes the expected net value can be computed by taking the first derivative of the ENV function with respect to x and setting it to zero:

$$x^* = -1/\alpha \ \ln \frac{a}{V \cdot \alpha \cdot p}$$

Without attempting to be rigorous, some analogies can be drawn between this simple example and the pre-exploration module design. In the pre-exploration module, repeated sampling of the redox length distribution $\epsilon(R)$ [which by the sampling of the conditional distribution $a(U|R)$ implies an endowment distribution] plays the role of the term p, the probability of occurrence of the target in the simplified example above. Passing through the remaining exploration phases with pre-established trial values for exploration efforts in each phase is equivalent to the conditional probability of detecting the target as a function of "x," d(x), in the example. Computation of the drilling costs, of course, is analogous to the term C(x). Also, filtering of the discoveries through the mining cost model to compute the net present value corresponds to the value of the target V in the example.

In the simplistic example, the determination of x*, the optimum level of effort, is trivial. Unfortunately, this is not so in the pre-exploration module for two reasons: (a) the equivalent to the "x" of the example is in reality a vector of exploration efforts in each of the remaining phases[1] and (b) the "function" relating the expected net present value to the exploration effort vector in the pre-exploration module is

1. Although the level of effort for the next immediate exploration activity is the only variable needed to be communicated to the exploration model in each consultation, the exploration efforts of each of the remaining phases are necessary for computation of the expected net present value in pre-exploration.

not mathematically tractable, but instead consists of a complex sequence of events that is integrated through Monte Carlo simulation.

Derivation of the net present value (including exploration costs) probability distribution for the exploration venture. Given the optimum vector of efforts for remaining exploration phases, the pre-exploration module is rerun to compute the companion distribution of net present value (including cost of remaining exploration phases) of the geographic entity under investigation, Φ(NPVX). This distribution represents the "prize size" distribution as perceived by the exploration agent. Operation of the decision submodel is based on the net present value distribution determined here.

Depending upon the current status of the exploration model in the exploration sequence at the time it consults the executive model, there are three slightly different cases to consider in design of the pre-exploration module. Table 10.1 serves to summarize these cases. Details on the pre-exploration module design for these cases are shown in Figures K.2, K.3, and K.4 of Appendix K. Basically, differences among the three cases can be referred back to operation 1 of the pre-exploration module described above. The "geologic feature" whose distribution is being updated in the exploration model and communicated to the executive model is different in each case.

Case 1 (Figure K.2, Appendix K) applies when the exploration model is in Phases 2, 3A, or 3B. In this case, where redox length is the pertinent geologic feature and endowment value is generated from the conditional endowment distribution given redox length, $a(U|R)$. Next, pods are created along the redox system to meet the endowment value.

Table 10.1. Cases in the pre-exploration module design.

Case number in the pre-exploration module design	Current status of the exploration model at time of consultation to the executive model	"geologic feature" whose revised size distribution is communicated from the exploration model to the executive model
1	Phase 2, 3A, and 3B	R, the length of the redox frontal system in the A_3 being investigated.
2	Phase 4A	$\ell(q_{min})$, the length of the mineralized target (at q_{min}, the minimum attractive cutoff grade) believed to occur along the redox system portion being explored.
3	Phase 4B and 5	$u_D(q_{min})$, the amount of uranium in the deposit at grades above q_{min}. q_{min} is the minimum attractive cutoff grade.

This operation is similar to operation 2 of the creation submodel described in Chapter 7, section 7.2.2.

Case 2 (Figure K.3, Appendix K) is employed when the exploration model is in Phase 4A. In this case, the pertinent geologic feature is the length of the target, $\ell(q_{min})$ (a target being defined in the model as mineralized material above the minimum attractive grade, q_{min}). Operation 1 of the pre-exploration module is accomplished by iterative sampling of the length of the target distribution and generating enough mineralized pods and gaps to meet the sampled value of target length.

In case 3 (Figure K.4, Appendix K) the pertinent "geologic feature" consists of the U_3O_8 quantity directly. In this case, the endowment inference operation (operation 1) is not needed.[1]

It is evident then that different geologic features are the focus of decision making as exploration proceeds. The three cases represent different paths, depending upon the stage of exploration, to determine the endowment implied by current expectations.

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1. As shown in Figure K.4, Appendix K, the usual passing of the deposit through the mining cost model obtain the net present value, NPV*, has been replaced by the response function:

$$\widehat{NPVOPT} = f_{NPV}[u(q_{min}), q_{min}, \overline{dpth}, \text{Price of } U_3O_8]$$

where

q_{min} = minimum attractive cutoff grade

$u(q_{min})$ = U_3O_8 amount in material above q_{min}

$\overline{dpth}$ = average depth of the deposit.

If one wanted to use the mining cost model as in previous cases, it is necessary to "decompose" the U_3O_8 quantity, $u(q_{min})$, into a series of mineralized pods and gaps.

10.4 The Decision Module

The objective of the decision module is to make the financial decision to continue of not continue exploration (i.e., to accept or reject the "uranium project"). Risk analysis is used because it is not sufficient to base a decision on expected net present value alone. A comprehensive procedure must also include a consideration of the risk of a project and its effect on the financial well-being of the firm. A comprehensive risk evaluation must include the following ingredients (Harris, 1970):

1. Value and overall risk of the firm at the time of the decision.
2. Expected profitability and risk of the proposed investment.
3. Expected effect of the investment on the risk position and value of the firm.
4. Re-evaluation of:
   a. The project at the new cost of capital of the firm, accounting for change in risk level of the firm due to adoption of the project
   b. Current operations of the firm at the new capitalization rate, a rate caused by adoption of the project.

The decision module integrates these elements. The net present value distribution of the "uranium project," Φ(NPVX), determined in the pre-exploration module, becomes the main input to the decision module to determine whether or not the remaining exploration phases should be conducted in the geographic entity currently being investigated.

10.4.1 The Theoretical Framework

The procedure used in this design is the heuristic adaptation of Markowitz' portfolio selection model described in Harris (1970).

There are uncertain quantities associated with exploration and mining ventures (existence of a deposit, detection, grade, tonnage, etc.). At the time of a decision, states of these quantities can only be estimated. Therefore, the net present value, p, of a particular project cannot be known with certainty. It can, however, be represented by a probability distribution. Likewise, future value of a firm is uncertain at any given point in time because the exact present value, V, of future flow of dividends cannot be known. Therefore, it also can be represented by a probability distribution.

The theoretical model assumes a firm facing a risk-cost of capital tradeoff, which can be represented by the following equation:

$$K = K^* + b \frac{\sigma_V}{\bar{V}} \tag{10-1}$$

where

K = cost of capital to the firm

K^* = risk free cost of capital

b = coefficient of risk of the firm

$\bar{V}$ = mean value of the firm

$\frac{\sigma_V}{\bar{V}}$ = coefficient of variation of value of the firm

At any point in time, the firm's cost of capital and coefficient of variation lie on the line described by this function.

Given a firm with a mean value $\overline{V}$, variance of value σ_V^2, and a cost of capital K, we can calculate the worth of the project. The project is subjected first to a risk analysis to produce a net present value probability distribution with mean p and variance σ_p^2. The worth of the project to the firm can only be determined by assessing the impact on the firm if it were adopted. This estimation is accomplished by computing the new expected value of the firm ($\overline{V}'$) and the new variance of value $(\sigma_V')^2$ after annexation of the project. Since both the project's value and the value of the firm are random variables, their sum is also a random variable, V', with the following mean and variance.

$$\overline{V}' = \overline{V} + p$$

$$(\sigma_V') = \sigma_V^2 + \sigma_p^2 + 2\ r\ \sigma_V\ \sigma_p$$

where

r = correlation coefficient between the value of the project and the value of the firm's existing operations.

The effect on the firm's cost of capital of adopting the project is calculated from the risk-cost of capital relationship:

$$K' = K* + b\,\frac{\sigma_V'}{V'}$$

The change in value of existing operations of the firm can be computed[1] after recapitalizing $\overline{V}$ at the new rate K':

$$\Delta V = \frac{K}{K'}\ \overline{V} - \overline{V}$$

where

ΔV = change in value of existing firm's operations solely due to new cost of capital K'.

The expected net present value of the project is recomputed at the new cost of capital, K'. For the project to be worthwhile to the firm, the net effect (i.e., net present value of the newly discounted project, p' plus ΔV in the firm's value has to be zero (firm's value unchanged) or positive:

- if $\overline{p}' + \Delta V \geq 0$, accept the project (i.e., continue exploration).
- if $\overline{p}' + \Delta V < 0$, reject the project (i.e., quit exploring current geographic entity)

where

$\overline{p}'$ = net present value of the project discounted at K'

ΔV = change in value of firm's existing operations solely due to new cost of capital, K'.

1. Assuming perpetuity.

For the sake of consistency with published literature, the decision procedure presented here has been given in terms of total value of the firm. In the decision module, net worth has been used as a measure of value of the firm. Furthermore, the value used is on a per-share basis (common shares).[1] Necessary changes are straightforward and are detailed in the decision module diagrammatic description, Figure K.5, Appendix K.

10.4.2 Estimation of the Adapted Markowitz' Model

To apply the modified Markowitz model, the parameters listed below must be known:[2]

1. K* and b, the coefficients of the risk-cost of capital equation:

$$K = K^* + b\,\frac{\sigma_{NWPS}}{\overline{NWPS}} \tag{10-2}$$

2. Definition of the firm[3] whose decisions are being modeled in terms of the following financial and risk parameters:

   a. $\overline{NWPS}$, the value per share of existing operations of the firm.

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1. This is because in the estimation procedure, it was desirable to remove the effect in the firm's value of changes in the number of shares over time.

2. These parameters are required by the potential supply system. Estimation of these parameters is completely external to the system.

3. These firm definitions imply the rate of return (cost of capital K) to be used everywhere in the system, for K is determined by substituting σ_{NWPS} and $\overline{NWPS}$ in the risk-cost of capital equation.

b. Number of shares outstanding.

c. σ_{NWPS}, the standard deviation of the net worth per share.

d. r, the coefficient of correlation of the value of a uranium project and the value of the firm's existing operations.

10.4.2.1 Estimation of K* and b

Using data available from Value Line Investment Survey, Moody's Industrial Manual, and company annual reports, measures for K, σ_{NWPS}, and $\overline{NWPS}$ were calculated for each of the companies in a sample of 25 firms involved in uranium production. Using these results, a linear function was fitted by regression analysis to obtain estimates for K* and b. Of course, before this function could be estimated, financial data were analyzed to obtain estimates of K, σ_{NWPS}, and $\overline{NWPS}$ for each firm.

Obtaining K for each firm. The current weighted average cost of capital for each firm, K, was obtained using the equation

$$K = \alpha_p K_p + \alpha_d K_d + \alpha_e K_e$$

where

K_p = cost of preferred stock

K_d = cost of debt

K_e = cost of equity

α = proportion of total financing provided by each source

The cost of preferred stock, K_p, was obtained by:

$$K_p = \frac{D}{P}$$

where

D = preferred dividends

P = price of preferred stock

The cost of debt, K_d was computed as follows:

$$K_d = i(1 - t)$$

where

i = weighted average of the interest rate in long term debts; weights are based on the amount outstanding of each debenture, bond, or note.

t = income tax rate

The cost of equity was obtained using the following valuation model:

$$K_e = \frac{D}{P} + g$$

where

D = dividends per share in last year reported

P = average price of stock in last year reported

g = growth rate of the firm

The dividend valuation model assumes perpetual constant growth of the firm.

Assuming that the past trend in net worth per share is meaningful to predict expected future growth of the firm, the growth rate, g, was estimated by fitting the following trend equation to the time series of net worth per share (1975-1979) in real dollars:

$$\ln(NWPS_t) = C_0 + C_1(t)$$

where

t = year

The estimate for g can be obtained by (Johnston, 1972):

$$g = e^{C_1} - 1$$

It must be clear that the procedure described provides only an approximation of the firms' cost of capital, K. The cost of equity capital is the most elusive term to obtain for the weighted average cost of capital computation. The dividend valuation model was assumed to be applicable to all firms. The difficulty of employing the dividend valuation model when a company pays no dividend or a negligible one is evident. As the dividend tends to zero, estimation of the growth rate, g, becomes more critical in estimating the cost of equity.

Obtaining $\overline{NWPS}$ and σ_{NWPS} for each firm. The coefficient of variation of value per share of the firm, $\sigma_{NWPS}/\overline{NWPS}$, is also needed for each firm. This is obtained by fitting a linear function to the time series (1975-1979) of net worth per share (NWPS) in real dollars:

$$NWPS_t = a_0 + a_1 (t) + \epsilon$$

$\overline{NWPS}$ is simply the expected value of NWPS conditional on time (t = 1979).

Variance of the value per share of the firm, σ^2_{NWPS}, can be calculated by the following expression:[1]

1. This is the variance in the prediction of an individual outcome (NWPS) drawn from the probability distribution of value per share of the firm.

$$\sigma^2_{NWPS} = MSE\left(1 + \frac{1}{n} + \frac{(t_h - \bar{t})^2}{\sum_i (t_i - \bar{t})^2}\right)$$

where

n = number of samples

t = time (year)

$\bar{t}$ = mean of t

t_h = last year reported (1979)

t_i = the i^{th} year $i = 1, 5$

MSE = mean square error of the linear trend fit

$$= \frac{\Sigma(\hat{NWPS}_t - NWPS_t)^2}{n - 2}$$

Table 10.2 shows the results obtained for the cost of capital K, mean net worth $\overline{NWPS}$, variance of net worth, σ^2_{NWPS}, and coefficient of variation of net worth for the sample of 25 firms.

A linear function was fitted to the data on K and $\sigma_{NWPS}/\overline{NWPS}$ in Table 10.2, giving the following results:

$$\underset{(0.00989)}{K = 0.04703} + \underset{(0.11642)}{0.49171}\left(\frac{\sigma_{NWPS}}{\overline{NWPS}}\right) \tag{10-3}$$

$r^2 = 0.43680$

Other measures[1] of the value of the firm were tried in the estimation of both the growth rate of the firm, g, and the coefficient

1. Also in real dollars.

Table 10.2. Data for risk-cost of capital relationship (source: based on 1975-79 data published by Value Line Investment Survey).

COMPANY	CODE	$\overline{NWPS}$	σ_{NWPS}	$\sigma_{NWPS}/\overline{NWPS}$	K
Exxon Corporation	EXXON	21.93690	0.50134	0.02285	0.06384
Atlantic Richfield	ARCO	11.68778	0.64633	0.05530	0.04596
UNC Resources	UNCRE	7.60359	1.63298	0.21476	0.11870
Mobil Corporation	MOBIL	20.81900	0.28591	0.01373	0.06691
Union Carbide	UNCARB	26.54995	0.94271	0.03551	0.06788
Gulf Oil	GULFO	19.02878	0.33065	0.01738	0.06399
Conoco, Inc.	CONOCO	15.01183	0.68735	0.04579	0.08397
Getty Oil	GETTY	18.00933	0.51400	0.02854	0.07166
Phillips Petroleum	PHILPE	11.77068	0.13015	0.01106	0.09167
Kerr-McGee Corporation	KERMG	20.36426	0.71581	0.03515	0.05302
Homestake Mining	HOMESK	5.11278	0.15661	0.03063	0.09934
Phelps Dodge	PHELPD	21.51896	0.41316	0.01920	0.03415
Denison Mines Ltd.	DENIM	3.95246	0.15120	0.03826	0.10027
Federal Resources Corporation	FEDRE	1.08418	0.02946	0.02717	0.03045
Tenneco Inc.	TENECO	27.02458	0.89721	0.03320	0.05641
Pioneer Corporation	PIONER	9.11669	0.23824	0.02613	0.10106
Cobb Nuclear Corporation	COBBNU	0.29887	0.05869	0.19636	0.18161
Energy Reserves Group	ENGYRS	0.83261	0.06171	0.07412	0.02610
Ranchers Exploration & Development	RANCHR	7.96280	0.33663	0.04227	0.03769
Rio Algom Mines Ltd.	RIOALG	15.99158	0.22158	0.01386	0.06727
U. S. Energy Corporation	USENGY	0.55044	0.10557	0.19178	0.22947
Newmont Mining Corporation	NEWMNT	12.30079	0.91318	0.07424	0.02902
Sabine Corporation	SABINE	3.75070	0.65061	0.17346	0.07816
St. Joe Minerals	STJOEM	10.69275	0.33223	0.03107	0.05368
Union Pacific Corporation	UPACIF	22.91718	0.24638	0.01075	0.04265

$\overline{NWPS}$ = expected value of net worth per share conditional on time (t = 1979) (dollars per share)

σ_{NWPS} = standard deviation of net worth per share

$\frac{\sigma_{NWPS}}{\overline{NWPS}}$ = coefficient of variation of net worth per share (decimal form)

K = weighted average cost of capital (decimal form) of variation of the firm's value. These measures were total net worth, total earnings, earnings per share, total market value, and market value per share (price of the stock). In addition, the growth rate of net worth per share and earnings per share directly reported by Value Line Survey, estimated by their own methodology (Bernhard, 1979), was analyzed.[1] The results of the fitting equation (10-2) were poor[2] because the correlation coefficient was, in most cases, nonsignificant at the 5% level.

10.4.2.2 Estimation of the Typical Firm

As stated earlier, use of the adapted Markowitz model requires definition of the firm in terms of the following financial and risk parameters:

a. $\overline{NWPS}$, the value per share of existing operations of the firm.

b. Number of shares outstanding.

c. σ_{NWPS}, the standard deviation of the net worth per share.

d. r, the coefficient of correlation of the value of a uranium project and the value of the firm's existing operations.

One plausible approach to identify parameters (a), (b), and (c) is to look at a sample of firms in the uranium industry, identify these parameters for each firm, and compute a weighted average for each parameter. An appropriate set of weights can be the proportion of total value of

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1. Value Line reports do not comprise all firms in the sample.

2. An exponential function of the form, $K = K^{*} \cdot e^{bt}$ was also tried, with no improvement in results.

uranium production of firms in the sample or the proportion of total uranium production. Unfortunately, neither data is available for all firms, especially for very large corporations. A simple average may be the only available choice.

The calculation of r, the correlation coefficient that reflects degree diversification of the typical firm with respect to uranium, is not trivial. One approach is to examine business cycles of the commodities produced by the typical firm. Commodity groups in which the typical firm is assumed to be involved are:

1. uranium
2. coal
3 copper
4. gold and silver
5. petroleum (crude oil and natural gas)
6. other (lumber and wood products, chemical and allied products, petroleum and coal products, rubber and miscellaneous plastic products, primary metal industry, and fabricated metal products).

Data for the value of U.S. production series of the groups above is shown in Table 10.3.

Let

i = index for the commodity groups as defined above;

$i = 1, \ldots 6$

V_{it} = value of U.S. production in the i^{th} commodity group in the t^{th} year

Table 10.3. Value of U.S. production for the commodity groups indicated.

(thousands of dollars unless otherwise stated)

Year	Uranium (1)	Coal (2)	Copper (3)	Gold and Silver (4)	Petroleum (5)	Other (mil. dol.) (6)
1956	70,601	2,648,789	938,532	98,994	8,380,572	44,539
1957	81,181	2,732,160	654,289	97,317	9,281,018	44,233
1958	116,397	2,184,089	515,127	91,746	8,696,563	42,298
1959	141,349	2,137,927	506,455	84,336	9,030,136	48,895
1960	152,188	2,097,541	693,468	86,182	9,210,151	48,605
1961	148,299	1,984,901	699,093	86,355	9,561,823	48,566
1962	138,294	2,025,647	756,707	93,919	9,914,123	52,214
1963	115,821	2,166,812	747,310	95,965	10,293,773	57,026
1964	111,707	2,314,230	812,901	97,951	10,404,767	61,687
1965	83,915	2,398,043	957,028	111,151	10,652,692	68,337
1966	152,281	2,521,956	1,033,850	119,582	11,429,182	75,169
1967	165,239	2,651,537	729,401	105,582	12,276,257	78,770
1968	182,698	2,643,585	1,008,195	128,229	12,963,514	85,442
1969	142,161	2,896,278	1,468,400	146,984	13,882,295	91,253
1970	149,464	3,878,003	1,984,484	143,136	14,919,406	90,113
1971	151,996	4,004,965	1,583,071	125,931	15,778,480	94,416
1972	162,272	4,647,234	1,704,796	147,704	15,886,972	110,433
1973	167,830	5,139,872	2,044,346	211,762	17,951,977	128,963
1974	192,560	9,647,042	2,468,964	339,027	28,153,951	153,261
1975	276,102	12,670,967	1,814,763	324,352	32,061,121	144,093
1976	404,830	13,398,715	2,234,975	280,668	35,801,316	167,308
1977	582,249	13,910,357	2,009,297	339,517	41,616,676	- 0
1978	643,800	14,664,071	1,990,323	406,005	46,678,965	- 0

- 0 = missing value

Sources: (1) to (5)- Minerals Yearbook. Prepared by the Staff of the U.S. Bureau of Mines, Various Issues.

(6)- U.S. Department of Commerce, Bureau of Census, 1975. Historical Statistics of the United States of America

U.S. Department of Commerce, Bureau of Census. Statistical Abstracts. Various Issues.

$$v_{it} = \frac{V_{it} - \overline{V}_i}{\sigma_{V_i}} = \text{"standardized"}^1 \; V_{it}$$

$\overline{V}_i$ = mean value of the i^{th} commodity group value series

σ_{V_i} = standard deviation of V_i

Define

$$X_t = \sum_{i=1}^{6} \alpha_i \; V_{it}$$

$$x_t = \frac{X_t - \overline{X}}{\sigma_X} = \text{"standardized"}^2 \; X_t$$

where

α_i = fraction of revenues of the (typical) firm due to production of the i^{th} commodity group.

The correlation coefficient r_{v1x} resulting from regressing v_1 with x provides a plausible estimate for r:

$$\hat{r} = r_{v_1,x}$$

Ideally, α_i should be a reflection of the composite of uranium industry firms. The annual reports of firms in the sample examined do not always show sufficient revenue disaggregation to support the computation of α_i

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1. To isolate the variation in business cycles, it is standardized to avoid scale effects.

2. To isolate the variation in business cycles, it is standardized to avoid scale effects.

on an individual basis. If they did, the approach suggested in the next section could replace the procedure described here. Instead, only partial information exists on individual firms' value of α_l, which at best can only support a reasonable assumption for the weights.

For demonstration purposes, the following values for the typical firm's weights were used:

$\alpha_1 = 0.05$ (uranium)

$\alpha_2 = 0.20$ (coal)

$\alpha^3 = 0.12$ (copper)

$\alpha_4 = 0.08$ (gold and silver)

$\alpha_5 = 0.45$ (petroleum)

$\alpha_6 = 0.10$ (other)

The regression of v_1 with x yields a coefficient of correlation, $r_{v1x} = 0.8479$. Hence, the estimated coefficient of correlation, r, of the value of a uranium project with the value of existing operations of the typical firm is taken to be 0.85: $\hat{r} \approx 0.85$. The high value estimate for r shows that there is a lack of diversification effect with respect to uranium projects in the typical firm whose r computation is demonstrated here. This may reflect interrelationships between prices of different fuels. It is advisable to investigate the sensitivity of r, as the weights, α_l, vary, to see how serious the distortions are that are caused by the assumptions of the typical firm's operations.

10.4.2.3 Addendum on the Estimation of r for the Typical Firm

Data problems prevent statistical estimation of the average (typical) firm to be used in the potential supply system. Consider the following approach, which would allow computation of r for the typical firm.

Let:

S_{it} = total revenue by the i^{th} firm in the t^{th} year

$(i = 1, \ldots, I)$

I = the total number of uranium producing firms in the U.S.

U_t = value of U.S. uranium production in the t^{th} year

β_{it} = weight to be given to the i^{th} firm in the t^{th} year to reflect its contribution to the uranium industry

$$\sum_{i=1}^{I} \beta_{it} = 1,$$

Define

$$A_t = \sum_{i=1}^{I} \beta_{it} \cdot S_{it}$$

The coefficient of correlation resulting from regressing standardized[1] versions of U with A provides an estimate for r, the coefficient of correlation of a uranium project with existing operations of a typical firm.

This approach requires data to compute the weights, β_{it}. An adequate weighting scheme is the amount of uranium produced by each firm in the sample:

$$\beta_{it} = \frac{\text{amount of uranium produced by firm i in year t}}{\sum_{i=1}^{I} \text{amount of uranium produced by firm i in year t}}$$

The only reason for mentioning this alternative procedure here is because DOE's confidential data files contain information that may be assembled to compute the coefficient of correlation, r, by this method. These data were not available for this study. An added benefit would be that the same data would also facilitate estimation of the first three parameters mentioned in section 10.4.2.2 of this chapter.

1.

$$u_t = \frac{U_t - \bar{U}}{\sigma_U}$$

$$a_t = \frac{A_t - \bar{A}}{\sigma_A}$$

$\bar{U}$ = mean of U

$\bar{A}$ = mean of A

σ_U = standard deviation of U

σ_A = standard deviation of A

CHAPTER 11

CONCLUSIONS

11.1 Major Conclusion

The major conclusion of this study is that it is feasible to design an exploration model and computer program that generates its own behavioral and decisions rules regarding the conduct and engineering of exploration activities internally in response to 1) exogenously determined and specified economic conditions, 2) the state of knowledge already acquired by previous exploration, 3) uncertainty about the economic value of potential deposits, and 4) the firm's aversion to risk.

However, this study found that the design and construction of such a system requires orders of magnitude greater effort and information than previously designed exploration models. Consequently, a modeling effort like that of this work should only be undertaken when it is required to achieve _a priori_ stated objectives that cannot be achieved by simpler models. One of these is a comprehensive description of potential supply for prices that are much higher than, say 3 or 4 times previous or current prices. Another is when the risk of exploration under the contemplated economic regimes is considered to be greater than current risk and the objective is to include the economic consequences of this increased risk in the estimate of potential supply. Clearly, when the relevant economic circumstances, such as product price, are very different from those of our experience, existing behavioral rules, such as drilling density, are not relevant. For such rules to be useful, they must be generated internally by the model itself in response to some optimization rationale. As explained below, this is one of the primary contributions of this study.

11.2 Contributions of This Work

The primary contributions of this system design are: (a) to comprehensively account for the major costs involved in exploration and production and (b) to articulate a system in which amount and intensity of exploration are responsive to economic conditions;[1] that is, to design a system that determines internally exploration and production strategies that maximize profit when explicit consideration is given to risk and to its economic consequences. Specifically, this system employs capital markets and statistical decision theory to estimate the expected value of the risk reduction from conducting the next stage of exploration. The structure of this potential supply system is generic; nevertheless, some components (discussed in section 11.6.1. of this chapter) have to be redesigned for the estimation of potential supply from other mineral deposit types.

A realistic simulation of exploration in both physical and economic terms is a goal that springs naturally from the achievements referred to above. Of course, this greatly compounds the design task.

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1. In the system only price is manipulated. Other economic variables (discount rate, royalties, cost index, etc.) that remain fixed could be manipulated.

Two design decisions are instrumental for a realistic simulation in which exploration is optimized in a comprehensive risk-based analysis:

* Activities involved in creating new supply are separated into their <u>physical</u> elements and <u>economic</u> elements. Both are integrated within the system for the optimization of exploration.

  This integration is facilitated by a mining cost model that is used to translate the state of nature (either true or perceived) into financial terms (present value).

* The mining cost model, a fairly complex and detailed one, accepts (1) a physical description of deposit morphology,[1] and (2) specified values of economic variables; e.g., price, and determines that cutoff grade that maximizes net present value. A design of a mining cost model made specifically for the system is necessary because of its key use in providing the link between the physical world and economics. Such a linkage is needed in several operations of the system (in the exploration model) besides the evident one that has to do with costing a discovery, determining its net present value, and its contribution to potential supply.

This highly structured system includes an exploration model that is unique in several ways when compared with existing models. There are exploration models that find the optimal drilling density but rely on the unrealistic assumption of a single stage exploration program. Others

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1. Morphology means the areal dimensions (length, width, and thickness), grade distribution, and a simplified spatial distribution of grades.

model the sequential aspects of exploration drilling, but since they rely on predetermined drilling rules they are unable to find the optimal exploration activity levels at each stage. As far as can be determined from models reported in published papers, the exploration model of this study is unique in the following ways:

* It is the most disaggregated sequential model designed to date.
* It is the first model that includes geologic features that are used as exploration guides by the model and that are related to the deposits that comprise the seeded endowment.
* It simulates exploration according to the exploration architecture actually used in the search for roll-type uranium deposits; this architecture includes the search in early stages for geologic guides to the deposits, followed by subsequent search for the deposits themselves.
* It is the only exploration model that separates the physical elements of the search process from the economic elements and links the two to find the optimal level of exploration activity at every stage; thus, it avoids the use of prescribed exploration rules of thumb and generates its own decision rules, rules consistent with profit maximization and the state of prescribed economic variables, such as price, when explicit consideration is given to risk.

11.3 Desirable Attributes of This System

The following attributes of the system design help to capture the essence of creating new supply:

1. Exploration is sequential and based upon a full description of the architecture of the exploration as practiced by industry in the search for roll front-type deposits. The exploration model recognizes the fact that before searching for a deposit, the agent searches--and actually drills--for geologic guides that will help him to "home in" and discover a deposit.

Exploration can cease at any point. The process starts with an _a priori_ distribution of a geologic attribute associated with the endowment which is subsequently updated to become the _a posteriori_ distribution. This revised probability distribution of the relevant attribute at the end of a particular exploration stage is required to simulate the perceived physical state that may be found in subsequent stages of the architecture alluded to above. The original idea was to use Bayesian revision at every stage; however, in the latest stages the straight forward revised probability is an easier question to answer than the elements involved in the Bayesian equation (likelihood and experience terms) according to our pilot elicitation.

2. The perceived state of nature upon completion of any exploration stage (a probability distribution of the size of a relevant geologic guide associated with endowment) can be translated into monetary units (present value distribution) using the mining cost model. This is a feature used throughout the exploration model.

3. Exploration budgets, optimal ones, are determined at every stage by a pre-exploration module according to the price regime that prevails in the system, the state of nature perceived by the exploration agent

being simulated, and the net present value distribution implied by that state. The model emulates the "pre-exploration" that, at least implicitly, should take place in industry to evaluate what rewards may be found and at what cost.

4. Risk is accounted for. The decision to continue or to cease exploration at any stage is executed by a decision module on the basis of the risk perceived (net present value distribution), the risk characteristics of a "typical" firm, and the impact on its cost of capital of continuing exploration in a particular drilling region. All risk analysis is based upon a probability distribution for net present value that includes those exploration disbursements still to be spent. Risk is considered by defining the cost of capital to the firm as a function of the uncertainties of the project and of the future earnings of the firm, and by recalculating the mean net present value at the new cost of capital implied by the annexation of the exploration project. Full accounting of the economic cost of risk is accomplished since only those exploration projects that render a positive contribution to the value of the firm are accepted. That provides the stopping rule in the search for a deposit. The use of the capital markets concepts mentioned above avoids the use of utility theory which would require the specification of a utility function in order to simulate the decision making process during exploration.

Thus, in addition to all the evident costs involved in exploration and production (including the rate of return), appropriate consideration is given to the cost of risk. This is a particularly valuable feature in simulating activities that are typically high-risk ventures.

5. The economic contamination of "rules of thumb," drilling density rules, and stopping rules--which are usually derived from experience under past or current economic conditions--are avoided. Instead, according to the price regime, the system creates its own rules (for example minimum attractive grade, minimum attractive deposit dimensions, number of holes to drill in a given stage, or when to quit an area) by consulting the executive model.

11.4 Work Status on System Construction

Separate computer test runs of most of the components of the system were made. Results showed the trend and behavior expected, but because of time and budget constraints, full assemblage of the system could not be accomplished.

A tradeoff had to be made of this assemblage and test results, against a full and detailed report of the system. The second option was selected because (1) the main commitment called for a report on methodology above all and (2) because a complete explanation of the philosophy of every piece of the system was important for further developments.

The following main elements of the system remain to be completed:

a. The pre-exploration module is partially done. Since it relies on Monte Carlo simulation procedures, response functions for net present value estimated externally from data generated by the full mining cost model are required to avoid invoking the complete model in each of the internal iterations. Also, a shortcut to the numerical optimization problem to find the level of exploration activity (number of drillholes) at every subsequent exploration phase that maximizes the expected net present value is needed.

b. The programming of the external routine that generates data on minimum attractive grade and deposit dimensions for the estimation of equations relating those attractiveness criteria functionally to price and depth.

11.5 Remarks on the Use of Subjective Opinion on Exploration Performance

An expert was used as a pilot study to procure two important sets of subjective opinion responses: (1) deposit morphology and geologic relationships; and (2) a measure of how the explorationist performs in each phase of the search process.

As explained below (section 11.6.1. of this chapter), the first set could have been avoided if hard data were available. It is the second set, the exploration performance questions, which are the core of the exploration model, that we are concerned with in this section.

Only one respondent was used because our objective was to structure the exploration model and make sure that all the exploration performance questions required to accomplish the design of the model can be answered.

The exploration model is designed to estimate what is discovered when a number of firms explore a region (here characterized by identical "typical firms" in terms of risk and cost of capital). For actual potential supply estimation runs, a sample of exploration experts in roll-type deposits would be preferred over the opinion of one expert.

The evidence from the pilot study conducted in this work is that nearly every exploration expert in this type of deposits should be able to answer the exploration performance questions. Different answers from different experts would be expected. As long as those responses only reflect differences in exploration efficiencies, one would be at ease, for what the exploration model ultimately attempts to capture--in terms of successes and failures--is the aggregate performance of industry.

However, one may end up dealing with a variety of overly optimistic or overly conservative exploration experts even if anonymity is granted. To dilute this unwanted effect, the respondents may be calibrated externally using hypothetical search circumstances for which the probability can be computed (e.g., geometric probabilities when searching for a hidden object) and elicit the subjective probability of discovery. The calibration factors so obtained can be used to adjust their responses in the exploration performance questionnaire.

A further refinement in the elicitation of exploration performance questions set is the use of an expert system "shell" to perpetuate the structure implied by those questions.

11.6 Other Uses and Further Research

11.6.1. Uses for Other Types of Occurrences

The methodology and system have a generic value. Using a simulator that models the creation of new supply--while preserving the essence of exploration and production activities--from undiscovered deposits should be appealing, not only for uranium deposits of the type referred here, but for all deposit types. The architecture of the system should be applicable to other type of uranium occurrences or to other kinds of minerals--even oil for that matter.

Although the system architecture can be preserved, the creation submodel and the exploration module are specific for roll-type deposits. To use the system for other modes of occurrence or for other minerals, requires both another exploration module and another creation submodel. This is so because geologic guides, exploration activities, and exploration architectures vary with geologic environment and type of deposit. On the other hand, the mining cost model should be applicable to most modes of occurrence provided that the confining prism methodology is adapted to those type of deposits. The drawback is that it is an open pit model; an underground mining cost model needs to be developed. Except for an update on the risk-cost of capital relationship, the decision module does not require any other changes.

A major difficulty lies in the demanding requirements of the system: the existence of a clearcut exploration architecture and its linkage with geologic guides, and above all, the modeling of deposit morphology and its relevant geologic relationships (relevant with respect to the exploration architecture). It is so because a simulator is required to "seed" the deposits that are implied by the endowment distribution (creation submodel); that is, deposits are created (with some simplified morphology) along with relevant geologic guides that will be searched for in one or several stages of exploration prior to the actual discovery of the deposit.

Exploration architecture: Some solutions may be available for these data requirements. Exploration architectures for different kinds of mineral deposits may be found in the literature, but often they are scarce and not very detailed. Although they often do not contain enough information on the geologic guides searched for at different stages, they could be complemented and/or modified by the respondent to exploration performance questions. In fact, it is necessaria to have the respondent agree with the exploration architecture for the sake of consistency and feasibility of the elicitation of exploration performance as explorationists may have architectures that differ in some regards. Such differences may compound the work considerably if a sample is needed. Nevertheless, for the roll-type uranium deposits of this study, no evidence was found to suggest any irreconcilable differences in architectures across a sample of explorationists or the need to change the exploration performance questionnaire.

Deposit morphology: To expedite design, deposit morphology descriptions were obtained via subjective opinion from the same expert used for the exploration performance questions. An appealing feature of such practice is that it promotes consistency between performance and morphology. Consistency is important because modeling requires disaggregating exploration experience into states of nature and into performance measures that later can be integrated internally by the models in the system. A hybrid could be considered in case one is concerned about the consistency alluded above by giving the geologist the opportunity to modify the hard data results; i.e., statistical compilations. An added benefit of such an approach is that he would not have to "stretch his mind" when he is asked questions about morphology at very low and unfamiliar grades (0.01% U_3O_8 was the cutoff grade used here for endowment purposes). In any case the "confining prism" methodology outlined in this work should provide a simple but adequate and flexible representation of deposit morphology for most types of deposits.

Geologic relationships: The most difficult piece of data required is that on geologic relationships. Early efforts in obtaining geologic relationships from "hard data" were directed toward external modelling of geochemical groundwater anomalies (helium and radon plumes) that would be generated by a deposit of a given size and grade. The purpose was to "seed" these deposit and anomalies to simulate the search for the latter during the early exploration stages. Solute transport models in porous media were used to represent the "anomaly formation" process. The use of a "hard" device rather than subjective opinion on geologic features is attractive. However, it was learned that the size and concentration of the anomaly are very sensitive to several highly speculative inputs whose values are subject to strong local variations. Due to the high level of information required by any solute transport model, its application in a highly generalized exploration model is not feasible.

Geologic relationships in this system were obtained by subjective opinion as an expedient to design. Peculiarities of roll-type deposits facilitated this task: clear and relatively simple geologic guides to exploration (redox fronts), their strong marriage with a well established exploration architecture, and a tractable "degree of association" between geologic guides and deposits.

In its purest form, the geologic relationships required for any mode of occurrence are: areal size of the geologic guides, its dependence upon the size of the deposit (i.e., endowment), the "degree of association" with the deposit; that is, the probability of having a deposit without a given geologic feature and the probability of having a given geologic feature when a deposit does not exists (false signal). The diversity of deposit sizes and dimensions of the geologic guides compounds the problem.

A more practical approach may be to consider a binary situation in which the "seeded" geologic feature is just a record of presence or absence of that feature, leaving its size implicit in the elicitation of exploration performance. This approach should be investigated further.

Given the aforementioned difficulties, the number of geologic guides to be "seeded" should be kept at a minimum and both the geologic interpretation and other geologic features that are not too relevant should be left implicit in the exploration performance questions.

The discussion on geologic relationships above underscores the need for hard data, not only for uranium but for any mineral. The geological profession should be made aware of the necessity for statistical analysis of data on deposit sizes, dimensions of geologic guides, and degree of association that is in their internal reports and, sometimes, in published articles, but that is seldom analyzed within a formal statistical framework.

11.6.2 Other Applications

The system was designed with a main problem in mind: finding an improved methodology for the assessment of resource adequacy. Other applications are possible and should encourage the assembly of a data bank on geologic relationships and conduct the statistical analysis referred to in the previous paragraphs.

A system of this nature could be employed as a support to evaluate the economic viability and potential of mineral leases or concessions. The estimation of the risk involved and the potential economic rent in a region or in a tract of land would provide a reference for negotiation of the terms of the concession. This would be particularly useful for federal leasing in industrialized countries and for mineral/oil concessions or risk contracts with foreign companies in resource-rich developing countries.

Another potential use is in the ranking of exploration projects by a company, especially for government owned monopolies where the overall budget is limited and rigid. This capital budgeting problem is sometimes solved either by informal subjective procedures or, at best, with simple exploration models. A system architecture of the nature developed in this work should provide a different ranking than the other two procedures mentioned, for it explicitly accounts for the economic cost of risk. The system could be adapted to select the optimal set of projects so as to maximize the contribution of their net present values to the value of the company so that exploration project risk enters in the selection.

The concepts used in the exploration model (including those in the pre-exploration module) could be applied to aid exploration budgeting and decisions at different stages during the exploration of a given area. In some companies, including large ones in developing countries, this process is still informal. The interesting elements in the model are the translation of both state of nature expectations and exploration performance in remaining exploration stages into present value distributions using a generalized but detailed cost model. This methodology could be incorporated in a local budgeting/decision system that includes more complex optimization and decision theory elements.

A system of this nature is perfectible. It could evolve--with better data on geologic relationships and a more "user friendly" approach with the advent of new hardware and software--into a computer assisted exploration design system. It could serve the companies or pertinent government entities in the evaluation of their current exploration strategies, rules of thumb, and drilling rules, and arrive at an improved exploration design.

APPENDIX A

THE "CONFINE" ROUTINE DESIGN

Statement of the Problem

As discussed in Chapter 4, section 4.2.3, given the information characterizing a mineralized pod, it is desirable in the system operation to be able to obtain the dimensions of a confining prism of any specified grade. This is precisely the objective of this routine.

Procedure

Let

$\lambda(.01)$, $w(.01)$, $th(.01)$ = length, width and thickness of the 0.01-confining prism.

$\lambda(q^*)$, $w(q^*)$, $th(q^*)$ = length, width and thickness of the q^*-confining prism.

q^* = a cutoff grade higher than 0.01% U_3O_8 used as reference in the elicitation of Appendix E.

$VR2(q)$ = the shrinkage function describing the proportional decrease of the confining prism volume as the cutoff grade q increases.

$\lambda(q_0)$, $w(q_0)$, $th(q_0)$ = length, width and thickness of the q_0-confining prism.

q_0 = any cutoff grade $\geq$ 0.01% U_3O_8.

$V(.01)$ = volume of the 0.01-confining prism.

$V(q^*)$ = volume of the q^*-confining prism.[1]

$V(q_0)$ = volume of the q_0 - confining prism.

Given a mineralized pod with 0.01-confining prism dimensions: $\lambda(.01)$, $w(.01)$, $th(.01)$, the q^*-confining prism dimensions: $\lambda(q^*)$, $w(q^*)$, $th(q^*)$, and the form of the shrinkage function: $VR2(q)$, the <u>confine routine</u> computes the value for the dimensions of the q_0-confining prism as follows:

(1) Compute the volume of the q_0-confining prism

$$V(q_0) = VR2(q_0) \cdot V(.01)$$

[1]For the sake of consistency it is required that $V(q^*) = VR2(q^*) \cdot V(.01)$

and also that:

$$\lambda(q^*) \leq \lambda(.01)$$
$$w(q^*) \leq w(.01)$$
$$th(q^*) \leq th(.01)$$

(2) Let

$$\lambda(q_0) = \lambda(q^*) + x[\lambda(.01) - \lambda(q^*)] \quad \text{(A-1)}$$

$$w(q_0) = w(q^*) + x[w(.01) - w(q^*)] \quad \text{(A-2)}$$

$$th(q_0) = th(q^*) + x[th(.01) - th(q^*)] \quad \text{(A-3)}$$

Since

$$\lambda(q_0) \cdot w(q_0) \cdot th(q_0) = V(q_0) \quad \text{(A-4)}$$

Then, it is necessary to solve for x the following cubic equation:

$$a_0x^3 + a_1x^2 + a_2x + a_3 = V(q_0) \quad \text{(A-5)}$$

where

a_0, a_1, a_2, a_3 = coefficients resulting from the multiplication and grouping implied by the substitution of eqns. A-1, A-2 and A-3 in A-4.

The solution of equation A-5 for x and the substitution of this value in equations A-1, A-2 and A-3 yield the desired dimensions.

One can use any conventional numerical procedure to find the value for x that satisfies relation A-5. Multiple real roots may be possible. However, by employing the following bounds, the solution is unique:

Case 1 if $0.01 \leq q_0 \leq q^*$, then $0 \leq x \leq 1$

These bounds on x guarantee the resulting q_0-confining prism dimensions are in between the corresponding dimensions of the 0.01% and the q*-confining prisms (e.g., $\lambda(.01) \leq \lambda(q_0) \leq \lambda(q^*)$, etc.) Also, it ensures that when q_0 = 0.01% U_3O_8, the resulting dimensions reproduce the original 0.01-confining prism dimensions. By the same token, when $q_0 = q^*$, the original q*-confining prism dimensions are met.

Case 2 if $q_0 > q^*$, then $\alpha < x < 0$ $(\alpha < 0)$

The value for α has to be such that negative dimensions for the q_0-confining prism are prevented.

Since

$$d(q_0) = d(q^*) + \alpha[d(.01) - d(q^*)] \geq 0$$

where d = any dimension (i.e., λ, w, or th) of the confining prism.

Then

$$\alpha \geq - \frac{d(q^*)}{d(.01) - d(q^*)}$$

Since none of the dimensions of the q_0 are allowed to take negative values, then

$$\alpha = \max\left[\frac{\lambda(q^*)}{\lambda(.01) - \lambda(q^*)}, \frac{w(q^*)}{w(.01) - w(q^*)}, \frac{th(q^*)}{th(.01) - th(q^*)}\right]$$

APPENDIX B

EXPLORATION ARCHITECTURE FOR ROLL TYPE DEPOSITS AS DESCRIBED BY THE EXPLORATIONIST INTERVIEWED

Phase 1. Regional Appraisal

1. Size of the area being investigated:

   A_1 = large size, may be the size of a state.

2. Objective of this phase:

   Identification of a favorable basin, A_2. The size of this basin typically ranges from 300-550 square miles.

3. Activities:

   Examination of generalized data which are fitting the roll front type deposit model:

   (a) Literature research (publications, oil tests, thesis, etc.)

   (b) Purchasing of oil well data and examination of the logs, including SP and resistivity logs. Gamma logs are often unavailable.

4. Exploration guides to achieve objective stated above:

   (a) Source of uranium.

   (b) Host of uranium.

   (c) Evidence of transport mechanism that moved the uranium from the source to the host rock.

5. Information typically available upon completion of the phase:

   (a) Basin (A_2) identified as a favorable setting for roll front type deposits.

   (b) Literature reviewed.

   (c) Published general geologic maps of the selected basin.

   (d) If you are lucky, some fifteen oil well logs (SP and resistivity) from the selected basin.

Phase 2. Geologic Reconnaissance of the Basin

1. Size of the area being investigated:

   A_2 = 300 to 500 square miles.

2. Objective:

   (a) Projection of the channel systems.

   (b) Identification of the permissive area, F, whose size is usually 40-50% of A_2. Elimination mostly takes care of those parts in the basin high in claystone and shale.

   Although the size of F is usually in the 120-275 square mile range, typically one ends up with about three blocks of A_3, approximately 7 square miles each.

3. Activities:

   (a) Airborne activities conducted are:

   (a.1) Airborne radiometry, particularly along the outcrop of the sandstones existing in the basin.

   (a.2) Color photography. Employed to conduct geologic mapping, to identify channel systems in the outcrops and, if possible, to map alteration patterns.

   (b) The following set of typical field activities are conducted simultaneously:

   (b.1) Water sampling program (springs, wells, etc.).

   (b.2) Sampling of iron concentrations and nodules in outcrops. Samples are assayed for ppm uranium content.

   (b.3) Geochemical sampling of the sandstone outcrops.

   (b.4) Field mapping.

   (c) Building of paleogeographic models and integration with geochemical data.

4. Exploration guides to achieve the objectives stated above:

   (a) Evidence of channel systems.

   (b) Geochemical anomalies in water wells.

   (c) Bleached zones in outcrops.

   (d) Evidence of carbonaceous debris in the outcrops.

5. Information typically available upon completion of the phase:

   Projection of channel systems in the basin. Typically, four or five channel systems are identified. The projected location of these systems constitutes the favorable area, F.

Phase 3. Stratigraphic Drilling

1. Size of the area being investigated:

   Three portions (A_3's) of favorable ground, approximately 7 square miles each.

   If the sandstone trend is discernible, the shape of the A_3 is approximately a 2 × 3.5 rectangle oriented parallel to the projected trend. If it is not, then the shape is usually square.

2. Objective of this phase:

   The overall objective is to "home in" on favorable geology. The Phase 3 budget is directed toward:

   (a) Evaluation of the lithology and the obtaining of a geologic overview of the area.

   (b) Identification of the general trend of the redox front.

3. Activities:

   Drilling, usually on a fence pattern perpendicular to the projected trend of the channel system. Spacing within a fence ranges from 0.25 to 0.5 miles. Spacing between fences is usually about 1 mile.

4. Exploration guides to achieve objective of this phase:

   (a) Favorable host.

   (b) Altered and unaltered ground.

   (c) Occasionally some mineralization (.02 - .04% U_3O_8) in the favorable host.

5. Information typically available upon completion of the phase:

   (a) Logs of every drill hole (gamma ray, resistivity, SP)

   (b) Detailed cross sections.

   (c) Correlation of the following factors:

      - alteration pattern

      - lithology (sandstone/shale ratio)

      - any mineralization

   (d) General trend of the redox front.

Phase 4. Prospect Evaluation

1. Size of the area being investigated:

   Only a fraction of the 7 square mile area (A_3) is favorable. Drilling activities are concentrated along the redox front.

2. Objective:

   The overall goal in this phase is to determine the geologic potential (pounds of U_3O_8) of the prospect.

   This is the discovery phase. Upon successful completion of this phase, ore grade mineralization has been encountered and a geochemical system has been identified which has a potential for containing "x" pounds of U_3O_8 at a stated cutoff and average grade. It may turn out to be noncommercial, but it is a discovery. The uncertainty in this estimate has two sources: uncertainty about the continuity of the mineralization and uncertainty about the continuity of ore grade mineralization. This uncertainty is to be dissipated in Phase 5.

   The idea of Phase 4 is to enhance the perceived size of the prospect, that is, to come up with as much geologic potential (pounds of U_3O_8) as possible. Due to the morphology of roll type uranium deposits, the way to enhance the perceived size of the prospect is to demonstrate length of the mineral deposit.

3. Activities:

   (a) Drilling.
   Drilling is conducted on a fence pattern.
   The spacing within a fence is affected by the width of the deposit existing (or presumed to exist) along the redox front. A "binary search" type of drilling is conducted in each fence looking for the altered/unaltered contrast. Drilling on any particular fence is halted either when one of the holes nicks an anomalous uranium concentration and encounters the proper roll front configuration (tail or nose of roll front deposit), or when two contrasting holes (altered/unaltered) are too close together (typically 100').

   The spacing between the fences varies with the perceived sinuosity of the redox front. Typically, it is on the order of 400', although a fairly straight redox interface could allow this figure to increase to 1000'.

   (b) One or two cores to test for equilibrium.

   (c) Computerization of all drill hole data.

   (d) Subsurface mapping.

4. Exploration guides to achieve the objectives stated above:

   (a) The redox interface.

   (b) Ore and sub-ore grade mineralization.

Phase 5. Continuing Evaluation

1. Size of the area investigated:

   The size is highly variable. It is composed mainly of the zone along the redox front system containing ore grade mineralization.

2. Objective:

   (a) To resolve the uncertainty about continuity of both ore and sub-ore grade mineralization, and to determine if an economic uranium deposit exists.

   (b) To obtain reserve estimates of the inferred and indicated category. Some minor amounts of proven reserves may be determined. However, it is in the next drilling phase (development drilling) in which additions to this reserve category will be substantial.

3. Activities:

   (a) Drilling.

   In this phase, drilling is very systematic; one can drill additional fences to end up with a between-fence spacing ranging from 100' to 200', or one can simply lay out a grid and drill at the intersections.

   (b) Computerization of drill hole data (including grade-thickness information).

   (c) Reserves computation.

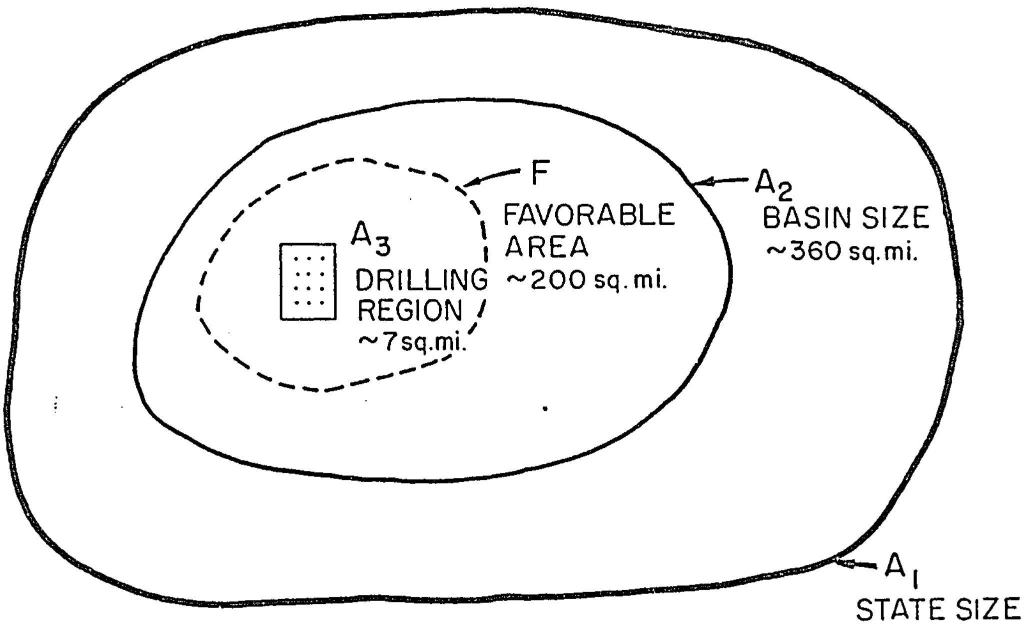
4. Exploration guides to achieve the objectives stated above:

   (a) Redox interface.

   (b) Ore grade mineralization.

APPENDIX C

SCHEMATIC DIAGRAM SHOWING THE AREAS INVOLVED IN EARLY EXPLORATION PHASES AS DESCRIBED BY EXPLORATIONIST INTERVIEWED



⋮⋮⋮ Indicates area reduction level at which formal drilling begins.

F = perceived permissive area after field/office work (Phase 2) has been conducted in A_2.

A_i = area subjected to investigation in the i^{th} phase.

A_{i+1} = area selected after the i^{th} phase has been conducted.

(no scale).

APPENDIX D

THE MODEL EXPLORATION ARCHITECTURE: A RESTATEMENT OF THE EXPLORATION ARCHITECTURE DESCRIBED IN APPENDIX B

Phase 1 — Regional Appraisal.

Major Objective: Select a basin favorable for the occurrence of economic roll front deposits.

Phase 2 — Geologic Reconnaissance of Basin.

Major Objective: Select drilling areas, A_3 (7 sq. miles), favorable for existence of economic roll front deposits.

Phase 3A — Reconnaissance Drilling and Redox Front Search.

Major Objective: Detect the redox interface, that is, to come up with at least one set of contrasting holes (altered/unaltered) in the intercepted favorable host.

Phase 3B — Following the Redox Front System General Trend.

Major Objective: Estimate the length of the redox front system.

Phase 4A — Deposit Search.

Major Objective: Detection of the mineralized target that may occur along the redox front.

Phase 4B — Preliminary Evaluation of the Discovery.

Major Objective: Determine the geologic potential of the prospect.

Phase 5 — Continuing Evaluation.

Major Objective: Determine inferred and indicated reserves of the prospect.

APPENDIX E

SUBJECTIVE OPINION ON DEPOSIT MORPHOLOGY QUESTIONNAIRE

QUESTION SM.1

TABLE SM.1 INSTRUCTIONS AND COMMENTS

Answering this table requires you to go through all Wyoming roll-front type uranium deposits you know of* and imagine a confining prism circumscribing those deposits. Questions will be asked about the width and thickness of that confining prism and how often those conditions occur in nature.

Question: Each box in Table SM.1 represents a particular combination of states of nature of a hypothetical confining prism enclosing mineralized material of grades $\geq .01$ % U_3O_8. You are asked to:

(a) Put 1000 in the box representing the most likely state of nature.

(b) Scale the other possible states of nature (boxes in the table) with respect to the 1000 box so that the scaled numerals reflect the relative frequency (e.g., 500 is 1/2 less frequent than 1000... etc...). If you feel that some states of nature are virtually impossible, you should enter zero in that box.

*Not necessarily economic.

QUESTION SM.1

TABLE SM.1. JOINT PROBABILITY TABLE FOR WIDTH AND THICKNESS OF THE CONFINING PRISM OF .01% U_3O_8 MATERIAL IN URANIUM DEPOSITS

		th(.01) = thickness of the confining prism circumscribing material at grades $\geq$ .01% U_3O_8					
		1-10	10-30	30-60	60-200	200+	
w(.01) = width of the confining prism circumscribing material at grades $\geq$ .01% U_3O_8 and no less than 1 ft. thick	1-27	700	80	20	5	0	
	27-66	650	800	70	20	5	
	66-200	600	950	750	250	50	
	200-400	70	880	1000	630	100	
	400-1060	20	330	500	920	200	
	1060-2660	0	5	30	80	300	

Note: all dimensions are in feet.

QUESTION SM.1

TABLE SM.1-T. CONDITIONAL PROBABILITY TABLE IN CUMULATIVE FORM

th(.01) \ w(.01)	1.0'	27'	66'	200'	400'	1060'	2660'
5.5'	0.0	0.3437	0.6622	0.9562	0.9902	1.0	
20'	0.0	0.026272	0.289005	0.600998	0.889988	0.998358	1.0
45'	0.0	0.00844	0.03794	0.3543	0.7763	0.9873	1.0
130'	0.0	0.00262	0.013118	0.144367	0.475076	0.958006	1.0
200+	0.0	0.0	0.007633	0.083971	0.236644	0.541987	1.0

Note: Each row represents Prob[$\tilde{w}(.01) \leq w(.01) | th(.01)$]

QUESTION SM.2

TABLE SM.2 INSTRUCTIONS AND COMMENTS

The objective of this table is to obtain the probability distribution of the thickness of a hypothetical confining prism circumscribing material down to the .01% U_3O_8 grade in Wyoming type roll-front deposits.

This probability distribution is in reality a conditional probability distribution, conditional upon the overall average thickness of the favorable lithology in the region.

Each row in the table corresponds to one probability distribution for that particular condition on the overall average thickness. It is suggested that you work on a row by row basis.

Question: (row by row) See example.

Example (row #1): Given that the overall average thickness of the favorable lithology in the region containing Wyoming roll-front type deposits is 200', what is the probability that the thickness of the confining prism enclosing material down to .01% U_3O_8 in Wyoming type roll-front deposits is no more than 15', 30', 60', 100', 200' and 300'?

QUESTION SM.2

TABLE SM.2. CONDITIONAL PROBABILITY DISTRIBUTIONS OF THE THICKNESS OF THE CONFINING PRISM CIRCUMSCRIBING MATERIAL OF GRADES $\geq$ .01%

Overall average thickness of the favorable lithology in the region	th(.01) = thickness of the confining prism enclosing mineralized material of grades $\geq$ .01%					
	$\leq$ 15'	$\leq$ 30'	$\leq$ 60'	$\leq$ 100'	$\leq$ 200'	$\leq$300'
200'	20%	45%	80%	92%	98%	100%
500'	*	25%	50%	80%	95%	100%
1000'	*	15%	30%	50%	80%	90%

* = 15' too small to give an answer.

QUESTION SM.3

Part (1)

The following two parts of question SM.3 will be dealing with dimensions of Wyoming roll-front deposits ALONG THE STRIKE of the Redox front system. Specifically these dimensions are : (a) the length of a composite pod down to the 0.01% U_3O_8 grade, (b) the "gaps" or total discontinuities (i.e., those portions along the strike of the Redox front system whose grade is below 0.01% or whose thickness is below 1 ft.)

Although we would like to define the length of a 0.01% U_3O_8 composite* pod** [λ(.01)] as a length containing absolutely no gaps, that is, a "pure continuous" length, we are aware of the limited information that is usually available which does not allow us to perceive the "true" state of nature of this continuous length. Instead, we ask you in this question to define this length.

Question: λ(.01) definition: What is in your view a "continuous"*** length λ(.01)? (i.e., up to what size gap (strike length) would you like to lump?)

Answer: x = up to 100 ft.

*The term composite implies the consideration of the combined total of all "ore zones" in the vertical direction, where "ore zone" is that mineralized portion whose grade is $\geq q$ (q in this question is .01%). This vertical "pancaking" or lumping of ore zones is both within and between sand lenses.

**It is preferable to use the term "pod" rather than "deposit" to emphasize the "continuous" length concept. Hence a deposit could be made up of a string of various "continuous" pods if they are close enough.

***The term continuous is quoted to emphasize that in fact the length we refer to as continuous includes gaps up to that size limit that you defined above.

QUESTION SM.3

Part (2)

TABLE SM.3 (2) CONDITIONAL PROBABILITY DISTRIBUTION OF "CONTINUOUS" LENGTH OF .01% U_3O_8 PODS [λ(.01)]

th(.01) = thickness of the confining prism enclosing material in a Wyoming roll-front type deposit down to the .01% U_3O_8 grade		λ(.01) = "continuous" length of the composite pod containing grades ≥ .01% and 1 ft minimum thickness					
		≤1000'	≤2000'	≤4000'	≤8000'	≤16000'	
th(.01)= 50'		20%	45%	65%	86%	99%	
th(.01)= 100'		10%	40%	60%	80%	95%	
th(.01)= 200'		5%	30%	45%	60%	80%	

INSTRUCTIONS AND COMMENTS:

Each row in the above table corresponds to one probability distribution for that particular condition on the thickness of the confining prism enclosing ≥ .01% material, th(.01). It is suggested that you work on a row by row basis.

Question: (row by row) Keeping in mind the definition for "continuous" length, λ(.01), that you stated in Part (1) of this question, state for each condition (row) the probability that the "continuous" length, λ(.01), of a pod* of grades, down to the .01% is no more than 1000', 2000', 4000', 8000' and 16,000'.

*Hence a deposit could be made up of a string of various "continuous" pods if they are close enough.

QUESTION SM.3

Part (3)

TABLE SM.3 (3) CONDITIONAL PROBABILITY DISTRIBUTION OF GAPS

thickness of the aggregate composite geochemical cell		gap(.01) (strike length)						
	50'	100'-300'	300'-900'	900'-2000'	2000'-6000'	6000'-9000'	9000'+	
		50	200	500	1000	700	200	
	100'	100'-300'	300'-900'	900'-2000'	2000'-6000'	6000'+		
		200	500	1000	500	200		
	200'	100'-300'	300'-900'	900'-2000'	2000'-6000'	6000'+		
		500	1000	600	400	200		

where gap(.01) = length of a transverse discontinuity, i.e., length of those portions along the strike of the Redox front system whose grade is below .01% or whose thickness is below 1 ft.

INSTRUCTIONS AND COMMENTS:

Each row in the table above corresponds to one probability distribution of gap(.01) for that particular condition on the thickness of the aggregate composite geochemical cell.

In this question you are asked to consider only those gaps whose size (length) is greater than the limit size gap (x) used to define "continuous" length of mineralized material down to .01%. It is suggested that you work on a row by row basis.

Question: (row by row) For each condition (row) in the table, assign 1000 to the most frequent event. Compare the remaining events in the row to that most frequent event and scale with respect to that 1000 so that the scaled numerals reflect the relative frequency (e.g., 500 is 1/2 less frequent than 1000, etc.)

QUESTION SM.4

TABLE SM.4 CONDITIONAL DISTRIBUTION OF WIDTH OF THE CONFINING PRISM

		w(.12) = width of the confining prism circumscribing material of grades ≥ 0.12%.		
		Low	Most Likely	High
w(.01) = width of the confining prism circumscribing material of grades ≥ .01%	w(.01) = 150'	10'	20'	35'
	w(.01) = 350'	15'	35'	50'
	w(.01) = 550'	40'	50'	70'

INSTRUCTIONS AND COMMENTS:

Each row in the table is to obtain your answers regarding the probability distribution of the width of the confining prism circumscribing material ≥ .12%, w(.12), in Wyoming roll-front type deposits for the particular condition on the width of the .01 confining prism, w(.01). It is suggested that the respondent fill in the table row by row.

Question: (row by row) Given the width w(.01) condition, state the low, most likely and high width of the confining prism circumscribing material of grades ≥ .12, w(.12).

QUESTION SM.5

TABLE SM.5 CONDITIONAL DISTRIBUTION OF THICKNESS OF THE CONFINING PRISM

		th(.12) = thickness of the confining prism circumscribing material of grades ≥ .12%		
		Low	Most Likely	High
th(.01) = thickness of the confining prism circumscribing material of grades ≥ .01%	th(.01)= 50'	20'	30'	40'
	th(.01)= 100'	50'	70'	80'
	th(.01)= 200'	120'	150'	170'

INSTRUCTIONS AND COMMENTS:

Each row in the table is to obtain your answers regarding the probability distribution of the thickness of the confining prism circumscribing material ≥ .12%, th(.12), in Wyoming roll-front type deposits for the particular condition on the thickness of the .01 confining prism, th(.01). It is suggested that the respondent fill in the table row by row.

Question: (row by row) Given the thickness of the .01 confining prism condition [th(.01)], state the low, most likely and high thickness of the confining prism enclosing material ≥ .12 in Wyoming roll-front type deposits, th(.12).

QUESTION SM.6

TABLE SM.6. RELATIONSHIP BETWEEN WIDTH OF THE CONFINING PRISM AND WIDTH OF THE MINERALIZED ZONE AT VARIOUS GRADES

$w(q)$ = width of confining prism enclosing material $\geq q$ in Wyoming roll-front type deposits	q = cutoff grade 0.01	0.05	0.15
100'	80'	50'	25'
200'	160'	115'	35'
350'	280'	200'	45'
500'	375'	300'	50'
700'	520'	420'	60'

INSTRUCTIONS AND COMMENTS:

Each box in the table represents a particular case.

Question: For that given grade q and width of the confining prism enclosing material $\geq q$ in Wyoming roll-front type deposits, $w(q)$, state in each box the most typical average width of the mineralized material $\geq q$, $\bar{\Omega}(q)$.

Example (box 2) What would be the most typical average width of mineralized material above .05%, $\bar{\Omega}(.05)$, when the width of the confining prism enclosing material $\geq .05$ in Wyoming roll-front type deposits is 100'.

Note: If you want a reference assume the following:

$\ell(.01)$ = length of the confining prism circumscribing this uranium deposit at .01% cutoff grade = 1 mile

th(.01) = thickness of the confining prism circumscribing the uranium deposit at .01% cutoff grade = 100 ft.

QUESTION SM.7

INSTRUCTIONS AND COMMENTS:

In this question you are given the dimensions of a hypothetical confining prism circumscribing material down to the .01% U_3O_8 grade in Wyoming type roll-front deposits. These reference dimensions are given below the table.

You can answer the question either in volume units or in tons, whatever you prefer.

Question:

Column 1: Mineralized material above q.

State the volume (tonnage) of mineralized material above the cutoff grades (q) listed that would typically exist inside the reference .01% confining prism.

[An alternative can be to answer this question for the .01% cutoff grade only and state the % of that volume (tonnage) that is made up of material with grades $\geq$.05, .10 and .15%.]

Column 2: Confining prism enclosing mineralized material above q.

The total volume V(.01) [and tonnage T(.01)] of the .01 confining prism is given in the first row. State the typical total volume V (or tonnage T) of the confining prism that circumscribes mineralized material above the cutoff grades (q) listed.

[An alternative is to state the % of the total volume (tonnage) of the .01% confining prism that is made up of other confining prism volumes (tonnages) at higher grades.]

QUESTION SM.7

TABLE SM.7. VOLUMETRIC RELATIONS IN WYOMING ROLL-FRONT DEPOSITS

		mineralized material of grades ≥ q		confining prism enclosing mineralized material of grades ≥ q	
		v(q) or	t(q)	V(q) or	T(q)
q = cutoff* grade	.01%	$56 \times 10^6 ft^3$		$87.5 \times 10^6 ft^3$	6.25×10^6 tons
	.05%	$14 \times 10^6 ft^3$		$52.8 \times 10^6 ft^3$	
	.10%	$3.5 \times 10^6 ft^3$		$40 \times 10^6 ft^3$	
	.20%	$0.7 \times 10^6 ft^3$		$20 \times 10^6 ft^3$	

where q = cutoff* grade;

v(q) = volume of mineralized material above q in Wyoming type roll-front deposits;

t(q) = tonnage of mineralized material above q (tonnage factor = 14 ft^3/ton);

V(q) = volume of the confining prism enclosing mineralized material above q;

T(q) = tonnage of the material inside the confining prism that circumscribes mineralized material above q.

REFERENCE: A confining prism circumscribing mineralized material above .01.

This confining prism has the following dimensions:

ℓ(.01) = 2500'
w(.01) = 350'
th(.01) = 100'

*Cutoff grade in this context does not refer to mining cutoff.

QUESTION SM.8

INSTRUCTIONS AND COMMENTS:

This question is designed to obtain an estimate of the longitudinal "room"* (i.e., length of the REDOX front system) needed for a given U_3O_8 endowment** level to occur in the mode of Wyoming roll-front type deposits.

Each row in the table corresponds to one probability distribution for that particular condition on the endowment occurring in the region. It is suggested that you work on a row by row basis.

Question: (row by row) Given E_T, the endowment level existing in the region,

a) State in column A the minimum Redox system length, L_A, required to contain that endowment.

b) State in column B the length, L_B, for which there is an 85% probability of being exceeded. Likewise, state in column E the length, L_E, for which there is a 15% probability of being exceeded.

c) Select in columns C and D two Redox system lengths, L_C and L_D, falling in between L_B and L_E. Thereafter, state the probabilities of exceeding those intermediate lengths.

*Note that we are not interested in what would be the volume (from which length could be derived by assuming some width and thickness) occupied by that endowment level. Our interest is in the length of that aggregate system of oxidation-reduction interfaces including grade discontinuities and trace portions along the system.

**A uranium deposit is defined as a discrete lens or a cluster of smaller lenses of uranium-bearing rock containing at least 10 tons (20,000 lbs) of U_3O_8 at a cutoff grade of .01% U_3O_8 and minimum thickness of 1 ft.

Endowment is defined as the U_3O_8 in a region contained in uranium deposits meeting the minimum requirements as defined above.

QUESTION SM.8

TABLE SM.8. CONDITIONAL PROBABILITIES OF REDOX SYSTEM LENGTH GIVEN ENDOWMENT

	A	B	C	D	E
E_T = 25	L_A = 4 mi	L_B = 15 mi $P(L \geq L_B)$ = 85%	L_C = 25 mi $P(L \geq L_C)$ = 75%	L_D = 45 mi $P(L \geq L_D)$ = 30%	L_E = 65 mi $P(L \geq L_E)$ = 15%
E_T = 75	L_A = 15 mi	L_B = 50 mi $P(L \geq L_B)$ = 85%	L_C = 73 mi $P(L \geq L_C)$ = 75%	L_D = 117 mi $P(L \geq L_D)$ = 35%	L_E = 175 mi $P(L \geq L_E)$ = 15%
E_T = 150	L_A = 35 mi	L_B = 125 mi $P(L \geq L_B)$ = 85%	L_C = 167 mi $P(L \geq L_C)$ = 73%	L_D = 233 mi $P(L \geq L_D)$ = 30%	L_E = 300 mi $P(L \geq L_E)$ = 15%

E_T = total endowment (million lbs of U_3O_8) existing in a region

L = redox system length (in miles) in the region

P = probability operator.

APPENDIX F

SUBJECTIVE OPINION ON EXPLORATION PERFORMANCE QUESTIONNAIRE

Phase 2: Geologic Reconnaissance of Basin

QUESTION P2.1

INSTRUCTIONS AND COMMENTS:

Part (1): Area A_3 contains some portion of the REDOX front system.

Two tables are given for the outcrop/no outcrop cases. In both cases we can assume that oil well logs are available.

Each square in the table represents a different case ("state of nature") indicated by the corresponding headings.

Assume:

You have conducted Phase 2 and you have performed all activities dictated by your "Best Practice." These activities*, including data interpretation and integration, have been performed over a large area A_2 (the basin).

You are now going to select** an A_3 area of size = 7 sq. mi. (as specified in your typical exploration sequence description) in which you will conduct the subsequent exploration phases.

Question: (for each square in table):

Suppose you are under case "x" and you are considering an area A_3 of size = 7 sq. mi. in which the average depth of the favorable lithology is "y" ft.; A_3 contains L miles of Redox front system. What is the probability that you will recognize this A_3 as being "favorable?"

Part (2): Area A_3 does not contain any portion of the REDOX front system.

Two tables are given for the outcrop/no outcrop cases. In both cases we can assume that oil well logs are available.

Each square in the table represents a different case ("state of nature") indicated by the corresponding headings.

Assume:

You have conducted Phase 2 and you have performed all activities dictated by your "Best Practice." These activities,* including data interpretation and integration, have been performed over a large area A_2 (the basin).

You are now going to select** an A_3 area of size = 7 sq. mi. (as specified in the previous question) in which you will conduct the subsequent exploration phases.

Question (for each square in table):

Suppose you are under case "x" and you are considering an area A_3 of size = 7 sq. mi. in which the average depth of the favorable lithology is "y" ft.; A_3 does not contain any portion of the Redox front system. What is the probability that this A_3 area would be incorrectly perceived as being "favorable?"

*Activities such as: airborne surveys and photography, water sampling, geochemical sampling of outcrop of favorable host (if outcrop available), field mapping, etc.

**A_3 is a contiguous area. You may select more than one A_3 area but this is not relevant for the answer to this question.

TABLE P2.1. PHASE 2 PERFORMANCE PROBABILITIES

Part (1): Area A_3 contains some portion of the REDOX front system.

Case I. Favorable lithologic grouping crops out in the basin

L = length (miles) of REDOX front system contained in area A_3	Average depth of the favorable lithologic grouping in the A_3 area: 100'	500'	1000'	2000'	
1 mi.	20%	15%	10%	2%	
3 mi.	35%	25%	15%	8%	
6 mi.	45%	30%	20%	12%	

Case II. Favorable lithologic grouping DOES NOT crop out in the basin

L = length (miles) of REDOX front system contained in area A_3	Average depth of the favorable lithologic grouping in the A_3 area: 100'	500'	1000'	2000'	3500'	5000'
1 mi.	10%	8%	5%	4%	2%	1%
3 mi.	22%	15%	8%	6%	3%	2%
6 mi.	28%	22%	12%	8%	6%	3%

TABLE P2.1 PHASE 2 PERFORMANCE PROBABILITIES

Part (2): Area A_3 does not contain any portion of the REDOX front system.

Case I. Favorable lithologic grouping crops out in the basin

	Average depth of the favorable lithologic grouping in the A_3 area				
	100'	500'	1000'	2000'	
L = 0	65%	80%	90%	92%	

Case II. Favorable lithologic grouping DOES NOT crop out in the basin

	Average depth of the favorable lithologic grouping in the A_3 area					
	100'	500'	1000'	2000'	3500'	
L = 0	70%	88%	90%	92%	95%	

Phase 3A: Reconnaissance Drilling and Redox Front Search

QUESTION P3A.1 CONDITIONAL PROBABILITY OF 3.A SUCCESS

TABLE P3A.1

n_c = total number of holes that you have drilled in Phase 3, Part (A). (Assume that n_c holes are ALL you have and you do the BEST you can with that many holes to achieve your objective of Phase 3, Part A.)

"state of nature"

L = length (miles) of REDOX front system contained in the A_3 = 7 sq. mi. area being explored

L \ n_c	15	25	35	45	55	70
.25 mi	5%	15%	30%	45%	60%	70%
.50 mi	15%	30%	45%	65%	80%	90%
1.5 mi	30%	60%	80%	85%	90%	100%
3 mi	40%	65%	85%	95%	100%	100%
5 mi	50%	70%	95%	100%	100%	100%

INSTRUCTIONS AND COMMENTS:

Assume: You are conducting Phase 3, Part A drilling in an area A_3 = 7 sq. miles.

Your objective is to obtain a geologic overview of your A_3 area and to encounter the oxidation-reduction interface, that is, to come up with at least one set of contrasting holes (one in oxidized ground and another in reduced ground).

Question: Each box in the above table represents a different set of conditions. For each box, state the probability that you will encounter at least one set of contrasting holes after having completed no more than n_c drill holes when the length of the REDOX front system contained in your search area A_3 is "L" miles.

QUESTION P3A.2 THE "EXPERIENCE" QUESTION

INSTRUCTIONS AND COMMENTS:

Each one of the 4 tables provided represent different cases for L, the miles of Redox front contained in the $A_3 = 7$ sq. miles area that you are exploring.

For each of the tables do the following:

<u>Question</u>: Each box in the table represents a particular combination of joint events (e.g., having a SUCCESS <u>and</u> drilling n_c Phase 3A holes) given that A_3 contains L miles of Redox front system. In each table you are asked to state the frequency that you have experienced each event.

QUESTION P3A.2 JOINT PROBABILITY TABLES OF NUMBER OF 3.A HOLES AND SUCCESS IN 3.A

TABLES P3A.2

L = 0 miles

	n_c 1-25	n_c 26-45	n_c 46-70
S_{3A}	X	X	X
$\bar{S}_{3A}$	0	3	0

L = >0 - 1.5 miles

	n_c 1-25	n_c 26-45	n_c 46-70
S_{3A}	2	8	2
$\bar{S}_{3A}$	0	1	0

L = 1.5 - 5.0 miles

	n_c 1-25	n_c 26-45	n_c 46-70
S_{3A}	0	3	4
$\bar{S}_{3A}$	0	0	0

L = 5.0 - 8.5 miles

	n_c 1-25	n_c 26-45	n_c 46-70
S_{3A}	0	3	0
$\bar{S}_{3A}$	0	0	0

where n_c = total number of holes drilled in Phase 3, Part A;

S_{3A} = success in 3.A, i.e., coming up with at least one set of contrasting holes (one hole in oxidized ground and another in reduced ground);

$\bar{S}_{3A}$ = non success in 3.A;

L = length (miles) of REDOX front system contained in the A_3 = 7 sq. mi. area being explored.

QUESTION P3A.2

TABLE P3A.2 - T. CONDITIONAL TABLE FOR RELATIVE FREQUENCIES OF NUMBER OF PHASE 3.A HOLES GIVEN THE LENGTH OF THE REDOX SYSTEM

	n_c		
	1-25	26-45	46-70
$L = 0$	0	3/3	0
$0 < L < 1.5$	2/13	9/13	2/13
$1.5 < L \leq 5$	0	3/7	4/7
$5 < L \leq 8.5$	0	3/3	0

n_c = total number of holes drilled in Phase 3.A

L = length of redox system occurring in an $A_3 \simeq 7$ sq. miles (in miles)

Note: This table is simply a restatement of responses in Table P3A.2.

TABLE P3A.2TN NEW CONDITIONAL TABLE FOR RELATIVE FREQUENCIES OF NUMBER OF PHASE 3.A HOLES GIVEN THE LENGTH OF THE REDOX SYSTEM

L	n_c 1-25	26-45	46-70
0	0.0	1.0	0.0
0-0.5	0.113620	0.824269	0.0621109
0.5-1.0	0.216050	0.619201	0.164749
1.0-1.5	0.137407	0.590401	0.272192
1.5-2.0	0.0	0.476326	0.52367
2.0-2.5	0.0	0.408194	0.591806
2.5-3.0	0.0	0.340062	0.659938
3.0-3.5	0.0	0.271932	0.728068
3.5-4.0	0.0	0.394378	0.605622
4.0-4.5	0.0	0.516825	0.483175
4.5-5.0	0.0	0.639270	0.360730
5.0-5.5	0.0	1.0	0.0
5.5-6.0	0.0	1.0	0.0
6.0-6.5	0.0	1.0	0.0
6.5-7.0	0.0	1.0	0.0
7.0-7.5	0.0	1.0	0.0
7.5-8.0	0.0	1.0	0.0
8.0-8.5	0.0	1.0	0.0

n_c = total number of holes drilled in Phase 3.A

L = length (in miles) of redox system occurring in an A_3 = 7 sq. m.

Note: This table is derived from Table P3A.2-T employing the procedure (external to the potential supply system) described in section 8.6 of chapter 8.

Phase 3B: Following the Redox Front System General Trend

QUESTION P3B.1 PROBABILISTIC DEFINITION OF THE REDOX TREND KNOWLEDGE OBJECTIVE IN PHASE 3 PART B

The stated objective for Phase 3 Part B is to determine the "general trend" of the REDOX front system.

State the ± percent error figure in the length estimate of the REDOX front system existing in A_3 that you try to achieve when conducting Phase 3, Part B. Also, state the confidence that you insist to have. (For example: Your objective may be to be 90% confident that the length estimate of the Redox front system is ± 30%.)

Answer:

Error: ± 20 %

Confidence: 80 %

QUESTION P3B.2

INSTRUCTIONS AND COMMENT:

Each table provided is for various levels of pre-existing holes (n_c).

Each box in a table represents a particular case of L miles of Redox front existing in the A_3 (7 sq. miles) area you are exploring and n_R holes drilled to evaluate this length.

Assume:

You have conducted Phase 3, Part A and drilled n_c holes.

You have achieved your Phase 3, Part A objective: You have encountered the oxidation-reduction interface, i.e., you came up with at least one set of contrasting holes (one in oxidized and another in reduced ground).

There are L miles of Redox front contained in the A_3 area.

The OBJECTIVE of the n_R holes drilled in Phase 3 Part B, is to evaluate the length of the Redox front system. Assume that n_R holes are ALL you have and you are going to do the BEST you can to evaluate the ENTIRE length.

Question:

State in each box of the table the low, most likely and high REDOX front system length estimate (in miles) with 80% confidence when in reality (state of nature) the Redox front system is L miles long and you drilled a total of n_R holes in Phase 3, Part B, to evaluate its entire length.

TABLES P3B.2 ACCURACY IN THE LENGTH ESTIMATE OF THE REDOX FRONT SYSTEM

n_c = 25 holes

"state of nature" L = length of the Redox front system existing in A_3 = 7 sq. mi.	n_R = 10 low	10 M.L.	10 High	20 low	20 M.L.	20 High	40 low	40 M.L.	40 High
0.5 mi.	.5	.7	1.0	.3	.6	.9	.4	.5	.7
1 mi.	.4	.6	1.5	.6	.8	1.2	.9	1.1	1.15
5 mi.	2.0	3.5	6.0	3.0	4.0	6.0	4.0	5.0	5.5

n_c = 35 holes

"state of nature" L = length of the Redox front system existing in A_3 = 7 sq. mi.	n_R = 10 low	10 M.L.	10 High	20 low	20 M.L.	20 High	40 low	40 M.L.	40 High
0.5 mi	0.4	0.6	0.9	.45	.55	.75	.5	.51	.6
1 mi	.5	1.3	1.5	0.9	1.1	1.2	.95	1.0	1.1
5 mi	3.0	4.0	6.0	4.0	5.0	6.0	4.5	5.0	5.5

n_c = 45 holes

"state of nature" L = length of the Redox front system existing in A_3 = 7 sq. mi.	n_R = 10 low	10 M.L.	10 High	20 low	20 M.L.	20 High	40 low	40 M.L.	40 High
0.5 mi	.5	.6	.8	.4	.5	.6	.5	.5	.5
1 mi	.8	1.1	1.2	.9	1.0	1.1	1.0	1.0	1.0
5 mi	4.0	4.5	6.0	4.5	5.0	5.5	5.0	5.0	5.0

where: n_R = total number of holes drilled in Phase 3, Part B, to evaluate the length of the Redox front system. Assume that n_R holes are ALL you have and you are going to do the best you can with that many holes to evaluate the entire length.

L = length of the Redox front system existing in A_3 = 7 sq. miles.

n_c = total number of pre-existing holes drilled in Part A of Phase 3.

Phase 4A: Deposit Search

QUESTION P4A.1

INSTRUCTIONS AND COMMENT:

This question is designed to obtain the average number of holes per fence after Phase 4A drilling has been conducted.

For each box in the tables answer the following question: suppose that you have drilled f_D fences searching for a deposit whose actual 0.01-confining prism width is W(.01) (if zero there is no deposit) along L miles of Redox front in the A_3 region you are exploring. The minimum average width of the mineralization attractive to you is $\overline{\Omega}_m$. State the low, most likely and high average number of holes per fence.

TABLES P4A.1 AVERAGE NUMBER OF HOLES PER FENCE IN PHASE 4, PART A

$f_D = 7$

L = 3 miles

minimum attractive average width ($\bar{n}_m$) of mineralization you are searching for	width W(.01) of the .01-confining prism of the deposit occurring somewhere along L: 0			200'			350'			700'		
	L	M.L.	H	L	M.L.	H	L	M.L.	H	L	M.L.	H
150'	5	6	7	5	6	7	6	7	8	7	8	9
300'	4	5	6	4	5	6	5	6	7	5	6	7
500'	4	5	6	4	5	6	4	5	6	4	5	6
700'	3	4	5	3	4	5	3	4	5	3	4	5

$f_D = 15$

L = 3 miles

minimum attractive average width ($\bar{n}_m$) of mineralization you are searching for	width W(.01) of the .01-confining prism of the deposit occurring somewhere along L: 0			200'			350'			700'		
	L	M.L.	H	L	M.L.	H	L	M.L.	H	L	M.L.	H
150'	4	5	6	4	5	6	5	6	6.5	5	6	7
300'	3.5	4.5	5	3.5	4.5	5	4.0	4.5	5.0	4.0	4.5	5.0
500'	3.5	4.0	4.5	3.5	4.0	4.5	3.5	4.0	4.5	3.5	4.0	4.5
700'	3	4	5	3	4	5	3	4	5	2.5	3.0	4.0

$f_D = 25$

L = 3 miles

minimum attractive average width ($\bar{n}_m$) of mineralization you are searching for	width W(.01) of the .01-confining prism of the deposit occurring somewhere along L: 0			200'			350'			700'		
	L	M.L.	H	L	M.L.	H	L	M.L.	H	L	M.L.	H
150'	3	4	5	3	4	5	3	4	5	3	4	5
300'	2	3	4	2	3	4	2	3	4	3	4	5
500'	2	3	4	2	3	4	2	3	4	3	4	5
700'	2	3	4	2	3	4	2	3	4	2	3	4

where: f_D = total number of fences drilled in Phase 4, Part A.
L = length of the REDOX front system that exists in area A_3 = 7 sq. mi. you are exploring.

QUESTION P4A.2 REFERENCE AVERAGE WIDTH TO BE USED WHEN FILLING IN THE COMING QUESTIONS

TABLE P4A.2

q	$\bar{\Omega}(q)$
.01	400'
.05	350'
.15	175'

where q = cutoff grade

$\bar{\Omega}(q)$ = average width (composite) of the mineralized material of grades $\geq$ q that you want to assume when filling in the coming tables for Phase 4, Part A.

INSTRUCTIONS AND COMMENTS:

This question is directly related to the next question corresponding to Phase 4, Part A.

When filling in tables in the coming questions you need to assume some reference average width $\bar{\Omega}(q)$ of the mineralized material of grades $\geq$ q in order to answer the question. We need to know what reference average widths [$\bar{\Omega}(q)$] you prefer to assume.

Question: State for each grade in the table above the reference average width [$\bar{\Omega}(q)$] of the mineralized material of grades $\geq$ q that you want to assume (can be the same for all grades if you wish).

QUESTION P4A.3

INSTRUCTIONS AND COMMENTS:

The objective of Phase 4, Part A, is to detect the mineralized target occurring somewhere along L, the Redox front system portion that is contained in A_3; the general trend of the Redox front system has been already determined in Phase 3. This question refers to the probability of success in Phase 4, Part A, i.e., the probability of detecting and recognizing a mineralized target whose dimensions and grade will be specified.

QUESTION:

Assume: You have conducted Parts A and B of Phase 3 successfully. You have determined the general trend of the Redox front system contained in the A_3 area.

Question: Suppose you are searching for a Wyoming roll front type deposit. You are interested in grades above q_m; the minimum attractive average width you are searching for is $\bar{\Omega}_m$, hence you design the spacing within a fence to search for a target with an average width of $\bar{\Omega}_m$ units.

Each row in the table represents a different state of nature. The target that actually occurs has the following characteristics: the average width of the mineralized material with grades $\geq q_m$ is $\bar{\Omega}_a(q_m)$ units wide; the length of a hypothetical confining prism circumscribing mineralized target material with grades $\geq q_m$ is $\ell(q_m)$ units. This target just described is the only one occurring along the Redox front system portion contained in the A_3 area. The length of the portion of the Redox front contained in the A_3 area is L units.

Each box in the tables represent a different case. State in each box the probability (%) of detecting and recognizing the specified target if you drill a total of f_D fences. Assume that f_D fences are ALL you have and you are going to do the BEST you can with that many fences. Also assume that those fences are "adequate", that is, they are appropriately designed to intercept a target of an average width of at least $\bar{\Omega}_m$ units. Answer this for each box in the tables.

QUESTION P4A.3

INSTRUCTIONS AND COMMENTS: (Continued)

Note: As an aid for the respondent, the geometric percent probability of detection* is given in the lower right hand side corner of each box.

The assumptions in this computed probability are the following:

(a) Target is a solid (no voids, no discontinuities or gaps) regular rectangular body of dimensions $\ell(q_m) \times \bar{\Omega}_a(q_m)$.

(b) Fences have adequate within-fence spacing to intercept the target.

(c) Spacing between the fences* is constant. Fences drilled orthogonal to the straight-line Redox front system. Initial fence position is chosen at random. Of course, these assumptions do not imply by any means that the respondent should assume equally spaced fences.

The respondent is asked to modify these simple geometric probabilities by incorporating the following factors:

Positive factors:

- Geological information and interpretation help to better determine the fences location as drilling unfolds.

Negative factors:

- Errors in recognition.
- Target is neither solid nor continuous; it has gaps and voids. It is not a regular rectangular shaped body either.

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*Probability is computed by the following expression:

$$P(\text{success}) = \frac{\ell(q_m)}{s} \quad \text{(set to 1.0 if result is greater than 1.0)}$$

where $\ell(q_m)$ = length of the target

s = spacing between fences $= \frac{L}{f_D}$

L = redox length (search length)

f_D = number of fences.

TABLES P4A.5 CONDITIONAL PROBABILITY OF SUCCESS IN PHASE 4, PART A

$\ell(q_m)$ (state of nature)	f_D = 1	3	5	8	10	15	30	40	55
.056 mi (300')	0 (2)	2 (6)	4.5 (9.3)	10 (15)	14 (19)	22 (28)	48 (56)	60 (75)	98 (100)
.15 mi (792')	3 (5)	8 (15)	13 (25)	31 (40)	39 (50)	68 (75)	90 (100)	100 (100)	(100)
.25 mi (1320')	6 (8)	13 (25)	26 (42)	51 (67)	67 (83)	90 (100)	100 (100)	100 (100)	(100)
.5 mi	13 (17)	28 (50)	56 (83)	82 (100)	95 (100)	100 (100)	100 (100)	100 (100)	(100)
1 mi	26 (33)	56 (100)	92 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)
2 mi	50 (66)	70 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)	(100)
2.5 mi	70 (83)	92 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)	(100)
3 mi	91 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)	(100)	(100)

Additional Conditions for this table:

q_m = .01

$\bar{n}_m$ = 400'

$\bar{n}_a(q_m)$ = 400' } state of nature

L = 3 miles } state of nature

$\ell(q_m)$ (state of nature)	f_D = 1	3	5	8	10	15	30	40	60
.056 mi (300')	0 (2)	0 (6)	3 (9.3)	7 (15)	10 (19)	20 (28)	35 (56)	50 (75)	90 (100)
.15 mi (792')	2 (5)	7.5 (15)	11 (25)	30 (40)	37 (50)	65 (75)	90 (100)	100 (100)	(100)
.25 mi (1320')	5 (8)	8 (25)	22 (42)	48 (67)	62 (83)	80 (100)	100 (100)	(100)	(100)
.5 mi	11 (17)	21 (50)	50 (83)	75 (100)	90 (100)	99 (100)	(100)	(100)	(100)
1 mi	20 (33)	49 (100)	83 (100)	91 (100)	99 (100)	100 (100)	(100)	(100)	(100)
2 mi	40 (66)	68 (100)	90 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)
2.5 mi	60 (83)	85 (100)	92 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)
3 mi	88 (100)	92 (100)	100 (100)	(100)	(100)	(100)	(100)	(100)	(100)

Additional Conditions for this table:

q_m = .05

$\bar{n}_m$ = 350'

$\bar{n}_a(q_m)$ = 350' } state of nature

L = 3 miles } state of nature

$\ell(q_m)$ (state of nature)	f_D = 1	3	5	8	10	15	30	40	60
.056 mi (300')	0 (2)	0 (6)	2 (9.3)	6 (15)	8 (19)	11 (28)	24 (.56)	30 (75)	75 (100)
.15 mi (792')	0 (5)	6 (15)	9 (25)	26 (40)	29 (50)	40 (75)	80 (100)	99 (100)	(100)
.25 mi (1320')	1 (8)	5 (25)	18 (42)	40 (67)	55 (83)	69 (100)	99 (100)	(100)	(100)
.5 mi	8 (17)	17 (50)	44 (83)	68 (100)	80 (100)	90 (100)	100 (100)	(100)	(100)
1 mi	14 (33)	39 (100)	76 (100)	88 (100)	95 (100)	100 (100)	(100)	(100)	(100)
2 mi	30 (66)	60 (100)	88 (100)	95 (100)	100 (100)	(100)	(100)	(100)	(100)
2.5 mi	56 (83)	80 (100)	89 (100)	97 (100)	100 (100)	(100)	(100)	(100)	(100)
3 mi	84 (100)	89 (100)	99 (100)	100 (100)	(100)	(100)	(100)	(100)	(100)

Additional Conditions for this table:

q_m = .15

$\bar{n}_m$ = 175'

$\bar{n}_a(q_m)$ = 175' } state of nature

L = 3 miles } state of nature

where q_m = minimum attractive grade you are <u>searching for</u>.

$\bar{n}_m$ = minimum attractive width of target you are <u>searching for</u>.

f_D = total number of fences drilled in Phase 4, Part A. Assume that f_D fences are ALL you have and you are going to do the BEST you can with that many fences. Also assume that those fences are adequately designed to intercept a target of an average width of $\bar{n}_m$ units.

$\bar{n}_a(q_m)$ = "actual" average width = average width of the mineralized target material $\geq q_m$. } state of nature

$\ell(q_m)$ = length of a hypothetical confining prism circumscribing all mineralized target material with grades $\geq q_m$. } state of nature

L = length of the REDOX front system contained in the A_3 area. Note: Assume there is <u>only</u> ONE mineralized target occurring along L. L is also called the search length. } state of nature

QUESTION P4A.4 ADJUSTMENTS TO TABLE P4A.3

INSTRUCTIONS AND COMMENTS:

When filling in the tables in question P4A.3, you were asked to assume that the average width $[\overline{\Omega}_a(q_m)]$ of the target at grades above q_m that actually exists is equal to the minimum average width that you are searching for $(\overline{\Omega}_m)$. This question is to elicit how the probability of success in the previous question changes when these two average widths are not equal.

Three sets of tables -- one for each reference grade -- will be asked. Every table in a set corresponds to a different length of the confining prism circumscribing mineralized target material with grades $\geq q_m$ $[\ell(q_m)]$. Every column in a table corresponds to a different state of nature.

Question: (row by row)

The center column in each table is already filled in. It reproduces your answers for the probability of success that you gave in the previous question (P4A.3) for $\overline{\Omega}_a(q_m) = \overline{\Omega}_m$ and all the other circumstances described in each table [i.e., f_D, q_m, $\ell(q_m)$, L].

For every row in the table, revise the probability of success stated in the center box to account for the fact that the average width of the mineralized material above q_m, $\overline{\Omega}_a(q_m)$ differs from the minimum attractive average width you are searching for $(\overline{\Omega}_m)$. [i.e., revise the probability to account for the fact that the fences you drill in Phase 4, Part A are designed to look for something else than what is actually there in terms of average width.]

TABLES P4A.4 - ADJUSTMENTS TO TABLE P4A.3

		$\bar{n}_a(q_m)$ [state of nature]		
		200'	400'	600'
f_D	1	2%	3%	4%
	10	30%	39%	44%
	30	84%	90%	99%

Additional Conditions:

q_m = .01

$\bar{n}_m$ = 400'

$\ell(q_m)$ = 0.15 miles } state of nature

L = 3 miles } state of nature

		$\bar{n}_a(q_m)$ [state of nature]		
		200'	400'	600'
f_D	1	9%	13%	14%
	5	47%	56%	60%
	10	80%	95%	97%

Additional Conditions:

q_m = .01

$\bar{n}_m$ = 400'

$\ell(q_m)$ = 0.5 miles } state of nature

L = 3 miles } state of nature

		$\bar{n}_a(q_m)$ [state of nature]		
		200'	400'	600'
f_D	1	40%	50%	60%
	3	60%	70%	75%
	5	92%	99%	100%

Additional Conditions:

q_m = .01

$\bar{n}_m$ = 400'

$\ell(q_m)$ = 2 miles } state of nature

L = 3 miles } state of nature

where $\bar{n}_m$ = minimum attractive average width that you are SEARCHING FOR.

f_D = fences drilled in Phase 4, Part A. Assume that f_D fences are ALL you have and you are going to do the BEST you can with that many fences. Also assume that those fences are adequately designed to intercept a target of an average width of $\bar{n}_m$ units.

q_m = minimum attractive grade that you are searching for.

$\bar{n}_a(q_m)$ = "actual" average width = average width of the mineralized target material $\geq q_m$. [state of nature]

$\ell(q_m)$ = length of a hypothetical confining prism circumscribing all mineralized target material with grades $\geq q_m$. [state of nature]

L = length of the REDOX front system portion contained in the A_3 area. [state of nature] Note: Assume there is only ONE mineralized target occurring along L. L is also called the search length.

} state of nature

TABLES P4A.4

		$\bar{n}_a(q_m)$ [state of nature] 200'	350'	500'
f_D	1	1%	2%	3%
	10	29%	37%	42%
	30	82%	90%	96%

Additional Conditions:

q_m = .05

$\bar{n}_m$ = 350'

$\ell(q_m)$ = 0.15 miles } state of nature

L = 3 miles }

		$\bar{n}_a(q_m)$ [state of nature] 200'	350'	500'
f_D	3	14%	21%	24%
	8	55%	75%	80%
	15	70%	99%	99%

Additional Conditions:

q_m = .05

$\bar{n}_m$ = 350'

$\ell(q_m)$ = 0.5 miles } state of nature

L = 3 miles }

		$\bar{n}_a(q_m)$ [state of nature] 200'	350'	500'
f_D	1	30%	40%	50%
	3	62%	68%	79%
	8	91%	99%	100%

Additional Conditions:

q_m = .05

$\bar{n}_m$ = 350'

$\ell(q_m)$ = 2 miles } state of nature

L = 3 miles }

where $\bar{n}_m$ = minimum attractive average width that you are SEARCHING FOR.

f_D = fences drilled in Phase 4, Part A. Assume that f_D fences are ALL you have and you are going to do the BEST you can with that many fences. Also assume that those fences are adequately designed to intercept a target of an average width of $\bar{n}_m$ units.

q_m = minimum attractive grade that you are searching for.

$\bar{n}_a(q_m)$ = "actual" average width = average width of the mineralized target material $\geq q_m$. [state of nature]

$\ell(q_m)$ = length of a hypothetical confining prism circumscribing all mineralized target material with grades $\geq q_m$. [state of nature]

L = length of the REDOX front system portion contained in the A_3 area. [state of nature] Note: Assume there is only ONE mineralized target occurring along L. L is also called the search length.

} state of nature

TABLES P4A.4

f_D	$\bar{n}_a(q_m)$ [state of nature] 125'	175'	300'
3	2%	6%	8%
15	30%	40%	46%
40	89%	99%	100%

Additional Conditions:

q_m = .15
$\bar{n}_m$ = 175'
$\ell(q_m)$ = 0.15 miles } state of nature
L = 3 miles

f_D	$\bar{n}_a(q_m)$ [state of nature] 125'	175'	300'
1	4%	8%	10%
8	58%	68%	74%
15	81%	90%	96%

Additional Conditions:

q_m = .15
$\bar{n}_m$ = 175'
$\ell(q_m)$ = 0.5 miles } state of nature
L = 3 miles

f_D	$\bar{n}_a(q_m)$ [state of nature] 125'	175'	300'
1	20%	30%	35%
3	50%	60%	65%
8	85%	95%	99%

Additional Conditions:

q_m = .15
$\bar{n}_m$ = 175'
$\ell(q_m)$ = 2 miles } state of nature
L = 3 miles

where $\bar{n}_m$ = minimum attractive average width that you are SEARCHING FOR.

f_D = fences drilled in Phase 4, Part A. Assume that f_D fences are ALL you have and you are going to do the BEST you can with that many fences. Also assume that those fences are adequately designed to intercept a target of an average width of $\bar{n}_m$ units.

q_m = minimum attractive grade that you are searching for.

$\bar{n}_a(q_m)$ = "actual" average width = average width of the mineralized target material $\geq q_m$. [state of nature]

$\ell(q_m)$ = length of a hypothetical confining prism circumscribing all mineralized target material with grades $\geq q_m$. [state of nature]

L = length of the REDOX front system portion contained in the A_3 area. [state of nature] Note: Assume there is only ONE mineralized target occurring along L. L is also called the search length.

} state of nature

QUESTION P4A.5 ESTIMATION OF THE LENGTH OF THE TARGET AFTER DRILLING 4.A PROGRAM

INSTRUCTIONS AND COMMENTS:

In Phase 4, Part A, the objective is to search for and discover (i.e., detect and recognize) the mineralized target. This question is to elicit the probability distribution for $\ell(q_m)$ (the length of the confining prism comprising the mineral deposit at grades above the minimum attractive grade you are searching for) after you have taken an action (i.e., drilled f_D fences) under a given set of circumstances (states of nature) and observed a given outcome (success or non success in 4A).

Two sets of tables will be given for the cases of non success and success in your Phase 4, Part A, objective. Each set consists of two tables: a ranking table and a probability distribution table.

<u>Question</u>:

RANKING TABLE: Each row in the table corresponds to a different set of circumstances [L, Th, D(q_m)], actions taken [q_m, $\bar{w}_m$ (q_m), f_D] and an outcome (success/non success in Phase 4, Part A).

For <u>example</u>, the fourth row in the table corresponds to the following condition: 25 fences have been drilled searching for a target of grades $\geq$ .05 and an average width of <u>at least</u> 350'. The circumstances (states of nature) prevailing are: the thickness of the aggregate geochemical cell is 100', the length of the REDOX front system portion contained in A_3 is 3 miles and in reality no deposit exists along those 3 miles [D(.05) = 0].

Rank each row in the table according to those combinations conducive to a large $\ell(q_m)$, the length of the mineralized target with grades $\geq q_m$. Assign #1 to that combination leading to the largest expected value for ℓ; assign #2 to the next and so on. The last rank number will correspond to the most unfavorable combination in terms of length of the target.

When the ranking is done, the interviewer writes the combination in the probability distribution table preserving the ranking order.

PROBABILITY DISTRIBUTION TABLE: For each row in the table, imagine you are dealing with that particular combination of conditions and state that numerical value of the length of the confining prism enclosing the mineral deposit at a grade above q_m, $\ell(q_m)$, for which there is a probability of 10% of being less than that.* [i.e., there is 90% probability of exceeding that stated $\ell(q_m)$]. Do this for all probability levels given in the column headings.

*That is, we are asking for the 10% fractile.

TABLE P4A.5(1) NON SUCCESS TABLE - RANKING

CONDITIONS				
state of nature				
$D(q_m)$	q_m	f_D		Rank
No	.01	25		16
No	.01	15		13
No	.01	5		7
No	.05	25		17
No	.05	15		14
No	.05	5		8
No	.15	25		18
No	.15	15		15
No	.15	5		9
Yes	.01	25		10
Yes	.01	15		4
Yes	.01	5		1
Yes	.05	25		11
Yes	.05	15		5
Yes	.05	5		2
Yes	.15	25		12
Yes	.15	15		6
Yes	.15	5		3

ADDITIONAL CONDITIONS FOR THE ENTIRE TABLE:

S = 0 i.e., you have not detected the mineralized target

for q_m = .01 $\bar{n}_m$ = 400'

for q_m = .05 $\bar{n}_m$ = 350'

for q_m = .15 $\bar{n}_m$ = 175'

Th = 100' state of nature

L = 3 miles

---

where: q_m = minimum attractive grade you are exploring for.

$\bar{n}_m$ = minimum attractive average width you are asked to assume when the minimum attractive grade is q_m.

S = success in 4.A. Detection and recognition of the target.

f_D = total number of survey fences drilled in Phase 4, Part A searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{n}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the [illegible] front system.

Th = aggregate thickness of the composite geochemical cell(s).

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although by definition in this table you have not found the deposit, this $D(q_m)$ condition is stated to make you think, on average, on the information you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present.

$\ell(q_m)$ = length of the confining prism circumscribing the mineralized target at grades $\geq q_m$.

TABLE P4A.5 (2)

NON SUCCESS TABLE - PROBABILITY DISTRIBUTION

CONDITIONS					PROBABILITY DISTRIBUTION FOR $\ell(q)$					
state of nature										
$D(q_m)$	q_m	f_D		Rank	10%	25%	50%	75%	90%	
Y	.01	5		1	1000'	2400'	4200'	5000'	6600'	
Y	.05	5		2	600'	2000'	3600'	4800'	6000'	
Y	.15	5		3	500'	1000'	2000'	3000'	4100'	
Y	.01	15		4	500'	1000'	1600'	2100'	2900'	
Y	.05	15		5	500'	1000'	1400'	2000'	2600'	
Y	.15	15		6	500'	900'	1200'	1700'	2100'	
N	.01	5		7	500'	900'	1200'	1700'	2000'	
N	.05	5		8	500'	900'	1200'	1600'	1900'	
N	.15	5		9	500'	800'	1100'	1500'	1600'	
Y	.01	25		10	500'	800'	1100'	1400'	1500'	
Y	.05	25		11	500'	800'	1100'	1300'	1400'	
Y	.15	25		12	400'	700'	1000'	1100'	1200'	
N	.01	15		13	400'	700'	800'	900'	1100'	
N	.05	15		14	400'	600'	700'	800'	1000'	
N	.15	15		15	400'	600'	700'	725'	900'	
N	.01	25		16	300'	400'	500'	500'	700'	
N	.05	25		17	300'	400'	400'	400'	600'	
N	.15	25		18	100'	200'	300'	400'	500'	

ADDITIONAL CONDITIONS FOR THE ENTIRE TABLE:

S = 0 i.e., you have not detected the mineralized target

for q_m = .01 $\bar{n}_m$ = 400'

for q_m = .05 $\bar{n}_m$ = 350'

for q_m = .15 $\bar{n}_m$ = 175'

Th = 100'
L = 3 miles } state of nature

---

where: q_m = minimum attractive grade you are exploring for.

$\bar{n}_m$ = minimum attractive average width you are asked to assume when the minimum attractive grade is q_m.

S = success in 4.A. Detection and recognition of the target.

f_D = total number of survey fences drilled in Phase 4, Part A searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{n}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the REDOX front system.

Th = aggregate thickness of the composite geochemical cell(s).

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although by definition in this table you have not found the deposit, this $D(q_m)$ condition is stated to make you think, on average, on the information you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present.

$\ell(q_m)$ = length of the confining prism circumscribing the mineralized target at grades $\geq q_m$.

TABLE P4A.5(3)

SUCCESS TABLE - RANKING

CONDITIONS			Rank
q_m	f_D		
.01	25+v		6
.01	15+v		4
.01	5+v		1
.05	25+v		8
.05	15+v		5
.05	5+v		2
.15	25+v		9
.15	15+v		7
.15	5+v		3

ADDITIONAL CONDITIONS FOR THE ENTIRE TABLE:

S = 1, i.e., you have detected the mineralized target.

for q_m = .01 $\bar{w}_m$ = 400'
for q_m = .05 $\bar{w}_m$ = 350'
for q_m = .15 $\bar{w}_m$ = 175'
Th = 100'
L = 3 miles } state of nature

where: q_m = minimum attractive grade you explored for.

$\bar{w}_m$ = minimum attractive average width you are asked to assume when the minimum attractive grade is q_m.

S = success in 4.A. Detection and recognition of the target.

f_D = total number of survey fences drilled in Phase 4, Part A searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{w}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the REDOX front system.

Th = aggregate thickness of the composite geochemical cell(s).

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although by definition in this table you have not found the deposit, this $D(q_m)$ condition is stated to make you think, on average, on the information you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present.

$\ell(q_m)$ = length of the confining prism circumscribing the mineralized target at grades $\geq q_m$.

v = an unstated but sufficient number of "verification" fences or holes drilled to corroborate the existence of a discovery

TABLE P4A.5(4)

SUCCESS TABLE - PROBABILITY DISTRIBUTION

CONDITIONS			Rank	PROBABILITY DISTRIBUTION FOR $\ell(q_m)$					
q_m	f_D			10%	25%	50%	75%	90%	
.01	5+v		1	1000'	2600'	4500'	5200'	7500'	
.05	5+v		2	700'	2400'	4000'	5000'	6800'	
.15	5+v		3	600'	2000'	3000'	4000'	5000'	
.01	15+v		4	500'	1500'	2100'	2600'	3100'	
.05	15+v		5	500'	1300'	1600'	2200'	2800'	
.01	25+v		6	500'	1100'	1600'	1800'	2100'	
.15	15+v		7	500'	1000'	1200'	1700'	2200'	
.05	25+v		8	500'	1000'	1200'	1600'	1900'	
.15	25+v		9	400'	800'	1100'	[illegible]	1500'	

ADDITIONAL CONDITIONS FOR THE ENTIRE TABLE:

S = 1, i.e., you have detected the mineralized target

for q_m = .01 $\bar{W}_m$ = 400'

for q_m = .05 $\bar{W}_m$ = 350'

for q_m = .15 $\bar{W}_m$ = 175'

Th = 100' } state of nature

L = 3 miles } state of nature

---

where: q_m = minimum attractive grade you explored for.

$\bar{W}_m$ = minimum attractive average width you are asked to assume when the minimum attractive grade is q_m.

S = success in 4.A. Detection and recognition of the target.

f_D = total number of survey fences drilled in Phase 4, Part A searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{W}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the REDOX front system.

Th = aggregate thickness of the composite geochemical cell(s).

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although by definition in this table you <u>have not</u> found the deposit, this $D(q_m)$ condition is stated to make you think, on average, on the information you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present.

$\ell(q_m)$ = length of the confining prism circumscribing the mineralized target at grades $\geq q_m$.

v = an unstated but sufficient number of "verification" fences or holes drilled to corroborate the existence of a discovery

QUESTION P4A.6 ADJUSTMENTS FOR QUESTION P4A.5

INSTRUCTIONS AND COMMENTS:

The objective of this question is to see how your expectations about $\ell(q_m)$, the length of the confining prism circumscribing the mineralized target, change when the reference conditions on Th, the aggregate thickness of the composite geochemical cell(s), and $\overline{\Omega}_m$, the minimum attractive width you are searching for, are modified.

Question:

The center box in the tables provided for this question, correspond to your answers to a case (a row) in question P4A.5, as specified at the bottom of the tables. The row and column headings of the center box correspond to the reference conditions on th and $\overline{\Omega}_m$ used in answering question P4A.5.

Each box in the table corresponds to different conditions on th and $\overline{\Omega}_m$. Write in each box the modified fractiles [the values for $\ell(q_m)$ corresponding to 10%, 25%, 50%, 75% and 90%].

TABLE P4A.6 ADJUSTMENTS FOR QUESTION P4A.5

$\bar{n}_m$ = minimum attractive width you are searching for	Th = 50' 10%	Th = 50' 25%	Th = 50' 50%	Th = 50' 75%	Th = 50' 90%	Th = 100' 10%	Th = 100' 25%	Th = 100' 50%	Th = 100' 75%	Th = 100' 90%	Th = 200' 10%	Th = 200' 25%	Th = 200' 50%	Th = 200' 75%	Th = 200' 90%
	Aggregate thickness of the composite geochemical cell(s) (state of nature)														
200'	200	250	300	550	700	400	600	700	800	900	600	700	800	1000	1100
400'	200	300	400	600	750	400	700	800	900	1100	500	800	900	1100	1300
600'	500	600	700	800	900	500	800	1000	1200	1400	800	900	1100	1500	1800

Note: All entries are in feet.

CASE IN QUESTION P4A.5:

Table in P4A.5 question: ☐ Success Table ☒ Non Success Table

S = No
Rank No. = 13
$D(q_m)$ = No
q_m = .01% U_3O_8
f_D = 15 fences
L = 3 miles

where:

q_m = minimum attractive grade you are exploring for.

$\bar{n}_m$ = minimum attractive width you are searching for.

S = success in 4.A. Detection and recognition of the target.

f_D = survey fences drilled in Phase 4, Part A, searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{n}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the REDOX front system.

Th = aggregate thickness of the composite geochemical cell.

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although in the case of non success you have not found the deposit, this $D(q_m)$ condition is stated to make you think, on average, on the information that you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present [$D(q_m)$ = No].

TABLE P4A.6 ADJUSTMENTS FOR QUESTION P4A.5

$\bar{n}_m$ = minimum attractive width you are searching for	Aggregate thickness of the composite geochemical cell(s) (state of nature) Th = 50'					Th = 100'					Th = 200'				
	10%	25%	50%	75%	90%	10%	25%	50%	75%	90%	10%	25%	50%	75%	90%
175'						500	1000	1200	1700	2000					
350'	450	900	1100	1700	2100	500	1000	1400	2000	2600	600	1200	1800	2500	3000
500'						700	1100	1800	2500	3000					

<u>Note</u>: All entries are in feet.

CASE IN QUESTION P4A.5:

Table in P4A.5 question: ☐ Success Table ☒ Non Success Table

S = No

Rank No. = 5

$D(q_m)$ = Yes

q_m = .05% U_3O_8

f_D = 15 fences

L = 3 miles

where:

q_m = minimum attractive grade you are exploring for.

$\bar{n}_m$ = minimum attractive width you are searching for.

S = success in 4.A. Detection and recognition of the target.

f_D = survey fences drilled in Phase 4, Part A, searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{n}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the REDOX front system.

Th = aggregate thickness of the composite geochemical cell.

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although in the case of non success you <u>have not</u> found the deposit, this $D(q_m)$ condition is stated to make you <u>think</u>, on average, on the information that you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present [$D(q_m)$ = No].

TABLE P4A.6 ADJUSTMENTS FOR QUESTION P4A.5

		Aggregate thickness of the composite geochemical cell(s) (state of nature)														
		Th = 50'					Th = 100'					Th = 200'				
		10%	25%	50%	75%	90%	10%	25%	50%	75%	90%	10%	25%	50%	75%	90%
$\bar{n}_m$ = minimum attractive width you are searching for	125'						350	500	550	600	800					
	175'	150	250	400	600	700	400	600	700	725	900	600	700	800	1000	1100
	300'						600	700	800	900	1100					

Note: All entries are in feet.

CASE IN QUESTION P4A.5:

Table in P4A.5 question: ☐ Success Table ☒ Non Success Table

S = No

Rank No. = 15

$D(q_m)$ = No

q_m = .15% U_3O_8

f_D = 15 fences

L = 3 miles

where:

q_m = minimum attractive grade you are exploring for.

$\bar{n}_m$ = minimum attractive width you are searching for.

S = success in 4.A. Detection and recognition of the target.

f_D = survey fences drilled in Phase 4, Part A, searching for a uranium deposit of grades $\geq q_m$ and average width of at least $\bar{n}_m$. Assume the fences are adequately designed to search for that average width and also assume that you have done the BEST you can with these fences.

L = search length, i.e., length of the REDOX front system.

Th = aggregate thickness of the composite geochemical cell.

$D(q_m)$ = existence of a uranium deposit of grades $\geq q_m$ somewhere along L. Although in the case of non success you have not found the deposit, this $D(q_m)$ condition is stated to make you think, on average, on the information that you may encounter when a deposit is present [$D(q_m)$ = yes] or on that information which is missing when a deposit is not present [$D(q_m)$ = No].

QUESTION P4A.7

This question refers to the verification holes, that is, those offset holes you drill after intercepting a target with the objective of corroborating its existence.

Question: What is typically the number of "verification" holes that you need to drill when the actual width of the mineralization above q_m is $\bar{\Omega}_a(q_m)$.

TABLE P4A.7 NUMBER OF VERIFICATION HOLES

.01%	
$\bar{\Omega}_a(.01)$	# of verification holes
200'	10 holes
400'	8 holes
600'	6 holes

.05%	
$\bar{\Omega}_a(.05)$	# of verification holes
200'	10 holes
350'	8 holes
500'	6 holes

.15%	
$\bar{\Omega}_a(.15)$	# of verification holes
125'	14 holes
175'	10 holes
300'	8 holes

Notation: q_m = minimum attractive grade.

$\bar{\Omega}_a(q_m)$ = average width of the mineralization above q_m (state of nature)

Phases 4B and 5: Preliminary Evaluation of the Discovery and Continuing Evaluation

QUESTION P4B5.1

PHASE 4 PROSPECT EVALUATION

Part 4.B EVALUATION OF THE PRELIMINARY DISCOVERY

Question:

The objective of Part B of Phase 4 is to determine the geologic potential of the prospect.

State the error figure in your lbs. of U_3O_8 geologic potential estimate that you try to achieve in your Phase 4, Part B evaluation.

Error: ± 50 %

Also state the confidence level that you try to achieve in your lbs. of U_3O_8 geologic potential estimate.

Confidence Level: 80 %

QUESTION P4B5.2

PHASE 5 CONTINUING EVALUATION

Question:

The objective of Phase 5 is to sample the mineral deposit to obtain reserve estimates especially of the inferred and indicated category. Basically, the question to be solved is that of continuity of mineralization.

State the error figure in your lbs. of U_3O_8 reserve estimate that you try to achieve in your Phase 5 evaluation:

Error: ± 32 %

Also state the confidence level that you try to achieve in your lbs. of U_3O_8 reserve estimate:

Confidence Level 80 %

QUESTION P4B5.3 ERROR IN THE GLOBAL RESERVES ESTIMATES

In this question, three tables are given, each one corresponding to different states of nature regarding $w(q_m)$, the width of the confining prism circumscribing material of grades $\geq q_m$.

Each box in the tables is a separate case which corresponds to particular conditions about the minimum attractive grade, q_m, and n_E, the number of holes drilled in Phase 4, Part B and Phase 5.

Assume:

You have just discovered (preliminary discovery) one portion of the mineralized target of grades $\geq q_m$. You have also drilled some "verification" fences or holes to rule out the possibility of "lucky holes."

You are now to conduct Phase 4, Part B to evaluate the "geologic potential" of the deposit and Phase 5 to continue evaluating the deposit.

The length of the confining prism, $\ell(q)$, that circumscribes the mineralized target of grades $\geq q_m$ is 1 mile.

The width of the confining prism that circumscribes the mineralized target of grades $\geq q_m$ is $w(q_m)$.

You are given "n_E" drill holes to conduct Phase 4, Part B drilling program and Phase 5 if possible.

Question:

Each box in the tables is a separate case corresponding to particular conditions about the minimum attractive grade, q_m, and n_E, the number of holes available to conduct Phase 4, Part B and Phase 5. Additional conditions regarding the "state of nature" accompany each table.

State in each box the low (L), most likely (M) and high (H) percent error figure (± x % error with 80% confidence) in the estimates of U_3O_8 contained in the mineral deposit described below after drilling n_E holes.

TABLES P4B5.5

n_E = number of holes drilled in 4.B and Phase 5 (assume these are all the holes you have)

q_m = minimum attractive grade	48 ± % error (80% conf.)			64 ± % error (80% conf.)			92 ± % error (80% conf.)			120 ± % error (80% conf.)			150 ± % error (80% conf.)			200 ± % error (80% conf.)			± % error (80% conf.)		
	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H
.01	60	65	70	55	60	65	50	55	60	45	50	55	25	30	40	15	20	28			
.05	55	60	65	45	50	60	40	45	55	35	40	45	24	29	37	14	19	26			
.15	62	67	72	60	62	67	51	56	61	46	51	56	35	40	46	20	25	30			

ADDITIONAL CONDITIONS: Dimensions of the confining prism circumscribing the mineral deposit.

$\ell(q_m)$ = 1 mile
$w(q_m)$ = 200 feet
} state of nature

n_E = number of holes drilled in 4.B and Phase 5 (assume these are all the holes you have)

q_m = minimum attractive grade	60 ± % error (80% conf.)			78 ± % error (80% conf.)			100 ± % error (80% conf.)			130 ± % error (80% conf.)			160 ± % error (80% conf.)			210 ± % error (80% conf.)			± % error (80% conf.)		
	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H
.01	60	64	68	54	58	62	48	53	58	44	49	54	25	29	34	15	19	24			
.05	56	59	62	49	52	55	40	45	50	34	39	44	24	28	33	15	18	23			
.15	65	68	70	59	62	65	50	55	60	45	50	55	40	45	50	25	30	35			

ADDITIONAL CONDITIONS: Dimensions of the confining prism circumscribing the mineral deposit.

$\ell(q_m)$ = 1 mile
$w(q_m)$ = 350' feet
} state of nature

n_E = number of holes drilled in 4.B and Phase 5 (assume these are all the holes you have)

q_m = minimum attractive grade	72 ± % error (80% conf.)			92 ± % error (80% conf.)			112 ± % error (80% conf.)			140 ± % error (80% conf.)			180 ± % error (80% conf.)			220 ± % error (80% conf.)			± % error (80% conf.)		
	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H	L	M	H
.01	59	64	67	54	57	60	45	49	52	40	45	49	26	30	35	18	22	24			
.05	50	55	60	46	48	52	40	44	48	35	40	44	24	27	32	15	18	22			
.15	63	66	69	58	61	64	50	55	60	46	51	56	38	42	46	28	32	36			

ADDITIONAL CONDITIONS: Dimensions of the confining prism circumscribing the mineral deposit.

$\ell(q_m)$ = 1 mile
$w(q_m)$ = 500' feet
} state of nature

where q_m = minimum attractive grade you are interested in. That is, any grade below q_m is unattractive due to price conditions, depth of deposit, etc.

n_E = number of "post discovery" holes drilled to evaluate the target. These holes may correspond to Phase 4, Part B only or may be numerous enough so that they correspond to both Phase 4, Part B and Phase 5. Assume that n_E holes are ALL you have and you are going to do the BEST you can with that many holes.

$\ell(q_m)$ = length of the confining prism circumscribing a mineral deposit at grades $\geq q_m$.

$w(q_m)$ = width of the confining prism circumscribing a mineral deposit at grades $\geq q_m$.

APPENDIX G

SCHEMATIC FLOW DIAGRAM OF THE POTENTIAL SUPPLY SYSTEM

START

READ IN:

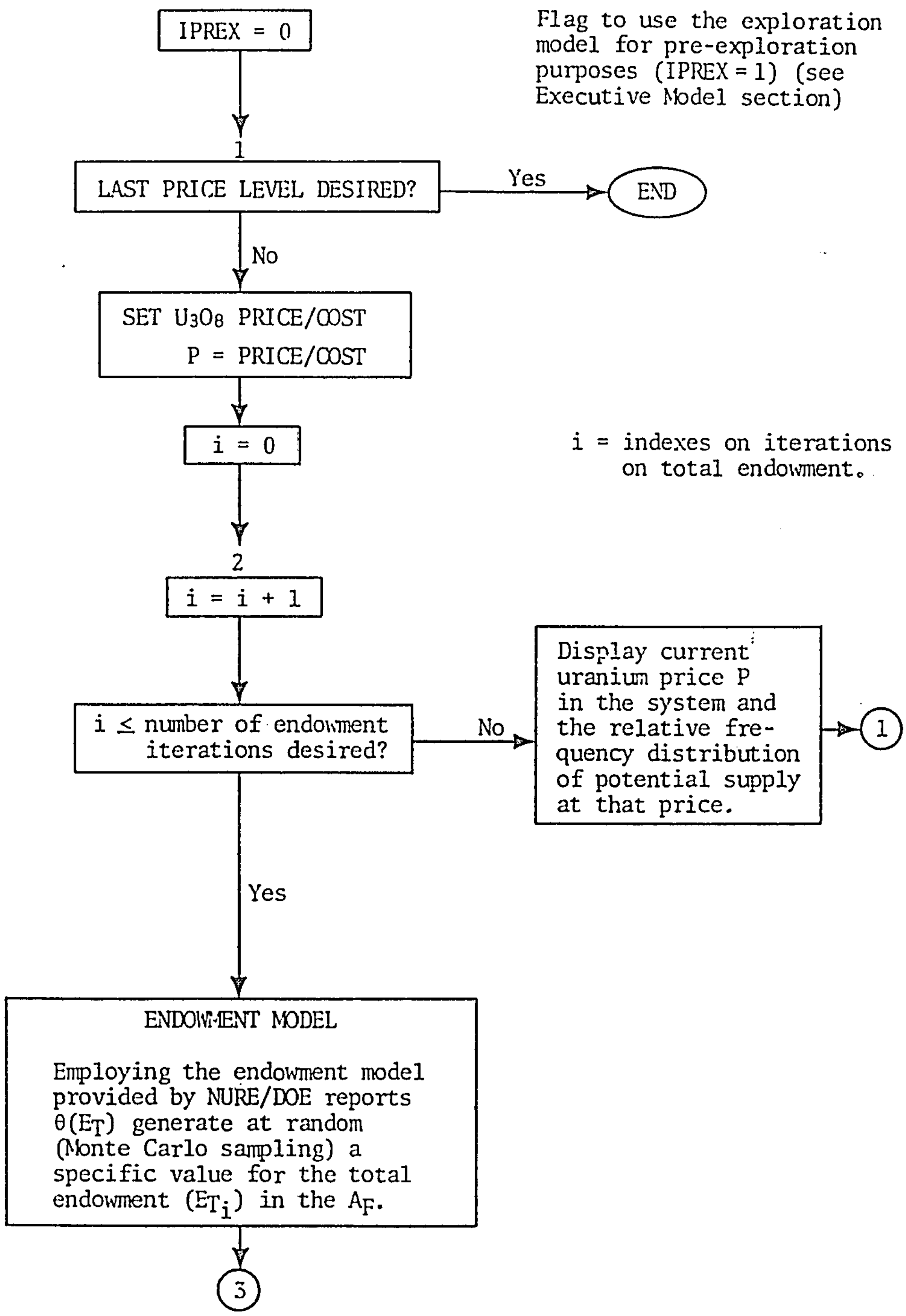
- DOE provided inputs: favorable area, endowment distribution, depths, etc. (Chapter 6).
- Economic related inputs: lead times, rate of return, cost indexes for date of evaluation, typical firm definition, state, state tax rate.
- Responses to the Subjective Opinion on Deposit Morphology questionnaire (see Appendix E).
- Responses to the Subjective Opinion on Exploration Performance questionnaire (see Appendix F).
- b(R,U), the joint distribution of redox length (R) and endowment (U) in an A_3. This distribution is obtained externally by feeding the prior expectations of the explorationist on the total endowment, $\gamma(U_T)$, to a simulation routine A3SAMP whose design is described in Appendix I.

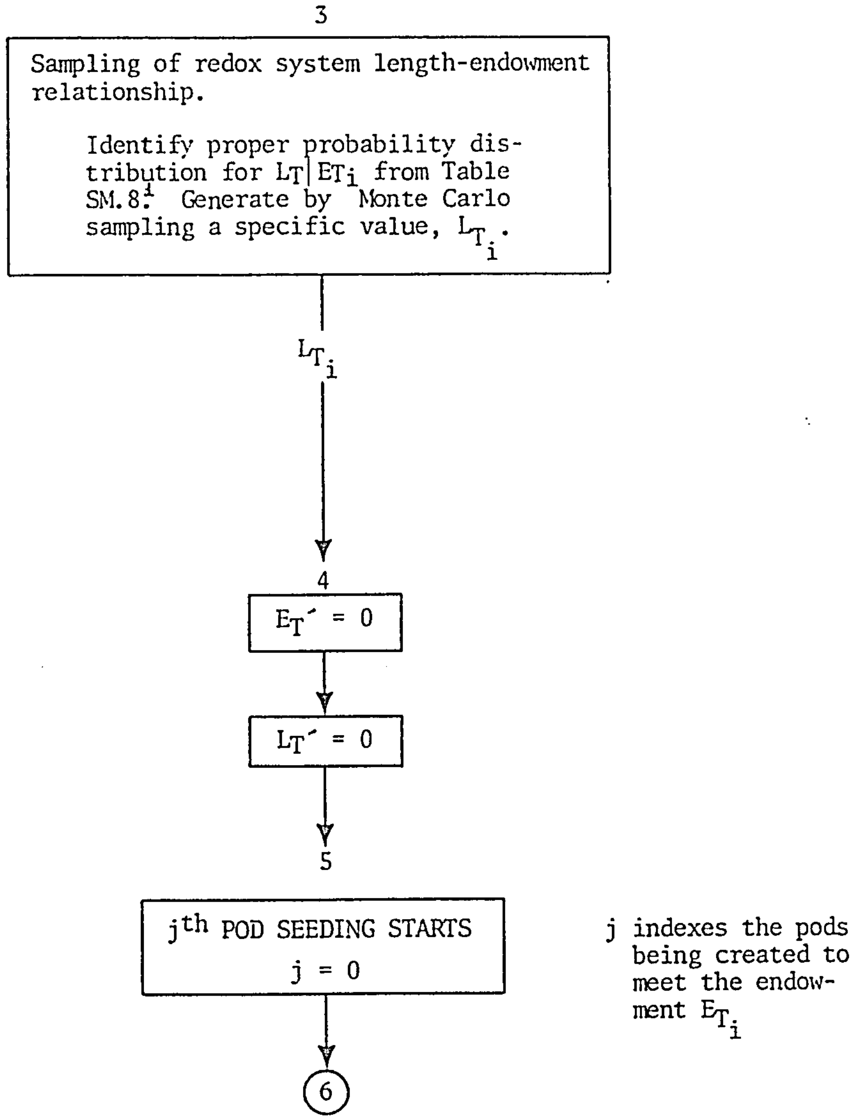
  The following distributions required in the potential supply system are derived from the joint b(R,U):

  - $\beta(R)$, the prior distribution of redox length in any A_3, before conducting any exploration activity.
  - $a(U|R)$, the conditional distribution for endowment in an A_3 given the redox length in an A_3.
- $\delta(L|L > 0)$, the censored distribution of redox length (L) per A_3. This distribution is obtained from an external routine, A3GRID, which employs a random walk design to simulate the redox length geometry. Details on design of this routine are provided in Appendix H.

1

FIGURE G.1 INITIAL PORTION OF THE POTENTIAL SUPPLY SYSTEM. READING OF INPUTS AND ENDOWMENT MODEL


IPREX = 0
Flag to use the exploration model for pre-exploration purposes (IPREX = 1) (see Executive Model section)
1
LAST PRICE LEVEL DESIRED?
Yes
END
No
SET U3O8 PRICE/COST
P = PRICE/COST
i = 0
i = indexes on iterations on total endowment.
2
i = i + 1
i ≤ number of endowment iterations desired?
No
Display current uranium price P in the system and the relative frequency distribution of potential supply at that price.
1
Yes
ENDOWMENT MODEL
Employing the endowment model provided by NURE/DOE reports θ(ET) generate at random (Monte Carlo sampling) a specific value for the total endowment (ETi) in the AF.
3



[1]All tables starting with "SM" are in Appendix E. Mention to the appendix is omitted hereafter.

FIGURE G.2 CREATION SUBMODEL: OPERATION No.1. SAMPLING OF REDOX SYSTEM LENGTH-ENDOWMENT RELATIONSHIP.

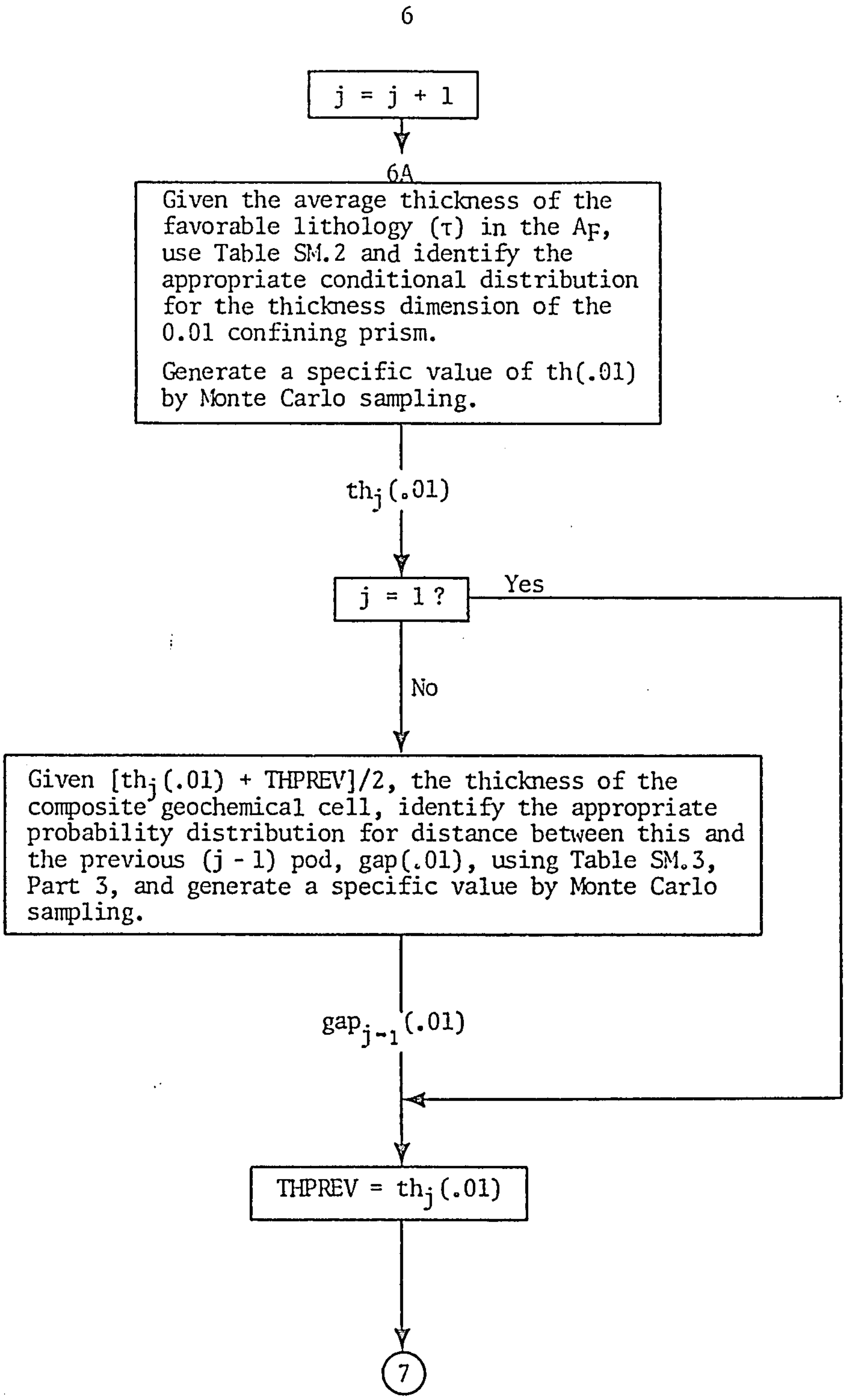


FIGURE G.3 CREATION SUBMODEL: OPERATION No. 2. SEEDING OF PODS ALONG THE REDOX SYSTEM

7

Given $th_j(.01)$, select the appropriate conditional distribution for the width of the confining prism [$w(.01)$] using Table SM.1. Generate a specific value for $w(.01)$ by Monte Carlo sampling.

$w_j(.01)$

8

Given $th_j(.01)$, identify the appropriate conditional probability distribution for the continuous length of the 0.01 confining prism of a pod [$\lambda(.01)$] from Table SM.3, Part 2. Generate a specific value by Monte Carlo sampling.

$\lambda_j(.01)$

10

Given $th_j(.01)$, identify the appropriate probability distribution for the thickness of the inner confining prism at grade $q^*(q^* > .01)$, $th(q^*)$ from Table SM.5. Generate a specific value by Monte Carlo sampling.

$th_j(q^*)$

11

11

Given $w_j(.01)$, identify the appropriate probability distribution for the width of the inner confining prism at grade q* [$w(q^*)$] using Table SM.4. Generate a specific value by Monte Carlo sampling.

$\downarrow w_j(q^*)$

12

Obtain the volume of the inner q*-confining prism of this pod as follows:

- compute the volume of the 0.01-confining prism

  $V_j(.01) = \lambda_j(.01) \cdot w_j(.01) \cdot th_j(.01)$

- using the shrinkage function derived from Table SM.7, obtain the fraction VR2 [$VR2(q^*) \equiv V(q^*)/V(.01)$] of the $V(.01)$ corresponding to the q*-confining prism volume [$V(q^*)$]

  $$V_j(q^*) = V_j(.01) \cdot VR2(q^*)$$

$\downarrow V_j(q^*)$

13

<u>Determination of the inner q*-confining prism dimensions for the j^th pod:</u>

Determine the length of the q*-confining prism by solving for it, i.e.,

$$\lambda_j(q^*) = V_j / [th_j(q^*) \cdot w_j(q^*)]$$

Ensure that the dimensions of the inner q*-confining prism [$q^* > .01$) do not exceed the 0.01-confining prism dimensions: If $\lambda_j(q^*) > \lambda_j(.01)$, make $\lambda_j(q^*) = \lambda_j(.01)$ and solve for width $w_j(q^*)$ instead. If the newly computed width $w_j(q^*) > w_j(.01)$, make $w_j(q^*) = w_j(.01)$ and solve for $th_j(q^*)$.

$\downarrow \lambda_j(q^*),\ w_j(q^*),\ th_j(q^*)$

14

14

Determination of the grade-tonnage curve of the j^{th} pod:

Given the dimensions of the 0.01-confining prism, the grade-tonnage curve for the endowment inside the j^{th} pod is determined from Table SM.7:

- From Table SM.7 the fraction (VR1) of the 0.01-confining prism made up of mineralized material above 0.01 is known, i.e.,

$$VR1 \equiv v(.01)/V(.01)$$

where $v(.01)$ = volume of mineralized material above 0.01% in the pod
$V(.01)$ = volume of the 0.01-confining prism.

- Obtain the tonnage of mineralized material above 0.01 by:

$$t_j(.01) = VR1 \cdot [\lambda_j(.01) \cdot w_j(.01) \cdot th_j(.01)]/T.F.$$

where $t(.01)$ = tonnage of mineralized material beyond 0.01% (short tons)
T.F. = tonnage factor (typically 14 ft^3/ton)

- Assign to the j^{th} pod the first three parameters μ, σ, d of the grade-tonnage curve determined from Table SM.7.

- Resource tonnage (T), the fourth parameter of the grade-tonnage curve is determined from $t(.01)$ and μ, σ, d:

$$\alpha = 1 - F\left(\frac{\ln(0.01 - d) - \mu}{\sigma}\right)$$

$$T = \frac{t(.01)}{\alpha}$$

$\mu_j \sigma_j \quad d_j \quad T_j$

15

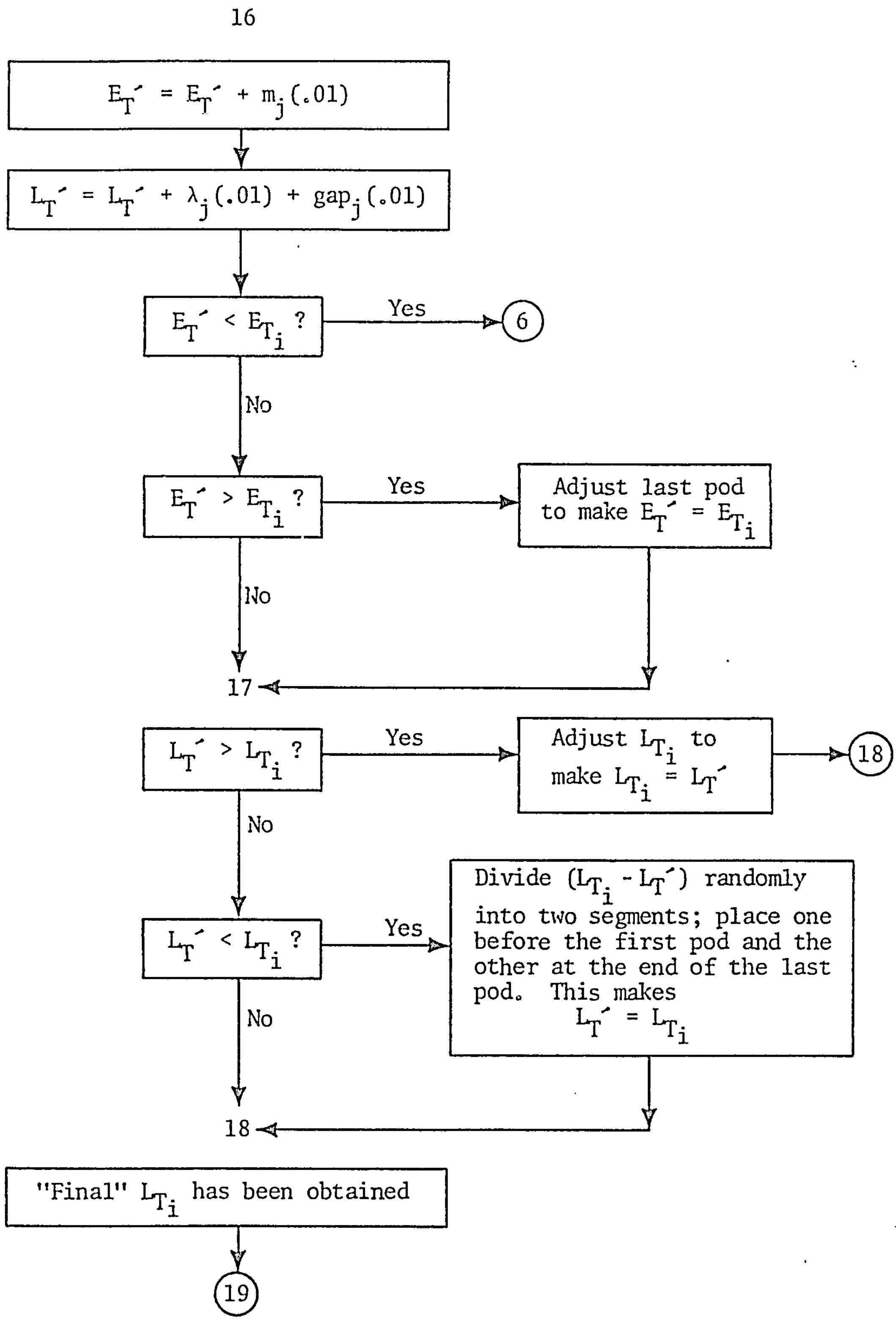
Determine U_3O_8 endowment in the j^{th} pod:

$$m_j(.01) = t_j(.01) \cdot \bar{q}_j(.01)/100.0$$

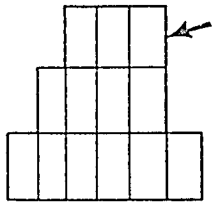
where $m_j(.01)$ = s. tons of U_3O_8 in the j^{th} pod contained in mineralized material beyond 0.01% U_3O_8.
$t_j(.01)$ = short tons of mineralized material beyond 0.01%
$\bar{q}_j(.01)$ = average grade (% U_3O_8) of mineralized material beyond 0.01%.

$m_j(.01)$

16


16
$E_T' = E_T' + m_j(.01)$
$L_T' = L_T' + \lambda_j(.01) + gap_j(.01)$
$E_T' < E_{T_i}$?
Yes
6
No
$E_T' > E_{T_i}$?
Yes
Adjust last pod to make $E_T' = E_{T_i}$
No
17
$L_T' > L_{T_i}$?
Yes
Adjust L_{T_i} to make $L_{T_i} = L_T'$
18
No
$L_T' < L_{T_i}$?
Yes
Divide $(L_{T_i} - L_T')$ randomly into two segments; place one before the first pod and the other at the end of the last pod. This makes $L_T' = L_{T_i}$
No
18
"Final" L_{T_i} has been obtained
19

19



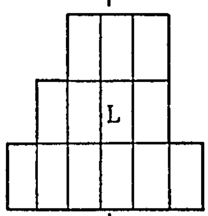
A_F = Favorable area (as "read in" from NURE inputs)

20

Start seeding of L_{T_i}.
Pick an A_3 at random.

21

Sample from the redox length per A_3 distribution, $\delta(L|L > 0)$ to assign a redox length L to the selected A_3.



22

Select another A_3 at random from the set of collateral A_3's. (Eliminate non-feasible A_3's to keep continuity in redox system length.)

23

FIGURE G.4 CREATION SUBMODEL: OPERATION 3.
SEEDING THE REDOX SYSTEM IN THE FAVORABLE AREA, A_F

23

Randomly sample from the redox length per A_3 distribution, $\delta(L|L > 0)$ to assign a redox length to the selected A_3.

24

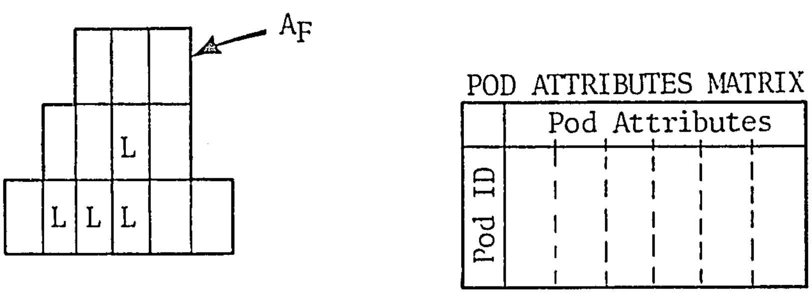
Store ID's of the selected A_3's in the sequence in which they were selected.

L_{T_i} not exhausted?

No → 22

Yes

CREATION SUBMODEL OPERATIONS HAVE BEEN COMPLETED



25

Proceed to simulate exploration.

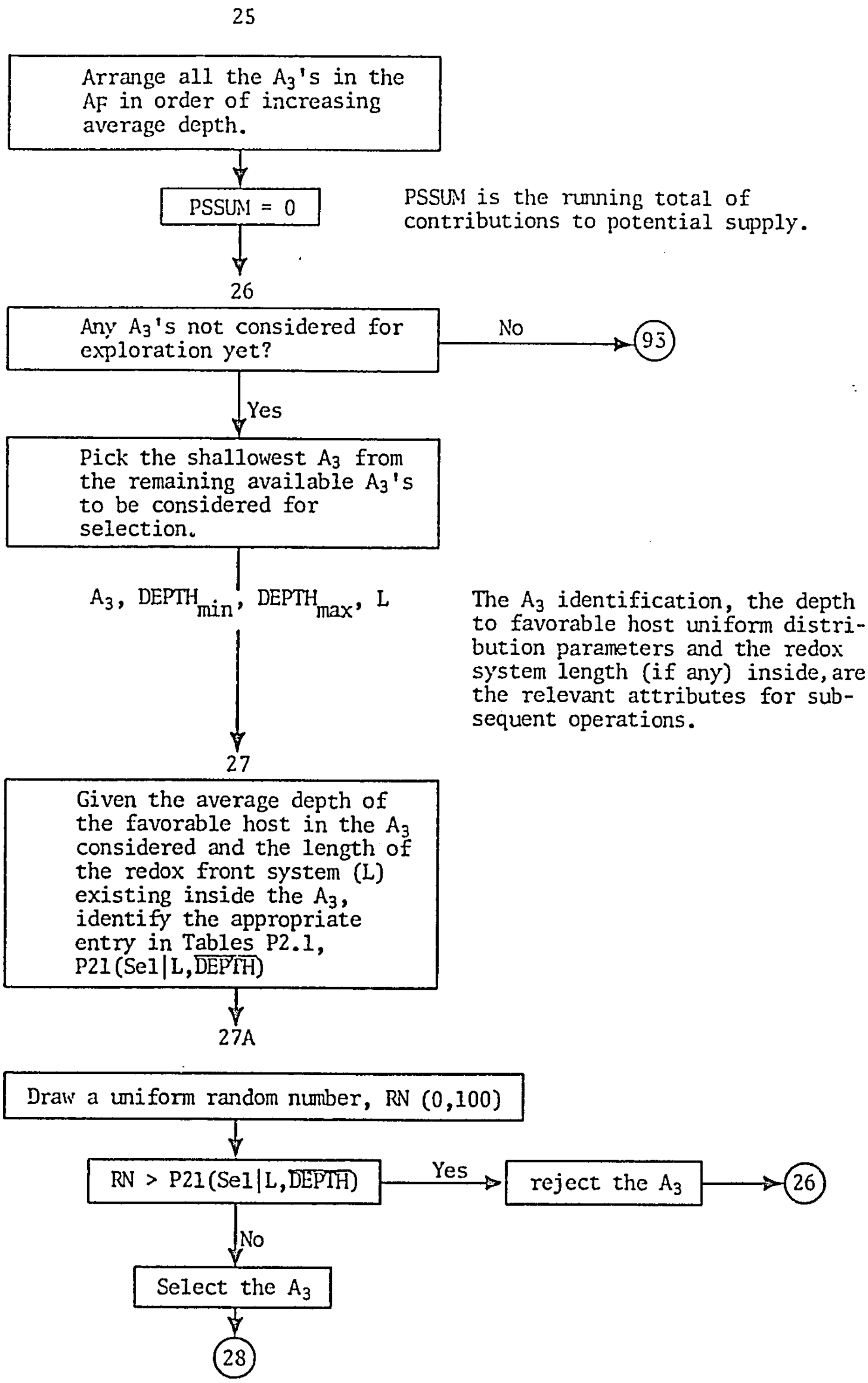


FIGURE G.5 EXPLORATION MODEL: PHASE 2

28

Revise $\beta(R)$ given that Phase 2 exploration has been conducted and the area A_3 whose average depth to the favorable host is $\overline{DEPTH}$, has been selected.

The following formula is used (non rigorous):

$$f(R|Sel,\overline{DEPTH}) = \frac{P21(Sel|R,\overline{DEPTH})\cdot\beta(R)}{\sum_R \beta(R)\cdot P21(Sel|R,\overline{DEPTH})}$$

where $\beta(R)$ = prior distribution of length of the redox system in any A_3 before conducting any exploration action (same for all A_3's).

$P21(Sel|R,\overline{DEPTH})$ = the probability of selecting an A_3 which contains R miles of redox system and which has an average depth to the favorable host of $\overline{DEPTH}$ feet. This value is obtained from the functions in table 8.2, chapter 8.

$f(R|Sel,\overline{DEPTH})$ = revised distribution of the length of the redox system in the A_3 selected.

30

Consult Executive Model: Pass current redox length expectations [i.e., $f(R|Sel,\overline{DEPTH})$], parameters of depth to host formation distribution, and current uranium price in the system.

Relevant variable returned by the model is:

n_{3A} = number of holes to drill in Phase 3.A

30A

$n_{3A} > 0$?

No → 2

If the decision is to halt exploration, there is no use in considering any of the remaining A_3's. The reason is that for any of the remaining A_3's, the average depth, $\overline{DEPTH}$, to the favorable host is higher, and higher depths imply worst endowment expectations. New endowment iteration is started.

Yes → 30B

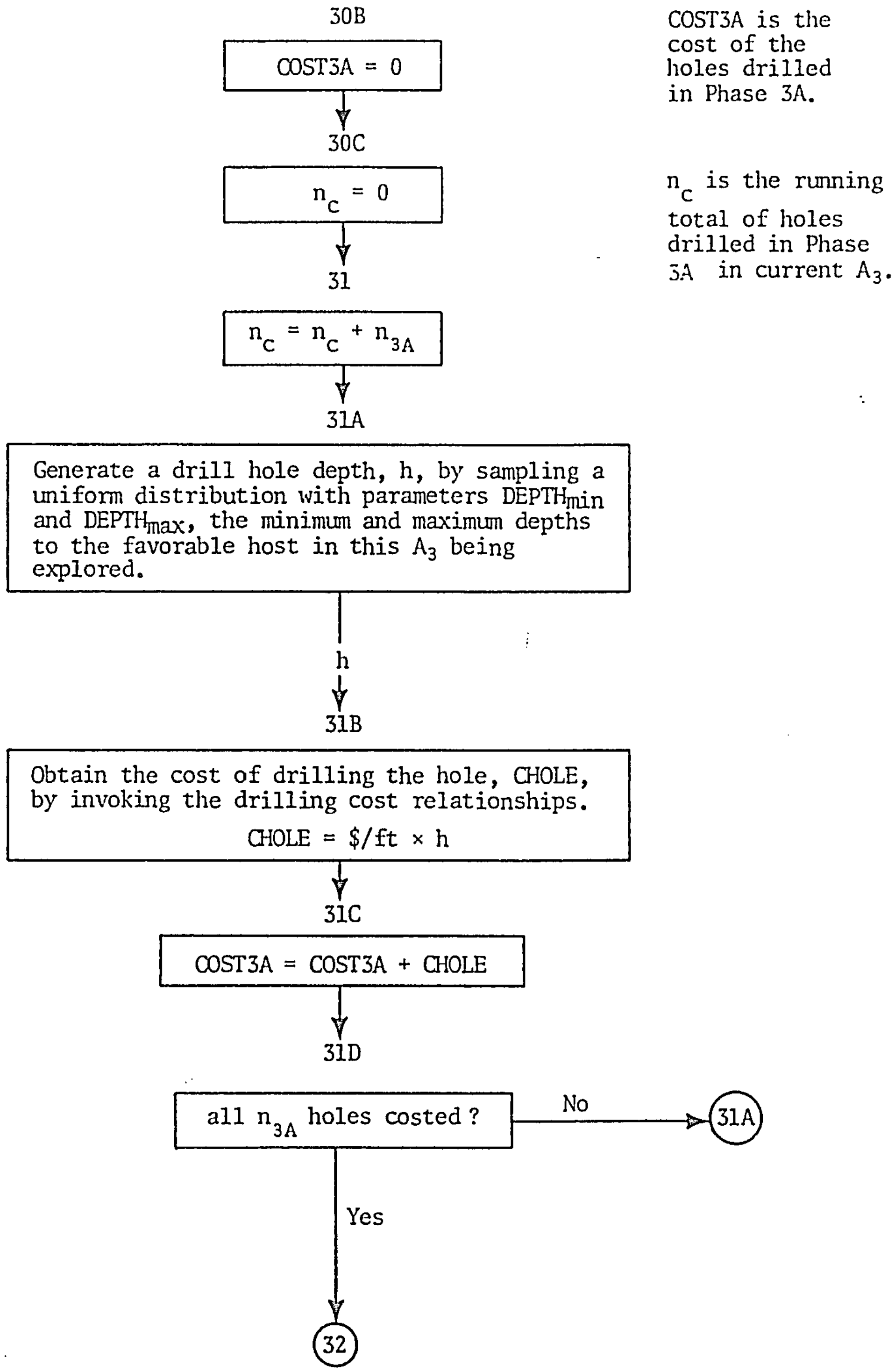
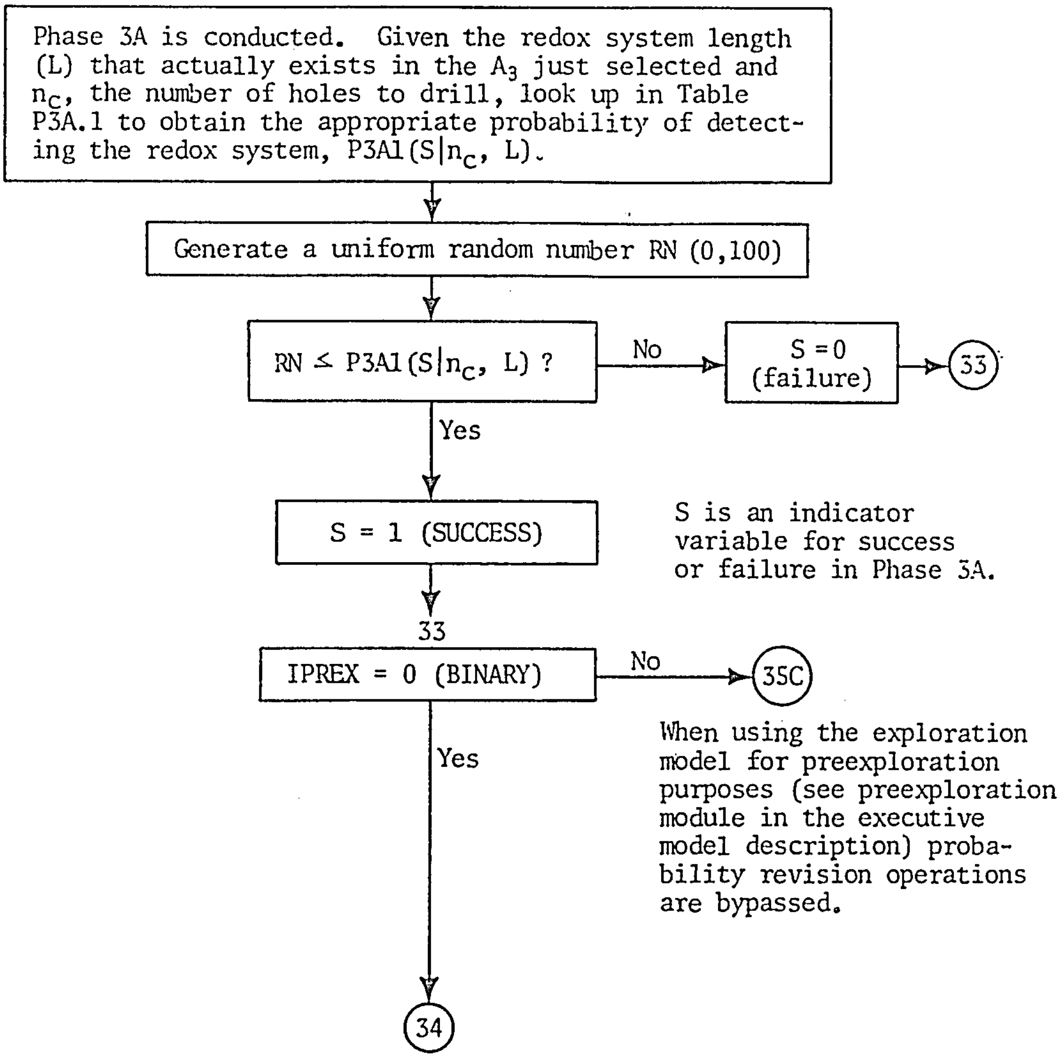


FIGURE G.6 EXPLORATION MODEL: PHASE 3A


Phase 3A is conducted. Given the redox system length (L) that actually exists in the A_3 just selected and n_c, the number of holes to drill, look up in Table P3A.1 to obtain the appropriate probability of detecting the redox system, $P3A1(S|n_c, L)$.
Generate a uniform random number RN (0,100)
$RN \leq P3A1(S|n_c, L)$?
No
S = 0 (failure)
33
Yes
S = 1 (SUCCESS)
S is an indicator variable for success or failure in Phase 3A.
33
IPREX = 0 (BINARY)
No
35C
Yes
When using the exploration model for preexploration purposes (see preexploration module in the executive model description) probability revision operations are bypassed.
34

34

Revise the prior distribution for redox length in the A_3, $f(R|Sel,\overline{DEPTH})$ in light of the new information in Phase 3A (i.e., success or failure and number of holes drilled). A posterior probability distribution is computed, $c(R|n_c,S)$ as follows (non rigorous):

$$c(R|n_c,S) = \frac{P(R,n_c,S)}{\sum_R P(R,n_c,S)}$$

where

$$P(R,n_c,S) = f(R|Sel,\overline{DEPTH})\cdot P(n_c|R)\cdot P(S|R,n_c)$$

$P(n_c|R)$ is the "experience" term, i.e., the relative frequency the explorationist has drilled n_c holes in Phase 3A when the length of redox front in the A_3 is "R" miles long. Table P3A.2-TN in appendix F is consulted here, identifying the appropriate entry corresponding to n_c holes and R miles of redox front.

$P(S|R,n_c)$ = probability of success when the length of the redox front in the A_3 is R miles and n_c holes are drilled in Phase 3A. Table 3A.1 is employed to obtain this term.

35

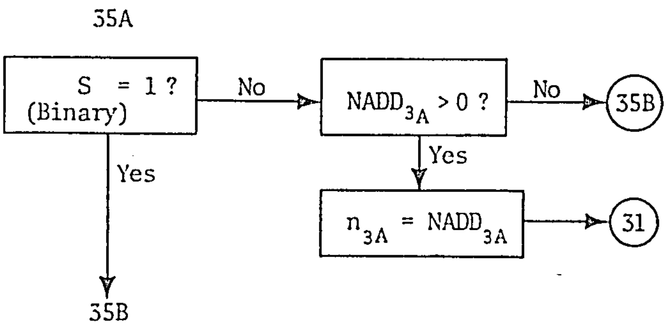
Consult the Executive Model. Pass current redox length (in the A_3) expectations (i.e., $c(R|n_c,S)$), the parameters of the depth to the host formation distribution, current value of uranium price in the system, the value of S (success or failure in Phase 3A), and n_c (the total number of holes drilled so far in Phase 3A).

Executive Model returns the following relevant value:

If S = 1, n_{3B} = number of holes to drill in Phase 3B

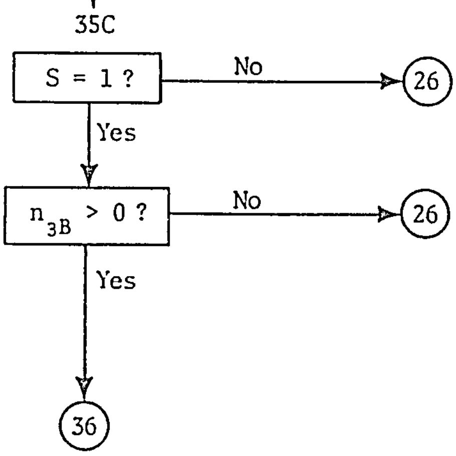
If S = 0, $NADD_{3A}$ = number of additional holes to drill in Phase 3A

35A



Update the "experience" Table P3A.2 (shown in Appendix F). Given L, the actual redox length in the current A_3, and n_C, the total number of holes that were drilled in Phase 3A, look up Table P3A.2-TN and identify the appropriate entry that represents the relative frequency of holes drilled in Phase 3A for the stated length, $\text{freq}(n_C|L)$. Add a "1" (i.e., one more experience) to both numerator and denominator. Add a "1" to the denominators of the remaining entries in that row (i.e., to all those entries with the same redox length, L, condition) so that the sum of the conditional probabilities still adds up to 1.0. The computation of numerator and denominator from the decimal entries is detailed in chapter 8, s. 8.6.2.

Note: if S = 0 (failure in Phase 3A), do the updating assuming that the "actual" length, L, is zero.



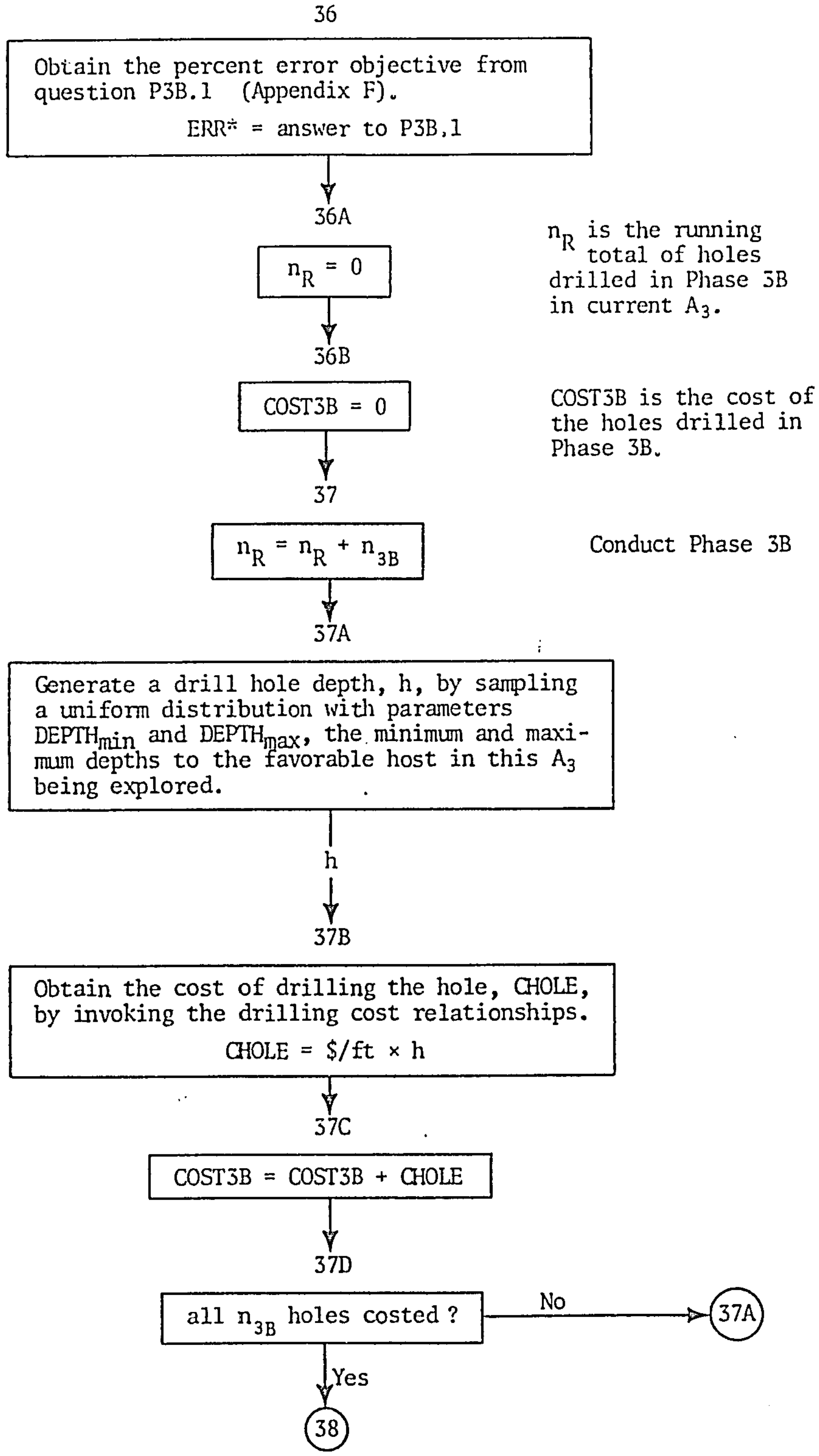
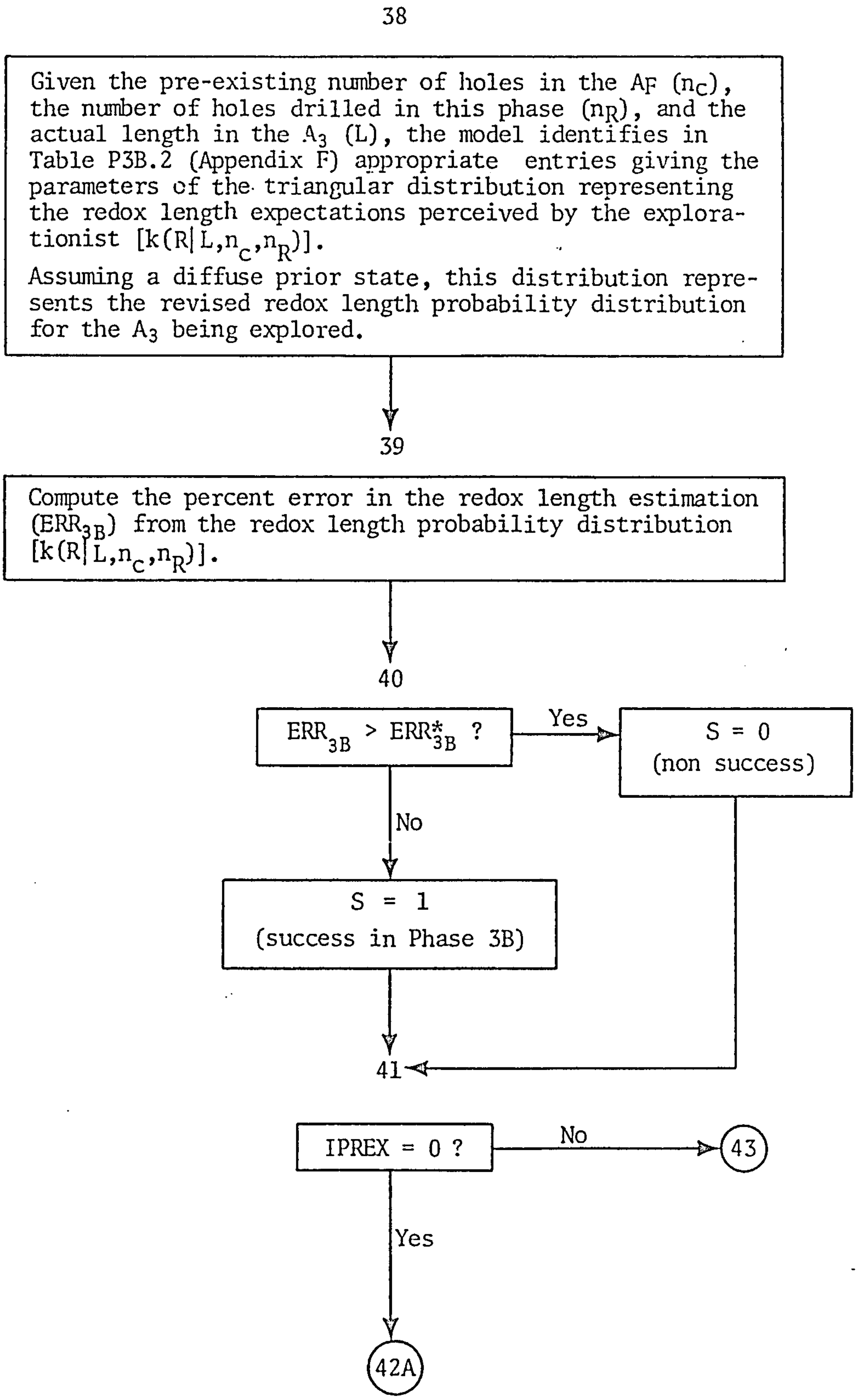


FIGURE G.7 EXPLORATION MODEL: PHASE 3B


38
Given the pre-existing number of holes in the AF (n_C), the number of holes drilled in this phase (n_R), and the actual length in the A_3 (L), the model identifies in Table P3B.2 (Appendix F) appropriate entries giving the parameters of the triangular distribution representing the redox length expectations perceived by the explorationist [$k(R|L,n_c,n_R)$].
Assuming a diffuse prior state, this distribution represents the revised redox length probability distribution for the A_3 being explored.
39
Compute the percent error in the redox length estimation (ERR_{3B}) from the redox length probability distribution [$k(R|L,n_c,n_R)$].
40
$ERR_{3B} > ERR^*_{3B}$?
Yes
S = 0
(non success)
No
S = 1
(success in Phase 3B)
41
IPREX = 0 ?
No
43
Yes
42A

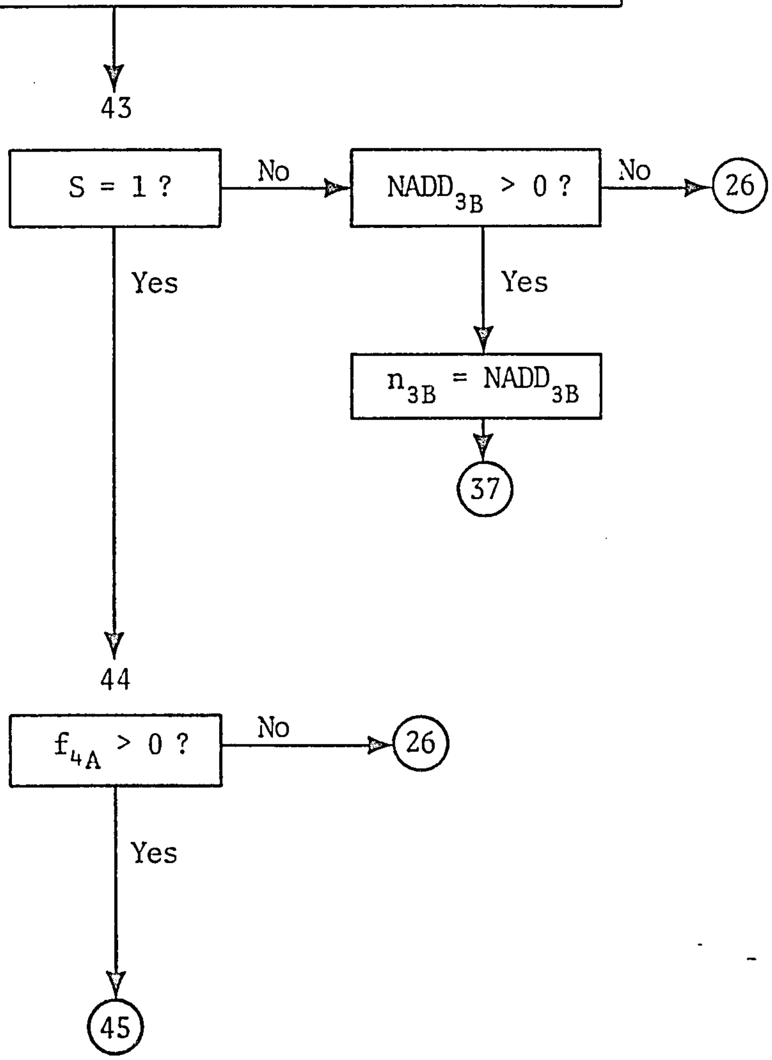
42A

Consult the Executive Model. Pass current redox length expectations [i.e. $k(R|L,n_C,n_R)$], the parameters of the depth to favorable host distribution in the current A_3, the current U_3O_8 price in the system, value for S (Phase 3B objective achieved or not), the number of holes drilled in Phase 3A (n_C) and the number of holes drilled so far in Phase 3B (n_R).

The executive model returns the following relevant values:

If $S = 1$, f_{4A} = number of fences to drill in Phase 4A.

If $S = 0$, $NADD_{3B}$ = number of additional holes to drill in Phase 3B.



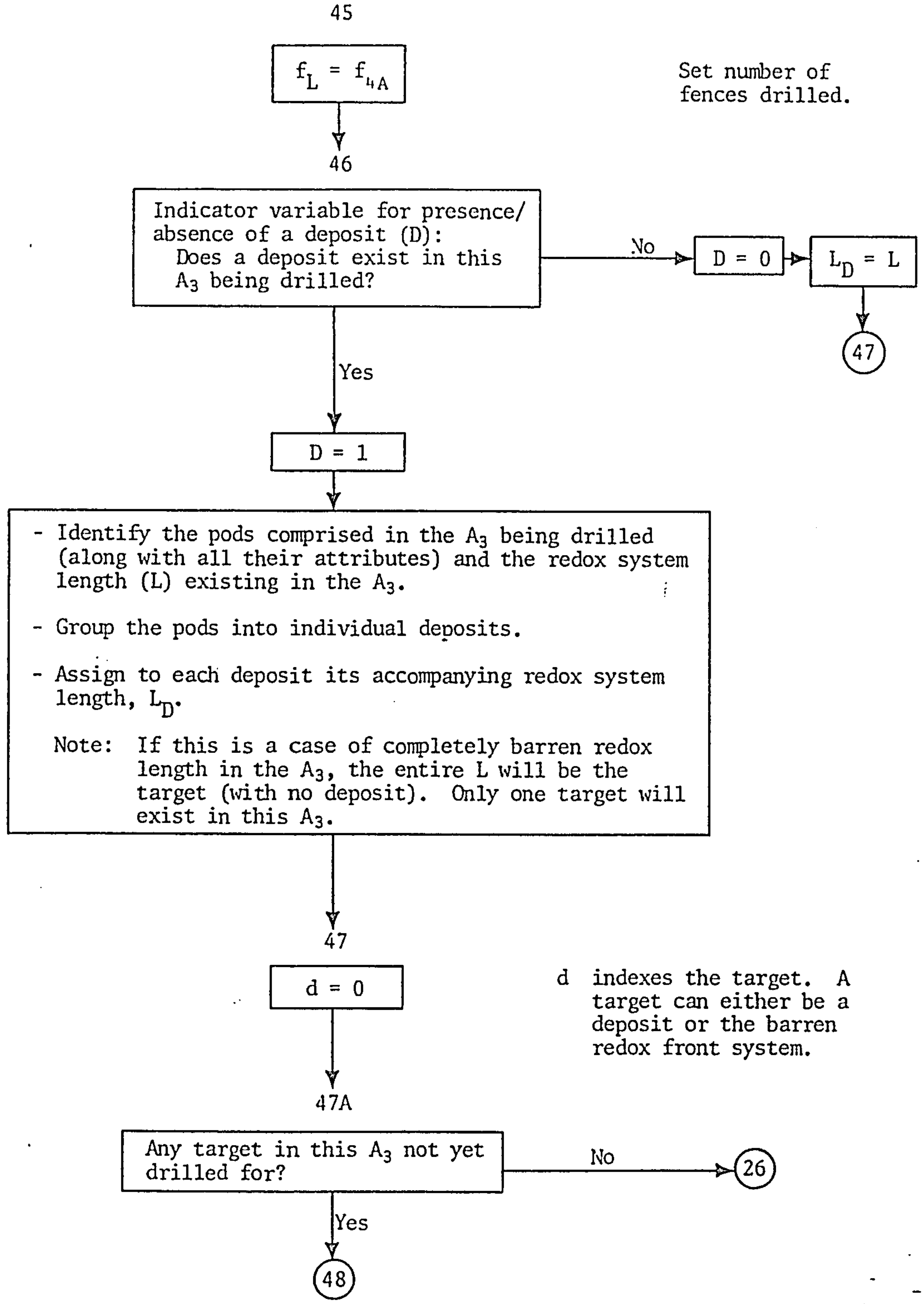
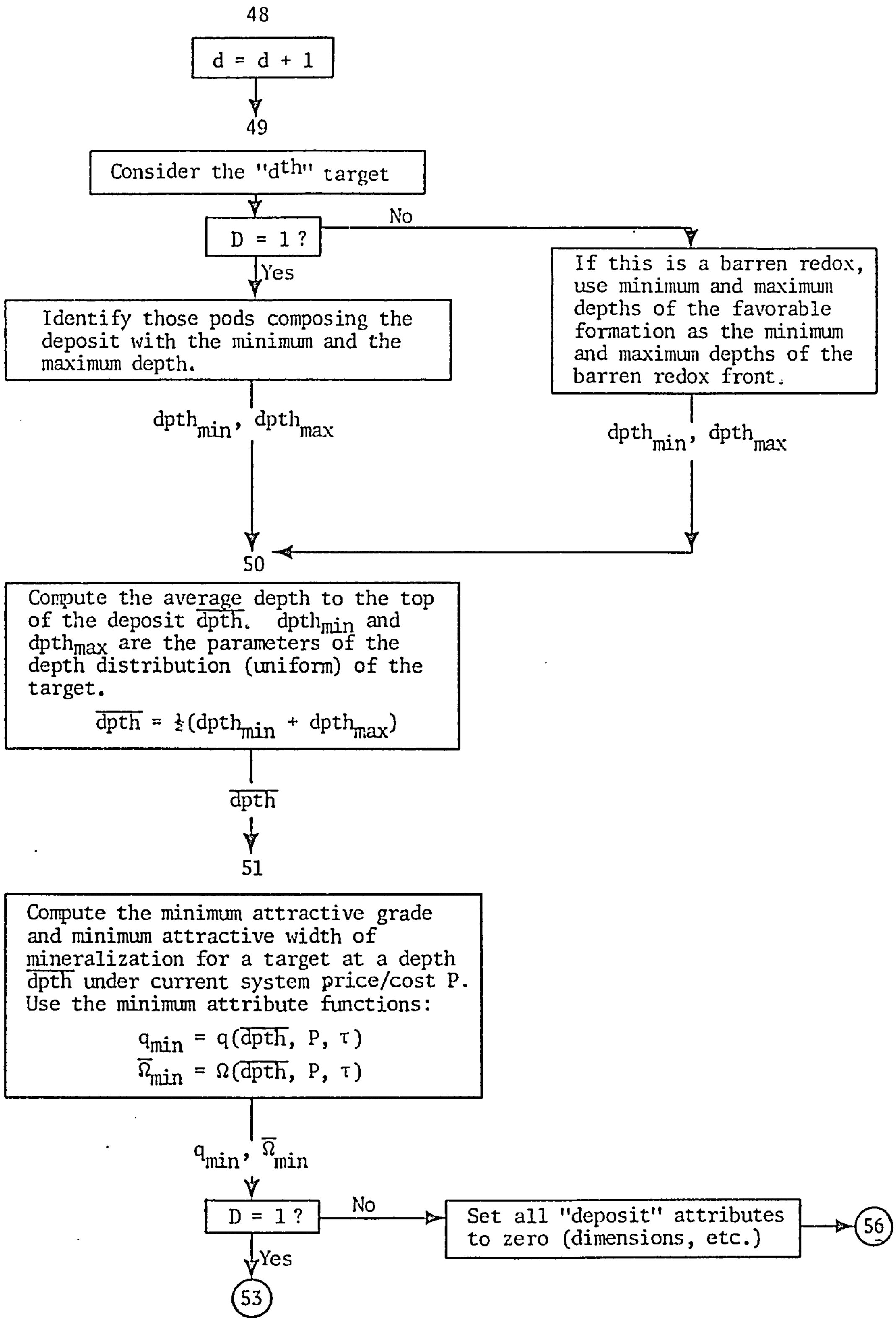


FIGURE G.8 EXPLORATION MODEL: PHASE 4.A


48
d = d + 1
49
Consider the "dth" target
D = 1 ?
No
Yes
Identify those pods composing the deposit with the minimum and the maximum depth.
If this is a barren redox, use minimum and maximum depths of the favorable formation as the minimum and maximum depths of the barren redox front.
dpthmin, dpthmax
dpthmin, dpthmax
50
Compute the average depth to the top of the deposit dpth. dpthmin and dpthmax are the parameters of the depth distribution (uniform) of the target.
dpth = ½(dpthmin + dpthmax)
dpth
51
Compute the minimum attractive grade and minimum attractive width of mineralization for a target at a depth dpth under current system price/cost P. Use the minimum attribute functions:
qmin = q(dpth, P, τ)
Ωmin = Ω(dpth, P, τ)
qmin, Ωmin
D = 1 ?
No
Set all "deposit" attributes to zero (dimensions, etc.)
56
Yes
53

53

Compute the length of the deposit at grade q_{min}, $\ell(q_{min})$, by adding up the length of the q_{min}-confining prism of each pod, $\lambda(q_{min})$, and the gap in between the pods comprising the deposit.

↓ $\ell(q_{min})$

54

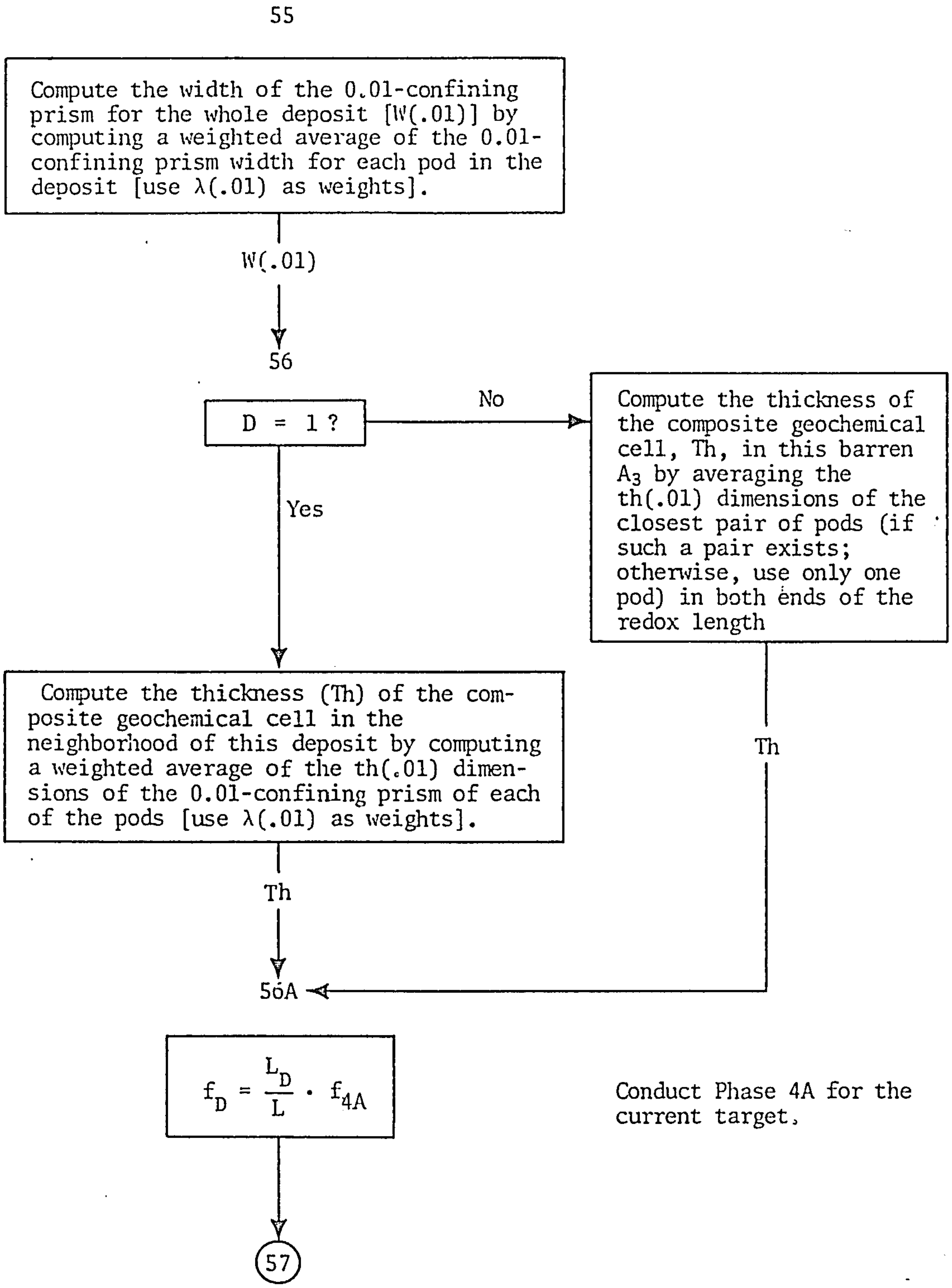
Compute the average width of the mineralization at cutoff grade q_{min} for the entire deposit, $\bar{\Omega}_D(q_{min})$, by:

1) Determine the width of the deposit confining prism at cutoff grade q_{min}, $w_D(q_{min})$, by computing a weighted average of the q_{min}-confining prism width for each pod [use $\lambda(q_{min})$ as weights].

2) Given $w_D(q_{min})$, look up the appropriate entry in Table SM.6 in Appendix E to obtain $\bar{\Omega}_D(q_{min})$.

↓ $\bar{\Omega}_D(q_{min})$, $w_D(q_{min})$

55


55
Compute the width of the 0.01-confining prism for the whole deposit [W(.01)] by computing a weighted average of the 0.01-confining prism width for each pod in the deposit [use λ(.01) as weights].
W(.01)
56
D = 1 ?
No
Yes
Compute the thickness of the composite geochemical cell, Th, in this barren A3 by averaging the th(.01) dimensions of the closest pair of pods (if such a pair exists; otherwise, use only one pod) in both ends of the redox length
Compute the thickness (Th) of the composite geochemical cell in the neighborhood of this deposit by computing a weighted average of the th(.01) dimensions of the 0.01-confining prism of each of the pods [use λ(.01) as weights].
Th
Th
56A
$f_D = \frac{L_D}{L} \cdot f_{4A}$
Conduct Phase 4A for the current target.
57

57

Given f_D, L_D, $\ell(q_{min})$, $\overline{\Omega}_{min}$, $\overline{\Omega}_D(q_{min})$, and q_{min}, use tables in question P4A.3 and the adjustment tables in question P4A.4 to identify the appropriate entry for the probability of success, $P4A3[S|f_D, L_D, \ell(q_{min}), \overline{\Omega}_{min}, \overline{\Omega}_D(q_{min}), q_{min}]$

58

Generate a uniform random number RN (0,100)

59

Generate an outcome:
RN < $P4A3[S|f_D, L_D, \ell(q_{min}), \overline{\Omega}_{min}, \overline{\Omega}_D(q_{min}), q_{min}]$?

No → Failure S = 0

Yes

S = 1 (Success)

59A

IPREX = 0 ?

No → 70

Yes

60

When using the exploration model for "preexploration" purposes (see preexploration module in Executive Model description), probability revision operations are bypassed.

Obtain $y[\ell(q_{min})|S, q_{min}, \overline{\Omega}_{min}, f_D, Th, D, L_D]$, the probability distribution representing the explorationist's expectations about the length of the deposit after conducting Phase 4A under the conditions noted there.

Given the values of all those conditions listed above, identify the appropriate entry in Tables P4A.5 for the $\ell(q_{min})$ probability distribution. Use Table P4A.6 to adjust the entry obtained above in case Th and $\overline{\Omega}_{min}$ are not the same as stated as reference in Tables P4A.5

61

61

D = 1 ?

No → 64

Yes

62

Compute the weighted average thickness of the 0.01 and q*-confining prism across all pods in the deposit [$\overline{th}$(.01) = Th]. Use length of the pods as weights.

Compute the expected cross sectional target dimensions when a deposit actually exists.

$\overline{th}$(.01), $\overline{th}$(q*)

63

Compute the weighted average width of the 0.01 and q*-confining prism across all pods in the deposit. Use length of the pods as weights.

$\overline{w}$(.01), $\overline{w}$(q*)

→ 69

64

$\overline{th}$(.01) = Th

Compute the expected cross sectional target dimensions when a deposit <u>is</u> <u>not</u> present.

65

Given $\overline{\Omega}_{min}$, make Ω'(.01) = $\overline{\Omega}_{min}$.

Look up in Table SM.6 (Appendix E) for that 01-confining prism width [w'(.01)] "heading" that would correspond to Ω'(.01).

66

$\overline{w}$(.01) = w'(.01)

67

67

Given $\overline{w}(.01)$, identify in Table SM.4 (Appendix E) the appropriate entry giving the parameters describing the triangular probability distribution for $w(q^*)$.

Given those parameters, compute the mean value for q^*-confining prism width, $\overline{w}(q^*)$.

$\overline{w}(q^*)$

68

Given $\overline{th}(.01)$, identify in Table SM.5 (Appendix E) the appropriate entry giving the parameters describing the triangular probability distribution for $th(q^*)$.

Given those parameters, compute the mean value for q^*-confining prism thickness, $\overline{th}(q^*)$.

$\overline{th}(q^*)$

69

Consult Executive Model. Pass current expectations regarding the length of the target, $y[\ell(q_{min})]$; the parameters of the target depth distribution, $dpth_{max}$, $dpth_{min}$; the expected cross sectional target dimensions $\overline{w}(.01)$, $\overline{th}(.01)$, $\overline{w}(q^*)$, $\overline{th}(q^*)$; the success indicator variable; the width of the 0.01-confining prism for the deposit, $W(.01)$; the number of fences drilled so far in Phase 4A for the d^{th} deposit; Ω_{min} and q_{min}.

Executive Model returns:

If $S = 1$, n_{4B} = number of holes to drill in Phase 4B.

If $S = 0$, $FADD_{4A}$ = number of additional fences to drill in Phase 4A.

70

IPREX = 1 ? — Yes → $FADD_{4A} = 0$ → 70A

No → 70A

70A

S = 1 ? — No → $FADD_{4A} = 0$? — No → $f_D = f_D + FADD_{4A}$ → 57

$FADD_{4A} = 0$? — Yes → 70B

S = 1 ? — Yes → 70B

70B

Obtain the average number of holes per fence from Tables P4A.1. Given $\bar{\Omega}_{min}$, f_D, L_D and W(.01), identify the appropriate entry in Table P4A.1. This entry contains the parameters of a triangular distribution describing the average number of holes per fence.

Sample (using Monte Carlo techniques) that triangular distribution to obtain HPF, a specific value for the average number of holes per fence.

70C

$n_{4A} = HPF \cdot f_D$

70D

S = 1 ?

No → NVER = 0 → 70F

Yes → 70E

70E

Consult Tables P4A.7 to obtain the number of verification holes (NVER) drilled after intercepting a mineralized target. Given $\bar{\Omega}_D(q_{min})$, identify the appropriate entry.

NVER

70F

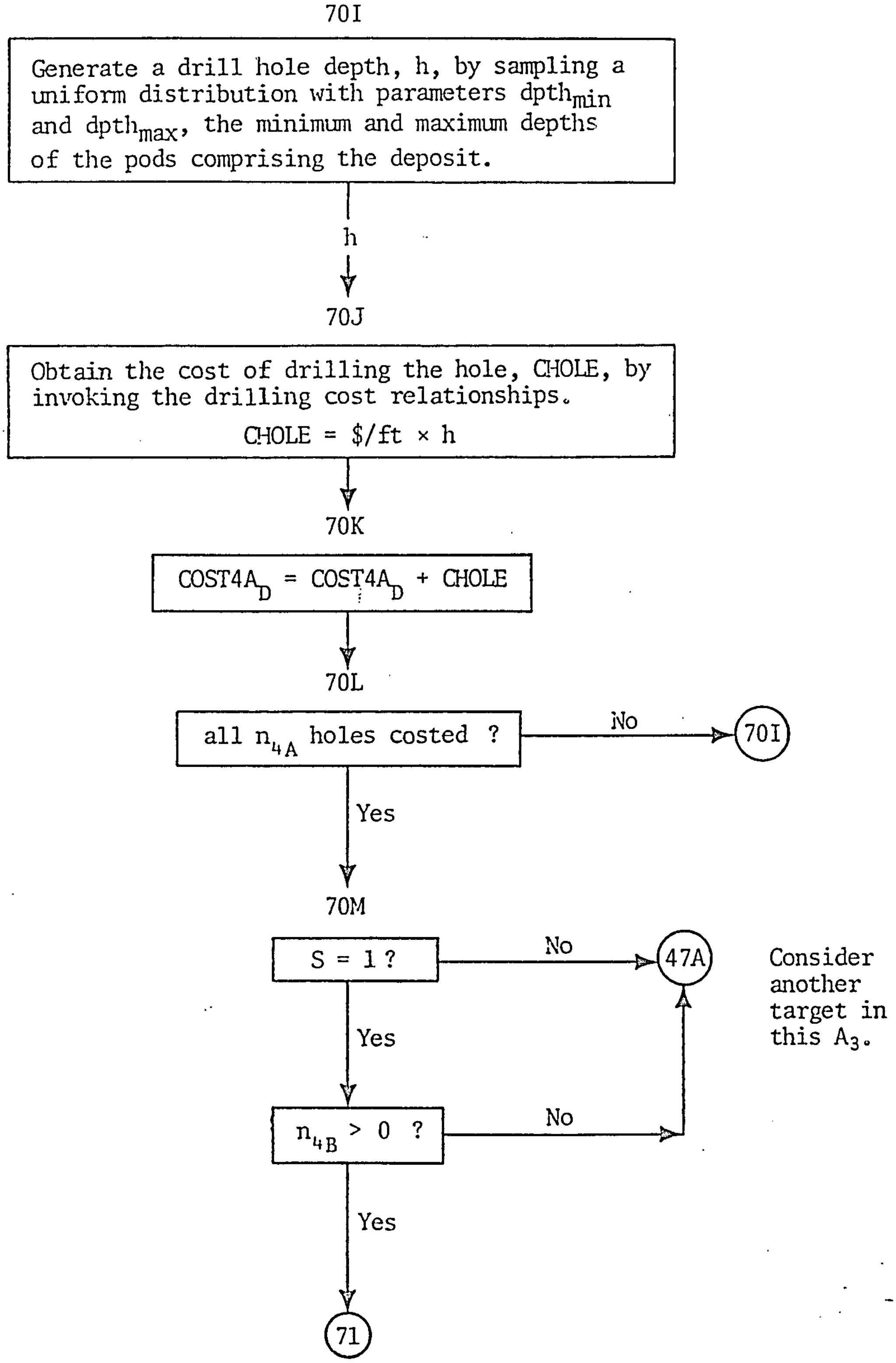
$n_{4A} = n_{4A} + NVER$

70H

$COST4A_D = 0$

Drilling cost expenditures in Phase 4A for this target "D".

70I


70I
Generate a drill hole depth, h, by sampling a uniform distribution with parameters $dpth_{min}$ and $dpth_{max}$, the minimum and maximum depths of the pods comprising the deposit.
h
70J
Obtain the cost of drilling the hole, CHOLE, by invoking the drilling cost relationships.
CHOLE = $/ft × h
70K
$COST4A_D = COST4A_D + CHOLE$
70L
all n_{4A} holes costed ?
No
70I
Yes
70M
S = 1 ?
No
47A
Consider another target in this A_3.
Yes
$n_{4B} > 0$?
No
Yes
71

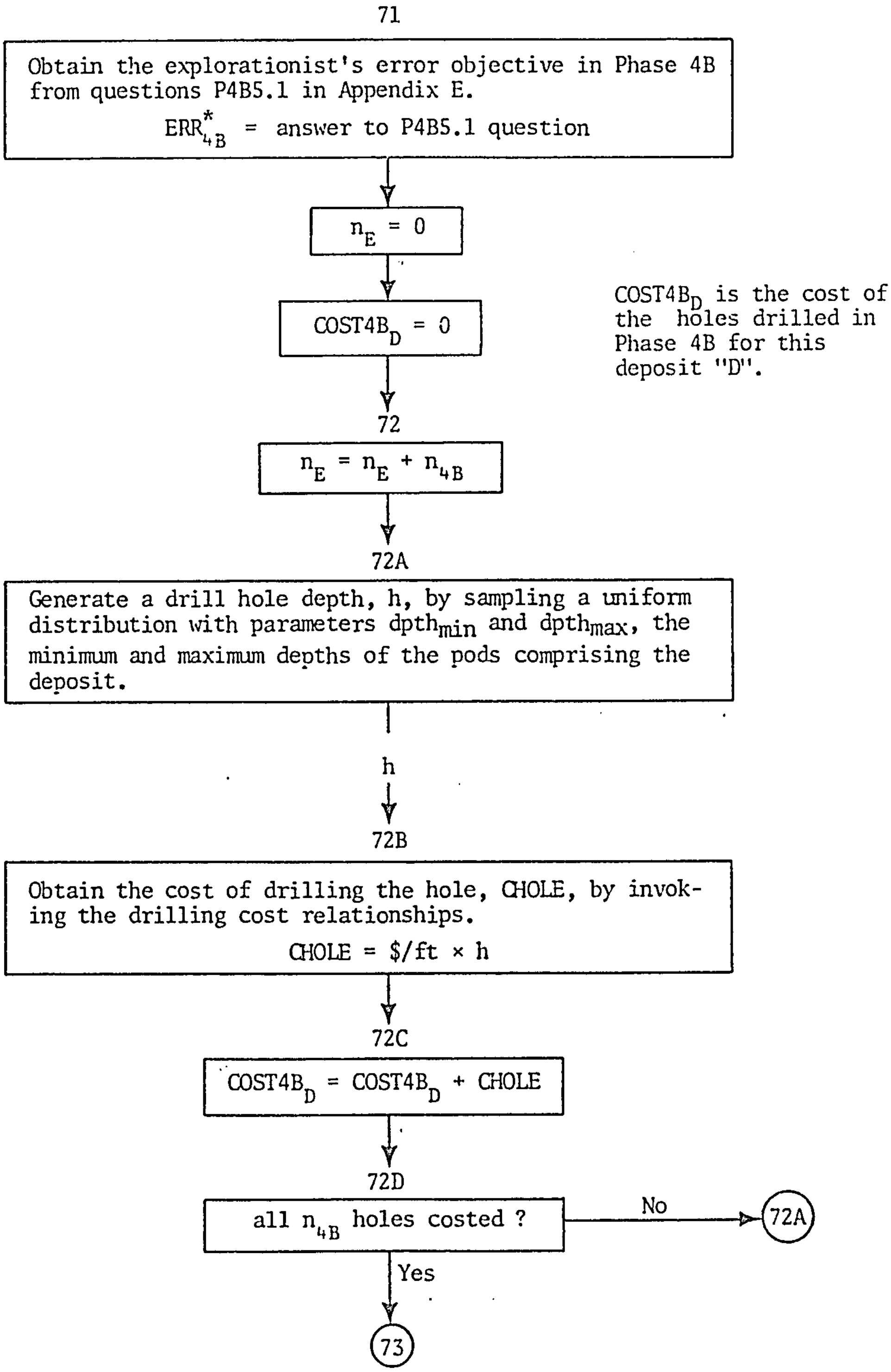
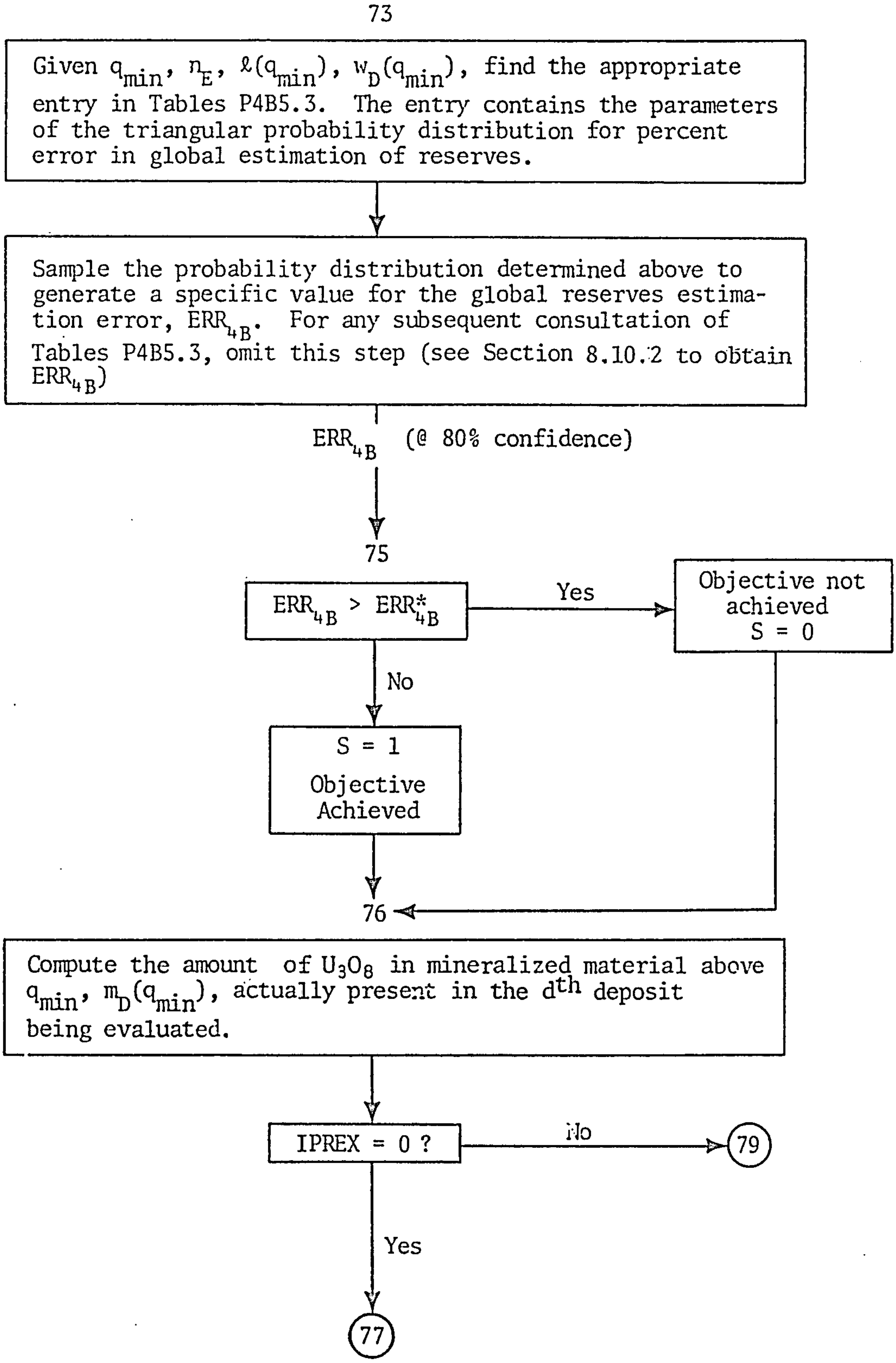


FIGURE G.9 EXPLORATION MODEL: PHASE 4B


73
Given q_{min}, n_E, $\ell(q_{min})$, $w_D(q_{min})$, find the appropriate entry in Tables P4B5.3. The entry contains the parameters of the triangular probability distribution for percent error in global estimation of reserves.
Sample the probability distribution determined above to generate a specific value for the global reserves estimation error, ERR_{4B}. For any subsequent consultation of Tables P4B5.3, omit this step (see Section 8.10.2 to obtain ERR_{4B})
ERR_{4B} (@ 80% confidence)
75
$ERR_{4B} > ERR^*_{4B}$
Yes
Objective not achieved
S = 0
No
S = 1
Objective Achieved
76
Compute the amount of U_3O_8 in mineralized material above q_{min}, $m_D(q_{min})$, actually present in the d^{th} deposit being evaluated.
IPREX = 0 ?
No
79
Yes
77

77

Assuming that the estimator for global reserves, $u_D(q_{min})$, is unbiased and normally distributed, fit a normal distribution to the random variable $\tilde{u}_D(q_{min})$. Use the actual (but unknown to the explorationist whose actions are being modeled) metal content, $m_D(q_{min})$, as the mean (μ_u). Use the value for the estimation error at 80% confidence, ERR_{4B}, to compute the variance (σ_u^2).

μ_u, σ_u^2

78

Consult the Executive Model. Pass q_{min}, n_E, the actual length and width dimensions $\ell(q_{min})$ and $w_D(q_{min})$, the current uranium price in the system, and the parameters of the probability distribution $N(\mu_u, \sigma_u^2)$ representing the current expectations of global reserves at q_{min}, $u_D(q_{min})$.

Executive Model returns:

If $S = 1$, n_5 = number of holes to drill in Phase 5.

If $S = 0$, $NADD_{4B}$ = number of additional holes to drill in Phase 4B.

79

$S = 1$?

No → $NADD_{4B} > 0$?

No → 47A

Look for another target in this A_3.

Yes → $n_{4B} = NADD_{4B}$ → 72

Yes → $n_5 > 0$?

No → 47A

Yes → 80

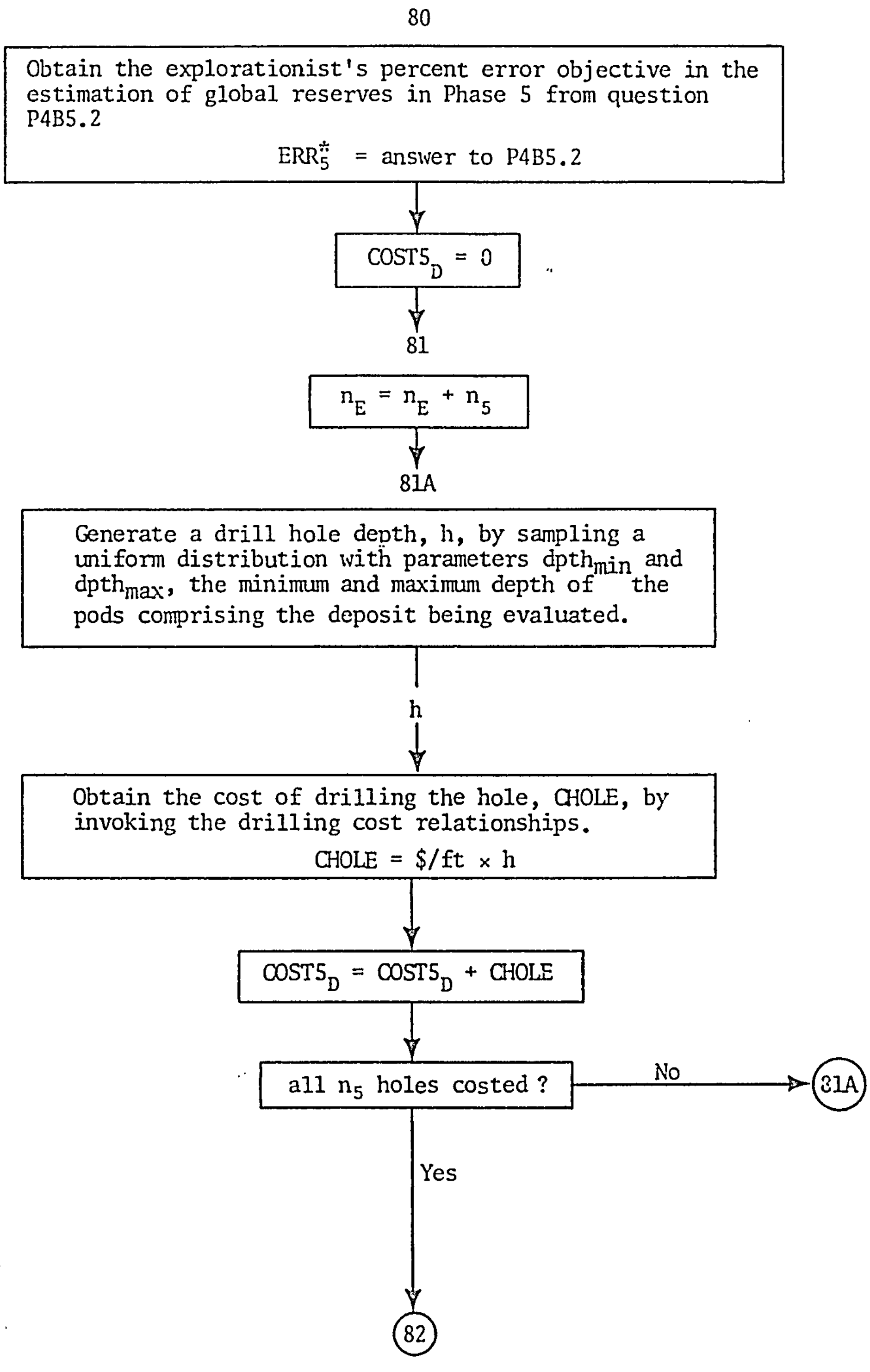
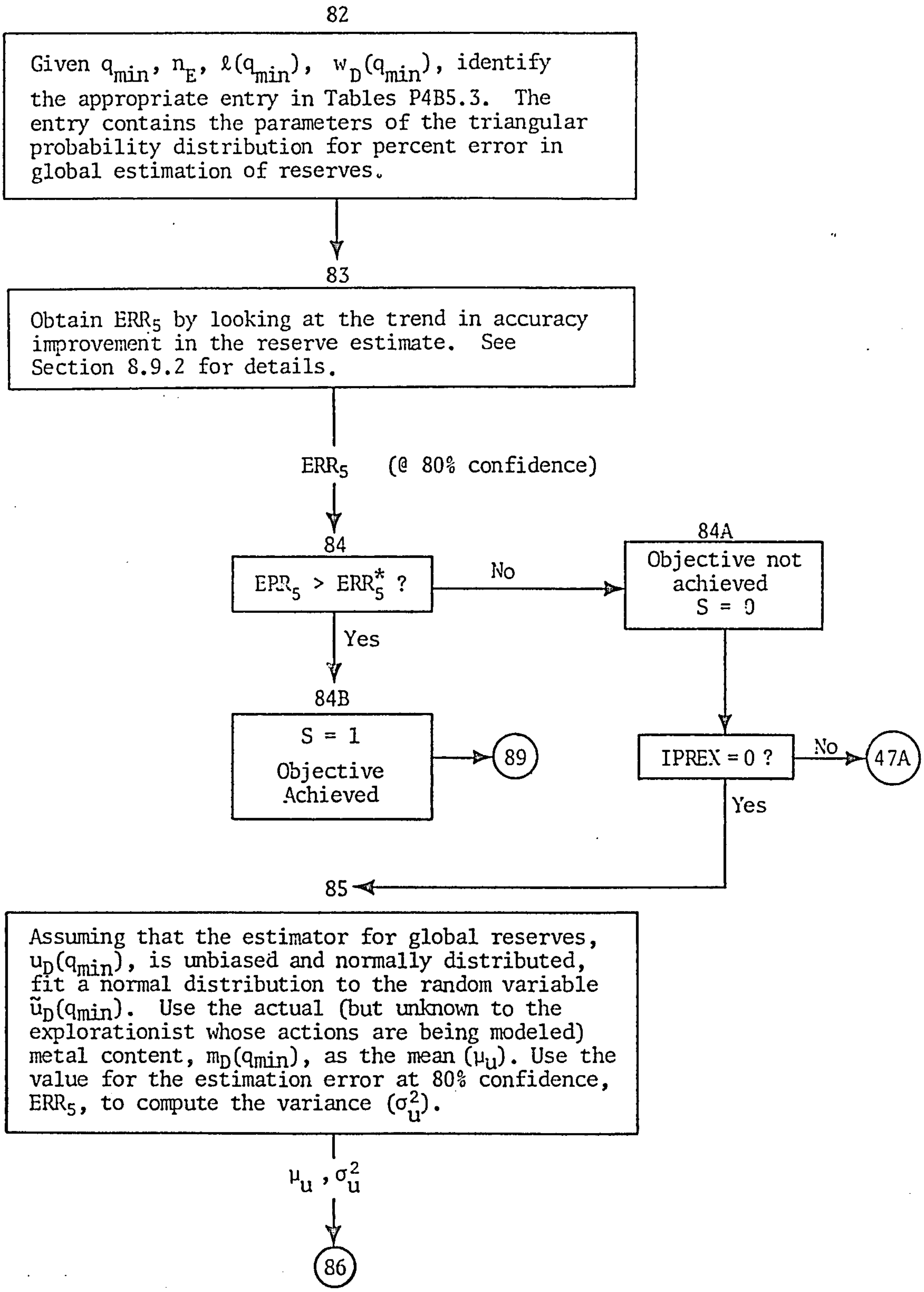


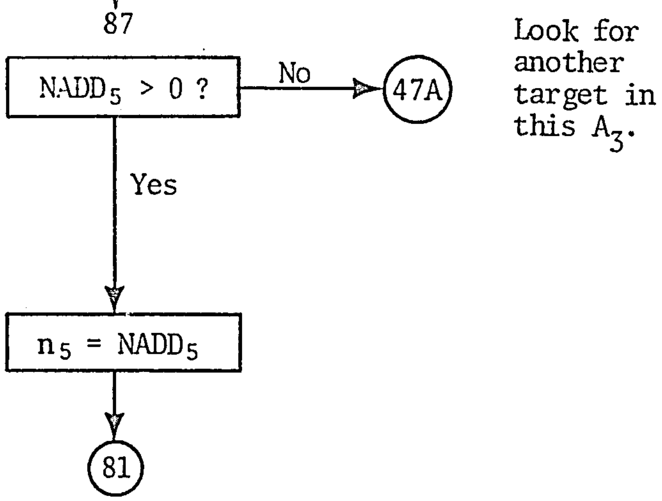
FIGURE G.10 EXPLORATION MODEL: PHASE 5


82
Given q_{min}, n_E, $\ell(q_{min})$, $w_D(q_{min})$, identify the appropriate entry in Tables P4B5.3. The entry contains the parameters of the triangular probability distribution for percent error in global estimation of reserves.
83
Obtain ERR_5 by looking at the trend in accuracy improvement in the reserve estimate. See Section 8.9.2 for details.
ERR_5 (@ 80% confidence)
84
$ERR_5 > ERR_5^*$?
No
84A
Objective not achieved
S = 0
Yes
84B
S = 1
Objective Achieved
89
IPREX = 0 ?
No
47A
Yes
85
Assuming that the estimator for global reserves, $u_D(q_{min})$, is unbiased and normally distributed, fit a normal distribution to the random variable $\tilde{u}_D(q_{min})$. Use the actual (but unknown to the explorationist whose actions are being modeled) metal content, $m_D(q_{min})$, as the mean (μ_u). Use the value for the estimation error at 80% confidence, ERR_5, to compute the variance (σ_u^2).
μ_u, σ_u^2
86

Consult the Executive Model. Pass q_{min}, n_E, the actual length and width dimensions for the deposit $\ell(q_{min})$ and $w_D(q_{min})$, the current uranium price in the system, and the parameters for the probability distribution $N(\mu_u,\sigma_u^2)$ representing the current expectations of global reserves at q_{min}, $u_D(q_{min})$.

Executive Model returns:

If $S = 0$, $NADD_5$ = number of additional holes to drill in Phase 5.



Pass the deposit through the MINING COST MODEL.

Provide the following information:

- For every pod making up the deposit:

  0.01-confining prism dimensions
  q*-confining prism dimensions
  grade-tonnage curve parameters
  depth to the top of the 0.01-confining prism
  gap distance between the pods
  shrinkage function parameters

- Economic related inputs:

  rate of return
  cost indexes
  lead times
  state
  state tax rate
  current price of uranium in the system

Mining Cost Model returns the following key information:

q_c^* = optimum cutoff grade

NPV* = maximum net present value (at grade q*)

$t(q_c^*)$ = tonnage of mineralized material beyond q_c^*

$\bar{q}(q_c^*)$ = average grade of mineralized material beyond q_c^*

↓

NPV*, q_c^*, $t(q_c^*)$, $\bar{q}(q_c^*)$

↓

(90)

FIGURE G.11 MINING COST MODEL

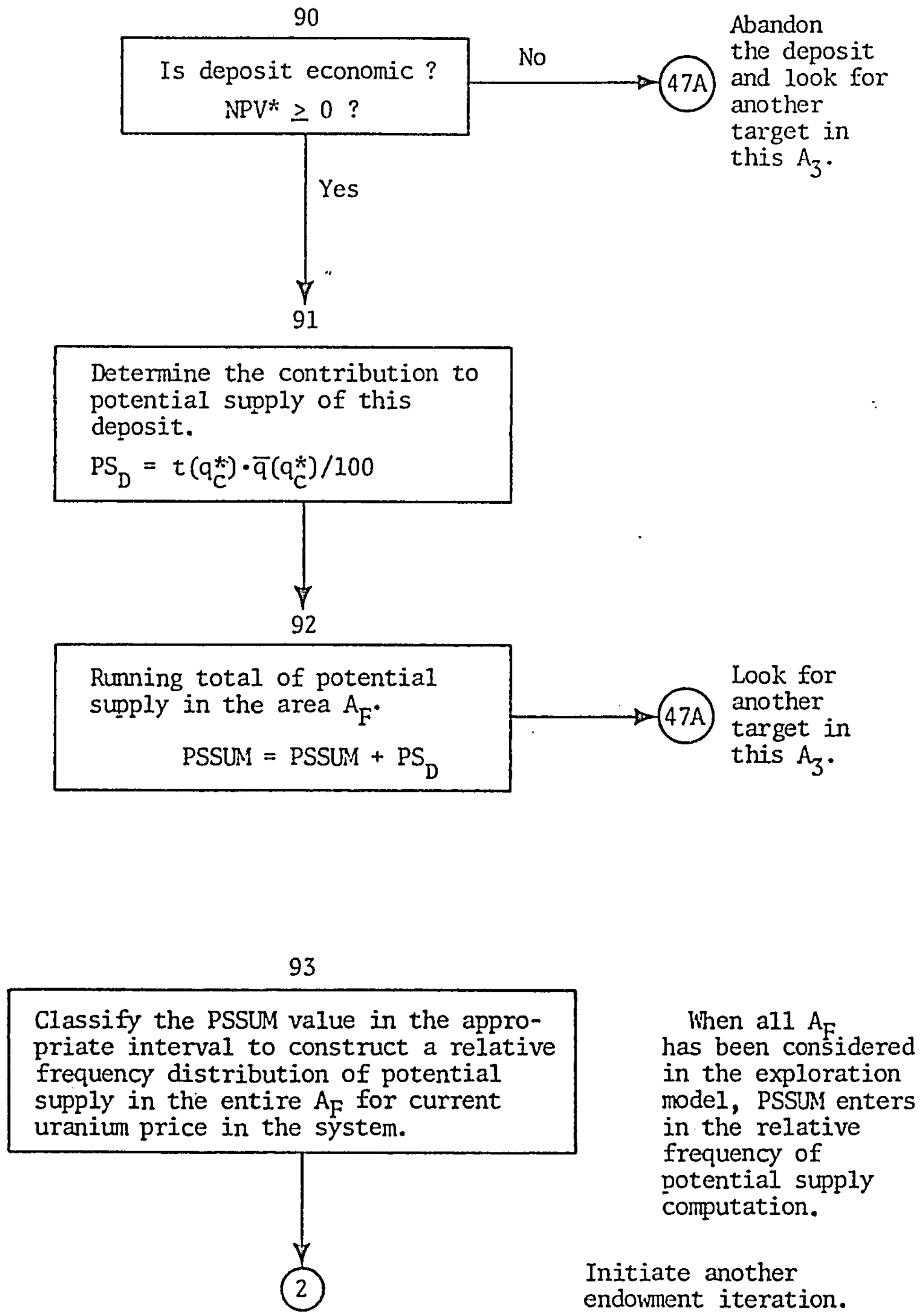


FIGURE G.12 CONTRIBUTION OF DEPOSIT TO POTENTIAL SUPPLY

APPENDIX H

A3GRID ROUTINE

A3GRID: Generating the Probability Distribution for Redox Length, $\delta(L|L>0)$, in a Drilling Area

INTRODUCTION

The outline of the favorable area delineated by NURE is an input to the potential supply system. A grid of rectangles, each corresponding in size to a drilling area, is placed over the outline. If the central point of a rectangle lies inside the outline, that rectangle is included in the coordinate set of rectangles which we shall call A_F. The individual rectangles of the grid will each be called an A_3. Whereas the boundary of the outline may be completely irregular, the boundary of A_F will consist entirely of horizontal and vertical lines. The individual A_3's within A_F will be referenced by a pair of coordinates (subscripts) according to their position in the grid overlay (see Figure H.1).

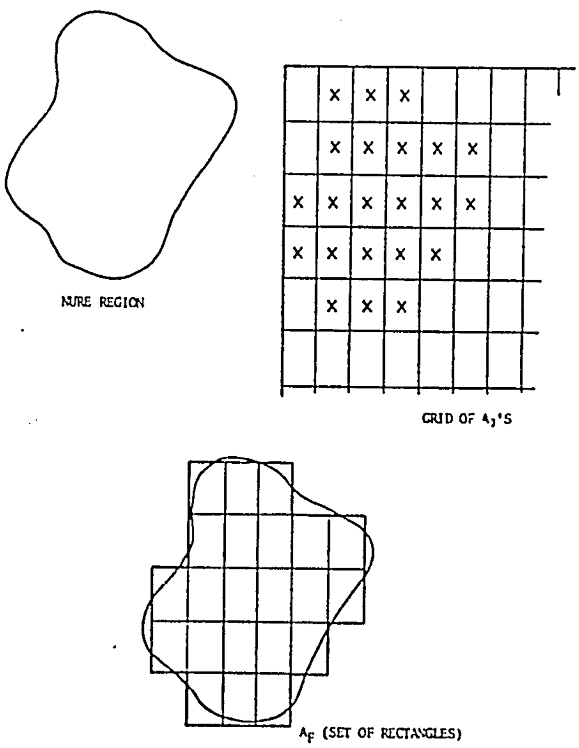


FIGURE H.1 NURE REGION AND A_F

The A3GRID routine is totally external to the potential supply system operation. The objective of this routine is to derive a probability distribution for length of redox within an A_3, $\delta(L)$. A truncated version of this distribution [i.e., $\delta(L|L > 0)$] is used in two tasks:[1]

(a) In the creation submodel of the potential supply system (see Section 7).

(b) In the A3SAMP routine, described in Appendix I.

Both tasks involve the "seeding"of the redox length in the A_F and keeping track of the redox lengths in each A_3 component of the grid. Although the random walk approach described in this appendix to "seed" the redox length could be used directly in the two tasks, the computer cost would probably be prohibitive. Instead, this auxiliary distribution, $\delta(L|L > 0)$ is employed in a much simpler "seeding" procedure.

The exploration model requires that the state of nature be reproduced. It is assumed that a redox front exists within the NURE region, and that we can estimate its length with a probability distribution; but what is its shape or position within the area? Having no such information about these features, it will be assumed that they are completely random (subject to certain restrictions) (see Figure H.2). To simulate the redox line we use a randomly generated set of connected line segments beginning at an arbitrary point within A_F. It is assumed that the distribution, $\delta(L)$, will correspond to a distribution of redox length existing within a drilling area placed randomly (at any point and with any orientation) within the NURE region (see Figure H.3). The arguments for this are intuitive: the randomness in the placement and shape of the (simulated) redox line is sufficient to ensure that its length in any grid rectangle will be representative of the length that would be found in a randomly placed rectangle of the same dimensions.

The method used is a form of random walk. To this end, the A_3 is further subdivided into small squares to be called A_0. These must be small enough to assure that the overall shape of the line will approximate those found in nature (see Figure H.4). A point is selected randomly on the perimeter of A_F, at which the generation of the redox line will commence.

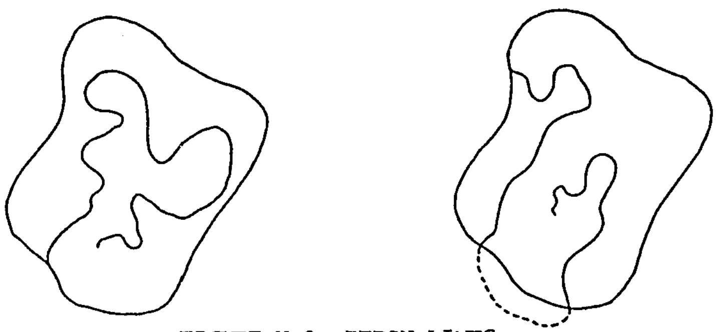


FIGURE H.2 REDOX LINES

[1]Additionally, $\delta(L)$ is used in the interpolation procedure for Table P3A.2 described in Chapter 8, section 8.6.2.

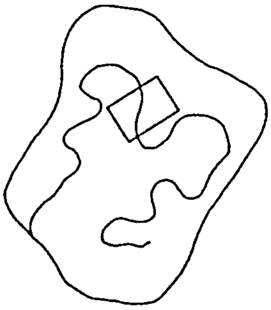


FIGURE H.3 RANDOMLY PLACED DRILLING AREA

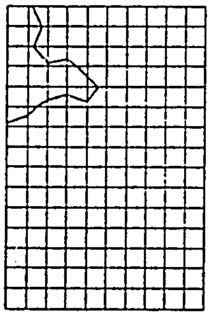


FIGURE H.4 GRID OF A_0'S ON A_3 WITH CONNECTED LINE SEGMENTS

Required as input to this routine are the shape of A_F, length and width of A_3, length of a side of A_0, and the distribution for redox length in the NURE region.[1] The shape of A_F is given by a simple indication of whether or not each A_3 in the grid is a member thereof (see Figure H.5)

---

[1]The distribution actually needed for the two tasks referred to above is the truncated redox length distribution within an A_3, $\delta(L|L > 0)$. It is assumed that this truncated distribution, in nature, remains invariant no matter the locality, size or shape of the roll-type favorable NURE region or the total endowment probability distribution, as long as the A_3 size and shape are the same (except, of course, in contrived situations that one can specify in modeling this geometric situation, like specifying an area size that is inadequate to contain the redox lengths being "seeded").

Thus, it is necessary to run the A3GRID simulation routine only once, using a (hypothetical) large A_F region (in the order of 500 square miles). Furthermore, the redox length for the entire NURE region does not need to be described by a probability distribution.

```
0 1 1 1 0 0 0 • • •
0 1 1 1 1 1 0 • • •
1 1 1 1 1 1 0 • • •
1 1 1 1 1 0 0 • •
0 1 1 1 0 0 0 •
0 0 0 0 0 0 0
0 0 0 0 0 0 0
• • • • •
• • •
```

FIGURE H.5 MATRIX INDICATING THE SHAPE OF A_F

In each iteration of the simulation a redox length is generated from the input distribution for A_F. A random starting point is selected. Then line segments are generated in successive A_0's (and, on the larger scale, in successive A_3's) until the total length of these (connected) segments matches or exceeds the redox length generated. Statistics on the lengths in the various A_3's are kept.

DEFINITIONS AND CONVENTIONS

1. Random number generation will follow the standard techniques used in the field of Monte Carlo simulation.

2. L_T is the redox length for the entire A_F during one iteration of the simulation, and is generated randomly from the input distribution.

3. LSUM is a running total of the lengths of line segments being generated in successive squares (A_0). When the value of LSUM reaches or exceeds the value of L_T, the generation of the entire redox line has been completed for this iteration.

4. The rectangles in the grid of which A_F is a subset will be called A_3. An A_3 may be identified by a pair of subscripts corresponding to its location in the grid, using ordinary matrix subscripting methods (see Figure H.6).

1,1	1,2	1,3	1,4
2,1	2,2	2,3	2,4
3,1	3,2	3,3	
4,1	4,2		

FIGURE H.6 GRID OF A_3's

5. An A_3 in which line segments are being generated is subdivided into a secondary grid of squares for that purpose. These are called A_0, and may be identified using subscripts, relative to the secondary grid, in a manner similar to the identification of A_3's (see Figure H.7).

1,1	1,2	1,3	1,4
2,1	2,2	2,3	2,4
3,1	3,2	3,3	
4,1	4,2		

FIGURE H.7 GRID OF A_0's WITHIN AN A_3

6. The walls, or sides, of an A_3 or an A_0 may be identified by direction. Here the standard cartographic convention is followed (see Figure H.8).

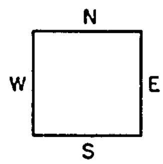


FIGURE H.8 WALL IDENTIFICATION

7. The starting point for a line to be generated in a square or rectangle will always fall on one of the sides of the figure; its position on that side is identified by the distance from the point to the first corner of the figure encountered when moving along the perimeter in a counterclockwise direction.

8. The starting wall is the side of A_0 or A_3 on which the starting point lies.

9. Given a starting wall and starting point in an A_0, an angle θ is to be generated within the square. This angle will be taken at the starting point, rotating in the counterclockwise direction from the starting wall, and will be no greater in magnitude than π radians, hence it will remain inside A_0. The line generated by the angle θ must intersect one of the remaining three walls of A_0 (after eliminating the starting wall), and it is this segment, between the starting point and the point of intersection with another wall, to which we refer as the line segment implied in A_0 by the angle θ.

10. The length of the line segment implied in A_0 by θ may be referred to simply as the "A_0 length."

11. The sum of A_0 lengths generated in a given A_3 will be called the "A_3 length."

12. An A_0 or an A_3 is occupied if the corresponding A_0 length or A_3 length, respectively, is not zero; otherwise, if no line segment(s) have been generated within it, the A_0 or A_3 is unoccupied.

13. A side of A_0 is accessible if it is (i) also a side of another A_0 which is unoccupied, or (ii) on a border of A_3 which is shared with another A_3 that is unoccupied. A side of A_0 is inaccessible if it is (i) also a side of another A_0 which is occupied, (ii) on a border of A_3 which is shared with another occupied A_3, or (iii) on a border of A_3 which is also a boundary of A_F.

14. A side of A_3 is accessible if it is also a side of another A_3 which is unoccupied; otherwise the side is inaccessible.

15. The statistics required for the A_3's are simply the A_3 lengths. These will be used to find the mean and variance of A_3 lengths, as well as to produce a frequency histogram and probability distribution for them, the latter to be used for sampling in subsequent programs.

PROCEDURE

0. Add together all those sides of A_3's which are on the border of A_F, to find the perimeter of A_F.

1. Generate a redox length L_T for A_F, from the input distribution.

2. Set LSUM equal to zero.

3. Select a starting point on the boundary of A_F:

   (a) Generate a uniform random number on the interval between zero and the perimeter of A_F;

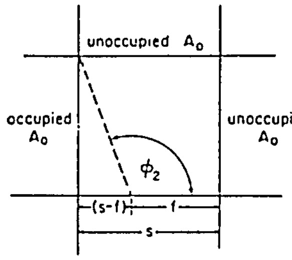
   (b) Using a fixed, arbitrary point and direction on the boundary of A_F, move away from that point in that direction along the boundary until the distance moved is equal to the random number generated in (a) -- this will be the starting point for the first segment of the simulated redox line;

   (c) Identify the A_3 and the wall on which the starting point occurs;

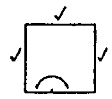
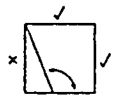
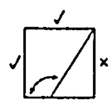
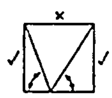
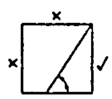
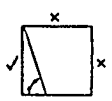
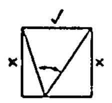
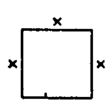
   (d) Identify the A_0 (and wall thereof) on which the starting point occurs, and compute the distance between the starting point and the first corner of A_0 encountered when moving in a counter-clockwise direction from it.

4. Set all A_0 lengths equal to zero.

5. Restrict the angle to be generated: determine the interval(s) in which the angle must lie, so that the line segment implied by it will not intersect any inaccessible side of A_0. The restrictions on the angle will be a function of the position of the starting point and the status (accessible or inaccessible) of the sides of A_0. In addition, the number of intervals involved (0, 1, or 2) must be determined. The eight possible cases are shown in Figure H.9.



Example: Let $\phi_1 = 0$ and ϕ_2 be as shown. Restrict selection of θ to the interval (ϕ_1, ϕ_2).

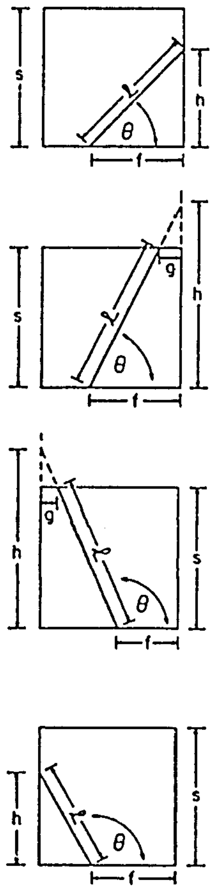
CASES:	RANGE OF θ	NUMBER OF INTERVALS
	$(0, \pi)$	1
	$[0, \pi - \tan^{-1}[s/(s-f)]]$	1
	$[\tan^{-1}(s/f), \pi]$	1
	$[0, \tan^{-1}(s/f)] \cup [\pi - \tan^{-1}[s/(s-f)], \pi]$	2
	$[0, \tan^{-1}(s/f)]$	1
	$[\pi - \tan^{-1}[s/(s-f)], \pi]$	1
	$[\tan^{-1}(s/f), \pi - \tan^{-1}[s/(s-f)]]$	1
	(. . .)	0

Note: ✓ = accessible
× = inaccessible

FIGURE H.9 RESTRICTIONS ON θ

6. Generate an angle θ on the interval(s) whose limits are established in (5).
   a) if the number of intervals is zero, $\theta = 0$;
   b) if there is one interval, say (ϕ_1, ϕ_2), generate θ uniformly on the interval;
   c) if there are two intervals, say $(0, \phi_1)$ and (ϕ_2, π), proceed as follows:
      (i) find the total number of radians in both intervals: $\phi_T = \phi_1 + (\pi - \phi_2)$;
      (ii) generate θ_0 uniformly on $(0, \phi_T)$;
      (iii) if $\theta_0 \leq \phi_1$, set $\theta = \theta_0$;
      (iv) if $\theta_0 > \phi_1$, set $\theta = \phi_2 + (\theta_0 - \phi_1)$.
7. If the number of intervals determined in Step 5 is zero, go to Step 13.
8. Find the length of the line segment implied by the angle generated in (6) (see Figure H.10).

<u>FOUR CASES</u>



I. $0 < \theta < \pi/2 \quad h \leq s$
$\tan(\theta) = h/f$, hence
$h = f \cdot \tan(\theta)$
$\ell = \sqrt{f^2 + h^2}$

II. $0 < \theta < \pi/2 \quad h > s$
$g/f = (h - s)/h$, hence
$g = f(h - s)/h$
(where $h = f \cdot \tan(\theta)$, as in I)
$\ell = \sqrt{(f - g)^2 + s^2}$

III. $\pi/2 < \theta < \pi \quad h > s$
$h = -(s - f)\tan(\theta)$
$g = (s - f)(h - s)/h$
$\ell = \sqrt{(s - f - g)^2 + s^2}$

IV. $\pi/2 < \theta < \pi \quad h \leq s$
$h = -(s - f)\tan(\theta)$
$\ell = \sqrt{(s - f)^2 + h^2}$

FIGURE H.10 LINE SEGMENT IMPLIED BY θ

9. Add the length found in (8) to LSUM.

10. If LSUM $\geq L_T$, go to Step 21.

11. If another A_0 has been encountered by the line segment, the point of intersection is the new starting point; identify the new A_0 and the wall on which the starting point occurs, and compute the distance between the starting point and the first corner of A_0 encountered when moving in a counterclockwise direction from it; then go to Step 5.

12. If a boundary of A_3 has been encountered by the line segment, the point of intersection is the new starting point. Record the total length generated in the current A_3, then move to the new A_3 thus encountered. Go to Step 3(c).

13. None of the sides of A_0 are accessible. Re-enter the previous A_0 (resetting the starting wall and starting-point-distance-to-corner for that square), and try again to set restrictions on the angle to be generated (as in Step 5), this time discounting the side which led us to a square having no accessible sides (see Figure H.11).

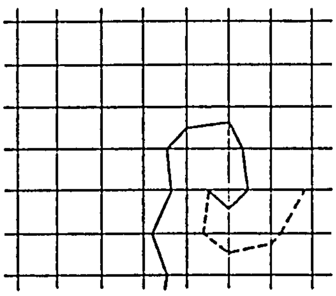


FIGURE H.11 BACKTRACKING, AFTER ENCOUNTERING A SQUARE (A_0) HAVING NO ACCESSIBLE SIDES

14. If the number of intervals for generation of θ is not zero, go to Step 6.

15. The simulated line has trapped itself: there are no adjacent A_0's which can be entered (see Figure H.12). In this case we can check the length generated so far in the current A_3, to see whether we might be able to reshape (redirect) the line and connect it to an unoccupied adjacent A_3 (see Figure H.13).

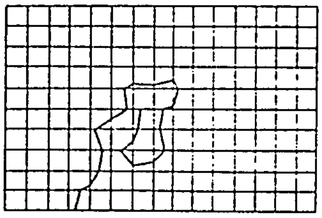
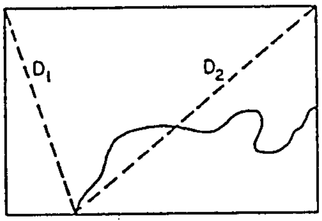


FIGURE H.12 A SIMULATED REDOX LINE WHICH HAS TRAPPED ITSELF



Example: diagonals D_1 and D_2 are drawn from the starting point to the opposite corners of A_3, as shown. If the sum of A_0 lengths in A_3 exceeds max $\{D_1, D_2\}$, then the redox line may be redirected to any point on any accessible wall of this A_3.

FIGURE H.13 RESHAPING THE LINE IN A_3

16. If at least one adjacent A_3 wall is accessible and the length already generated in the current A_3 is sufficient to reach it, select a new A_3 and starting point randomly from among those which can be reached from the starting point in the current A_3; then go to Step 3(c).

17. No accessible side of the current A_3 can be reached from the starting point for the A_3, given the length thus far generated in the A_3. Record the total length generated in this A_3.

18. Add together all those sides of A_3's which are on the border of A_F, provided the A_3 is not occupied.

19. If the sum produced in (18) is zero, i.e., all A_3's bordering A_F are occupied, reject this iteration: set all A_3 lengths to zero and return to Step 2.

20. Use the sum produced in (18) to select randomly a new A_3, starting wall, and starting point for the continuation (re-entry into A_F) of the redox line. The method of selection is similar to that found in Steps 3(a) and 3(b), but omitting the sides of occupied A_3's from consideration. Go to Step 3(c).

21. Truncate the length in the current A_3 so that the sum of A_3 lengths equals L_T; i.e., subtract from the last A_3 length the difference, LSUM - L_T.

22. Collect the required statistics on the A_3's. If the specified number of iterations for the simulation has not been attained, reset all A_3 lengths to zero and return to Step 1.

23. Compute the mean, variance, and histogram form of the probability distribution for length of redox to be found in an A_3.

NUMERICAL EXAMPLE OF A3GRID PROCEDURE

Assume we have A_F in the shape indicated in Figure H.1, input as in Figure H.5; the east and west walls (E and W) of A_3 are three miles in length, and walls N and S are each two miles long; a side of A_0 has length 0.1 mile, hence the grid of A_0's runs 30 squares vertically by 20 squares horizontally. The perimeter of A_F (Step 0) is computed to be 54 miles. Our reference point and direction for Step 3(b) will be the westernmost corner of the northernmost side, and clockwise. To begin we generate a redox length of L = 23.164 miles and set LSUM = 0.

Next we would like a starting point on the boundary of A_F. First, we generate a random number between 0 and 54 (the perimeter), say 30.44 miles. Then we follow the boundary away from the northwest corner in the clockwise direction until we have traversed 30.44 miles, to locate the starting point of the redox line in A_F. It can be seen in Figure H.14 that this point falls on the south wall of $A_3(5,4)$. If we apply the A_0 grid overlay to this A_3, we can see that the point falls on the south wall of $A_0(30,6)$, and its position is 0.04 miles from the southeast corner thereof (see Figure H.15).

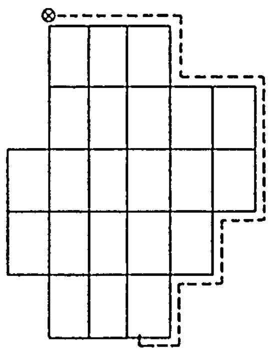


FIGURE H.14 STARTING POINT IN A_F

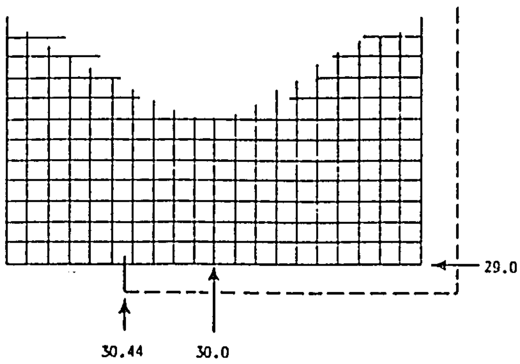
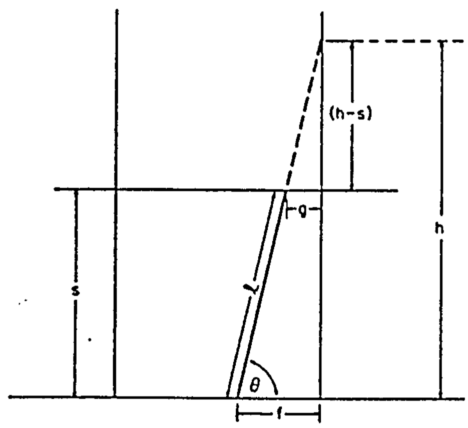


FIGURE H.15 STARTING POINT IN A_3

All A_0 lengths are set to zero. Since the adjacent squares, $A_0(30,5)$, $A_0(29,6)$, and $A_0(30,7)$, are all unoccupied, the interval for generation of θ is 0 to π(see Figure H.9, first case). Suppose we generate θ = 1.34 radians. This implies a line segment of length 0.1027 which intersects the north wall of A_0 (see Figure H.16). Now LSUM is augmented to 0.1027. The new starting point is located on the south wall and 0.0165 miles from the southeast corner of $A_0(29,6)$.

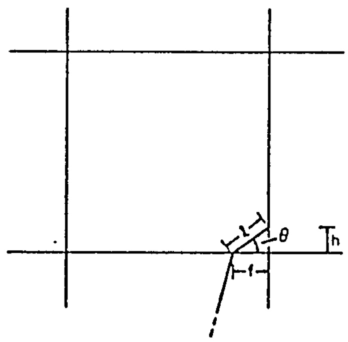


$A_0(30,6)$:

$s = 0.1$
$f = 0.04$
$\theta = 1.34$ (rad)
$h = 0.1702$ [$= f \cdot \tan(\theta)$]
$h-s = 0.0702$
$g = 0.0165$ [similar triangles]
$\ell = 0.1027$ [Pythagoras theorem]

FIGURE H.16 DETAIL ON $A_0(30,6)$

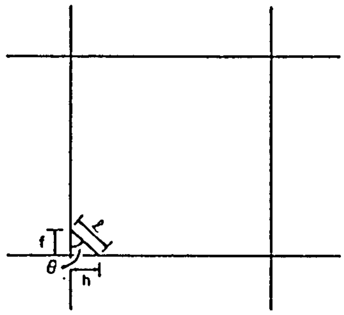
Again the interval in which θ may lie is $(0,\pi)$, and this time we generate $\theta = 0.62$ radians. This gives us a line segment of length 0.0203, intersecting the east wall (see Figure H.17). LSUM is augmented to 0.1230. We are now in $A_0(29,7)$, on the west wall, 0.0118 miles from the southwest corner. The restrictions on θ still give us the full range of $(0, \pi)$: let the sampled value for $\theta = 0.81$ radians. Now $\ell = 0.0171$ and the segment intersects the south wall (see Figure H.18). LSUM becomes 0.1401, and our new starting point is on the north wall, 0.0124 miles from the northwest corner, of $A_0(30,7)$.



$A_0(29,6)$:

$f = 0.0165$ [g, from previous square]
$\theta = 0.62$ (rad)
$h = 0.0118$
$\ell = 0.0203$

FIGURE H.17 DETAIL ON $A_0(29,6)$



$A_0(29,7)$:

$f = 0.0118$ (h, from previous square]
$\theta = 0.81$ (rad)
$h = 0.0124$
$\ell = 0.0171$

FIGURE H.18 DETAIL ON $A_0(29,7)$

Here we have a situation where θ is more restricted. The west wall of this A_0 is inaccessible because $A_0(30,6)$ is occupied; the south wall is inaccessible because it is a boundary of A_F. The east wall is accessible, however, because $A_0(30,8)$ is unoccupied. Thus, the line will inevitably be directed into that square, for which reason θ must fall between 2.2904 and π radians (see Figure H.19). We generate 2.65 radians. The line segment is 0.0994 miles in length, and intersects the east wall of A_0 (see Figure H.20). The new starting point lies on the west wall of $A_0(30,8)$, 0.0531 miles from the southwest corner. The accessible walls are N and E, hence θ must be generated on the interval (1.0828, π)(see Figure H.21).

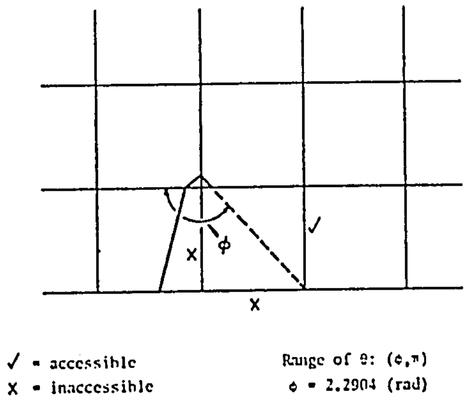


FIGURE H.19 RESTRICTIONS ON θ IN $A_0(30,7)$

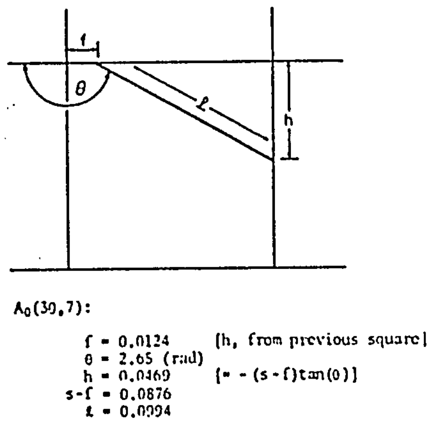


FIGURE H.20 DETAIL ON $A_0(30,7)$

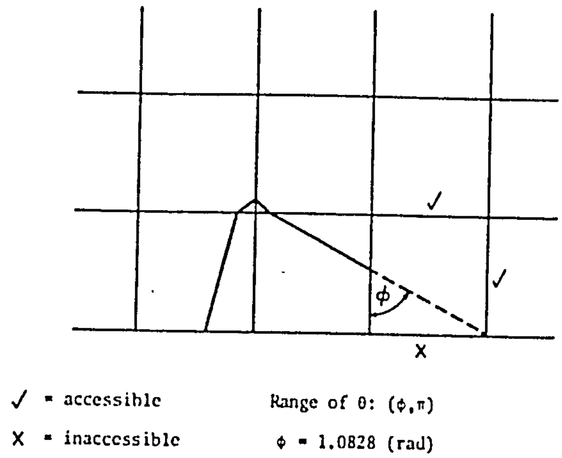


FIGURE H.21 RESTRICTIONS ON θ IN $A_0(30,8)$

As this process continues, LSUM becomes larger, approaching the redox length of 23.164 miles originally established for this iteration. Before reaching that length, the line will probably cross into $A_3(5,3)$ or $A_3(4,4)$, and may continue on to subsequent adjacent A_3's. When LSUM finally is greater than 23.164, the truncation (Step 21) occurs, and all A_3 lengths for this iteration will be entered into the frequency histogram.

APPENDIX I

A3SAMP ROUTINE

The prior joint probability distribution for redox length and uranium endowment in A_3, $b(E,L)$,[1] as well as the marginal and conditional distributions associated with it, are required for use in the exploration and executive models. These are generated external to the potential supply system in A3SAMP, which is a simulation program using the procedure in the creation submodel of the potential supply system (see chapter 7) and the results of A3GRID for sampling. The procedure for generating a redox line in A_F is similar to that already described in A3GRID, except that now the A_0 grid and piecewise segments are eliminated altogether. The A_3 lengths are generated from the distribution $\delta(L|L > 0)$, which was derived in A3GRID; their placement in successive A_3's is limited only by a few restrictions for the sake of plausibility.

Required as input to the system are the shape of A_F (as described in the introduction to A3GRID), length and width of A_3, distribution of A_3 lengths, $\delta(L|L > 0)$ (results of A3GRID), prior distribution for uranium endowment (see Section 8.2), distribution of redox length in the NURE region (see Section 7.2, Operation 1), and all input required for the creation submodel of the potential supply system. The procedure is as follows:

1. Generate an endowment E_T and a redox length L_T for A_F, from the input distributions.

2. Generate a sequence of pods and gaps matching the endowment and length generated in (1), using the procedure described in the creation submodel (see Section 7.2, Operation 2).

3. Use A3SAMP, the routine for sampling from the input distribution of A_3 lengths, to establish a sequence of A_3's through which the redox line moves, and the length in each A_3 of the sequence.

   (a) Set all A_3 lengths equal to zero.

   (b) Set LSUM equal to zero.

   (c) Randomly select a starting A_3 from among those in A_F.

   (d) Randomly select an A_3 length A3L from the input distribution, $\delta(L|L > 0)$.

   (e) Add A3L to LSUM.

   (f) Place the coordinates (subscripts) of the current A_3 on a list, so that the sequence of entry may be preserved.

   (g) If LSUM $\geq L_T$, go to (k).

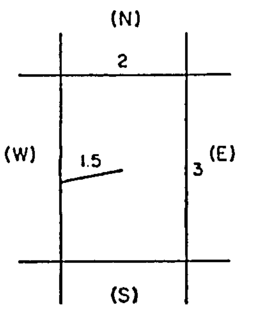
---

[1] E and L refer in this appendix to the endowment and redox length perceived by the geologist.

(h) Ascertain the number and identities of accessible walls of the current A_3. A wall of A_3 is accessible if and only if

(i) it is also a wall of another A_3 which is unoccupied;

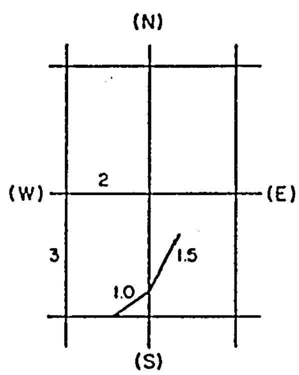
(ii) it is the wall opposite the starting wall, A3L matches or exceeds the length of the side between the two, i.e., A3L is sufficient to reach the other side of A_3 (see Figure I.1);



Example: if walls W and E are 3 miles in length, walls N and S are 2 miles long, the starting wall is W, and the length generated in $A_3(I,J)$ is 1.5 miles, then wall E is inaccessible, that is, we may not proceed to $A_3(I, J + 1)$.

FIGURE I.1 INACCESSIBLE WALL (E), CASE ii

(iii) it is the wall opposite the starting wall of the previous A_3. Then, the sum of A3L and the previous A_3 length must be greater than or equal to the length of the wall perpendicular to it, i.e., again, the summed lengths must be sufficient to reach the side in question (see Figure I.2).



Example: if the length in $A_3(I,J)$ is 1.5 miles, that in the previous A_3 is 1.0 miles, the previous starting wall was S, and the lengths of the sides of A_3 are as shown, then wall N is inaccessible: we may not proceed to $A_3(I-1, J)$.

FIGURE I.2 INACCESSIBLE WALL (N), CASE iii

(i) If the number of accessible sides is zero, randomly select a new A_3 from among those remaining unoccupied, and go to (d). [If no unoccupied A_3's remain, reject the experiment, returning to (a).]

(j) Choose randomly from among the accessible walls ascertained in (h); identify the new A_3 and starting wall thereof, then go to (d).

(k) Truncate the length in the last A_3 on the list so that the sum of A_3 lengths equals L_T; i.e., subtract from the last A_3 length the difference, $LSUM - L_T$.

4. Using the pod information collected in (2), follow the redox line through the sequence of A_3's and lengths generated in (3), to observe the joint occurrence of length and endowment in A_3.

5. Record these joint occurrences, so that the joint frequency distribution b(E, L) may be obtained. If the specified number of iterations for the simulation has not elapsed, return to Step 1.

APPENDIX J

MINING COST MODEL SCHEMATIC FLOW DIAGRAM

START

Inputs Required:

Economic, location and operation:

- uranium price/cost (P), $/lb.
- discount rate (cost of capital K, in agreement with the financial and risk parameters defining the firm assumed to be conducting uranium operations and the risk-cost of capital relationship, as discussed in the executive model, in chapter 10)
- cost indexes for the desired date of evaluation (a total of twelve). No inflation projections are considered.
- depletion rate (DPLRAT)
- state code
- land type code
- state income tax rate (STRTE)
- duration of the following pre-production activities and cost if appropriate (listed in order of precedence implicit in the model):
    1. Feasibility study and property consolidation. Include cost (TCFA).
    2. Permits and licences. Include cost (TCPL).
    3. Land acquisition. Include cost (LANDK).
    4. Development drilling. Cost generated in model.
    5. Mine development. Cost generated in model.
- mill construction indicator variable (MILL = 0 or 1). If no mill is constructed, input the milling charge, TOLMIL (toll milling assumed)
- pit slope angle (ϕ)
- working days per year (WD)

FIGURE J.1. MINING COST MODEL. INPUTS REQUIRED

Inputs required: (continued....)

Physical description of the geologic deposit:

- number of pods composing the deposit (NPODS)
- attributes of each pod:
  - 0.01-confining prism length, width, and thickness ($\lambda(.01)$, $w(.01)$, $th(.01)$) for each pod
  - q^*-confining prism length, width, and thickness ($\lambda(q^*)$, $w(q^*)$, $th(q^*)$) for each pod
  - grade tonnage curve (lognormal) parameters (μ, σ, d, T) for each pod
  - depth to the top of the 0.01-confining prism (DT01) for each pod
  - gap distance to next pod (along longitudinal axis) (gap(.01))
  - parameters of the "shrinkage function" (same for all pods)

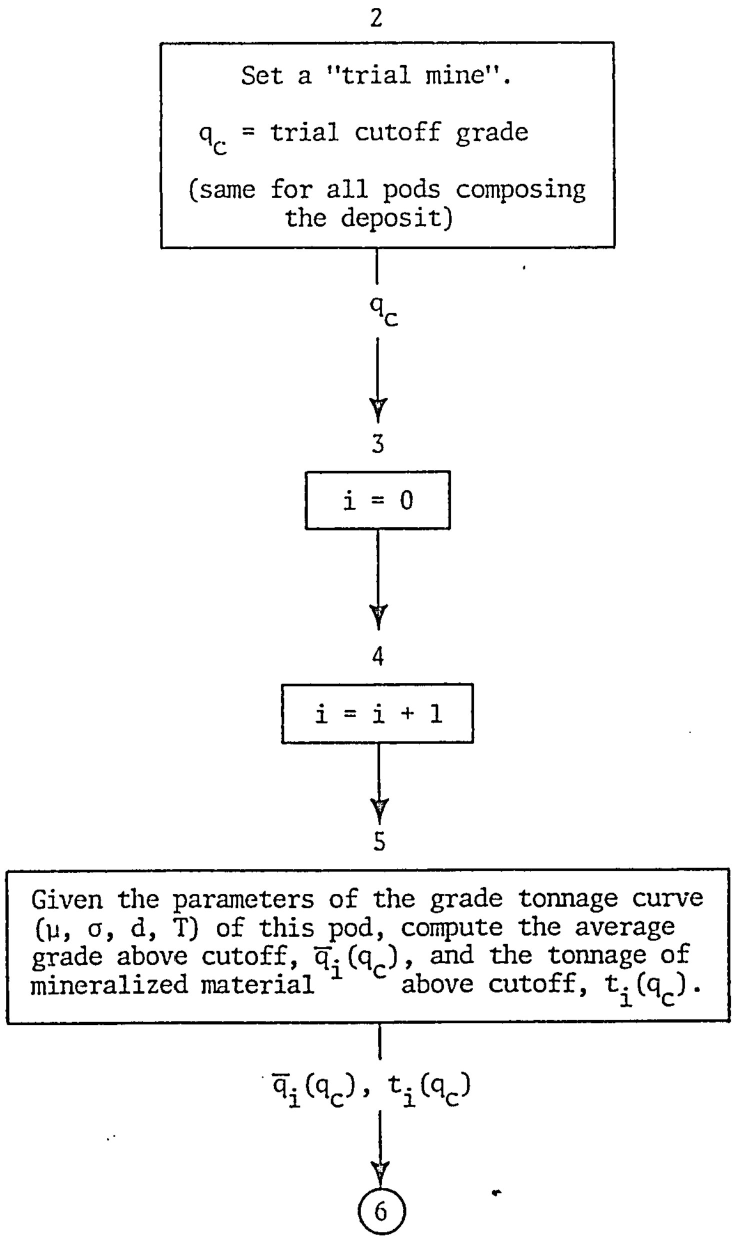


FIGURE J.2. MINING COST MODEL. COMPUTATION OF THE DIMENSIONS OF THE "ORE" DEPOSIT AND OTHER CHARACTERISTICS AT THE CURRENT TRIAL MINING CUTOFF GRADE, q_c

6

Employ the CONFINE routine to compute the length, width and thickness dimensions of the confining prism at a cutoff grade q_c for this pod: (routine described in App. A): $\lambda_i(q_c)$, $w_i(q_c)$, $th_i(q_c)$

$\lambda_i(q_c)$, $w_i(q_c)$, $th_i(q_c)$

7

Compute the depth to the top of the q_c-confining prism of this pod (DTQC):

$$DTQC_i = DT01_i + \tfrac{1}{2}(th_i(.01) - (th_i(q_c))$$

$DTQC_i$

8

i = NPODS ?

yes → 11

no

9

For each pod, except the last one, compute the gap distances to the q_c-confining prism of next pod, $gap_i(q_c)$

$$gap_i(q_c) = gap_i(.01) + \tfrac{1}{2}\left(\lambda_i(.01) - \lambda_i(q_c)\right) + \tfrac{1}{2}\left(\lambda_{i+1}(.01) - \lambda_{i+1}(q_c)\right)$$

$gap_i(q_c)$

10

10

i < NPODS ?

yes → 4

no

11

Compute the total tons in the deposit ($t(q_c)$) of mineralized material above the cutoff grade q_c:

$$t(q_c) = \sum_{i=1}^{NPODS} t_i(q_c)$$

12

Compute the average grade above cutoff for the entire deposit ($\bar{q}(q_c)$):

$$\bar{q}(q_c) = \sum_{i=1}^{NPODS} \frac{t_i(q_c)}{t(q_c)} \cdot \bar{q}_i(q_c)$$

13

13

Compute total area (PVA) of the "ore" deposit's projection (i.e. at cutoff grade q_c) onto a horizontal plane:

"gap area" (GA):

$$GA = \sum_{i=1}^{NPODS-1} \tfrac{1}{2}\left(w_i(q_c) + w_{i+1}(q_c)\right) gap_i(q_c)$$

"pod area" (PA):

$$PA = \sum_{i=1}^{NPODS} \lambda_i(q_c) \cdot w_i(q_c)$$

$$PVA = GA + PA$$

14

Compute the semimajor (A_B) and semiminor (B_B) axis of the ellipse representing the bottom section of the inverted truncated cone (of elliptical section) that simulates the ultimate pit needed for the extraction of the "ore" deposit at cutoff grade q_c (i.e. the extraction of all q_c-confining prisms):

$$A_B = \tfrac{1}{2} \sum_{i=1}^{NPODS} \lambda_i(q_c) + \tfrac{1}{2} \sum_{i=1}^{NPODS-1} gap_i(q_c)$$

$$B_B = PVA/(A_B \cdot 3.1416)$$

15

FIGURE J.3. MINING COST MODEL. COMPUTATION OF PIT GEOMETRY, VOLUMES AND TONNAGES OF MATERIALS NEEDED TO BE REMOVED

15

Compute the average depth (weighted average, using the areal projection as weights) to the bottom of the "ore" deposit (d_B) and to the top of the "ore" deposit (d_T)

$$d_T' = \sum_{i=1}^{NPODS} \left(\frac{\lambda_i(q_c) \cdot w_i(q_c)}{PVA} \right) \cdot DTQC_i$$

$$d_T'' = \sum_{i=1}^{NPODS-1} \left(\frac{0.5(w_i(q_c) + (w_{i+1}(q_c)) \cdot gap_i(q_c)}{PVA} \right) \cdot \left(\frac{DTQC_i + DTQC_{i+1}}{2} \right)$$

$$d_T = d_T' + d_T''$$

$$d_B' = \sum_{i=1}^{NPODS} \left(\frac{\lambda_i(q_c) \cdot w_i(q_c)}{PVA} \right) \cdot \left(DTQC_i + th_i(q_c) \right)$$

$$d_B'' = \sum_{i=1}^{NPODS-1} \left(\frac{0.5(w_i(q_c) + w_{i+1}(q_c)) \cdot gap_i(q_c)}{PVA} \right) \cdot \left(\frac{DTQC_i + th_i(q_c) + DTQC_{i+1} + th_{i+1}(q_c)}{2} \right)$$

$$d_B = d_B' + d_B''$$

d_B, d_T

16

16

Compute the semimajor (A_T) and semiminor (B_T) axis of the ellipse representing the top section of the inverted truncated cone (of elliptical section) that simulates the ultimate pit needed for the extraction of the deposit at cutoff grade q_c.

$$A_T = (d_B + A_B \cdot \tan\phi)/\tan\phi$$

$$B_T = (d_B + B_B \cdot \tan\phi)/\tan\phi$$

where ϕ = pit slope angle

A_T, B_T

17

17

Compute the semiaxis of the ellipses representing the top and bottom sections of the "preproduction" cone" (an internal cone inside the ultimate cone; it is also an inverted truncated cone of elliptical section) that simulates the preproduction stripping pit needed to reach the top of the "ore" deposit.

bottom ellipse (keeping the same shape as the bottom ellipse of the ultimate cone)

$$b_B = \sqrt{\frac{\alpha_{min}}{\frac{A_B}{B_B} \cdot 3.1416}}$$

where α_{min} = minimum preproduction pit floor area

$$a_B = \frac{A_B}{B_B} \cdot b_B$$

top ellipse

$$a_T = (d_T + a_B \cdot \tan\phi)/\tan\phi$$

$$b_T = (d_T + b_B \cdot \tan\phi)/\tan\phi$$

where ϕ = pit slope angle

a_B, b_B, a_T, b_T

18

Compute volumes of materials to be removed:

total confining prism volume (all pods) at cutoff grade q_c

$$V(q_c) = \sum_{i=1}^{NPODS} \lambda_i(q_c) \cdot w_i(q_c) \cdot th_i(q_c)$$

volume of ultimate cone

$$v = \frac{d_B}{3}\left(3.1416 \cdot A_B \cdot B_B + 3.1416 \cdot A_T \cdot B_T + (3.1416^2 \cdot A_B \cdot B_B \cdot A_T \cdot B_T)^{\frac{1}{2}}\right)$$

volume of preproduction cone

$$v_S = \frac{d_T}{3}\left(3.1416 \cdot a_B \cdot b_B + 3.1416 \cdot a_T \cdot b_T + (3.1416^2 \cdot a_B \cdot b_B \cdot a_T \cdot b_T)^{\frac{1}{2}}\right)$$

volume of intradeposit waste

$$v_W = V(q_c) - \frac{t(q_c)}{TF}$$

where TF = tonnage factor

volume of overburden to be stripped during production

$$v_U = v - v_S - V(q_c)$$

v_S, v_U, v_W

19

19

Compute tonnages of materials to be removed:

preproduction stripping material

$$S = \frac{V_S}{TF}$$

where TF = tonnage factor

stripping material removed during production

$$U = \frac{V_U}{TF}$$

internal waste material

$$W = \frac{V_W}{TF}$$

S, U, W

20

20

Using the size of ore deposit - production rate relationship, obtain the daily ore production rate (TOPD).

$$TOPD = 507.58378 \left[\frac{t(q_c)}{1 \times 10^6} \right]^{0.68298945}$$

TOPD

21

Obtain the life of the operation in years.

$$Y = t(q_c)/(TOPD \cdot \text{number of working days per year})$$

Y

22

Compute the rate of preproduction stripping material removal, TSPD.

$$TSPD = S/\text{duration of mine development activities}$$

23

Compute the rate of stripping material removal during production.

$$TUPD = U/(Y \cdot \text{number of working days per year})$$

24

Compute the rate of removal of internal waste material.

$$TWPD = W/(Y \cdot \text{number of working days per year})$$

25

FIGURE J.4. MINING COST MODEL. RATES OF MATERIALS REMOVAL AND LIFE OF THE OPERATION COMPUTATION.

25

Compute the rate of removal of ore and internal waste.

$$TOWPD = TOPD + TWPD$$

26

Compute the haulage distances to the waste dump, H_U, and the ore pile, H_O, by assuming an 8% maximum grade. Assume 3048 ft. level road (1 kilometer) for the waste dump and 4572 ft. (1.5 kilometers) for the ore pile:

$$H_U = \frac{d_T}{.08} + 3048'$$

$$H_O = \frac{d_B}{.08} + 4572'$$

The average haulage distance for ore and internal waste, H_{OW}, is:

$$H_{OW} = \frac{TWPD}{TOWPD} \cdot H_U + \frac{TOPD}{TOWPD} \cdot H_O$$

27

27

In this open pit mining cost model, there are three distinct material removal operations for costing purposes:

(1) Preproduction stripping material

(2) Stripping of overburden material during production.

(3) Removal of ore and internal waste (i.e. removal of the ore's confining prisms).

For the purpose of equipment selection, the equipment used for each one of these operations is classified into two major categories:

(1) Drilling and blasting equipment (DB)

(2) Excavating, loading and hauling equipment (LH)

The type of equipment to be used for each category in each of the material removal operations depends on the rate of material removal. The equipment selection is performed according to the following table based on STRAAM handbook:

	Drilling and blasting		Excavating, loading and hauling	
	Crawler drills $DB_x = 1$	Rotary drills $DB_x = 2$	Front end loaders and trucks ($LH_x = 1$)	Electric shovels and trucks ($LH_x = 2$)
Preproduction stripping material ($x = S$)	TSPD < 9000 m.t.	TSPD ≥ 9000	TSPD < 9000	TSPD ≥ 9000
Stripping material during production ($x = U$)	TUPD < 9000	TUPD ≥ 9000	TUPD < 9000	TUPD ≥ 9000
Ore and internal waste material ($x = OW$)	TOWPD < 9000	TPWPD ≥ 9000	TOWPD < 9000	TOWPD ≥ 9000

28

FIGURE J.5. MINING COST MODEL. EQUIPMENT SELECTION

28

Direct cost of ore and internal waste removal (DCOWPD) (dollars/day):

Given the type of equipment to be used for this material removal operation, the haulage distance H_{OW}, the depth to the ore deposit (d_B) and the rate TOWPD, compute DCOWPD using "STRAAM" relationships (see section 9.3.6.1).

29

Direct cost of overburden removal during production (DCUPD) (dollars/day):

Given the type of equipment to be used for this material removal operation, the haulage distance to the waste dump (H_U), the depth to the top of the ore deposit (d_T) and the daily rate of material removal (TUPD), compute DCUPD using "STRAAM" relationships (section 9.3.6.2 in chapter 9).

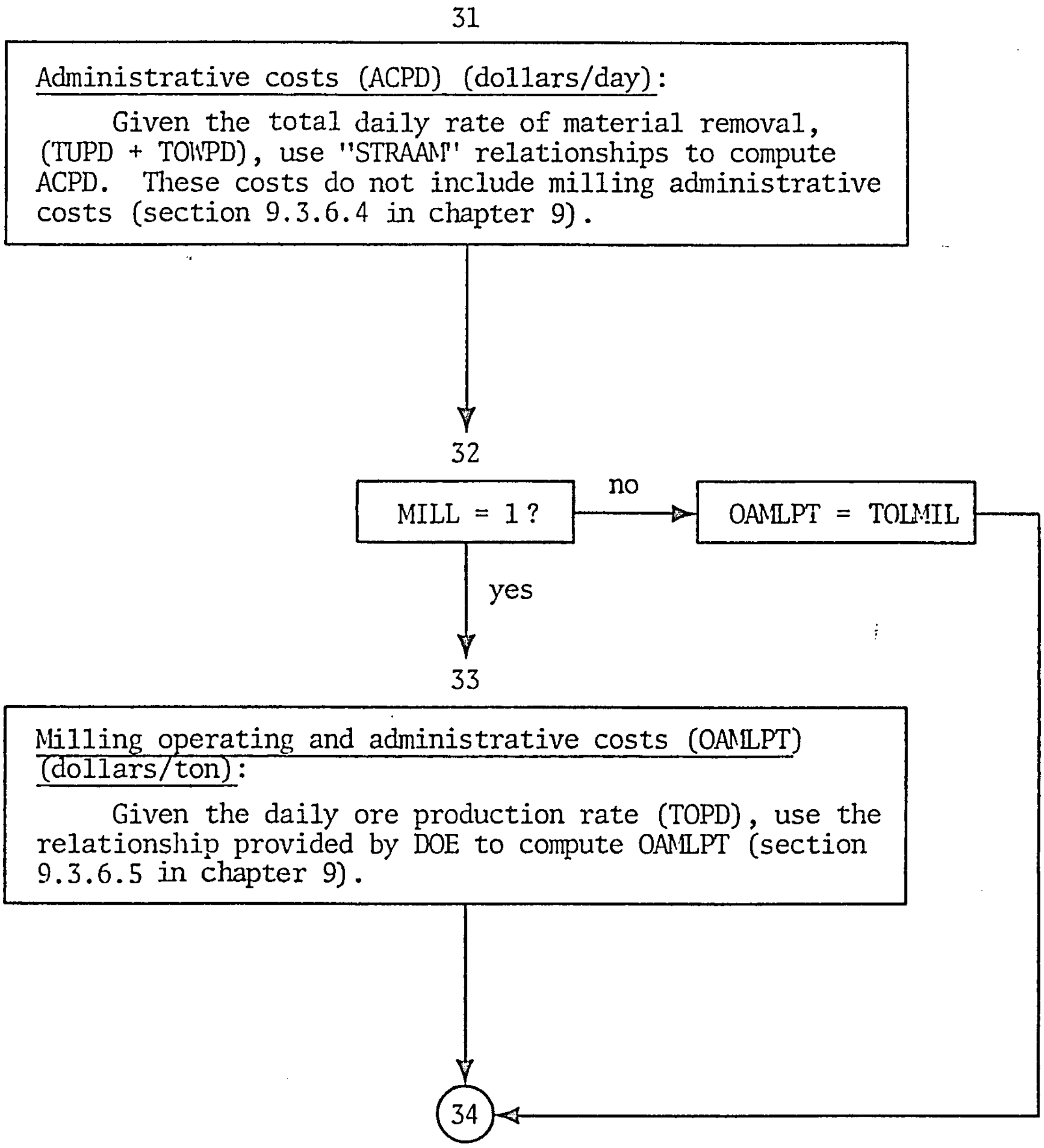
30

Other direct costs (DCHPD) (dollars/day):

Given the total daily rate of material removal (TUPD + TOWPD), use "STRAAM" relationships to compute DCHPD (clearing cost, general items, water supply....) (section 9.3.6.3 in chapter 9).

31

FIGURE J.6. MINING COST MODEL. OPERATING COSTS DETERMINATION


31
Administrative costs (ACPD) (dollars/day):
Given the total daily rate of material removal, (TUPD + TOWPD), use "STRAAM" relationships to compute ACPD. These costs do not include milling administrative costs (section 9.3.6.4 in chapter 9).
32
MILL = 1 ?
no
OAMLPT = TOLMIL
yes
33
Milling operating and administrative costs (OAMLPT) (dollars/ton):
Given the daily ore production rate (TOPD), use the relationship provided by DOE to compute OAMLPT (section 9.3.6.5 in chapter 9).
34

Development Drilling Cost (TCDD):

Assume that the "ore" deposit will be drilled at the intersection of a 50' x 50' square grid. Employ the total area of the "ore" deposit's projection onto a horizontal plane (PVA) to approximate the number of holes required by the following expression:

$$n_{DD} = \frac{PVA}{50' \times 50'}$$

Assuming that the drilling depths will be to the bottom of the geologic deposit, compute the weighted average drilling depth ($\bar{h}_{DD}$). Use the area of each pod and gap as weights:

$$\bar{h}_{DD}' = \sum_{i=1}^{NPODS} \frac{\lambda_i(q_c) \cdot w_i(q_c)}{PVA} (DT01_i + th_i(.01))$$

$$\bar{h}_{DD}'' = \sum_{i=1}^{NPODS-1} \frac{\frac{1}{2}(w_i(q_c) + w_{i+1}(q_c)) \cdot gap_i(q_c)}{PVA} \cdot \frac{1}{2}\left(DT01_i + th_i(.01) + DT01_{i+1} + th_{i+1}(.01)\right)$$

$$\bar{h}_{DD} = \bar{h}_{DD}' + \bar{h}_{DD}''$$

Employ the drilling cost relationship (see description in section 8.12) to obtain the cost per ft. drilled at a depth $\bar{h}_{DD}$. Total cost of development drilling (TCDD) is given by:

$$TCDD = \text{cost per ft} \times \bar{h}_{DD} \times n_{DD}$$

35

FIGURE J.7. MINING COST MODEL. CAPITAL COSTS DETERMINATION.

35

Preproduction stripping:

Given TSPD and assuming that prepoduction stripping operations are subcontracted, compute the total cost of preproduction stripping, TCSTRP, using "STRAAM" relationships (see section 9.3.7.2 in chapter 9).

36

Mine plant and buildings:

Given TOWPD + TUPD, compute the mine plant and buildings cost, TCMPD, using "STRAM" relationships (see section 9.3.7.3 in chapter 9).

37

Restoration during construction:

Assuming an area to be restored equal to the area of the surface section of the inverted truncated cone that simulates the ultimate pit for this mine, compute the cost of restoration during construction, TCREST, using "STRAAM" cost figure of $8750/hectare (1975 dollars).

38

MILL = 1 ?

no → TCMILL = 0 → 40

MILL is the indicator variable for the construction of a mill.

yes

39

Mill:

Given the ore production rate, TOPD, compute the capital cost of a mill, TCMILL, using the relationship provided by DOE (see section 9.3.7.4 in chapter 9).

40

40

<u>Equipment cost</u>:

Given the equipment type to be used during the production period and the rates TOWPD and TUPD compute the cost of the equipment for overburden removal (EQKU) and ore and internal waste removal (EQKOW) using "STRAAM" relationships (see section 9.3.7.5 in chapter 9).

$$TOTEQK = EQKU + EQKOW$$

41

<u>Engineering and construction management fee</u>:

Given the costs of production stripping (TCSTRP), mine plant and buildings (TCMPB), restoration during construction (TCREST) and equipment (TOTEQK), compute the engineering and construction management fee (TCECMF) using "STRAAM" relationships (see section 9.3.7.2 in chapter 9).

42

<u>Working capital (60 days) (WKCAP)</u>:

Given all the operating costs (DCOWPD, DCUPD, DCHPD, ACPD, OAMLPT) compute the cost of 60 days operation, WKCAP.

$$WKCAP = (DCOWPD + DCUPD + DCHPD + ACPD + OAMLPT + TOPD) \cdot 60$$

43

43

Given the duration of the activities in the following list, compute the beginning and finishing date for each.

Beginning of calendar is activity #1. The name of the variable for the total cost of the activity is given in parenthesis.

1. Feasibility study and property consolidation (TCFA)
2. Permits and licenses (TCPL)
3. Land acquisition (LANDK)
4. Development Drilling (TCDD)
5. Mine development (TCMD) (33% overlap with previous activity). Includes:
   - Pre-production stripping (TCSTRP)
   - Mine plant and building construction (TCMPB)
   - Restoration during construction (TCREST)
   - Mill construction (CMILL)
   - Engineering construction and management fees (TCECMF)

   TCMD = TCSTRP + TCMPB + TCREST + CMILL + TCECMF
6. Production (duration of production is the mine life)

$$B_1 = 0$$

$$F_i = B_i + D_i, \ \forall i$$

$$B_i = F_{i-1} - \frac{\alpha_i}{100} \cdot D_{i-1}, \quad i > 1$$

where B_i = beginning date of the i^{th} activity

F_i = finishing date of the i^{th} activity

D_i = duration of the i^{th} activity

α_i = % overlap of the i^{th} activity with the $(i-1)^{th}$ activity

44

FIGURE J.8. MINING COST MODEL. PREPRODUCTION ACTIVITIES CALENDAR

44

The timing of the following activities is specified using the beginning of mine production as a reference:

7. Mine equipment purchase

   Instantaneous disbursement (i.e. $B_7 = F_7$) assumed to occur 6 months before production begins

8. Working capital disbursement

   Instantaneous disbursement (i.e. $B_8 = F_8$) assumed to occur at the time of beginning production

↓

45

45

For the purpose of equipment replacement and depreciation, the equipment used for each of the two material removal operations (overburden removal and ore + internal waste removal) is divided into three categories:

(1) excavating and loading equipment

(2) hauling equipment

(3) drilling and blasting equipment

Given the physical life assumed for each equipment category (according to the type of equipment selected, i.e. crawler drills, rotary drills, front end loaders, etc.), compute the number of purchases ($NPUR_{ij}$) needed to meet the life of the mine. Life of the last purchase is adjusted to meet the required mine life.

$NPUR_{ij}$ = number of purchases required for the i^{th} equipment category ($i = 1, 3$) used in the j^{th} material removal operation ($j = 1, 2$)

46

FIGURE J.9. MINING COST MODEL. PREPARATION FOR CASH FLOW ANALYSIS.

46

Compute the cost of each equipment category in each material removal operation, $FRAC_{ij}$, employing "STRAAM's" proportion of each equipment category according to the equipment selected (section 9.3.9 in chapter 9).

47

Compute the mill recovery using the mill recovery-average grade relationship provided by DOE.

$$RMILL = \frac{1}{100}\left(95.2955 - \left(\frac{0.629347}{\overline{q}(q_c)}\right)\right)$$

48

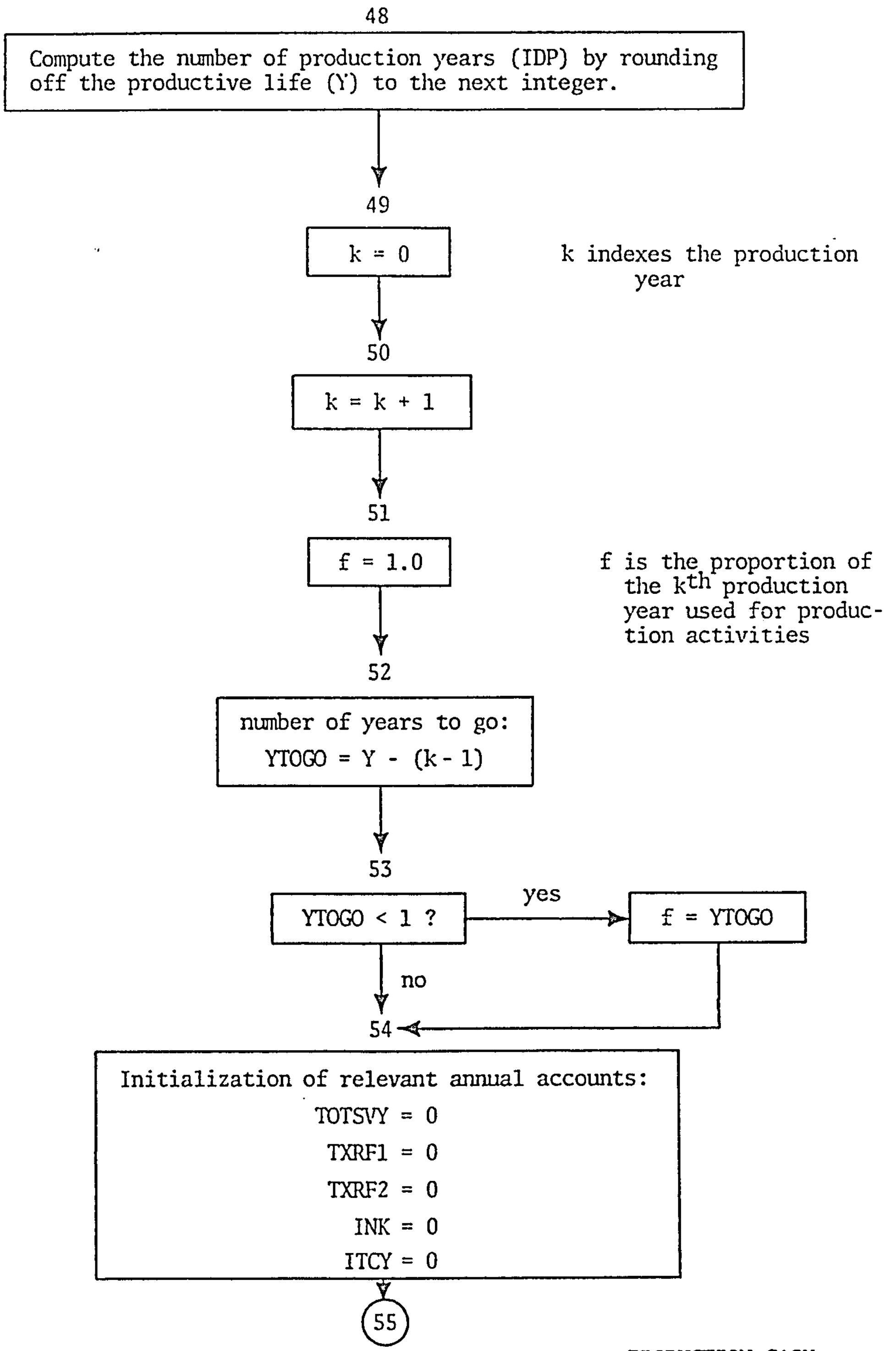


FIGURE J.10. MINING COST MODEL. GENERATION OF THE PRODUCTION CASH FLOW STREAM

55

Annual Revenue (TRVY)

$$TRVY = (TOPD * WD)\,\frac{\bar{q}(q_c)}{100} \cdot 2205 \cdot RMILL \cdot P \cdot f$$

56

Annual Royalty Payments (RYY)

Determine the royalty payment (RYPTO) according to the land type, state, and average grade as pertinent.

$$RYY = RYPTO \cdot (TOPD * WD) \cdot f$$

57

Gross Income for Mining (GIFMY)

$$GIFMY = TRVY - RYY$$

58

Annual Mining Operating Cost (TOCY)

$$TOCY = WD \cdot (DCOWPD + DCUPD + DCHPD + ACPD) \cdot f$$

59

Annual Milling Operating Cost (OAMLY)

$$OAMLY = OAMLPT * TOPD * WD \cdot f$$

60

Annual Gross Profit (GPY)

$$GPY = GIFMY - TOCY + OAMLY$$

61

61

Annual Depreciation (other than equipment) (DPROY):

Assume straight line depreciation, life equal to the life of the mine and no salvage value:

$$DPROY = \frac{1}{IDP}(TCDD + TCSTRP + TCREST + TCECMF + TCPL + TCMPB + CMILL) \cdot f$$

62

Annual Equipment Depreciation (DPREQY):

$$DP_{ij} = \frac{FRAC_{ij} - SVCEV_{ij}}{LFDP_{ij}} \cdot f$$

where:

DP_{ij} = annual depreciation of the i^{th} equipment category (i = 1, 3) used in the j^{th} material removal operation (j = 1, 2)

$FRAC_{ij}$ = cost of equipment i j

$SVGEV_{ij}$ = salvage value of equipment i j (assumed to be equal to the undepreciated value. In most cases it is equal to the "scrap" value except when equipment is sold before completing the "depreciation life" LFDP.

$LFDP_{ij}$ = depreciation life. Usually it is equal to the physical life, except when the life has to be "shortened" or "lengthened" to meet the mine life. In any case, LFDP is within the asset depreciation range specified by U.S. tax laws. LFDP is used for tax deduction purposes.

$$DPREQY = \sum_{i=1}^{3} \sum_{j=1}^{2} DP_{ij}$$

63

63

Total annual depreciation

DPRY = DPREQY + DPROY

64

Is this a year when any equipment should be salvaged?

no

Equipment is salvaged when it is replaced or when productive life of the mine terminates

yes

65

Add the salvage value of the appropriate equipment category(ies) to the total annual salvage value.

$$TOTSVY = SVGEV_{ij}$$

66

Net income before taxes and depletion

NIBTDY = GPY - DPRY

67

Annual Depletion Charge (DPLY)

Assume that percentage depletion will always be greater than unit depletion (i.e. percentage depletion will always be taken)

a = NIBTDY

if NIBTDY < 0 $\longrightarrow$ a = 0

$$DPLY = \min\left\{0.5 \cdot a \; , \; GIFMY \cdot \frac{DPLRAT}{100}\right\}$$

68

68

Annual severance tax (SVTXY)

According to the state, determine the severance tax (tax base can be gross value, gross value at the mine, net profits, etc., depending on the state).

69

Annual state income tax (STITY)

$$TI = NIBTDY - SVTXY$$

$$STITY = 0$$

$$\text{if } (TI > 0) \longrightarrow STITY = \frac{STRTE}{100} \cdot TI$$

70

Annual federal income tax (FITY)

$$FTI = NIBTDY - DPLY - STITY - SVTXY$$

$$\text{If } (FTI \leq 0) \longrightarrow FITY = 0$$

$$\text{If } (FTI \leq 25000) \longrightarrow FITY = 0.20 \cdot FTI$$

$$\text{If } (25000 < FTI \leq 50000) \longrightarrow FITY = 0.20 \cdot 25000 + 0.22(FTI - 25000)$$

$$\text{If } (FTI > 50000) \longrightarrow FITY = 0.42 \cdot 25000 + 0.48(FTI - 50000)$$

71

71

FTI < 0 ?

no → 73

yes

72

Carry back operating losses for this year (maximum of 3 years) and obtain a tax refund (if possible) (TXRF1).

Carry the portion of the loss not used to offset past income forward to be applied against positive earnings in the next five years.

→ 74

73

Check for any operating losses being carried forward and recompute the federal income taxes (FITY) if necessary.

74

Annual Replacement Capital (INK)

The cash flow model checks if any replacement equipment will be purchased this year (to be put into use next year). Value of all equipment purchased in the year is stored in a variable named INK.

75

Investment tax credit (ITCY):

If any new equipment was put in service this year compute the portion of the cost which qualifies for investment tax credit.

$$QI = \alpha \cdot FRAC$$

where: QI = qualifying amount for investment tax credit

FRAC = cost of the equipment

α = fraction of the cost of the equipment eligible for investment tax credit (determined by the IRS according to the physical life of the asset)

Physical Life	α
> 7 years	1.0
5 - 7 years	2/3
3 - 5 years	1/3
< 3 years	0

The total investment tax credit (ITCY) is equal to 10% of the sum of the qualifying investments for the year (QI).

76

Carry back and carryforward of investment tax credits:

Deduct the investment tax credit (ITCY) from the tax liability (FITY) and compute the net federal income tax (NFITY). Note the following maximum-credit limitations:

$$\text{If } (\text{FITY} \leq 25000) \longrightarrow \text{ITCY} \leq \text{FITY}$$

$$\text{If } (\text{FITY} > 25000) \longrightarrow \text{ITCY} \leq 25000 + \tfrac{1}{2}(\text{FITY} - 25000)$$

In cases where the investment tax credit cannot be fully applied in a given year because of these maximum-credit limitations, the remaining investment is carried back (maximum of 3 years) to obtain a tax refund (TXRF2). The portion of the investment tax credit not used to offset past tax liabilities is carried forward to be applied against tax liabilities in the next five years.

The cash flow model checks, in every year, for any investment tax credits being carried forward and adjusts the net federal income tax (NFIYT) if necessary.

77

77

Annual net profit

PRFNET = GPY - DPRY - DPLY - STITY - SVTXY - NFITY

78

Annual cash flow

CF_k = PRFNET + DPRY + DPLY - INK + TOTSVY + TXRF1 + TXRF2

79

k = IDP ?

(i.e. last production year)

no → 50

yes → 80

Present value of preproduction investment (PVI)

Given the percent discount rate (cost of capital K) desired, the beginning date, finishing dates and the total cost of each investment activity, the following discounting procedure is applied (assume uniform flow and continuous compounding).

Let $D_i = F_i - B_i$

where D_i = duration of the i^{th} preproduction investment activity

F_i = finishing date of the i^{th} preproduction investment activity

B_i = beginning date of the i^{th} preproduction investment activity

$$r = \ln\left(\frac{K}{100} + 1\right)$$

where r = force of interest (i.e. the nominal rate in decimal form compounded continuously to yield the effective rate K)

$$PA = \frac{e^{r \cdot D_i}}{r \cdot D_i e^{r \cdot D_i}}$$

where PA = present worth of \$1 (at time B_i) flowing uniformly throughout the duration of the activity (set to 1.0 if $D_i = 0$)

$$PV_i = PA \cdot \left(\frac{1}{\left(1 + \frac{K}{100}\right)^{B_i}}\right) \cdot TC_i$$

where TC_i = total cost of the i^{th} preproduction investment activity

PV_i = present value of the i^{th} preproduction investment activity

Finally: $PVI = \sum_i PV_i$

81

FIGURE J.11. MINING COST MODEL. NET PRESENT VALUE COMPUTATION

81

<u>Present value of the production cash flow stream</u> (PVCFP)

Given the discount rate (cost of capital K) desired and the k^{th} production year, the following discounting procedure is applied (assumes uniform flow and continuous compounding):

Let $BCF_k = B_6 + k - 1$

$FCF_k = B_b + k$

where BCF_k = beginning date of the k^{th} production cash flow

FCF_k = finishing date of the k^{th} production cash flow

k = indexes the production year

B_6 = beginning date of production activity

The computation of the present value for each cash flow (CFO_k) is the same as the one employed in the computation of the pre-production investment present values. The present value of all production cash flows is:

$$PVCFP = \sum_k CFO_k$$

82

82

Net present value for this trial mine

NPV = PVCFP - PVI

83

Optimum value (net present value maximization criteria) found?

no → 2

Numerical optimization routine keeps setting trial values for cutoff grade until the optimum is found.

yes

84

q_c^*, the optimum cutoff grade has been determined. Also, the following important measures are determined:

$t(q_c^*)$ = tonnage of mineralized material above optimum cutoff grade

$\bar{q}(q_c^*)$ = average grade of mineralized material above cutoff grade

Y^* = mine life

NPV^* = maximum net present value

85

END

APPENDIX K

EXECUTIVE MODEL SCHEMATIC FLOW DIAGRAMS

START

Invoke the PRE-EXPLORATION MODULE to obtain:

(a) the optimum level of effort of next exploration activity (ex*).

(b) Φ(NPVX), the frequency distribution of net present value (including remaining exploration costs). This distribution represents the "prize" size in the geographic entity being explored and is jointly determined with (a) above. The discount rate used is K, the current cost of capital of the (typical) firm.

Note: ex* is a generic name to indicate the optimum number of holes (or fences) to drill next, according to the current state in the exploration model.

Φ(NPVX), ex*

Invoke the DECISION MODULE to decide whether or not to adopt the "uranium project" (i.e., to continue or to stop exploring the geographic entity under investigation).

If decision is to continue, ex* is left unchanged.

If decision is to stop, ex* is set to zero. This signals the exploration model to quit exploring that geographic entity.

END

FIGURE K.1 EXECUTIVE MODEL. DIAGRAM SHOWING THE CALLING SEQUENCE OF THE TWO MAJOR COMPONENTS OF THE EXECUTIVE MODEL.

START

INPUTS REQUIRED BY THE MODULE:

- Economic related inputs: current price of uranium in the system, lead times, current cost of capital (K), cost indexes for date of evaluation, risk and financial characteristics of the firm (typical) assumed to be conducting the exploration, state, state tax rate.

- Responses to the deposit morphology questionnaire (Appendix E).

- Responses to the exploration performance questionnaire (Appendix F).

- $a(U|R)$, the conditional distribution of endowment in an A_3, given redox length in an A_3. Details on this distribution are given in section 8.3.

- $\varepsilon(R)$, the distribution representing the current redox length expectations (in the A_3 being investigated) in the exploration model.

- Minimum and maximum depth to the favorable formation in the A_3 being investigated ($DEPTH_{min}$ and $DEPTH_{max}$).

- PHASE, the current phase in the exploration model and the value for S, the success/failure indicator variable in the current phase.

- The exploration effort (number of holes, n_c, n_R , as pertinent) spent so far in the exploration activities as simulated by the exploration model.

FIGURE K.2 PRE-EXPLORATION MODULE. CASE 1: CONSULTATION OCCURS WHEN EXPLORATION MODEL IS AT END OF PHASE 2, OR DURING PHASE 3A OR PHASE 3B

PERTINENT DECISION VARIABLES AND THEIR CONSTRAINTS

Current State in the Exploration Model	Pertinent Decision Variables	Constraints
If PHASE = 2 (end of phase)	$h_{3A}, h_{3B}, r_{4A}, h_{4B}, h_5$	
If PHASE = 3A and S = 0	$h_{3A}, h_{3B}, r_{4A}, h_{4B}, h_5$	$h_{3A} \geq n_C$
If PHASE = 3A and S = 1	$h_{3B}, r_{4A}, h_{4B}, h_5$	
If PHASE = 3B and S = 0	$h_{3B}, r_{4A}, h_{4B}, h_5$	$h_{3B} \geq n_R$
If PHASE = 3B and S = 1	r_{4A}, h_{4B}, h_5	

h_x = number of holes drilled in Phase x in the pre-exploration module.

r_{4A} = number of fences drilled in Phase 4A in the pre-exploration module (for the entire A_3 region).

Note: "h" is used instead of the customary "n" for number of holes to make clear the difference between the pre-exploration module aiding the executive model and the exploration model in the main stream of the potential supply system. The same is true for the usage of "r" instead of "f" for number of fences.

1

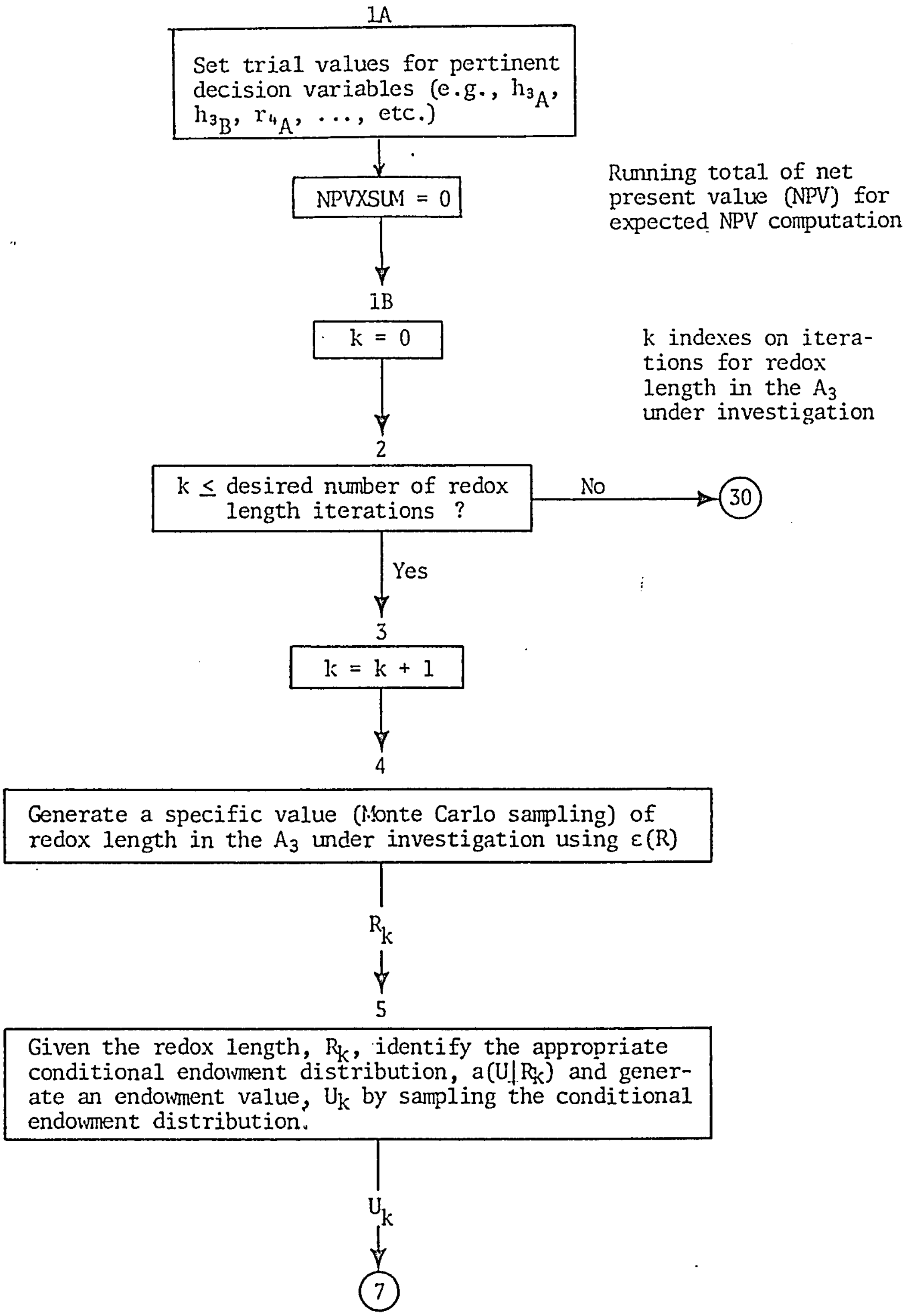
$\downarrow$

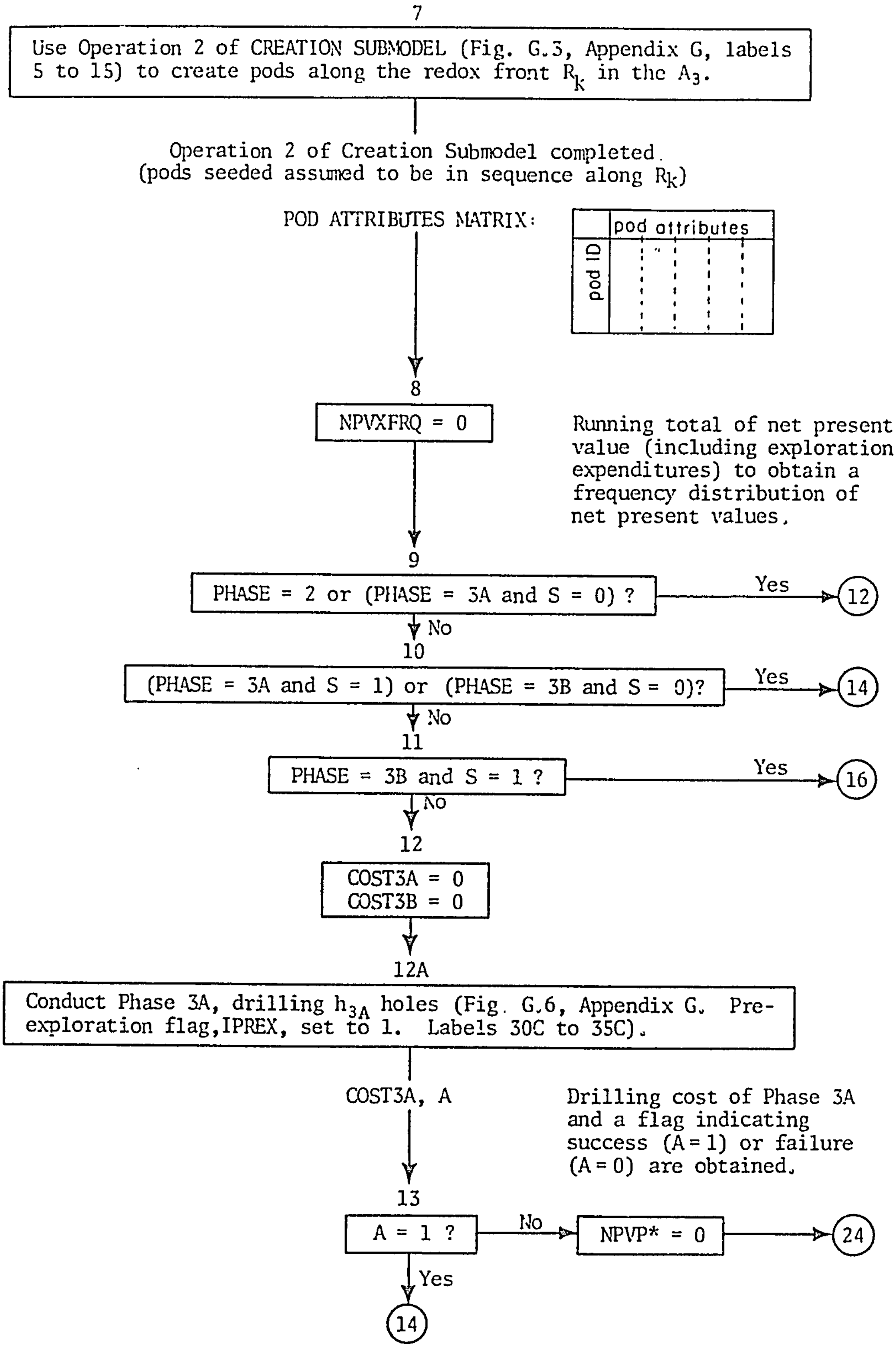
IDISPV = 0

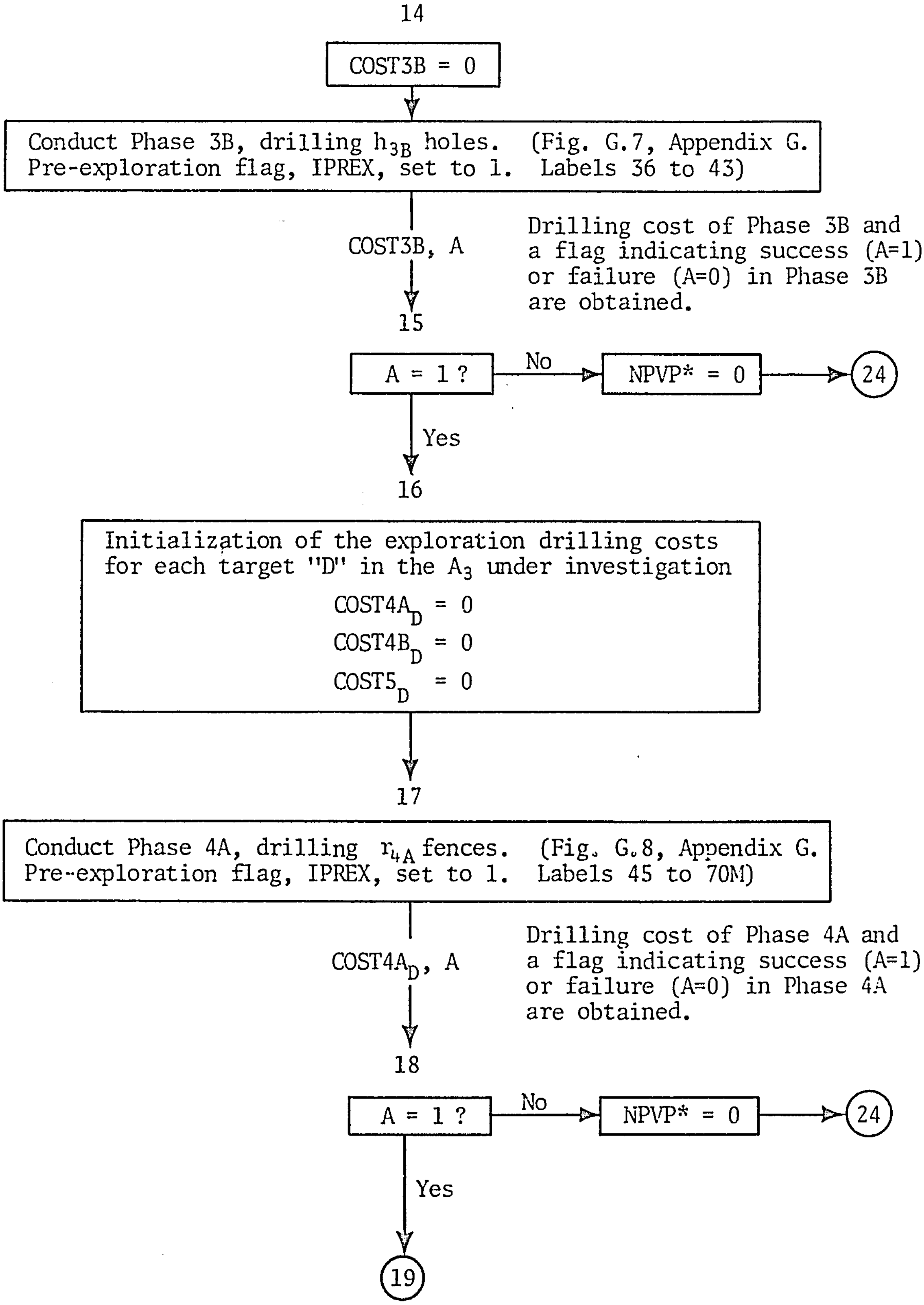
$\downarrow$

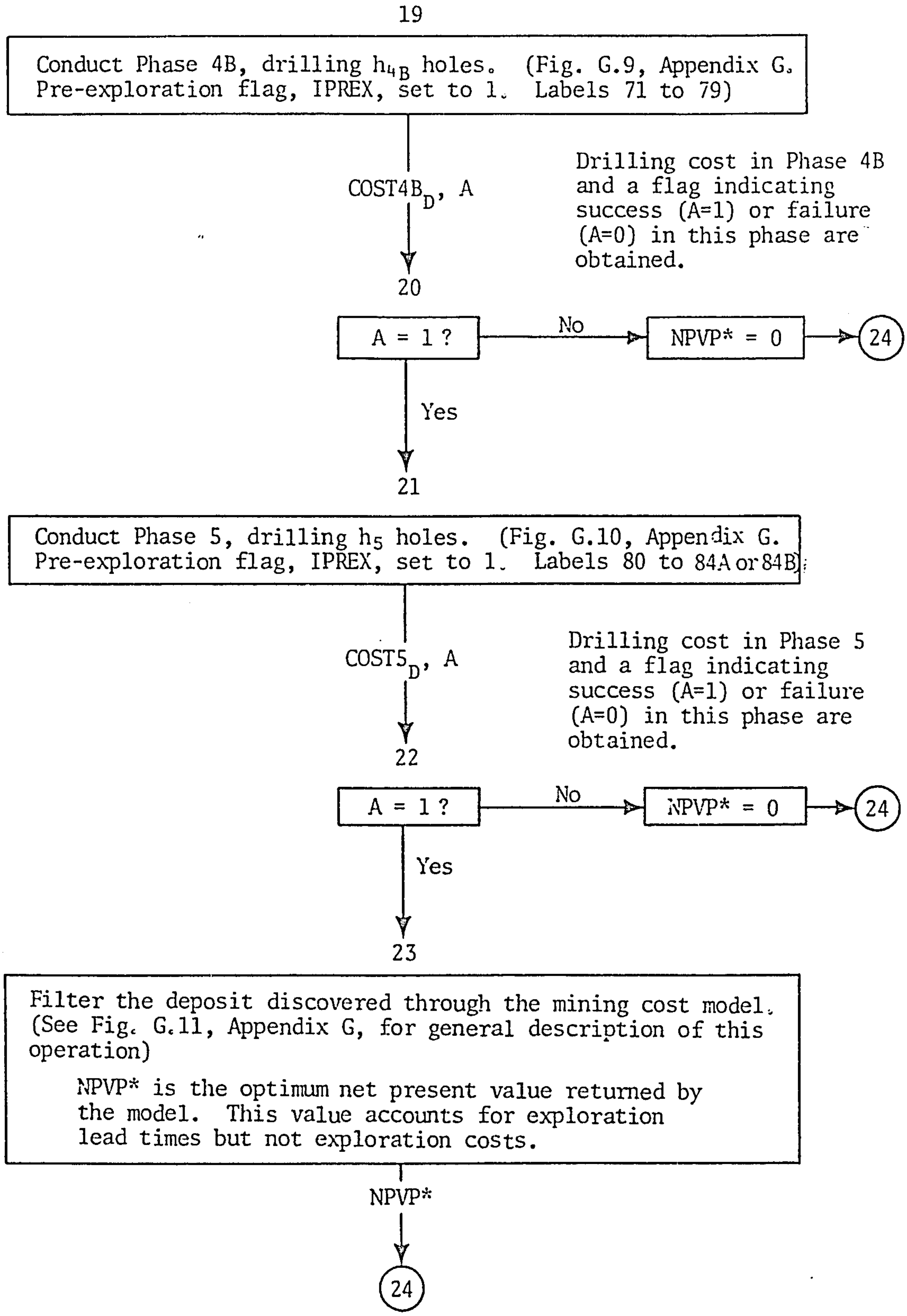
1A

Net present value distribution computation flag 0 = No
1 = Yes


1A
Set trial values for pertinent decision variables (e.g., h_{3_A}, h_{3_B}, r_{4_A}, ..., etc.)
NPVXSUM = 0
Running total of net present value (NPV) for expected NPV computation
1B
k = 0
k indexes on iterations for redox length in the A_3 under investigation
2
k ≤ desired number of redox length iterations ?
No
30
Yes
3
k = k + 1
4
Generate a specific value (Monte Carlo sampling) of redox length in the A_3 under investigation using $\varepsilon(R)$
R_k
5
Given the redox length, R_k, identify the appropriate conditional endowment distribution, $a(U|R_k)$ and generate an endowment value, U_k by sampling the conditional endowment distribution.
U_k
7


7
Use Operation 2 of CREATION SUBMODEL (Fig. G.3, Appendix G, labels 5 to 15) to create pods along the redox front R_k in the A_3.
Operation 2 of Creation Submodel completed.
(pods seeded assumed to be in sequence along R_k)
POD ATTRIBUTES MATRIX:
pod attributes
pod ID
8
NPVXFRQ = 0
Running total of net present value (including exploration expenditures) to obtain a frequency distribution of net present values.
9
PHASE = 2 or (PHASE = 3A and S = 0) ?
Yes
12
No
10
(PHASE = 3A and S = 1) or (PHASE = 3B and S = 0)?
Yes
14
No
11
PHASE = 3B and S = 1 ?
Yes
16
No
12
COST3A = 0
COST3B = 0
12A
Conduct Phase 3A, drilling h_{3A} holes (Fig. G.6, Appendix G. Pre-exploration flag, IPREX, set to 1. Labels 30C to 35C).
COST3A, A
Drilling cost of Phase 3A and a flag indicating success (A = 1) or failure (A = 0) are obtained.
13
A = 1 ?
No
NPVP* = 0
24
Yes
14


14
COST3B = 0
Conduct Phase 3B, drilling h_{3B} holes. (Fig. G.7, Appendix G.
Pre-exploration flag, IPREX, set to 1. Labels 36 to 43)
COST3B, A
Drilling cost of Phase 3B and a flag indicating success (A=1) or failure (A=0) in Phase 3B are obtained.
15
A = 1 ?
No
NPVP* = 0
24
Yes
16
Initialization of the exploration drilling costs for each target "D" in the A_3 under investigation
$COST4A_D = 0$
$COST4B_D = 0$
$COST5_D = 0$
17
Conduct Phase 4A, drilling r_{4A} fences. (Fig. G.8, Appendix G.
Pre-exploration flag, IPREX, set to 1. Labels 45 to 70M)
$COST4A_D$, A
Drilling cost of Phase 4A and a flag indicating success (A=1) or failure (A=0) in Phase 4A are obtained.
18
A = 1 ?
No
NPVP* = 0
24
Yes
19


19
Conduct Phase 4B, drilling h_{4B} holes. (Fig. G.9, Appendix G. Pre-exploration flag, IPREX, set to 1. Labels 71 to 79)
$COST4B_D$, A
Drilling cost in Phase 4B and a flag indicating success (A=1) or failure (A=0) in this phase are obtained.
20
A = 1 ?
No
NPVP* = 0
24
Yes
21
Conduct Phase 5, drilling h_5 holes. (Fig. G.10, Appendix G. Pre-exploration flag, IPREX, set to 1. Labels 80 to 84A or 84B)
$COST5_D$, A
Drilling cost in Phase 5 and a flag indicating success (A=1) or failure (A=0) in this phase are obtained.
22
A = 1 ?
No
NPVP* = 0
24
Yes
23
Filter the deposit discovered through the mining cost model. (See Fig. G.11, Appendix G, for general description of this operation)
NPVP* is the optimum net present value returned by the model. This value accounts for exploration lead times but not exploration costs.
NPVP*
24

24

Compute the contribution of the discovery (zero if there was no discovery) to overall net present value (includes exploration costs) by subtracting the discounted cost of all exploration drilling activities at a rate K%, the cost of capital of the firm (typical) assumed to be conducting the exploration.

$$NPVX = NPVP^* - [COST3A + COST3B + COST4A_D + COST4B_D + COST5_D]$$

(all costs discounted at K%)

25

NPVXSUM = NPVXSUM + NPVX

26

IDISPV = 0 ?

No → NPVXFRQ = NPVXFRQ + NPVX

Yes

27

Any target in this A_3 not yet drilled for ?

Yes → 16

No

28

IDISPV = 0 ?

No → 29

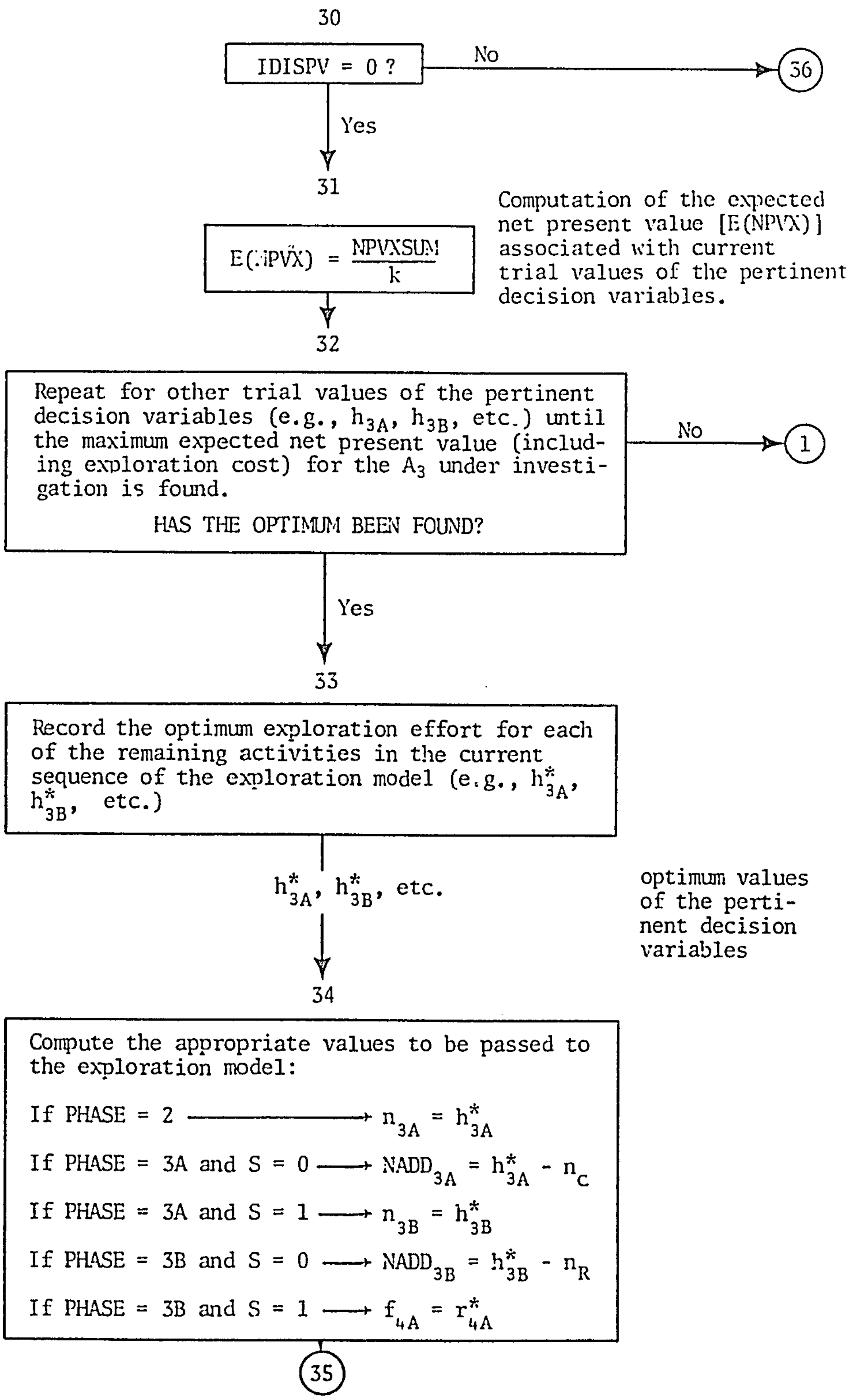
29

Classify NPVXFRQ in the appropriate net present value interval to produce a net present value frequency distribution, Φ(NPVX)

Yes

2

Generate another redox length value and redo the exploration.


30
IDISPV = 0 ?
No
36
Yes
31
$E(NPVX) = \frac{NPVXSUM}{k}$
Computation of the expected net present value [E(NPVX)] associated with current trial values of the pertinent decision variables.
32
Repeat for other trial values of the pertinent decision variables (e.g., h_{3A}, h_{3B}, etc.) until the maximum expected net present value (including exploration cost) for the A_3 under investigation is found.
HAS THE OPTIMUM BEEN FOUND?
No
1
Yes
33
Record the optimum exploration effort for each of the remaining activities in the current sequence of the exploration model (e.g., h^*_{3A}, h^*_{3B}, etc.)
h^*_{3A}, h^*_{3B}, etc.
optimum values of the pertinent decision variables
34
Compute the appropriate values to be passed to the exploration model:
If PHASE = 2 → $n_{3A} = h^*_{3A}$
If PHASE = 3A and S = 0 → $NADD_{3A} = h^*_{3A} - n_C$
If PHASE = 3A and S = 1 → $n_{3B} = h^*_{3B}$
If PHASE = 3B and S = 0 → $NADD_{3B} = h^*_{3B} - n_R$
If PHASE = 3B and S = 1 → $f_{4A} = r^*_{4A}$
35

35

Compute the net present value frequency distribution associated with the optimum level of exploration effort (e.g., h^*_{3A}, h^*_{3B}, r^*_{4A}, etc.)

In order to do this, set the net present value distribution computation flag (IDISPV) equal to 1 and redo the iterations, setting the trial values to the optimum.

↓

IDISPV = 1

↓

1A

36

The values to be passed to the exploration model representing the optimum level of exploration effort for the next activity have been calculated. Also, the companion net present value (including exploration costs) frequency distribution [Φ(NPVX)] has been determined.

↓

37

END

START

INPUTS REQUIRED BY THE MODULE:

- Economic related inputs: current price of uranium in the system, lead times, current cost of capital (K), cost indexes for date of evaluation, risk and financial characteristics of the firm (typical) assumed to be conducting the exploration, state, state tax rate.

- Responses to the deposit morphology questionnaire (Appendix E).

- Responses to the exploration performance questionnaire (Appendix F).

- $y[\ell(q_{min})]$, distribution representing the current expectations, in the exploration model, about the length of the target after Phase 4A has been conducted.

- Expected cross sectional target dimensions: $\overline{w}(.01)$, $\overline{th}(.01)$, $\overline{w}(q^*)$, $\overline{th}(q^*)$.

- $dpth_{min}$ and $dpth_{max}$, the minimum and maximum depths of the target.

- $W(.01)$, the width of the 0.01 confining prism of the target (zero if target occurring in the A_3 investigated is a a barren redox).

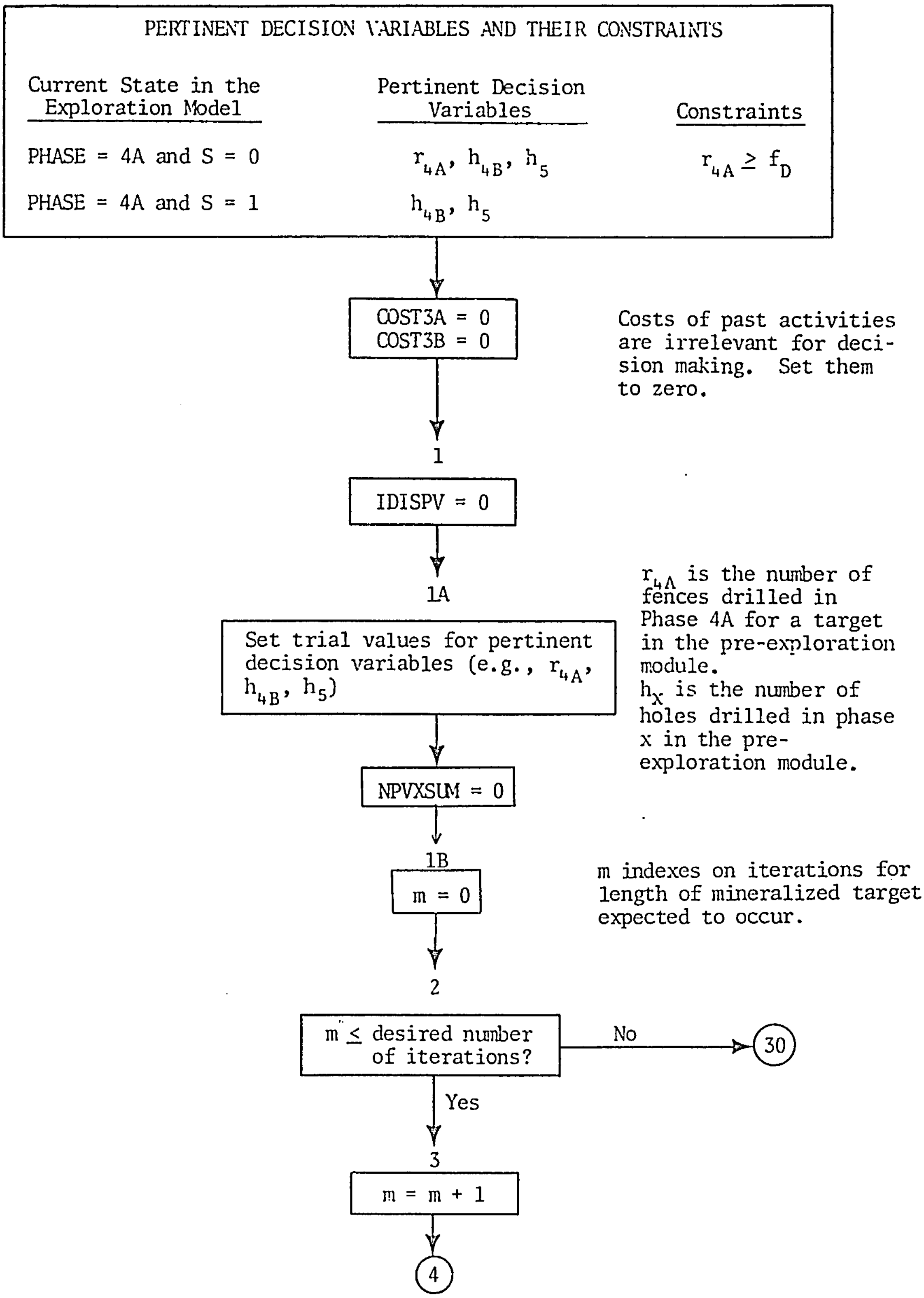
- L_D, the length of the redox along which the target occurs.

- f_D, the number of fences drilled so far in Phase 4A for the search of the target.

- q_{min}, the minimum cutoff grade economically attractive.

- $\overline{\Omega}_{min}$, the minimum average mineralization width economically attractive.

FIGURE K.3 PRE-EXPLORATION MODULE. CASE 2: CONSULTATION OCCURS WHEN EXPLORATION MODEL IS IN PHASE 4A


PERTINENT DECISION VARIABLES AND THEIR CONSTRAINTS
Current State in the Exploration Model
Pertinent Decision Variables
Constraints
PHASE = 4A and S = 0
r_{4A}, h_{4B}, h_5
$r_{4A} \geq f_D$
PHASE = 4A and S = 1
h_{4B}, h_5
COST3A = 0
COST3B = 0
Costs of past activities are irrelevant for decision making. Set them to zero.
1
IDISPV = 0
1A
Set trial values for pertinent decision variables (e.g., r_{4A}, h_{4B}, h_5)
r_{4A} is the number of fences drilled in Phase 4A for a target in the pre-exploration module.
h_x is the number of holes drilled in phase x in the pre-exploration module.
NPVXSUM = 0
1B
m = 0
m indexes on iterations for length of mineralized target expected to occur.
2
m ≤ desired number of iterations?
No
30
Yes
3
m = m + 1
4

4

Generate a specific value of length of mineralized target by sampling (Monte Carlo techniques) the $y[\ell(q_{min})]$ distribution.

$\ell(q_{min})_m$

5

Create enough pods of fixed cross sectional dimensions [i.e., $\overline{th}(.01)$, $\overline{th}(q^*)$, $\overline{w}(.01)$, $\overline{w}(q^*)$] so that the sum of the lengths of their q_{min}-confining prism plus the gap distances in between pods meet $\ell(q_{min})_m$.

Note: Assume that the group of pods constitutes the only deposit along the redox length, L_D.

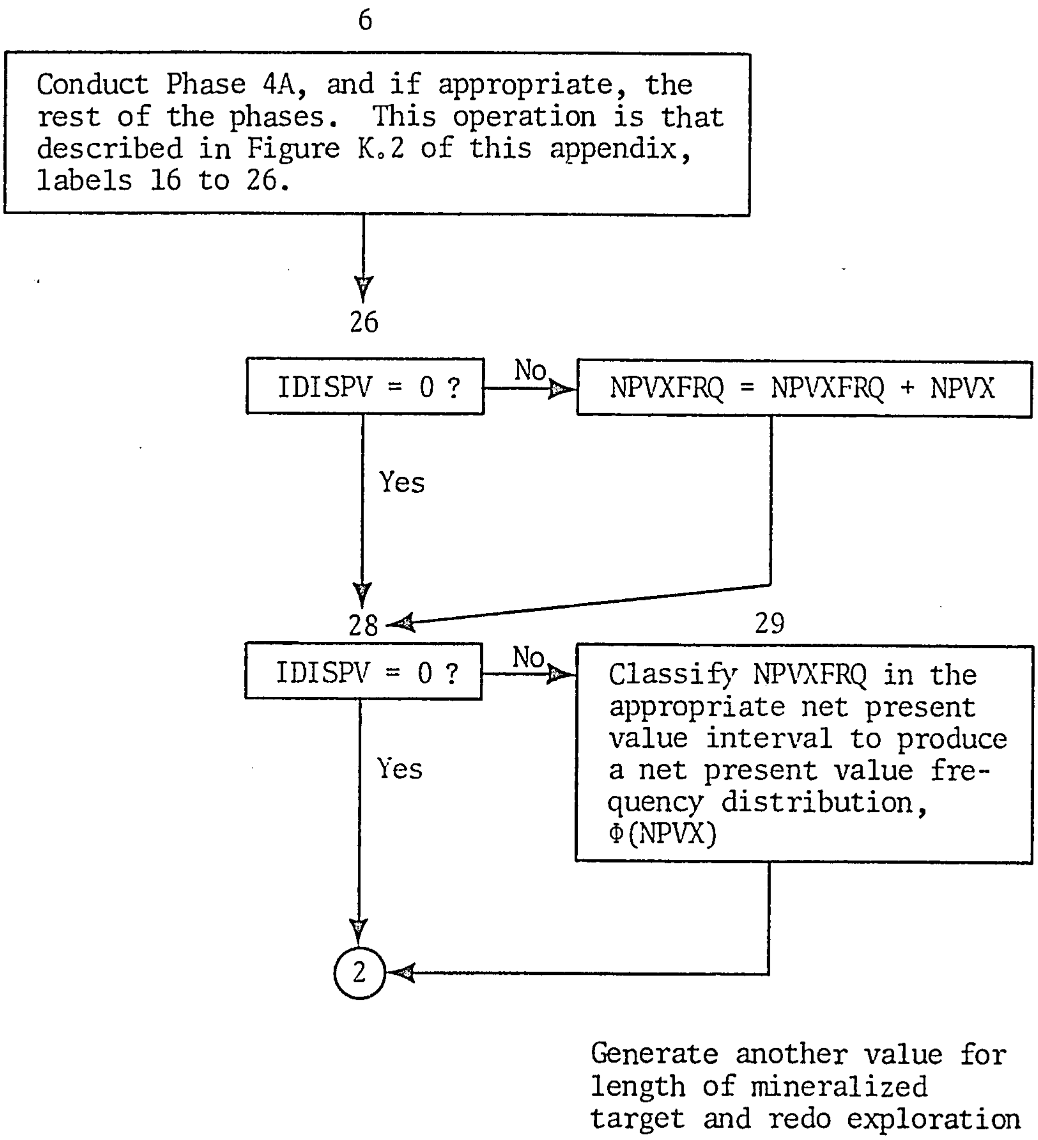
(This operation is similar to the Operation 2 of the creation submodel shown in Figure G.3)

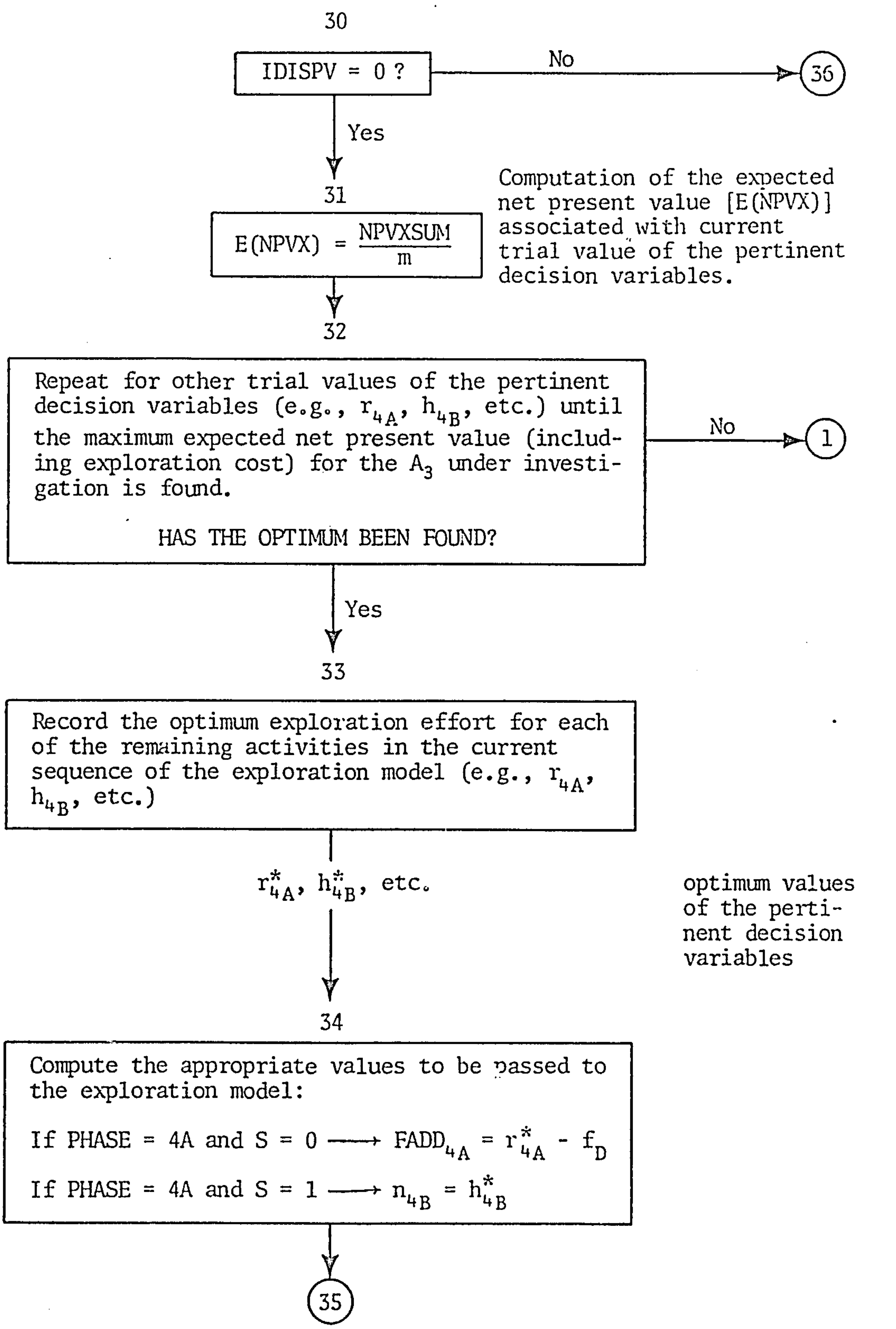
matrix of pod attributes representing the target that the exploration agent "thinks" is present along the redox length, L_D.

NPVXFRQ = 0

running total of net present value (including remaining exploration expenditures) to obtain a frequency distribution of net present values

6


6
Conduct Phase 4A, and if appropriate, the rest of the phases. This operation is that described in Figure K.2 of this appendix, labels 16 to 26.
26
IDISPV = 0 ?
No
NPVXFRQ = NPVXFRQ + NPVX
Yes
28
29
IDISPV = 0 ?
No
Classify NPVXFRQ in the appropriate net present value interval to produce a net present value frequency distribution, Φ(NPVX)
Yes
2
Generate another value for length of mineralized target and redo exploration


30
IDISPV = 0 ?
No
36
Yes
31
$E(NPVX) = \frac{NPVXSUM}{m}$
Computation of the expected net present value [E(NPVX)] associated with current trial value of the pertinent decision variables.
32
Repeat for other trial values of the pertinent decision variables (e.g., r_{4A}, h_{4B}, etc.) until the maximum expected net present value (including exploration cost) for the A_3 under investigation is found.
HAS THE OPTIMUM BEEN FOUND?
No
1
Yes
33
Record the optimum exploration effort for each of the remaining activities in the current sequence of the exploration model (e.g., r_{4A}, h_{4B}, etc.)
r^*_{4A}, h^*_{4B}, etc.
optimum values of the pertinent decision variables
34
Compute the appropriate values to be passed to the exploration model:
If PHASE = 4A and S = 0 $\longrightarrow$ $FADD_{4A} = r^*_{4A} - f_D$
If PHASE = 4A and S = 1 $\longrightarrow$ $n_{4B} = h^*_{4B}$
35

35

Compute the net present value frequency distribution associated with the optimum level of exploration effort (e.g., r^*_{4A}, h^*_{4B}, etc.)

In order to do this, set the net present value distribution computation flag (IDISPV) equal to 1 and redo the iterations, setting the trial value to the optimum.

IDISPV = 1

1A

36

The values to be passed to the exploration model representing the optimum level of exploration effort for the next activity have been calculated. Also, the companion net present value (including exploration costs) frequency distribution [Φ(NPVX)] has been determined.

37

END

START

INPUTS REQUIRED BY THE MODULE:

- Economic related inputs: current price of uranium in the system, lead times, current cost of capital (K), cost indexes for date of evaluation, risk and financial characteristics of the firm (typical) assumed to be conducting the exploration, state, state tax rate.

- Responses to the exploration performance questionnaire (Appendix F).

- PHASE, the current phase in the exploration model and the value for S, the success/failure indicator variable in the current phase.

- The exploration effort (n_E) spent so far in the current phase in the exploration model. Also current global reserves estimation error, ERR, achieved.

- The parameters of the probability distributions, $N(\mu_u, \sigma_u)$, (normal distribution) representing the current expectations of global reserves at q_{min}, $u_D(q_{min})$.

- $dpth_{min}$ and $dpth_{max}$, the actual minimum and maximum depths of the target.

- q_{min}, the minimum attractive grade.

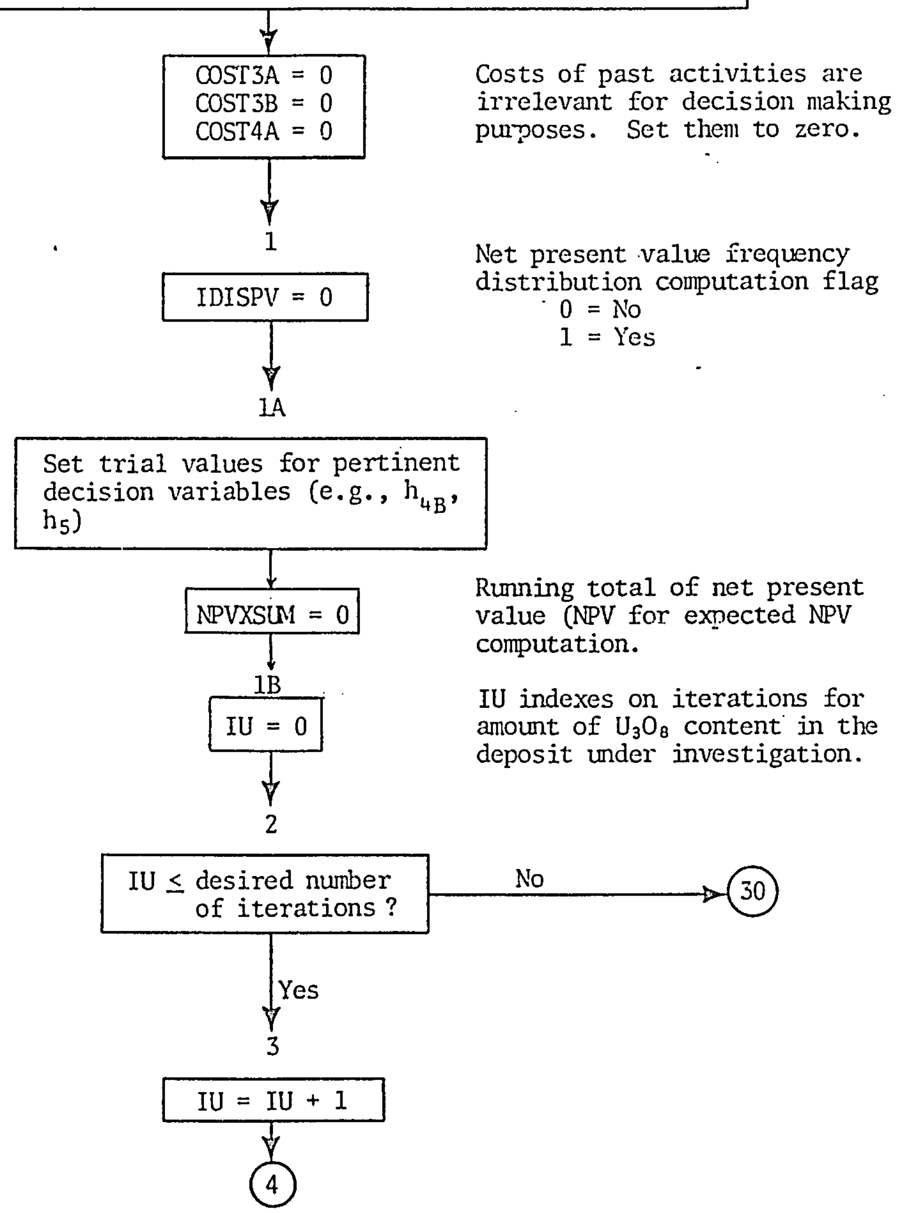
- $\ell(q_{min})$, the actual length of the q_{min}-confining prism circumscribing the deposit.

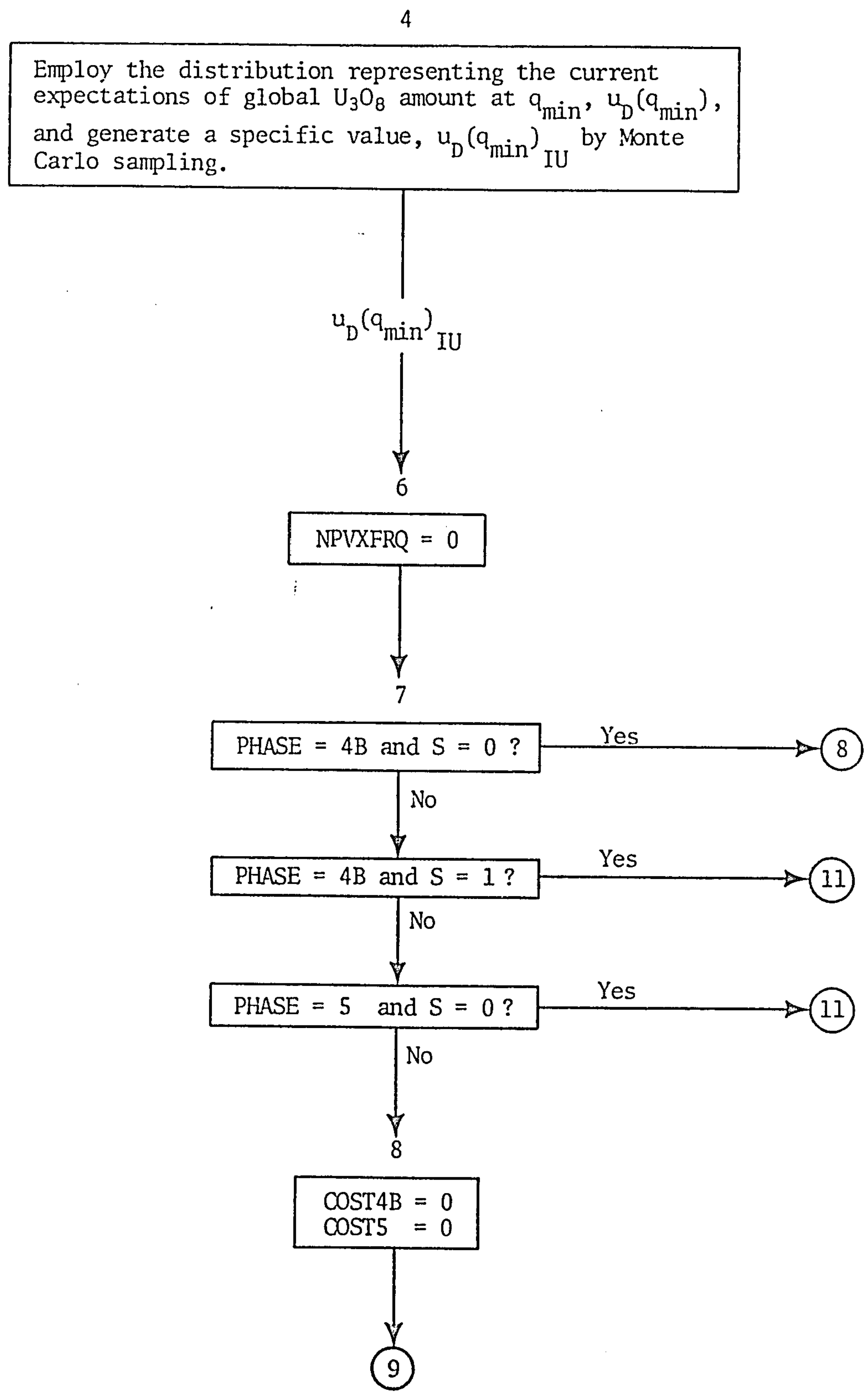
- $w(q_{min})$, the actual width of the q_{min}-confining prism circumscribing the deposit.

FIGURE K.4 PRE-EXPLORATION MODULE. CASE 3: CONSULTATION OCCURS WHEN EXPLORATION MODEL IS IN PHASE 4B OR PHASE 5

PERTINENT DECISION VARIABLES AND THEIR CONSTRAINTS

Current State in the Exploration Model	Pertinent Decision Variables	Constraints
If PHASE = 4B and S = 0	h_{4B}, h_5	$h_{4B} \geq n_E$
If PHASE = 4B and S = 1	h_5	$h_5 \geq n_E$
If PHASE = 5 and S = 0	h_5	$h_5 \geq n_E$




4
Employ the distribution representing the current expectations of global U_3O_8 amount at q_{min}, $u_D(q_{min})$, and generate a specific value, $u_D(q_{min})_{IU}$ by Monte Carlo sampling.
$u_D(q_{min})_{IU}$
6
NPVXFRQ = 0
7
PHASE = 4B and S = 0 ?
Yes
8
No
PHASE = 4B and S = 1 ?
Yes
11
No
PHASE = 5 and S = 0 ?
Yes
11
No
8
COST4B = 0
COST5 = 0
9

9

Conduct Phase 4B, drilling h_{4B} holes. (Fig. G.9, Appendix G. Pre-exploration flag IPREX is set to 1. Labels 71 to 76)

$COST4B_D$, A

10

A = 1 ? — No → NPVP* = 0 → (24)

Yes

11

COST5 = 0

12

Conduct Phase 5, drilling h_5 holes. (Fig. G.10, Appendix G. Pre-exploration flag IPREX set to 1. Labels 80 to 84B)

$COST5_D$, A

13

A = 1 ? — No → NPVP* = 0 → (24)

Yes

14

Using a response function illustrated next, obtain the optimum net present value (excluding exploration costs). The response function is obtained externally by repeatedly running the mining cost model to generate data to fit the function by regression analysis.

$$\widehat{NPVOPT} = f_{NPV}\left[u(q_{min})_{IU},\quad q_{min},\ \overline{dpth},\ \text{Price of } U_3O_8\right]$$

$\widehat{NPVOPT}$

(15)

15

$NPVP^* = NP\hat{V}OPT$

24

Compute the contribution of the discovery (zero if there was no discovery) to overall net present value (includes exploration costs) by subtracting the discounted cost of all exploration drilling activities at a rate K%, the cost of capital of the firm (typical) assumed to be conducting the exploration.

$$NPVX = NPVP^* - [COST3A + COST3B + COST4A_D + COST4B_D + COST5_D]$$

(all costs discounted at K%)

25

NPVXSUM = NPVXSUM + NPVX

26

IDISPV = 0 ?

No → NPVXFRQ = NPVXFRQ + NPVX → 28

Yes → 28

28

IDISPV = 0 ?

No → 29

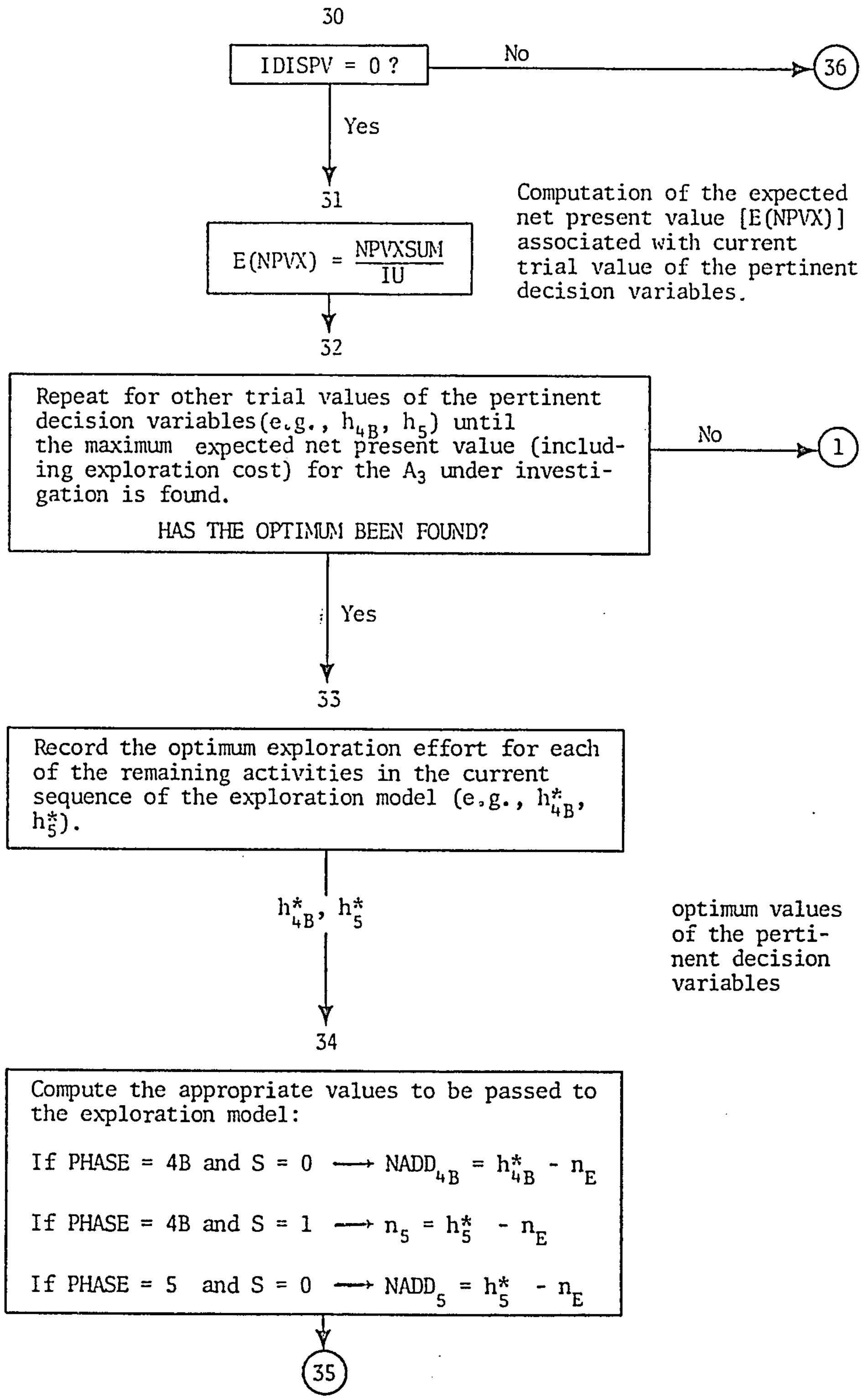
Yes → 2

29

Classify NPVXFRQ in the appropriate net present value interval to produce a net present value frequency distribution, Φ(NPVX)

2

Generate another value for U_3O_8 amount in the deposit and redo the pertinent exploration phases.


30
IDISPV = 0 ?
No
36
Yes
31
$E(NPVX) = \frac{NPVXSUM}{IU}$
Computation of the expected net present value [E(NPVX)] associated with current trial value of the pertinent decision variables.
32
Repeat for other trial values of the pertinent decision variables (e.g., h_{4B}, h_5) until the maximum expected net present value (including exploration cost) for the A_3 under investigation is found.
HAS THE OPTIMUM BEEN FOUND?
No
1
Yes
33
Record the optimum exploration effort for each of the remaining activities in the current sequence of the exploration model (e.g., h^*_{4B}, h^*_5).
h^*_{4B}, h^*_5
optimum values of the pertinent decision variables
34
Compute the appropriate values to be passed to the exploration model:
If PHASE = 4B and S = 0 $\longrightarrow$ $NADD_{4B} = h^*_{4B} - n_E$
If PHASE = 4B and S = 1 $\longrightarrow$ $n_5 = h^*_5 - n_E$
If PHASE = 5 and S = 0 $\longrightarrow$ $NADD_5 = h^*_5 - n_E$
35

35

Compute the net present value frequency distribution associated with the optimum level of exploration effort (e.g., h^*_{4B}, h^*_5).

In order to do this, set the net present value distribution computation flag (IDISPV) equal to 1 and redo the iterations, setting the trial values to the optimum.

IDISPV = 1

1A

36

The values to be passed to the exploration model representing the optimum level of exploration effort for the next activity have been calculated. Also, the companion net present value (including exploration costs) frequency distribution [Φ(NPVX)] has been determined.

37

END

START

1

INPUTS REQUIRED BY THIS MODULE:

- The optimum level of effort of next exploration activity ex* as determined by the pre-exploration module.
  Note: ex* is a generic name to indicate the number of holes (or fences) to drill next, according to the current state in the exploration model (e.g., if the phase to be initiated in the exploration model is Phase 3A, then $ex^* = n_{3A}$).

- Φ(NPVX), the distribution of net present value (including remaining exploration costs). This distribution represents the "prize" size in the geographic entity being explored and is jointly determined with ex*.
  The discount rate used is K, the current cost of capital of the (typical) firm.

- The risk-cost of capital relationship (value-per-share basis).

$$K = K^* + b \frac{\sigma_{NWPS}}{\overline{NWPS}}$$

  where

  K^* = the risk free cost of capital

  b = coefficient of risk.

  The determination of this relationship is external to the system and is detailed in the text (Section 10.4.2.1).

- Definition of the (typical) firm assumed to be conducting the exploration (also external to the system):
  - $\overline{NWPS}$, the expected net worth per share
  - σ_{NWPS}, the standard deviation of net worth per share
  - number of shares
  - r, the coefficient of correlation of the value of a uranium project and the value of the firm's existing operations
  - K, the current cost of capital of the firm. Note that since the risk-cost of capital relationship is given, K is determined by substitution of $\overline{NWPS}$ and σ_{NWPS} above in that relationship

3

FIGURE K.5 THE DECISION MODULE

3

Compute the first moment about the origin and second moment about the mean (i.e., mean and variance) of the NPVX distribution, $\Phi(NPVX)$.

$E(NPVX)$, $VAR(NPVX)$

4

Compute the new mean ($\overline{NWPS}'$) and standard deviation (σ'_{NWPS}) of the value (net worth per share basis) of the firm after project annexation:

$$\overline{NWPS}' = \overline{NWPS} + \frac{E(NPVX)}{\#\text{ of shares}}$$

$$\sigma'_{NWPS} = \left[\sigma^2_{NWPS} + \frac{VAR(NPVX)}{(\#\text{ of shares})^2} + 2\,r\left(\sigma_{NWPS} \cdot \frac{[VAR(NPVX)]^{\frac{1}{2}}}{\#\text{ of shares}}\right)\right]^{\frac{1}{2}}$$

$\overline{NWPS}'$, σ'_{NWPS}

5

Using the risk-cost of capital relationship, compute the new cost of capital if the uranium project were adopted

$$K' = K^* + b\left(\frac{\sigma'_{NWPS}}{\overline{NWPS}'}\right)$$

<u>Note</u>: K^* is the risk-free cost of capital, b is the coefficient of risk. The determination of these coefficients is detailed in the text (Section 10.4.2.1)

K'

6

6

Compute $\overline{NWPS}_0$, the present value of current operations (net worth per share basis) of the firm if the new cost of capital (K´) prevails

$$\overline{NWPS}_0 = \overline{NWPS} \times \frac{K}{K'}$$

[Note: This formula is an approximation which effectively calculates an implicit perpetuity at the original discount rate (K) and recapitalizes it at the new rate K´.]

↓

$\overline{NWPS}_0$

↓

7

Recompute the expected net present value of the project, E(NPVX´) discounted at the new cost of capital (K´). The following approximation may be used:

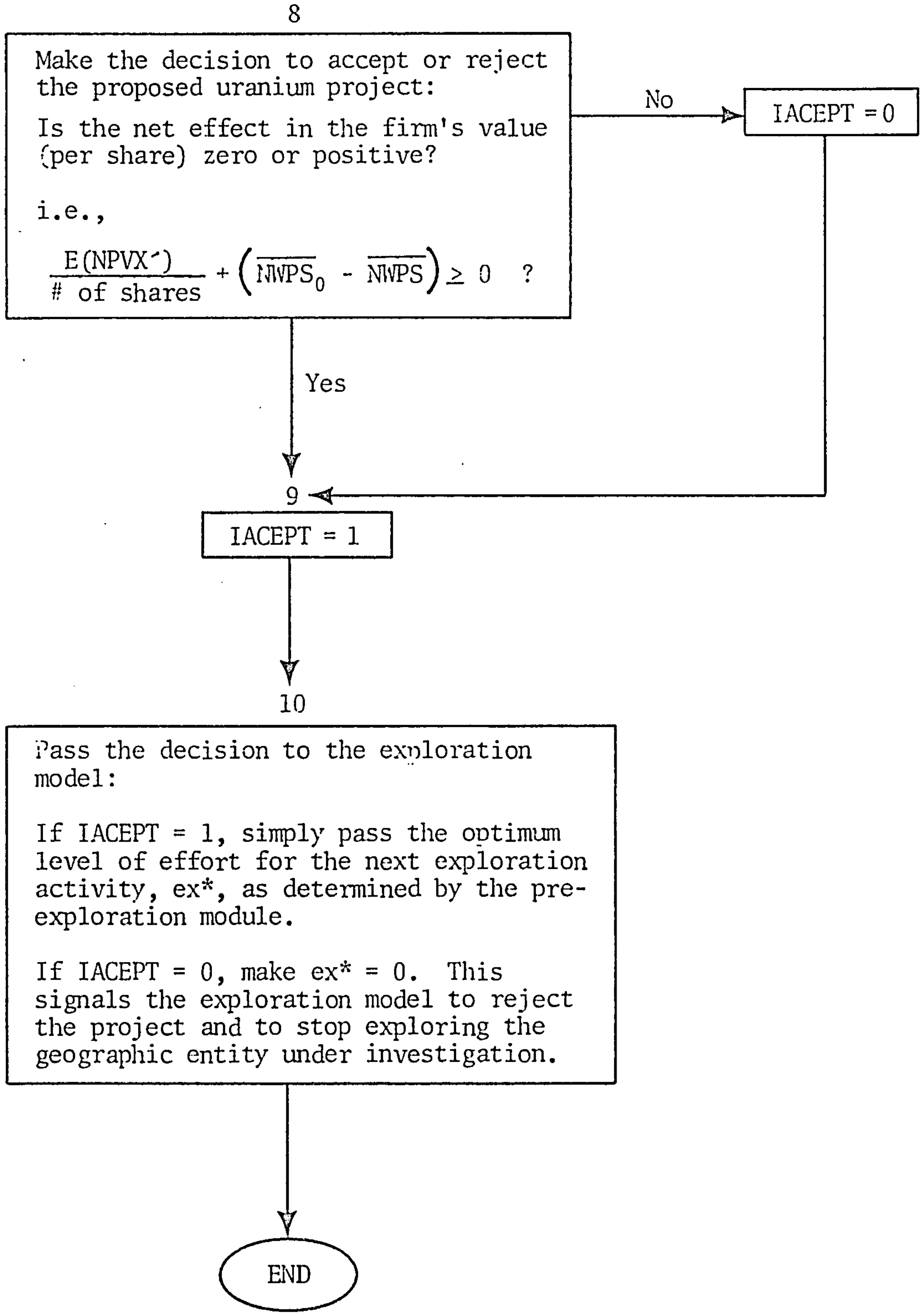
$$E(NPVX') = E(NPVX) \times \frac{K}{K'}$$

↓

E(NPVX´)

↓

8


8
Make the decision to accept or reject the proposed uranium project:
Is the net effect in the firm's value (per share) zero or positive?
i.e.,
$\frac{E(NPVX')}{\text{\# of shares}} + \left(\overline{NWPS}_0 - \overline{NWPS}\right) \geq 0$?
No
IACEPT = 0
Yes
9
IACEPT = 1
10
Pass the decision to the exploration model:
If IACEPT = 1, simply pass the optimum level of effort for the next exploration activity, ex*, as determined by the pre-exploration module.
If IACEPT = 0, make ex* = 0. This signals the exploration model to reject the project and to stop exploring the geographic entity under investigation.
END

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