AN INITIAL STUDY OF AN OPEN PIT MINE

A thesis presented to the Department of Mining Engineering of the University of the Witwatersrand for the Degree of Master of Science.

Submitted by -

HOWARD ROBINSON

June, 1949.
ERRATA

Page 33: \[ A = \frac{((Y* X Z) - (Y*Y)) / ((NS**Z) - (X**Z))}{((X*Z) - (X*Y))) / ((NS**Z, - (X**Z))} \] should be:
\[ A = \frac{(((Y*XZ) - ((X) - (X*Y))) / ((NS**Z, - (X**Z))} \]

Page 50: First line - 41000 should read 48000.

Page 53: Bottom - \( V \) is substituted for \( PV \) in the formula
\[ PV - V = P - (d*V*T) \] without comment and is changed back again on page 86, again without comment.

Page 97: GMC = ....... should read:
\[ GMC = \text{On if } \text{One} \lor \text{On}
= \text{On if } \text{One} \lor \text{On}
= \text{On otherwise} \]
The phrase "different than" is used in pages 21 and 120.

Page 116: Second last line - "past" should read "last".

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I would like to thank my department, Professor R. F. Thomas and W. L. Dobson, Glee of the Mining Department for their guidance and help with this work throughout.

I am also indebted to Dr. B. Bennett of the Statistics Department who assisted in the use of the editing program I. The preparation of the abstract, the papers linked to revised and edited by one

Finally, I would like to add that the research project of this paper have been carried out have been reviewed.
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The model described ... an economic planning tool designed to ... for the evaluation of open pit mining as follows:

The model evaluates the plant capacity that can be used on a given mineral deposit, according to a specific proposal. The proposal in question is divided into the following steps:

1) The production capacities of the mine, concentrator, and refinery,

2) The open pit production volume

- the sequence of excavation
- the location of the pit bench
- the angle of the safe and stable wall slopes
- the number of ore working faces and exposed ore benches required to supply the plant, listed in sub-sections:

3) the calculation and projection of equity and long-term capital to meet the capital requirements.

This proposed model is based on a course of action devised to exploit the given mineral deposit. Each proposal is evaluated by performing the calculations to find the cost of production, which includes the purchase cost of inventories, capital costs, and the cost of action, which will provide the capital required to meet the plan.

In addition to the proposal, the following information on the mine's operation is required to evaluate the plan:

1) Mine capacity
2) Ore reserves
3) Geologic and physical features, and
4) Peculiar costing elements

The model is an instrumental aid for working upon these facts according to the selected mining proposal. These input data are resolved by the model from the complex mine structure into simple components for the exact determination of an open pit mining solution. Consequently, the model is constructed to be a "total system" analogue of an open pit mine. The "total system" analogue is a duplication of the interdependent economic, mathematical, mining and design concepts, forming the integral open pit mining concept. In this way, the dependance of the computations on one another were accounted for so that the model would be a realistic duplicate of an actual open pit mine.

The model produces the following information which makes it the elements of the open pit mining solution:

1) A mineral grade cut-off point is computed to make the ore or waste decision. This cut-off grade depends on the mining method of the material exposed for mining at any given time and the current value of costs and selling price at such time. Consequently, it is computed on an annual basis.

2) The final pit surface is designed. This surface is the result of mining at the given deposit. The surface is designed to achieve the maximum total profit.

3) The pit production is simulated to give the excavation time of the material within the final pit surface, and annual topographic maps of the pit.

4) Production, cost and revenue schedules are produced on an annual basis.

This solution is produced to study the way in which the deposit's exploitation reacts to the alternative mining proposals. The reaction is measured by computing the profitability of the solution from an analysis of the cash generated.

The model's practical application lies in the investigation of its solution to exploit the given mineral deposit and
In this way, the user seeks to become intimately acquainted with how the particular deposit behaves through adjustment of the mining proposal. Two opposite extremes of the range of situations which can be evaluated by the Model are:

1) the examination of a long term proposal that may cover the entire life of the mine, and

2) the examination of a short term proposal, for example, a two or three year project to mine an additional section of an existing mine.

To make these investigations, the mutual relationship between the constituent parts of the mine structure are simulated to establish the peculiar nature of the given deposit. The simulation is a computer analogue designed to account for the physio-economic differences between deposits. The pit and deposit are simulated with a computer analogue of both a three dimensional network of lines and specially devised spatial data sets. This analogue is used to imitate the physical structure of the deposit's mineral distribution and topography; the pit surface; the important geologic features; and the spatial distribution of any density functions.

The three dimensional network of lines creates a grid of interrelated blocks or intersections for which certain values are calculated to describe the deposit. The grid-intersections are the geometric centre of the grid-blocks formed from perpendicular planes bisecting the lines between adjacent grid-intersections. The following elements of the mine and deposit are simulated using this grid system and the spatial data sets:

1) Topography and geology: Computer programming techniques are devised to make the grid system resemble the topography and important geologic features.

2) Mineral Distribution: The deposit's mineral distribution is simulated by forecasting the division, arrangement and dispersal of mineral values through the grid system.

3) Value Distribution: The cash value of each grid-block is calculated with the standard cost method. The
DECLARATION

I, [Name], hereby declare that the information set out in my [document] has not been altered or amended in any way and that the signature below is genuine.

[Signature]

[Date]
grid-block values include all costs and revenue accrued by the production of the minerals occurring in the particular grid-blocks. The standard cost method is devised by assigning a standard cash value to the grid-block in proportion to its mineral content. Then, this standard is adjusted according to the specific conditions under which the particular grid-block is mined. For example, the standards could be adjusted to vary with location, metallurgical penalties or time.

4) Pit Production Method: The pit production is simulated by imitating the real sequence of grid-block excavations. A record is kept of each grid-block excavation time. Annual production schedules and topographic maps are computed with this time record.

The simulation of the deposit's physical structure and pit production methodtailors the Model to the user's deposit. Only simple modifications are required to adapt the simulation to a different deposit or pit production method. This flexibility was an objective in the design of the Model to facilitate that no loss of accuracy would occur from the physio-economic differences between mineral deposits.

The Model is capable of producing an open pit mining solution when exploration drilling has delimited the extent of mineralization. The profitability of this solution is measured with a cash flow analysis. The solution and its potential value are examined in terms of the user's needs. This analysis can be rerun repetitively as additional information becomes available from further drilling. The solution computed by the Model becomes more accurate as the exploration data increase. The exploration work can be controlled by making the decision to continue or not at each analysis of the ever increasing exploration data. Ultimately, the final decision is made to mine the deposit or not, when the exploration data yields enough information to achieve the desired accuracy level.

The Model is described with a detailed account of its characteristics in the following Chapters. The literature survey in Chapter 2 describes the current practices of mine planning and pit design. The Model is presented by describing the construction and derivation of its parts in Chapters 3 and 6. Chapters 4, 5 and 8 list the Model's computer programs. The program listings follow the
Chapters in which their computations are discussed: an example of an application of the Model is presented in Chapter 7. Chapter 6 contains a criticism, the limiting provisions and some recommendations on the Model. Finally, the list of references appears in Chapter 10.

The computer programmes used in the Model were written in the FORTRAN language which consists of the set of statements described in "IBM System/360 FORTRAN IV Language" (2). FORTRAN is a body of expressions which communicate the computations required from the computer. The means used to initiate and control the program used in this thesis are described in "IBM System/360 Operating System Job Control Language" (4). The job control language is the method of commanding the computer to execute the action required on a particular program. The Chapter listing computer programmes contain the following information:

1) a list of steps required to prepare the programmes for execution;
2) a chronological description of the programme computations;
3) a programme list and notes describing how the programmes are tailored, for different deposits;
4) a glossary to define the FORTRAN variables; and
5) a list of the job control language for the Mine Plan Programme only.

The following discussions assume that the reader is familiar with FORTRAN and the job control language.
A broad view of the customary mine planning and pit design methods is presented in "Surface Mining" (1). It is the most recent and inclusive summary of the generally accepted surface mining practices. The two articles in the book which are applicable to this thesis are 'Preliminary Evaluations' by H. R. Sundem, page 94, and 'Pit Planning and Layout' by A. Soderberg and D. O. Rausch, page 141.

Sundeen describes the factors required to evaluate an open pit mining venture. He states: "The following outline lists the basic factors which must be taken into account for a preliminary evaluation of a prospective mine:

1) Geology,
2) Mining conditions,
3) Economic analysis,
4) Ore treatment requirements,
5) Geography,
6) Legal status of land and mining rights, and
7) Historical, political and sociological factors".

The evaluation of any potential deposit is updated continuously as additional data becomes available from exploration of the factors in the above list. Consequently, the point where a preliminary evaluation develops into a comprehensive evaluation is peculiar to the individual making the analysis. The model described by this thesis is capable of taking over the continuously expanding evaluation at the point when exploration drilling has marked the boundaries of mineralization.

The first five items in Sundeen's list are used by the model..."
to produce the optimum open pit solution for the user's deposit. This solution and the return on capital investment are compared with the user's company policy. The user adopts a course of action according to the results of the comparison between company policy and the model's evaluation. The model is not able to analyze the entire data of the continuous evaluation, which is predominantly socio-political. Therefore, the last two items, i.e. list of included in the model's evaluation indirectly, is so far as they affect the costs, revenue, pit design, plant production capacities and cash flow analysis.

The article 'Pit Planning and Layout' by Soderberg and Rausch (9, page 141) describes the current practice in designing the final surface of an open pit mine. This design is contrived by finding the physical limits of the pit beyond which the mineralization ceases to be profitable. The final pit surface designed with this logic constitutes all the points which define the ore boundary. These pit limits are found in the following manner:

The entire deposit is divided into sections, for example, the deposit illustrated in Fig. 2-1. The sections' end boundaries conform generally to the final pit shape. The wider boundaries are formed by the cuts which divide the deposit into sections. To simplify the calculations, the sections are assumed to be two-dimensional by eliminating the direction perpendicular to the sides. The consecutive two-dimensional sections are assumed to be small enough parts of the deposit to approximate the final pit surface. An economic limit is found for each section by moving its end boundaries in increments to the break-even point between profit and loss. The pit limits of each section are defined when the configuration of end boundaries produces the break-even point. Finally, the limits of adjacent sections are smoothed so that they do not exceed the allowable safe wall angles. The resulting pit limits become the final pit surface.

This 'pit limit' concept of pit design is a marginal examination for the boundaries beyond which the deposit's mineralization ceases to be profitable. The design of the final pit surface at the marginal limit between ore and mineralization is not the ideal objective. The objective should be to find the final pit surface yielding the maximum total profit. Another obstacle in the 'pit limit' design...
A. PLAN VIEW

B. SIDE VIEW OF A TYPICAL SECTION

FIG. 2.1 DESIGN SECTIONS OF THE 'PIT LIMITS' CONCEPT
concept is that the boundaries are determined separately for each section. This design is rather like examining each leg, in turn, of a three-legged stool to find out which leg supports the stool. If the legs were examined by isolating them individually from the system, the conclusion would be that each leg alone holds the stool upright. This is foolish, of course, but the right conclusion, that all three legs are necessary to hold the stool upright, can be made only if the stool is examined as a total system. Similarly, there are inherent errors in the sectional approach to pit design. The sections' large size and the two-dimensional assumption for their magnitude oversimplify the design, which creates the following errors:

1) The sections do not simulate the pit shape as the pit is excavated. Each section is evaluated independently on its own mineralization. Consequently, the economic potential of any mineralization occurring in a section must include the cost of exposing that mineralization. However, the sides of the sections are formed with planes that do not coincide with pit walls of the excavation. The sides are commonly vertical as shown in Fig. 2. The difference between these sides and the actual pit walls is not considered in the evaluation of each section. Therefore, the potential value of a particular section does not include the exact overburden stripping costs. Consequently, the designed pit limits will be greater than they should be.

2) The economic boundaries found for each section must be smoothed to get the final pit surface. Often the boundaries of adjacent sections differ enough to make the wall-angles between sections exceed the allowable extreme. If this is the case, the section boundaries are smoothed to achieve the safe wall-angles. This smoothing alters the pit limits and the final surface is no longer the result of the design concept.

3) The sections are too bulky to simulate the deposit's structural features. The sections are not capable of readily responding to the influence of structural features on the pit design, pit production method, or the cash value and ore reserve calculations. For example, topography, fault zones or an irregular mineral distribution are difficult to manage with the regular shape of the sections. Additionally, the sec-
tions would be difficult to construct for deposits requiring variable intermediate and final wall-angles. Consequently, the pit design loses accuracy through the cumbersome shape of the sections.

In the final analysis, it can be understood that the 'pit limits' design concept is unacceptable for two reasons: Firstly, the original limit objective is improper because it is not a basis for maximizing profits. Secondly, the sections used as a design tool are inflexible and not representative of an actual pit.

T. Pana (7) suggests the following pit design method which overcomes some of the disadvantages of the 'pit limits' design concept:

1) Construct a three-dimensional grid system to simulate the deposit. The grid system forms evaluation units or grid-blocks. The grid-blocks are smaller, more versatile and more representative evaluation units than the sections of the 'pit limits' design concept.

2) Estimate the mineral grades for each grid block by interpolation from the bore-hole samples.

3) Evaluate each grid-block "by weighting the recoverable (mineral) against the total costs including a minimum acceptable profit but excluding stripping related" (7) to the grid-block.

4) Classify the grid-block as ore if the evaluation is positive, or as waste if negative.

5) Construct removal increments from a family of overlapping cones. The geometry of the pit is approximated by a network of interconnecting cones. The material contained within each cone is defined as a removal increment. (7)

6) Total the grid-block values occurring in each removal increment.

7) Include the removal increments in the pit with a total positive value and exclude those with a total negative value.
The group of positive removal increments forms the final pit surface.

Lana's pit design method solves the following, two disadvantages of the 'pit limits' design concept: Finally, the burden of waste stripping costs is distributed to the mineralization exactly. The three-dimensional removal increments simulate the pit shape, whereas the two-dimensional sections do not. Secondly, the grid system responds to deviations in the deposit's mineral distribution, structural features and topography.

However, Lana's pit design method does not provide for a maximum profit objective. E. Lerchs and E. Grossman (5) state "the objective is to design the contour of a pit so as to maximize the difference between the total value of ore extracted and the total extraction cost of ore and waste." Lerchs and Grossman (5) have derived the theory, in the form of a graph algorithm, to design the final pit surface which will yield the maximum profit from a given deposit. The theory is an arithmetical set of rules designed to find the maximum value of the points in a special graph. This graph is described and defined by the (6) as follows:

"The term 'graph' denotes something quite different from the graphs you may be familiar with from analytical geometry or function theory. The kind of graph you probably have dealt with consists of the set of all points in the plane whose co-ordinates (x, y), in some co-ordinate system, satisfy an equation in x and y. The graphs (in the present context) are simply geometric figures consisting of points and lines connecting some of these points; they are sometimes called 'linear graphs.' It is unfortunate that two different concepts bear the same name, but this terminology is now so well established that it would be difficult to change it.

Definitions (See Fig. 2.2):

Graph: A figure consisting of points (vertices) and segments connecting some of these vertices. (The connecting segments may be straight line segments or curved segments and are called edges.)

Directed Graph: A graph with directed edges. The directed edges are called arcs.

Definitions (See Fig. 2.2):
Locher and Grossman (5) construct one of these graphs for an open-pit mine in the following manner:

"Let the entire pit be divided into a set of volume elements $V(I)$. This division can be quite arbitrary, but may also be obtained by taking for $V(I)$ the unit volumes defined by a three-dimensional grid. Associate to each volume element $V(I)$ a mass

$$m(I) = p(I) - c(I)$$

where $p(I)$ and $c(I)$ are the mine revenue and the extraction cost of element $V(I)$. Let each element $V(I)$ be represented by a vertex $x(I)$; of a graph. Draw an arc $(x(I), x(J))$ if $V(J)$ is adjacent to $V(I)$, that is, $V(I)$ and $V(J)$ have at least one point in common, and if the mining of volume $V(I)$ is not permissible unless volume $V(J)$ is also mined. We thus obtain a directed three-dimensional graph $G = (X, A)$ with a set of vertices $X$ and a set of arcs $A$. Any feasible contour of the pit is represented by a "closure" of $G$, that is, a set of vertices $Y$ such that if a vertex $x(I)$ belongs to $Y$ and if the arc $(x(I), x(J))$ exists in $A$ then the vertex $x(J)$ must also belong to $Y$. If a mass $m(I)$ is associated to each vertex $x(I)$, and if $m(Y)$ is the total mass of a set of vertices $Y$, then the problem of optimum pit design comes to finding in a graph $G$ a closure $Y$ with maximum mass or, shortly, a maximum closure of $G"$ (See Fig. 2)."

This maximum closure is found by following a set of rules which identify the vertices of grid-blocks existing within the final pit surface.

The graph algorithm designs the final pit surface with the cash value and extraction cost given for each grid-block. The given information represents an assumption of the economic circumstances under which the deposit exists. Locher and Grossman, however, that "besides pit design, planning also depends on questions such as:

1) what upgrading plants to install;
2) what quantities to extract, as a function of time;
3) what mining methods to use;
4) what market to select; and
5) what transportation facilities to provide.

There is an intimate relationship between all the above points, and it is meaningless to consider any one component separately. A mathematical model taking into account all possible alternatives simultaneously would, however, be of formidable size and its solution would be beyond the means of present knowledge.

This deficiency in mathematical knowledge is overcome by technological development of computer, mathematical, simulation and design methods. These methods are programmed to constitute the model described by this thesis. They are techniques devised to duplicate the physio-economic environment of an open pit mine. The Model designs, evaluates and partially plans the items in the preceding list, in the following manner:

1) The mine environment is given by the user in terms of the product market prices, the mine's cost structure, the approximate production capacity and the pit production method.

2) The Model performs the following functions:
   A. selects the exact production capacities of the upgrading plants,
   B. computes annual production targets and cut-off grades,
   C. designs the final pit surface,
   D. simulates the pit production method to give a time and location record of the annual production, and
   E. computes the profitability of the venture.

For some mining situations, it will be possible to vary the environmental conditions listed in the first subparagraph. These conditions, which are variables, are changed one at a time to find the ideal solution for exploiting the user's deposit. The ideal solution is chosen if the user from the combination of environmental conditions yielding the greatest profit, as computed by the Model.

There is one last open pit design reference that should be
A paper describing an open pit design model which also is due to the Lehigh - Brown algorithm. Their model performs only the design function for the final pit surface. Additionally, they make some suggestions for the methods of grade prediction and routing that are employed in the model described by this thesis.
CHAPTER 1  SIMULATION OF THE SURFACE MINERAL DEPOSIT

3.1 OPENING DISCUSSION

The surface mineral deposit given by the user is simulated to provide a record of:

1) the important structural, geologic and physical features, and

2) the spatial distribution of minerals.

The simulation uses a computer analogue of a three-dimensional grid system superimposed on the user's deposit. This computer analogue is devised with three-dimensional FORTRAN arrays. The array subscripts correspond to the intersections on the grid system's fixed network of lines. For example, the point 1000 feet south, 400 feet east, and 300 feet deep of the grid system illustrated in Fig. 3.1 would furnish the subscript (1, 4, 3) of the array address (1, 4, 3). The grid system is dimensioned further in Section 3.2.

The structural, geologic and physical features requiring simulation are all those which could affect the development of an open pit mining solution. For example, the method of mineral grade prediction must be sensitive to the mineralization zones which have evolved from different geologic processes. Consequently, it is necessary to identify the boundaries of these zones so that the prediction of mineral values in a particular zone is not influenced by mineral samples coming from another zone. These zones could be bounded by faults, intrusions, the water table and so forth. The boundaries delimiting these mineralization zones are simulated by reporting the presence or non-presence of each feature at the intersections of the grid system's line network. The simulation of these surfaces is described in Section 3.1.
Every new deposit evaluated with this model will have a different configuration of structural, geologic and physical features. Therefore, the model's computer programs have to be tailored to the unique deposit. The structural simulation techniques and computer programs are discussed in detail to explain how the programs are modified for different deposits.

The spatial distribution of minerals is a record containing the accumulation of recoverable minerals at each intersection of the grid system's line network. (The recoverable minerals are those capable of being extracted from the deposit with a profit). Each estimate of the recoverable mineral accumulation is a point density in the spatial distribution of minerals. The two computer programs discussed in Section 14 are used to estimate these mineral accumulations from the samples taken out of the exploration bore-holes.

3.2 SELECTION OF A GRID SYSTEM

The grid system is an imaginary network of lines superimposed on the specified deposit. The network of lines is a three-dimensional arrangement forming a set of reference points, planes and sections. This spatial reference set is used to define the essential nature of the given deposit at the reference points. Any data collected to describe a point in the deposit is stored in a data set by referencing that data with the coordinate location of the given point. Thus, the spatial distribution of any value is defined with reference to the fixed co-ordinates of the grid system.

The spatial features of the grid system used to reference data sets are defined below. These definitions are illustrated in Fig. 3.1.

1) Grid Co-ordinates: The grid co-ordinates are the number of unit interval distances from the origin along each axis.

2) Grid-intersections: The grid-intersections are the points common to the grid system's fixed network of lines.
3) **Grid-blocks**: The grid-block boundaries are the planes intersecting the grid points between grid-intersections. The grid-blocks represent a unit mass of the deposit in its unmined state. The grid-intersections are at the geometrical center of the grid-blocks.

4) **Grid-section**: A grid-section constitutes all the grid-blocks occurring between the parallel planes which are perpendicular to any of the three co-ordinate directions, are centered on any particular grid co-ordinate, and have a gap equal to the unit interval distance.

5) **Grid-level**: A grid-level is a horizontal grid-section.

The grid co-ordinates are scaled down distances from the actual distances to create small integer values suitable for FORTRAN subscripts. These subscripts become the addresses for the spatial data sets stored in FORTRAN arrays. The scaling is in proportion to the unit interval distances.

3. 2 1 GRID SYSTEM TYPES

The grid system is designed by planning the unit interval distances between grid co-ordinates and the angular separation between the axes. There are four basic grid system types which are constructed from the combinations of:

1) regular or irregular unit interval distances, and
2) rectangular or non-rectangular axes.

The most commonly used grid system is constructed with regularly spaced unit interval distances and rectangular axes. These unit interval distances are equally spaced along each axis, although they need not be the same interval for all axes. The illustration in Fig. 3.1 is one of these regularly spaced rectangular grid systems. The zero grid-level is designed to ensure that the highest point on the deposit's surface is within the grid system.

The regularly spaced and rectangular grid system is a good simulation for massive deposits with uniform mineral quantities. For these deposits, grade prediction can be calculated by estimating the mineral quantities in an,
particular grid-level solely from the assays occurring on that grid-level. However, this horizontal interpolation is not acceptable for deposits of which the grid system must portray any of the following characteristics:

1) zonal mineral trends,
2) erratic mineral quantities, and
3) some specific structural features.

Soderburg and Rauch (3, page 141) have warned that "Horizontal interpolations of ore values must be made with caution; some ore bodies have a zonal trend with a definite strike direction; others have ore values trending at dips up to vertical, with definite cut-offs in values normal to the dip zones. In such cases, horizontal interpolation of values will be misleading." For these situations, Soderburg and Rauch recommend that the grade prediction calculations be made with vertical grid-sections and that the information be transferred to the horizontal grid-sections. However, this merely changes the direction of the interpolation calculations; and therefore the same problems will still exist, except in a different dimension.

Another solution to this representation problem can be formulated by constructing grid systems with different configurations from the traditional regularly spaced and rectangular grid system. The answer is to design the grid system to represent the mineral-bearing rock structures. These grid systems are constructed by:

1) specifying the co-ordinate intervals equal to the rock structure spacing, and
2) orienting the grid axes parallel to the major mineral trends.

The following two examples illustrate the type of grid system which can be constructed to overcome the representation problem. Firstly, consider a regularly spaced but non-rectangular grid system:

The vein deposit and grid system illustrated by the cross-section view in Fig. 3.2A. The grade prediction of minerals is calculated individually for each grid-level parallel to
SURFACE

CROSS-SECTION OF A VEIN DEPOSIT

CROSS-SECTION OF A NON-RECTANGULAR GRID SYSTEM

FIG. 5.2A A NON-RECTANGULAR GRID SYSTEM USED TO SIMULATE A VEIN DEPOSIT

SURFACE

CROSS-SECTION OF A LAYERED DEPOSIT

CROSS-SECTION OF AN IRREGULARLY SPACED GRID SYSTEM

FIG. 5.2B AN IRREGULARLY SPACED GRID SYSTEM USED TO SIMULATE A LAYERED DEPOSIT
the vein.* The mineral quantities of a particular grid-level are estimated from the assay composites taken in that grid-level.* In this way, the zonal mineral trend of the vein deposit is simulated to avoid estimating mineral quantities from assays taken in different mineral bearing rock structures.*

Secondly, consider a rectangular grid system of which the horizontal grid axes are equally spaced but the vertical axis is irregularly spaced:

The stratified deposit and grid system illustrated by the cross-section view in Fig. 3.2.* The co-ordinate intervals in the vertical direction have been constructed to correspond to the layer thickness of the rock strata.* The mineral quantities of a particular grid-level can be estimated from the assay composites taken from the same grid-level.* In this way, assay composites taken from different rock layers will not influence the grade prediction calculations.*

The construction of the grid system to represent the deposit is important to insure the accuracy of the grade prediction calculations.*

3.2 2 Dimensions of the Grid System

The grid system and the co-ordinate interval dimensions must be compatible with both the deposit size and the storage capability of the computer used to execute the Model.* Obviously, the grid system must be large enough to contain entirely any pit that could be derived from the deposit.* The number of grid co-ordinates along the axes determine the total number of grid-intersections.* The data sets created to describe the pit and deposit have a member for each of these grid-intersections.* Consequently, the unit interval distances must be large enough to enable the computer to accommodate the data sets.* The computer capacity required for the largest programme in the Model is twenty-two bytes per grid-intersection for the data sets describing the pit and deposit, and an additional 216,000 bytes for the programmes and running data sets.* These capacity estimates are slightly generous to allow for uncertain events.* The total capacity could be trialled for a particular trial of the Model.*

The unit interval distances are designed to yield grid
blocks, whose boundaries contain any convenient unit amount of the pit production capacity. This grid-block unit is adopted as a measurement standard for the mine's tonnage and volume computations. The grid-block dimensions should be small enough so that the unit tonnages and volumes can accurately measure the mine production and simulate the pit surface. Therefore, the vertical and transverse unit interval dimensions should be an integer divisor of the bench height and width as set forward for solution by the mining plan. Additionally, the longitudinal unit interval should approximate an integer divisor of the ore working face advance per year. In other words, the production capacity per year of the smallest excavator planned for the mine should be a multiple of the material within the grid-block boundaries. It is not essential that the grid-blocks be designed in this way; however, it does facilitate the pit production simulation.

3.2.3 SPATIAL DATA SETS

The data sets used to describe the deposit and pit are addressed by using the grid co-ordinates to direct the attention of the computer to the storage location of the data set members. Accordingly, there is a storage location corresponding to each value or quantity computed at the grid-intersections or grid-blocks. In this way, the density of any spatial distribution is given at the points defined by the grid system.

There are two types of data sets used to store these spatial distributions. The first type is a FORTRAN array which "is a set of variables identified by a single variable name. A particular variable in the array may be referred to by its position in the array; each variable in this array consists of the name of the array, immediately followed by numbers enclosed in parentheses, called subscripts." (2, page 13) For example, consider the array, named 'VALUE,' which has three subscripts corresponding to the three coordinate directions of the grid system illustrated in Fig. 1. If it is desired to address the point density of 'VALUE' at grid co-ordinate 1, 3, and 2, this would be:

\[ \text{VALUE}(1, 3, 2) \]

"Ordinary mathematical notation might use VALUE 1, j, k to represent any element of the array VALUES" (2, page 19) In
FORTAN, this is written as VALUL(I,J,K).

The second data set type used to store the spatial distributions is called a direct access data set. These data sets are kept on external storage devices, for example, a magnetic guide, instead of the internal computer storage. These data sets provide a permanent record and minimize the internal computer storage requirements. Additionally, they allow the user to specify the location within a data set from which data is to be read, or into which data is to be written" (2, page 51). The FORTAN statements 'READ' and 'WRITE' "cause transfer of data into or from internal storage" (2, page 51). The 'READ' and 'WRITE' statements have the following format:

READ(a'r) list
WRITE(a'r) list

where: 'a' represents the data set reference number followed by an apostrophe
       'r' represents the position of a record within
       the data set associated with 'a'
       'list' is a series of variable or array names

For example, the following 'WRITE' statement will cause the internal value of 'GRADE' to be written on data set ten in location 'LOC':

WRITE(10'LOC) GRADE

If it is desired to 'WRITE' the point density of 'GRADE' at grid co-ordinate I = 2, J = 4 and K = 1, the record location 'LOC' is found with the following equation:

LOC=K*J*N+I*(J-1)+J

Where: 'K' and 'N' are the maximum grid co-ordinates in the I and J co-ordinate directions.

For the grid system illustrated in Fig. 3-1:

LOC=7*3*(3-1)+7*4*(4-1)+1
LOC=149

When the 'WRITE' statement is executed with the corresponding values of 'LOC' and 'GRADE', it will cause the record to be transferred from internal storage to the appropriate
location on the direct access device.

Thus, the two data types—array and direct access—are created to simulate the spatial distribution functions of the pit and deposit. The importance of these data sets is in the construction of a technique to correlate the computer storage location of the data set members with their location in the deposit.

3.2.4 SOME PROGRAMMING CONSIDERATIONS

The grid co-ordinates and the actual distances represent two different measurement scales for the user's deposit. The two scales are mixed in some of the computer program expressions. Consequently, scaling factors are included in these computations to convert between actual distances and the grid co-ordinates. This is simply a matter of scaling the distances in proportion to the unit interval distances between grid co-ordinates. However, the scale conversion must take into account the angular separation of the axes, if a non-rectangular grid system is used. The way in which these scale conversions are made is explained in Section 4.1.

Complicated grid system designs should and probably can be avoided to keep the computer programs simple. The irregularly spaced grid systems are an unnecessary complication, if a regular grid can be designed by reducing all the unit interval distances to equal the smallest significant rock layer. Then, the thicker rock layers are simulated by building up the equal unit interval distances. Additionally, the non-rectangular grid can be used conveniently for the Grade Prediction Program, and then substituted by a regular grid for the remaining computations of the Model. This change is made easily by transferring the point densities at the grid intersections in the non-rectangular grid to the same position in the rectangular grid.

3.3 SIMULATION OF THE DEPOSIT'S STRUCTURAL FEATURES

Most deposits have structural features that interfere with some of the Model's computations, as explained previously in Section 3.1. These structural features are the rock formations, topography, fault zones, water table, and so forth.
that will influence, in any way, the development of an open pit mining solution. To overcome this problem, the model is provided with a decision capability to branch the computer program to the computations which are appropriate for the particular physical conditions. This is accomplished by programming the model to recognize the regions distinguished by the structural features.

The following two examples demonstrate the need to simulate the structural features:

1) Consider an intruded rock structure that has a different mineral composition than the remainder of the deposit. This difference makes it necessary to insure that assays taken in the two zones are not used together to produce the mineral grade predictions.

2) Consider a pit surface whose safe wall-angles depend on the relative orientation of the pit walls to the bedding planes of the rock formations. The variable safe wall-angles make it necessary to define the pit zones limited by a particular safe wall-angle.

The shape of any structural features can be simulated to make a computer analogue of the pit and deposit. In addition to these physical or structural features, there may be legal restrictions on the pit, for example, property lines. The same simulation techniques are employed to make certain that the pit surface does not extend beyond the property lines, as demonstrated in Subsection 6.5.2.

3.3.1 INDICATOR ARRAYS

There are two techniques available for simulating the shape of the structural features and legal boundaries. The first technique is an indicator array. These arrays are similar to the spatial density arrays described in Subsection 3.2.3. They have an array member corresponding to each grid-intersection. The array members are pointers which indicate the presence of the particular structural features. The deposit's topography is a good example for illustrating the indicator array. A zero value is given to the array members representing grid-intersections above the deposit's surface. Any positive integer value is assigned to the remaining array members. The positive array members represent grid-intersections below the deposit's surface.
The input data used to create the topographic file are the
intercepts of the deposit's surface with the grid system.
The intercepts are read into a data set called 'SCORK' -
'SCORK' in an array with one subscript which identifies the
horizontal grid co-ordinate location of the intercepts. The
single subscript defines this horizontal location by specifying
an array member for each intercept. The vertical
co-ordinates of the intercepts become the array members. The
'SCORK' variable is given in Table 3.1 for the grid system
illustrated in Fig. 3.1.

<table>
<thead>
<tr>
<th>Surface Intercepts</th>
<th>Input</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 1 2</td>
<td>SCORK (1) = 2</td>
</tr>
<tr>
<td>2 1 2</td>
<td>SCORK (2) = 2</td>
</tr>
<tr>
<td>3 1 1</td>
<td>SCORK (3) = 1</td>
</tr>
<tr>
<td>4 1 1</td>
<td>SCORK (4) = 1</td>
</tr>
<tr>
<td>5 1 1</td>
<td>SCORK (5) = 1</td>
</tr>
<tr>
<td>6 1 1</td>
<td>SCORK (6) = 1</td>
</tr>
<tr>
<td>7 2 2</td>
<td>SCORK (7) = 0</td>
</tr>
<tr>
<td>2 2 2</td>
<td>SCORK (9) = 2</td>
</tr>
<tr>
<td>3 2 2</td>
<td>SCORK (10) = 2</td>
</tr>
<tr>
<td>7 8 1</td>
<td>SCORK (56) = 1</td>
</tr>
<tr>
<td>9 2 2</td>
<td>SCORK (57) = 2</td>
</tr>
<tr>
<td>9 2 2</td>
<td>SCORK (58) = 2</td>
</tr>
<tr>
<td>3 9 1</td>
<td>SCORK (59) = 1</td>
</tr>
<tr>
<td>5 9 1</td>
<td>SCORK (60) = 1</td>
</tr>
<tr>
<td>6 9 1</td>
<td>SCORK (62) = 1</td>
</tr>
<tr>
<td>7 9 1</td>
<td>SCORK (63) = 1</td>
</tr>
</tbody>
</table>

where:

\[ N = NI \times (I - 1) + I \]

'NI' is the maximum value of 'I'

Table 3.1

The indicator array 'TCPC' is created from 'SCORK' by...
executing the following FORTRAN statements:

```fortran
READ(5,1) NI,NJ,NK
1 FORMAT (3I4)
NK=NI+NJ
READ (5,10) (SCORK(N),N=1,NK)
10 FORMAT (7I2)
DO 104 K=1,NK
   N=0
   DO 104 J=1,NK
   DO 104 I=1,NI
   IF (SCORK(N)-K) 105, 103, 103
103 II=I
   GO TO 104
104 TOPC(I,J,K) = II
```

'NI', 'NJ' and 'NK' are the maximum grid co-ordinates in the 'I', 'J' and 'K' co-ordinate directions, respectively. The '7' in statement number ten is equal to 'NI'.

This topographic indicator array is constructed to control all computations of the spatial density functions. The density computations are by-passed for any particular array member, if the 'TOPO' array indicates the air space above the deposit's surface. For example, assume the model is executing the computations to estimate the mineral quantities at each grid-intersection. Obviously, it is a redundant exercise to make this estimate for the grid-intersections representing air space. Consequently, the model is programmed to recognize these grid-intersections and to assign them a zero mineral quantity without going through the computations.

All indicator arrays are created and used in a similar way to the 'TOPO' example.

3.3 2 POLYNOMIAL EQUATIONS

The second technique for simulating structural features is the derivation of polynomial equations to represent the surfaces and boundaries. The polynomial equations are constructed on the co-ordinates of the grid system. They are only an effective alternative to the indicator arrays for surfaces with few irregularities.
Consider, for example, a fault through which there has been relative rock movement. As a result of this rock movement, the fault defines a boundary on either side of which the grade predictions must be independent of each other. The fault is simulated with a polynomial equation to identify the relative location of the bore-hole samples to the boundary. This fault could be simulated with a plane, as illustrated in Table 3.2.

<table>
<thead>
<tr>
<th>Bore-hole No.</th>
<th>Grid Co-ordinate Intercept</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>I  J  K</td>
</tr>
<tr>
<td>1</td>
<td>29.50 18.25 4.40</td>
</tr>
<tr>
<td>28</td>
<td>17.50 14.00 1.60</td>
</tr>
<tr>
<td>38</td>
<td>3.00 12.00 3.80</td>
</tr>
</tbody>
</table>

Matrix of the Bore-hole Intercepts:

\[
\begin{bmatrix}
29.50 & 18.25 & 4.40 \\
17.50 & 14.00 & 1.60 \\
3.00 & 12.00 & 3.80
\end{bmatrix}
\]

Matrix Solution of the Fault plane:

\[66.31 I - 27.75 J - 37.625 K + 1274.075 = 0\]

**TABLE 3.2**

The grid co-ordinate location of the bore-hole samples compared with the fault plane equation. Then, the mineral quantities on each side of the fault are predicted from only those samples coming from the same side of the fault. In this way, samples existing on different sides of the fault are prevented from affecting the grade predictions unfavourably.

### 3.3 SOME PROGRAMMING CONSIDERATIONS

The employment of these structural simulation techniques is discussed and demonstrated in the next section. The following list itemizes their advantages and disadvantages.
The indicator arrays:
1) are simple to programme,
2) are rapidly executed by the computer,
3) can accurately simulate irregular surfaces, but
4) require a great amount of internal computer storage space.

The polynomial equations:
1) require little internal computer storage space, but
2) are difficult to derive and programme, and
3) are not accurate for highly irregular surfaces.

The large storage requirements for the indicator arrays is partially alleviated by allocating two computer storage locations per array member instead of the usual four. As a result, the maximum value that any of these variables can adopt is $12,10^4$. This allocation is executed by dimensioning indicator arrays with the FORTRAN specification 'INTEGER*2'.

3.4 SPATIAL DENSITY OF THE DEPOSIT'S MINERAL VALUES

Description of the two computer programmes used to estimate the spatial density of the mineral distribution for the user's deposit is contained in this section. The spatial mineral distribution is prerequisite to the development of the open pit mining solution described in Chapter 6. The following sections correspond to the two computer programmes. They describe the input, output, and basic computations, and how these computations cope with interference from the given deposit's structural features.

3.4.1 COMPUTATION OF THE ASSAY COMPOSITES

The Assay Composites Programme divides each bore-hole into the grid-levels defined by the FORTRAN array addresses on the vertical axis of the grid system. The assays of each grid-level from each bore-hole are averaged in proportion to
their sample lengths to furnish composite values. The composites are identified by their bore-hole, grid-level and mineralization zone. Then, they are passed to the grade prediction program.

The mineral samples from the exploration bore-holes are input data to the Assay Composites Program. The format for these data consists of the proportional mineral content per bore-hole length for each recoverable mineral. Table 3-4 presents the input data from the bore-hole illustrated in Fig. 3-1. The vertical unit intervals are one hundred feet. The samples are reported from the top of each borehole to the bottom and their lengths are measured in feet. The first sample length includes the distance from the zero grid-level to the deposit's surface, and additionally, any barren rock from the surface down to the first reported mineralization. The first sample length is reported in this manner so that the measurements conform to the same base. In this way, the cumulative sample lengths of any bore-hole will correspond to a relative location in the grid system.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Sample Length</th>
<th>Proportional Mineral Content</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Mineral 1</td>
</tr>
<tr>
<td>1</td>
<td>340*6</td>
<td>0*0</td>
</tr>
<tr>
<td>2</td>
<td>84*2</td>
<td>0*025</td>
</tr>
<tr>
<td>3</td>
<td>20*4</td>
<td>0*003</td>
</tr>
<tr>
<td>4</td>
<td>50*2</td>
<td>0*041</td>
</tr>
<tr>
<td>5</td>
<td>39*8</td>
<td>0*035</td>
</tr>
<tr>
<td>6</td>
<td>47*6</td>
<td>0*049</td>
</tr>
<tr>
<td>7</td>
<td>74*6</td>
<td>0*052</td>
</tr>
<tr>
<td>8</td>
<td>29*2</td>
<td>0*053</td>
</tr>
<tr>
<td>9</td>
<td>13*6</td>
<td>0*049</td>
</tr>
<tr>
<td>10</td>
<td>158*6</td>
<td>0*035</td>
</tr>
<tr>
<td>11</td>
<td>285*8</td>
<td>0*075</td>
</tr>
<tr>
<td>12</td>
<td>62*8</td>
<td>0*041</td>
</tr>
<tr>
<td>13</td>
<td>35*8</td>
<td>0*033</td>
</tr>
<tr>
<td>14</td>
<td>86*0</td>
<td>0*005</td>
</tr>
</tbody>
</table>

Table 3-3
The input bore-hole samples are averaged in proportion to their length to give a single mineral value for each grid-level and bore-hole. This average becomes the "assay composite" for the given grid-level. For instance, the assay composite of Mineral 1 for the fifth grid-level illustrated in Fig. 3-3 is calculated as shown in Table 3-4. Note that the lengths of samples two and five are not equal to their reported length. These sample lengths are truncated by the upper and lower boundaries of the fifth grid-level. Sample No. 2 is 47.8 ft and Sample No. 5 is 54.2 ft from the zero grid-level. Consequently, only the bottom 27.8 ft of Sample No. 2 and the top 16 ft of Sample No. 5 are included in the average for the fifth grid-level's assay composite.

---

**Example Calculations for an Assay Composite**

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Mineral Content</th>
<th>Sample Length</th>
<th>Product</th>
</tr>
</thead>
<tbody>
<tr>
<td>2</td>
<td>0.0025</td>
<td>27.8</td>
<td>0.0695</td>
</tr>
<tr>
<td>3</td>
<td>0.0028</td>
<td>20.4</td>
<td>0.0571</td>
</tr>
<tr>
<td>4</td>
<td>0.0041</td>
<td>50.2</td>
<td>0.2058</td>
</tr>
<tr>
<td>5</td>
<td>0.0038</td>
<td>1.6</td>
<td>0.0050</td>
</tr>
</tbody>
</table>

| Total      | 100.0 | 0.3385 |

**Total Product**

**Assay Composite** = \[
\frac{0.3385 \times 100.0}{100.0} = 0.3385
\]

**Total Sample Length**

**TABLE 3-4**

The assay composites for all the grid system types discussed in Subsection 3.2.1 are computed as illustrated in the above example. There is one simple difference in the programming for irregularly spaced grid systems: The vertical unit intervals for these grids are stored in a FORTRAN array to replace the constant interval size of the regularly spaced grid systems. This modification is jointed...
out in Section 4.1.

When the assay composites have been computed, it is necessary to define the geologic region in which they occur. This is required to identify the different mineralization zones, as explained in Section 3.3. The Assay Composites Programme executes this identification with the following routine:

```plaintext
I=0
DO 10 K=1,NBH
DO 10 F=1,NK
I=I+1
IF (3274*10**6*63+CORI(N)-1*3875+CORJ(N)-37*625)*K 6,7,7
6 IF (K-10) 1,1,2
7 IF (K-10) B,B,9
8 M=1
GO TO 3
9 M=2
GO TO 3
1 M=3
GO TO 3
2 M=4
3 ASAY1(K,F,N)*SACO(K,1,F)
ASAY2(K,F,N)*SACO(K,2,F)
WRITE(10)'ASAY1(K,F,N),ASAY2(2,F,N),F'
10 CONTINUE
```

Definitions:

*I* is a counter for the storage location of the direct access data set number eleven.

*N* is a subscript which identifies the bore-holes.

*NBH* is the total number of bore-holes.

*K* is a subscript which identifies the grid-level.

*NK* is the total number of grid-levels.

*CORI(N)* and *CORJ(N)* are the I and J bore-hole coordinates in feet.

*F* is a subscript which identifies the mineralization zones.
'SACO(K,1,M)', and 'SACO(K,2,M)' are the unzoned assay composites for two minerals, respectively.

'ASAY1(K,N,M)', and 'ASAY2(K,N,M)' are the assay composites for two minerals, respectively. These data sets include the subscript 'N' to identify the mineralization zones.

The preceding routine sorts the assay composites into four mineralization zones. The four zones are delimited by a fault and the water table. The first IF statement locates the assay composites relative to the fault. The IF statements '6' and '7' locate the assay composites relative to the water table which occurs at the tenth grid level.

Note that the expression in the first IF statement has mixed scales. 'CCPI' and 'COPJ' are measured in feet and the remaining terms in grid co-ordinates. The mixed scales are compensated for by the constants. Expressions with mixed scales occur frequently in the simulation routines; therefore, care must be exercised, if they are modified.

The programme accommodates as many minerals as required by specifying the appropriate dimensions for the FORTRAN variables. The dimensions are listed and explained in section 4.4.

The above example serves to illustrate the method of sorting assay composites into their mineralization zones. In this example, the given fault and water table must be replaced with new boundaries of any different deposit. The construction of the surface expressions for these new boundaries was explained in section 3.3. A simple example will serve to illustrate how the substitution is made. The fault expression in the first IF statement could be replaced by the following indicator array, if the fault surface is highly irregular:

\[
\text{FAULT}(\text{CC}(N), \text{COPJ}(N), K)
\]

where 'CC' and 'COPJ' are measured in grid co-ordinates instead of feet, as above. This 'FAULT' indicator array would be created the same way as explained for the 'TOPO' array in Section 3.3.

The output data from the Assay Composites Programme are the assay composites for each potentially recoverable mineral.
The 'WRITE' statement in the above routine causes the assay composites to be stored in a direct access data set for the Grade Prediction Program.

3.4 2 INTERPOLATION OF THE SPATIAL MINERAL DISTRIBUTION

An interpolation method is used to estimate the point densities of the spatial mineral distribution at the grid intersections. The computations described in this subsection are executed by the Grade Prediction Program listed in Chapter 5.

The interpolation method was suggested by R.C. Weaver (11), and it was chosen over the trijonal and polygonal methods. The interpolation method offers search parameters that adapt it to the peculiar grade prediction conditions of different mineralization zones. These parameters define the search for the assay composites to be included in each estimate of a point mineral density. The capability for varied execution makes the interpolation method more desirable than the trijonal and polygonal methods in terms of increased accuracy and adaptability. Nevertheless, the interpolation method is limited to massive and relatively uniform mineral deposits.

The point densities of the spatial mineral distribution are averages of the surrounding assay composites on the same grid-level and in the same mineralization zone. The assay composites are weighted by the inverse proportion of their distance from each point density. The basic interpolation formula follows; it is written in FORTRAN notation:

\[
\begin{align*}
X &= 0 \\
Y &= 0 \\
\text{DO 10 } &N = 1, \text{MAC} \\
X &= X + (G(N)/D(N)**P)) \\
10 &Y = Y + (1/(D(N)**P)) \\
\text{GRAD}(I,J,P) &= X/Y
\end{align*}
\]

where:

'N' is a subscript which identifies the assay composites to be included in the average.

'MAC' is the total number of assay composites.

'G(N)' is the proportional mineral content of assay compo-
The routine is executed at each grid-intersection (I,J,K) to estimate the joint mineral density at grid-intersection (I,J,K). This estimate is repeated for all recoverable minerals reported in the user's deposit. The assay composites 'G' included in each estimate are identified by the search parameters mentioned previously. These parameters are:

1) A maximum distance limit: The assay composites for each estimate are limited to those within a maximum distance of the particular grid-intersection. This is a circular limit so that each estimate is made only from the assay composites occurring on the same grid-level or mineral horizon. The grid-levels will correspond to the mineral trends or horizons, if the grid system is constructed as described in Section 3.2.

2) A minimum distance limit: If an assay composite exists within a minimum distance of a particular grid-intersection, then grid-intersection's joint mineral density assumes the value of the assay composite.

3) A weighting factor: The reciprocal distance between grid-intersection and assay composites is weighted with a power factor 'I', as demonstrated in the preceding routine.

4) A maximum number of assay composites: The number of assay composites per estimate is limited to those closest to the particular grid-intersection, up to the maximum.

5) A minimum angular separation: The minimum angular separation between assay composites must be exceeded; if not, the assay composite furthest from the particular grid-intersection is deleted from the average.
The function of these parameters can be demonstrated with the grid-intersection illustrated in the center of Fig. 3.4 and the surrounding assay composites. Of the seven assay composites illustrated, the following ones are rejected from the estimate of the grid-intersection's point mineral density:

1) AC6 and AC7, because they exceed the maximum distance limit.

2) AC2, because it is farther from the grid-intersection than AC3 and the angular separation between the two is less than the minimum.

3) AC1, because it is the furthest from the grid-intersection of the remaining assay composites; and without it, the maximum number of assay composites per average is not exceeded.

These search parameters are defined by the user before the Grade Prediction Programme is executed. They can be determined with a trial and error solution in the following manner: A random sample is taken of the assay composites from each different mineralization zone. The mineral values of each random sample are estimated with the remaining assay composites in the same zone with assured search parameters. Alternatively, the assay composites could be used to estimate the mineral values of a previously mined section, if one exists. The random samples' estimated values are compared with their actual values using a correlation test. The search parameters are redefined until the correlation test becomes significant.

The correlation test is made from a regression curve of the estimated and actual values. The actual values are the independent variables on the curve and the estimated values are the dependent variables. A least square regression line is calculated from this data. If a straight line relationship can be assumed between the variables, the regression line is expressed in FORTRAN as:

\[ EY = A + (B \times AV) \]

where:
- \( EY \) is the estimated value,
- \( AV \) is the actual value, and
- \( A \) and \( B \) are constants.
AC = ASSEMBLY COMPOSITE

GRID-INTERSECTION

MAXIMUM DISTANCE LIMIT

FIG. 3.6 THE SURROUNDING ASSAY COMPOSITES OF A PARTICULAR GRID-INTERSECTION
The constants --A and Y---can be calculated with the following FORTRAN routine:

```
X=0
Y=0
X2=0
XY=0
DO 10 N=1,NS
  X=X+AV(N)
  Y=Y+EV(N)
  XY=XY+(EV(N)*AV(N))
10 X2=X2+AV(N)**2
A={N*AV(N)**2}-N*(X*Y)-(X**2)
B={(N*XY)-(X*Y)}/(N*NS2)-(X**2)
```

where:

'X' is the summation of the actual values,
'Y' is the summation of the estimated values,
'X2' is the summation of the actual values squared,
'XY' is the summation of the product of the actual and estimated values, and
'NS' is the total number of samples.

The degree of association between the estimated and actual values is measured to find out how well the current search parameters estimate the actual mineral quantities. The relative intensity of this association is given by the coefficient of correlation. The correlation coefficient can be computed with the following FORTRAN routine:

```
C=0
D=0
DO 10 N=1,NS
  C=C+(A*(B+AV(N)))-N*2
10 D=(EV(N)-Y)**2
R=SQR(D/C)
```

where:

'NS' is the total number of samples,
'A' and 'B' are the constants from the regression line,
The correlation coefficient 'R' is dimensionless and varies between -1 and +1. The plus sign shows positive linear correlation and the minus sign shows negative linear correlation. Either extreme of the coefficient is perfect correlation, and any value in between is less than perfect. Additionally, a zero correlation indicates no association between the estimated and actual mineral quantities.

The search parameters are varied until the correlation analysis indicates that the parameters have produced a good association between the estimated and actual mineral quantities. However, the association may not be linear as assumed, in which case the correlation coefficient may approximate zero. If so, the linear relationship is replaced with a non-linear regression equation.

The number of random samples required and a significant correlation coefficient are calculated for each mineralization zone. Student’s distribution can be used for this in the following manner: Firstly, a correlation coefficient is calculated on an assumed sample size. Next, the assumed size is compared with the minimum sample size required for a significant correlation coefficient. If the assumed size is larger than the minimum, then the correlation coefficient is significantly different than zero on a particular confidence level. This minimum is computed from the following statistic expressed in FORTRAN:

\[ T = \frac{R \cdot \text{SQR}(N-2)}{\text{SQR}(1-(R^2))} \]

where:

'R' is the given correlation coefficient from the assumed sample.
FIG. 3.3 VERTICAL CROSS-SECTION OF A BORE-HOLE
'n' is the minimum sample size, and
'T' is Student's statistic.

The minimum sample size 'n' must be such that the right hand side of the above expression is equal to the critical value 'T' from Student's distribution on a given confidence level and with 'n - 2' degrees of freedom. When 'n' has been calculated, it is compared with the assumed sample size. The correlation coefficient 'R' is significant if the assumed sample size is greater than 'n'. Otherwise, the assumed sample size is enlarged until it is greater than the minimum size.

The search parameters are defined for each of the mineralization zones in the preceding manner. M. R. Spiegel's "Theoretical Problems of Statistics" (10) is recommended for further discussion of the sampling theory of correlation.

The mineralization zones are recognized by the Grade Prediction Programme with a duplicate of the simulation routine used to sort the assay composites. This routine is demonstrated in Subsection 3.1. The simulation routine recognizes the distinct structural features to identify the mineralization zone of the point mineral densities being estimated. Then, the routine identifies the zones of the assay composite. The assay composites, whose zones are not the same as the particular point mineral density, are excluded from the estimate. The mineralization zones used for demonstration purposes in Chapters 4 and 5 will be undoubtedly different for any other deposit. Consequently, this simulation routine must be tailored to the user's deposit by creating the appropriate indicator arrays or polynomial equations, as explained in Section 3.3.

In addition to the mineralization zones, it is necessary to simulate the deposit's surface for the Grade Prediction Programme. This simulation is in line with the 'TOPO' indicator array used as an example in Section 3.1. The interpolation for the point mineral density at any given grid-intersection is executed only if the 'TOPO' array indicates that the point is below the deposit's surface. Otherwise, the interpolation computations for that point are bypassed and the Programme moves to the next grid-intersection.

Thus the spatial mineral distribution is computed. The point
Mineral densities estimated at the grid-intersections are stored in a permanent record. This record is a direct access data set which is kept to provide the succeeding Mine Plan Programme with some of its input data.
distances between grid co-ordinates and the direction of the axes, as explained in Section 3.2. The grid system dimensions that must be planned to execute the Assay Composites Programme are:

1) the angular separation of the co-ordinate axes,
2) the relative position of the zero grid-level with the deposit's surface, and
3) the unit interval distance of the more nearly vertical axis.

The zero grid-level is defined to correspond with the highest point on the given deposit's surface. Elevations are measured from the zero grid-level with depth being the positive direction. The vertical unit intervals are given as explained in Step 1.

STEP 2:

The dimensions must be specified for the FORTRAN arrays and direct access data sets. The dimensions:

1) are defined for the FORTRAN arrays in Section 4.5,
2) are given for the DEFINE FILE statement in Section 4.3 for the direct access data sets.

STEP 3:

A scale factor must be defined to convert between actual vertical distances and grid co-ordinates. This scale factor is called 'KINT' in the Assay Composites Programme.

A) 'KINT' is a constant and equal to the unit interval distance between the vertical grid co-ordinates for a regularly spaced and rectangular grid system.

B) 'KINT' is a variable with an array member equal to each interval distance between grid co-ordinates for an irregularly spaced vertical axis. 'KINT' is dimensioned as illustrated in Fig. 4.1 for this situation.

C) 'KINT' is a constant and equal to the vertical distance between the more nearly horizontal axis of the non
THE SCALE FACTOR 'KINT' FOR THE ABOVE DEPOSIT

KINT(1)=50
KINT(2)=160
KINT(3)=30
KINT(4)=130
KINT(5)=120

FIG. 4.1 SCALE FACTOR 'KINT' FOR AN IRREGULARLY SPACED GRID SYSTEM
STEP 4:

The appropriate number of variables must be added at Lines 086 and 102 for each potentially recoverable mineral reported in the deposit.

STEP 5:

The boundaries of the distinct mineralization zones are simulated with indicator arrays or polynomial equations, as described in Section 3.3. These simulated surfaces are substituted into the Programme’s simulation routine between Lines 94 and 101. In this way, the Programme is tailored to locate the specific mineralization zones of the given deposit’s assay composites. The substitution of these zones is described in Section 3.4.1.

Care must be exercised to insure that the surface simulations are made with the correct units. Simple scale factors are used to convert actual distances to grid co-ordinates or conversely, The constant scale factor described in Step 3.3 is used to convert scales on regularly spaced axes. The array described in Step 3D is used to convert scales on irregularly spaced axes. Finally, an equation is derived to convert from actual rectangular co-ordinates to non-rectangular grid co-ordinates. This equation for the grid system illustrated in the cross-section view of Figs 4.2 is:

\[
K = \frac{\tan (a)}{SFK} \left( I + \frac{V}{\tan (a)} \right)
\]

where:

- \( K \) is the grid-level
- \( I \) is the horizontal grid co-ordinate
- \( V \) is the vertical distance in feet
- \( SFK \) is a scale factor equal to the interval distance
Fig. 4.2 Cross-section of a non-rectangular grid.
between the I grid co-ordinates in feet.

'SFK' is a scale factor equal to the interval distance between the K grid co-ordinates in feet.

'a' is the angle between the horizontal and the K co-ordinate axes.

The scale factors are included in any of the FORTRAN expressions that have terms with mixed scales. These expressions have been noted in Section 4.3.

4. 2 SEQUENCE OF PROGRAMME COMPUTATIONS

__LINES 001 - 003:__
Specification statements.

__LINES 004 - 015:__
Input statements.

__LINE 014:__
The horizontal bore-hole co-ordinate locations, in feet, are stored on a magnetic disk for rapid access in subsequent programs.

__LINES 016 - 022:__
DO loop 16 composites the bore-hole samples 'SAM' between grid-levels to give an average mineral grade 'SACD' for each recoverable mineral (M), bore-hole (N) and grid-level (K).

__LINES 020 - 022:__
These statements calculate the cumulative distance 'CUMINT' from the zero grid-level to each bore-hole sample.

__LINES 029 - 031:__
These statements simulate the deposit's surface by assigning any negative value to 'SACD', if the assay composite should represent an empty space. Whenever 'SACD' is used, its array
members are tested for negative values and the programme branches past the appropriate calculations, if the array member is negative.

LINES 032 - 075:

These statements contain an implied computational loop to identify the bore-hole samples 'SAMP' to be included in each assay composite 'SACO'.

LINES 076 - 081:

The bore-hole samples 'SAMP(K,N,LC(LL))', 'SAMPA' and 'SAMPB' occurring on the given grid-level (K) are averaged in proportion to their sample lengths 'INT(N,LC(LL))', 'INTA' and 'INTB' to yield that grid-level's assay composite 'SACO'.

LINES 083 - 105:

The assay composites 'SACO' are given an additional array dimension (K) to identify their mineralization zones.

4.3 A LIST OF THE ASSAY COMPOSITES PROGRAMME

The following variables were dimensioned for the open pit proposal described in Section 7.1. They must be changed for any new deposit, as defined in Section 4.4.

LINE

001 REAL SAM2(2,40,18),INT(60,18),CORI(60),CORJ(60),
   1CUMINT(17),INTA,INTB,SACO(19,4,60),
   2ASAY1(19,60,4),ASAY2(19,60,4),KINT
   002 INTEGER LC(20),SURFAC(60)

----------COMMENT----------

The lower case constants in the following line are: ii = NBR * NK, jj = (number of recoverable minerals + 1) * 4, kk = NBR *
A dimension must be added to the constant 'KINT', if the vertical grid co-ordinate intervals are irregularly spaced. The intervals, in feet, between grid co-ordinates are stored in the new array 'KINT(K)', for DO loop 16. 'KINT(K)' must be substituted for 'KINT' wherever it appears in the program.

```plaintext
004   READ (5, J) KINT, MIN, MBH, N5AMP, NK, NZ
005   3 FORMAT (5*1,5I4)
006   DO 901 NS = 1, NZ
007   901 READ (5, 1) SAMP(1, NS, L), SAMP(2, NS, L), INT(N, L),
008          SAMP(1, L+1), SAMP(2, L+1), INT(N, L+1),
009          SAMP(1, L+2), SAMP(2, L+2), INT(N, L+2)
010   1 FORMAT (7F7*4, 2F7*4, 2F7*4, 2F7*4)
011   DO 803 NS = 1, NZ
012   803 X = CORI(N)
013   Y = CORJ(N)
014   2 FORMAT (12*N) X, Y
015   DO 152 L = 1, N5AMP
016   152 CURNT(L) = 0
017   DO 150 K = 1, MIN
018   150 DO 149 J = 1, MBH
019   149 CURNT(1) = INT(N, 1)
020   DO 10 L = 7, N5AMP
021   DO 22 L = INT(N, L) + INT(N, L)
022   4 CURNT(L) = INT(N, L)
023   DO 30 K = 1, KS
024   SUM = 0
025   INTA = 0
026   ITRB = 0
027   SAMPB = 0
028   SAMPB = 0
029   IF (SURFAC(N) - K) = 131, 131, 130
030   130 SACO(1, 2, 1) = 10 + 0
031   GO TO 16
032   131 SUZ = 0
033   DO 51 LL = 1, NSAMP
034   51 LC(LL) = 0
035   I = 0
```
036 \ L = 1
037 \ IF (K*KINT-CUMINT(L)) 6,8,7
038 \ INTA=INT(M,L)-CUMINT(L)+K*KINT
039 \ IF (INTA-KINT) 14,9,9
040 \ SAMP=SAMP(M,1,L)
041 \ GO TO 10
042 \ IF (L-4*SAMP) 101,100,100
043 \ 101 \ L=L+1
044 \ IF (INT(N,L)) 52,52,5
045 \ INTA=CUMINT(L)+K*KINT
046 \ IF (INTA-KINT) 53,112,112
047 \ 53 \ SAMP=0
048 \ GO TO 10
049 \ 100 \ IF ((K*KINT-CUMINT(L)-KINT) 111,111,112
050 \ 111 \ INTA=INT-CUMINT(L)
051 \ SAMP=0
052 \ SUM=INT(N,L)+INT(N,L)
053 \ SUM=INT(N,L)+SAMP(M,N,L)
054 \ IF (SUM-KINT) 150,150,151
055 \ 112 \ SACO(K,M,N)=0*0
056 \ GO TO 10
057 \ 8 \ IF (KINT-INT(...) 9,9,110
058 \ 9 \ SACO(K,M,N)=0*0
059 \ GO TO 10
060 \ 110 \ INTA=INT(L,L)
061 \ SAMP=SAMP(K,M,L)
062 \ 10 \ SUM=INT(L)
063 \ 150 \ LL=0
064 \ 11 \ IF (SUM-KINT) 12,151,13
065 \ 12 \ IF (L-1) 13,13,102
066 \ 102 \ SUM=SUMINT(N,L-1)
067 \ 151 \ INTA=INT-SUM
068 \ SAMP=SAMP(K,M,L)
069 \ IF (E) 120,120,140
070 \ 140 \ DO 1 \ LL=1,L
071 \ 1 \ IF (LC(LL)) 120,120,15
072 \ 15 \ SUM=SUMP=SAMP(M,L,LC(LL))INT(N,LC(LL))
073 \ 120 \ SUM=SUMP=SAMP*INTB
074 \ 120 \ SACO(K,M,N) SUM/INT

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A variable 'ASAYn' is added for each recoverable mineral at lines 086 and 087. In other words, ASAY1(K,N,M)=C, ASAY2(K,N,M)=0, ASAYn(K,N,M)=0 is added at line 086; and ASAY1(K,N,M) = SACO(K,1,N), ASAY2(K,N,M) = SACO(K,2,N), ASAYn(K,N,M) = SACO(K,n,N) is added at line 102, where 'n' equals the number of recoverable minerals.

The given mineral deposit was divided into four mineralization zones by a fault (line 092) and a water table (lines 093 and 094). The portion of this Program from lines 092 and 101 is tailored to the user's deposit, as explained in Section 3.4.1. Note the mixture of scales in line 092. 'CORT' and 'CORJ' are measured in feet but 'K' is measured in grid coordinates. This difference is accounted for by including scale factors in the expression's constants.
CHAPTER 4 ASSAY COMPOSITES PROGRAMME

The computer programme used to execute the assay composites computations described in Section 3.4.1 is listed in Section 4.3. The remaining Sections of Chapter 4 supplement this list with:

1) a description of how the Programme is prepared for execution in Section 4.1,
2) a chronological description of the Programme's computations in Section 4.2,
3) comments within the Programme list in Section 4.3, and
4) a glossary of the FORTRAN variables in Section 4.4.

4.1 Preparation of the Programme for Execution

The Assay Composites Programme, as listed in Section 4.3, was written for a regularly spaced and rectangular grid system. Additionally, its physical simulation routine was written for the deposit described in Section 7.1. The steps required to adapt the Programme to another deposit's:

1) grid system,
2) size and
3) mineralization zones

are listed below. These steps are cross referenced with the Programme list in Section 4.3 to locate the exact position of the modifications.

STEP 1:
The grid system is defined by choosing the unit interval
This glossary defines the FORTRAN variables of the Assay Composites Programme. The glossary is not complete; some transfer variables used to facilitate programming are not included in the list. The entries appear in the following format:

\[
\text{NAME}(i, j, k) = \{ \} \text{ definition } \{ \};
\]

\[
\begin{align*}
&i - \text{the name of the first subscript} \\
&j - \text{the name of the second subscript} \\
&k - \text{the name of the third subscript}
\end{align*}
\]

This arrangement of each variable’s definition is described as:

1) The variable’s FORTRAN name appears first.

2) The first set of brackets displays the number of subscripts or does not appear, if the variable is not an array.

3) The variable’s precision is displayed between the second set of brackets with the following abbreviations:

\[
\begin{align*}
a &\text{ 'D' is double precision, with eight storage locations;} \\
b &\text{ 'R' is real precision, with four storage locations;} \\
c &\text{ 'I' is integer precision, with four storage locations;} \\
d &\text{ 'Is2' is integer precision, with two storage locations.}
\end{align*}
\]
This glossary defines the FORTRAN variables of the Assay Composites Program. The glossary is not complete; some transfer variables used to facilitate programming are not included in the list. The entries appear in the following format:

\[ \text{NAME}(i,j,k) - \{ \} \text{ definition ( )}; \]

- \( i \) - the name of the first subscript
- \( j \) - the name of the second subscript
- \( k \) - the name of the third subscript

This arrangement of each variable's definition is described as:

1) The variable's FORTRAN name appears first.
2) The first set of brackets displays the number of subscripts or does not appear, if the variable is not an array.
3) The variable's precision is displayed between the second set of brackets with the following abbreviations:

- 'D' is double precision, with eight storage locations;
- 'F' is real precision, with four storage locations;
- 'I' is integer precision, with four storage locations; and
- 'I=2' is integer precision, with two storage locations.
1. A statement of the variable's significance appears next.

2. The third set of brackets displays the subroutines in which the name appears, if the same name is used for different variables. The brackets do not appear, if there is no conflict.

3. Finally, the dimensions of the variable's array are named. These dimensions only appear as required.

------------------------GLOSSARY------------------------

**ASAY**(i,j,k), **ASAY**(i,j,k), • • • ASAYn(i,j,k) - (R) the assay composites for n recoverable minerals;

i - grid-level
j - bore-hole identification number
k - mineralization zone identification number

**CRI**(i), **COR**(i) - j: the rectangular I and J co-ordinate locations, respectively, of the bore-holes in feet;

i - bore-hole identification number

**CUXINT**(i) - (R) the cumulative distance from the zero grid-level to each bore-hole sample;

i - sample identification number

**INT**(i,j) - (R) the sample length;

i - bore-hole identification number
j - sample identification number

**INTA**, **INTB** - (R) two variables used to adjust the top and bottom sample lengths between the grid-level boundaries, when the boundaries do not coincide with the cumulative sample length 'CUXINT'.

**KINT** - (R) the interval distance between the vertical grid co-ordinates

**LC**(i) - (R) a counter of the number of samples per grid-level per bore-hole.
i - sample identification number

Nbh - (I) the total number of bore-holes

Nk - (I) the total number of grid-levels

Nmin - (I) the total number of recoverable minerals

Nsample - (I) the total number of samples in the bore-hole with the most samples

Nz - (I) the number of mineralization zones

Saco(i, j, k) - (M) assay composites;
  i - grid-level
  j - mineral identification number
  k - bore-hole identification number

Samp(i, j, k) - (M) the proportional mineral content of each bore-hole sample. This array must have all sample values from the zero grid-level to the bore-hole bottom including zero values for air space, waste rock and unused array members, as explained in Section 3.4.1;
  i - mineral identification number
  j - bore-hole identification number
  k - sample identification number

Surfac(i) - (I) the bore-hole collar elevation in grid co-ordinates;
  i - bore-hole identification number
CHAPTER 5  GRADE PREDICTION PROGRAMME

The Grade Prediction Programme computes the spatial density of the given deposit's mineralization as described in subsection 3.4.2. A list of the source programme is given in Section 5.1. The remaining Sections of Chapter 5 supplement these lists with:

1) a description of how the Programme is prepared for execution in Section 5.1,
2) a chronological description of the Programme's computations in Section 5.2,
3) comments within the Programme list in Section 5.3, and
4) a glossary of the FORTRAN variables in Section 5.4.

5.1 PREPARATION OF THE PROGRAMME FOR EXECUTION

The Grade Prediction Programme, as listed in Section 5.3, must be tailored to a new deposit's:

1) grid system,
2) size, and
3) mineralization zones.

The following steps are cross-referenced with the Programme list to locate the exact position of these modifications:

STEP 1:

The grid system was defined partially for the Assay Composites Programme as noted in Section 4.1. The grid system dimensions planned to execute that Programme were the angular separation of the co-ordinate axes, the relative...
position of the zero grid-level with the deposit's surface and the co-ordinate intervals on the more nearly vertical axis. The co-ordinate intervals of the two horizontal axes are planned for the Grade Prediction Programme to complete the definition of the grid system. The ideal grid system dimensions are discussed in Subsection 3.2.2. Briefly, these dimensions must be small enough to give grid-block volumes that can accurately approximate the topography and pit surface. However, they also govern the total number of grid-blocks which must be less than the maximum INTEGER value of 32000. The limit of the grid system size and the co-ordinate intervals are given to the Programme by reading the values of the following constants (See line 005, Section 5.3):

1) 'IS' and 'JS' are the co-ordinate intervals of the I and J axes, respectively. These scale factors are used to reduce the actual distances to small integer values which are suitable for subscripts of FORTRAN arrays. These scale factors are constant regardless of the grid system type. For the irregularly spaced grid systems, only the axis perpendicular to the mineral trends needs to be irregularly spaced. The scale factors 'IS' and 'JS' specify the co-ordinate intervals in the plane perpendicular to the irregular axis. Therefore, they may be constants. For the non-rectangular grid systems, the input data are modified to read the bore-hole locations in grid co-ordinates instead of actual distances. The scale factors are unnecessary with this change. In which case, they may be deleted from the Programme expressions containing them or they may be set equal to one.

2) 'NI', 'NJ' and 'NK' are the maximum number of co-ordinate intervals in the I, J and K co-ordinate directions, respectively.

The co-ordinate intervals 'IS' and 'JS' must be mutually tolerant with the total grid size 'NI', 'NJ' and 'NK' to produce less than the maximum number of grid-blocks. This limit is 32000 and is imposed by the maximum value permitted for INTEGER variables. For example, assume that the given deposit is 1000 ft x 5000 ft x 1600 ft, and that the ideal grid system is regularly spaced with co-ordinate intervals of IS = 100 ft, JS = 100 ft, and the vertical interval equal to 50 ft. These dimensions produce a grid
with 41000 grid-blocks which exceeds the maximum 32000. The vertical scale factor is already defined to be suitable for the Assay Composites Programme. Therefore, the 'JS' and 'JS' scale factors must be increased to reduce the number of grid-blocks to the maximum allowable limit.

**STEP 2:**

The following search parameters must be defined for each mineralization zone, as described in Subsection 3.4.2:

1) 'NCLUS' - the maximum number of assay composites per estimate,
2) 'MAXD' - the maximum distance limit,
3) 'MEAR' - the minimum angular separation,
4) 'F' - the inverse proportional weighting factor, and
5) 'MIND' - the minimum distance limit.

These data are read in at line 007 in the Programme list. The interpolation in a particular mineral horizon automatically will be made only from the assay composites in that horizon, if the grid system is constructed as explained in Step 1 of Sections 4.1 and 5.1.

**STEP 3:**

The dimensions must be specified for the FORTRAN arrays and direct access data sets.

A) The dimensions are defined for the FORTRAN arrays in Section 5.4.

B) The dimensions are given for the DEFINE FILE statement in Section 5.3 for the direct access data sets.

**STEP 4:**

The boundaries of the mineralization zones simulated in the Assay Composites Programme are also simulated in this Programme with the same indicator arrays and polynomial equations. These surfaces are simulated between lines 052 and 062 of the Programme list in Section 5.3. The derivation
and substitution of these surfaces are described in Sections 3.4 and 4.1.

STEP 5:

The appropriate number of variables must be added for each potentially recoverable mineral as described by the comment in the Programme list at Line 008

5.2 SEQUENCE OF PROGRAMME COMPUTATIONS

LINES 001 - 039:
Specification and input statements

LINES 040 - 166:
The nested DO loops 001 and 05 estimate the point mineral densities 'GRAD1' and 'GRAD2' at each grid-intersection (i,j,k) from the assay composites 'ASAY1' and 'ASAY2'.

LINE 040:
The 'LCC' expression calculates the location in a direct access data set of the member (i,j,k).

LINE 050:
This IF statement branches the Programme around the remaining grade prediction computations, if the grid-intersection (i,j,k) is one of those above the deposit's surface.

LINES 051 - 075:
DO loop 17 calculates the radial distance 3 between the grid-intersection (i,j,k) and the assay composites (CORI, CORJ) in the same mineralization zone.

LINES 052 - 054:
These statements define the mineralization zone M in which the grid-intersection (i,j,k) occurs.

LINES 055 - 062:
The computations for the radial distance between grid-intersection and assay composite are bypassed, if the particular assay composite is not in the same zone as the given grid-intersection (I, J, K).

LINES 063 - 065:

If there is an assay composite 'ASAY1' and 'ASAY2' within a minimum distance 'MIND' of the grid-intersection (I, J, K), the point mineral density 'GRAD1' and 'GRAD2' of that grid-intersection is given the value of that assay composite.

LINE 076:
This statement selects the appropriate maximum number of assay composites 'NCLUS' per interpolation for the mineralization zone 'K'.

LINES 077 - 081:

DC loop 24 ranks the radial distances 'R' from the grid-intersection (I, J, K) to the nearest 'LMAX' assay composites in increasing order.

LINES 090 - 100:
The radial distances 'R' are lost in DO loop 24. Consequently, DO loop 26 is required to recalculate them, but only for the nearest 'LMAX' assay composites. In this way, the assay composites existing outside the maximum distance limit 'LMAX' are deleted from the grade prediction at the grid-intersection (I, J, K).

LINES 101 - 143:

DO loop 29 calculates two bearings 'B1' and 'B2' from the grid-intersection (I, J, K) to the selected 'NCLUS(L)' assay composites. The two bearings are computed from north and south base directions, respectively. The reason for this is that two assay composites close to and on either side of a base line will have a small angular separation but a large difference between their bearings. Therefore, the two bearings are required to test the angular separation close to the opposite base line.

LINES 145 - 149:
GO loops 17 and 18 test the angular separation between the selected assay composites \((RC(j))\). If the angular separation is less than the minimum \(\theta_{MIN}\) the assay composite furthest from the grid-intersection \((I,J,K)\) is deleted from the grade prediction by reassigning its radial distance a zero value.

LINES 165 - 173:

The point mineral densities \((GRAD1,GRAD2)\) at the grid-intersection \((I,J,K)\) are zero, if there are no assay composites within the maximum distance limit \(MAXD\). The 'WHITE' statement in Line 171 is included to display these grid-intersections. Normally, the grid-intersections on the outer edges of the grid system will be out of range of the search parameters, if the grid system extends past the exploration drilling, however, definition of the search parameters should be re-examined, if a grid-intersection that is within the deposit is out of range.

LINES 174 - 184:

The assay composites that have not been excluded by previous tests are averaged in inverse proportion to their radial distances \(\theta\) by the power \(P\) to estimate the point mineral densities \(GRAD1\) and \(GRAD2\).

5.3 A LIST OF THE GRADE PREDICTION PROGRAMME

001 REAL COP1(60),CP2(60),MAXD(4),ASAY1(19,60,4),
1ASAY2(18,60,4),R(60),G(60),R1(60),ATAN,ABS, SQRT,
2STCOS(10)
002 INTEGR MCLOS(4),NC(60),T,SCORE(750)
003 INTG2*2 TOPO(10,25,18)

The parameters of the following DEFINE FILE statement are:

1) \(ii = M1\#M2\#M3\),
2) \(jj = (\text{number of recoverable minerals})\#1\),
3) \(kk = M2\#M4\),
4) \[ n = (\text{number of recoverable minerals} + 1) \times 4, \]

5) \[ m = \text{MBH} * 4 \]

---

```
004 \hspace{1cm} \text{DEFINE FILE 10(ii, jj, L, IJ), 11(kk, ll, L, JI),}
005 \hspace{1cm} 112(m, h, L, JJ)}
006 \hspace{1cm} \text{READ (5, 1) NI, NJ, KK, NBH, NZ, IS, JS}
007 \hspace{1cm} 1 \text{ FORMAT (7F4)}
008 \hspace{1cm} \text{READ (5, 1004) (NCLUS(M), MAXD(M), BEAR(M), P(M),}
009 \hspace{1cm} \text{1IND(m), m=1, 4)}
010 \hspace{1cm} 1004 \text{ FORMAT (12, 6*1, F7*4, F3*1, F4*2)}
```

---

The appropriate number of variables and expressions must be added to the program for each recoverable mineral. Parallel expressions are created for \text{ASSAY}(K,N,M), \text{ASSAY}, \text{GRADN} and \text{ASSAY}, \text{where 'n' identifies each recoverable mineral, at Lines 012, 018, 020, 073, 075, 169, 180, 184 and 185.}
N = N
1055 II = I
1053 II = 0
1054 TOPO(I, J, K) #=II
090 EOCVAR=0
091 DO 901 K = 1, NK
092 DO 901 J = 1, NJ
093 DO 95 I = 1, NI
094 GRAD1 = 0
095 GRAD2 = 0
096 DO 161 N = 1, NBJ
097 IF (TOPO(I, J, K) ) = 46, 46, 999
098 IF (11096*3 + 2*652*CORI(N) - 5*55*CORJ(N) - 150*5*K ) = 17, 75, 75
099 2015
100 IF (13096*3 + 2*652*CORI(N) - 5*55*CORJ(N) - 150*5*K ) = 75, 17, 17
101 2016
102 IF (13096*3 + 2*652*CORI(N) - 5*55*CORJ(N) - 150*5*K ) = 75, 17, 17
103 IF (ABS(CORI(N) - IS*1) = MIND(X) ) 90, 90, 91
104 IF (ABS(CORJ(N) - JS*J) = MIND(X) ) 1016, 1016, 32
105 IF (ABS(CORJ(N) - JS*J) = MIND(X) ) 93, 93, 1015
106 IF (ABS(CORI(N) - 15*1) )
107 GO TO 17
108 GO TO 17
109 GO TO 17

Lines 052 - 062 are tailored to the user's mineralization zones.
070 1045 \[R(N) = \text{SORT}\left(\{\text{ABS} \left(\text{CORI}(N) - IS*1\right)\}**2 + \{\text{ABS} \left(\text{CORJ}(N) - IS*1\right)\}\right)\]

071 \GO TO 17

072 1016 GRAD1 = ASAY1(K, M, N)

073 GRAD2 = ASAY2(K, M, N)

074 \GO TO 46

075 17 \CONTINUE

076 LMAX = NCLSUS(M)

077 LD 20 L = 1, LMAX

078 DO 1018 J = 1, NBH

079 IF \(R(N)\) \(=\) 1018, 1015, 19

080 1018 \CONTINUE

081 19 SMEL = R(N)

082 20 DO 21 J = 1, NBH

083 IF \(R(N)\) \(=\) 23, 23, 21

084 21 IF \(R(N) - SMEL\) \(=\) 22, 22, 23

085 22 NC(L) = N

086 \GO TO 23 \CONTINUE

087 23 \CONTINUE

088 171 R(NC(L)) = 90000000

089 24 \CONTINUE

090 DO 26 L = 1, LMAX

091 IF \(\text{ABS} \left(\text{CORI} \left(\text{NC} \left(\text{L} \right) \right) - IS*1\right) - \text{NIND} \left(\text{K} \right)\) \(=\) 180, 180, 181

092 180 R(NC(L)) = ABS(CORJ(NC(L)) - JS*1)

093 \GO TO 184

094 181 IF \(\text{ABS} \left(\text{CORJ} \left(\text{NC} \left(\text{L} \right) \right) - JS*1\right) - \text{NIND} \left(\text{K} \right)\) \(=\) 192, 192, 183

095 182 R(NC(L)) = ABS(CORI(NC(L)) - IS*1)

096 \GO TO 184

097 183 R(NC(L)) = SQRT((ABS(CORI(NC(L)) - IS*1)**2 + (ABS(CORJ(NC(L)) - JS*1))**2)

098 184 I" = (3(NC(L)) - LAXD(K)) 26, 26, 25

099 25 R(NC(L)) = 0

100 26 \CONTINUE

101 DO 29 L = 1, LMAX

102 IF \(R(NC(L))\) \(=\) 29, 29, 27

103 27 IF \(\text{CORI} \left(\text{NC} \left(\text{L} \right) \right) - IS*1\) \(=\) 1102, 100, 1101

104 1101 IF \(100\text{CORI} \left(\text{NC} \left(\text{L} \right) \right) - IS*1\) \(=\) 106, 103, 103

105 1102 IF \(\text{CORI} \left(\text{NC} \left(\text{L} \right) \right) - IS*1\) \(=\) 103, 304, 104

106 103 B(NC(L)) = ATAN(1(ABS(CORJ(NC(L)) - JS*1)) / \(1(\text{ABS} \left(\text{CORI} \left(\text{NC} \left(\text{L} \right) \right) - IS*1\right)\))

107 \GO TO 28

108 104 B(NC(L)) = 4*71239*ATAN(1(ABS(CORJ(NC(L)) - JS*1)) / \(1(\text{ABS} \left(\text{CORJ} \left(\text{NC} \left(\text{L} \right) \right) - JS*1\right)\))

109 \GO TO 28

110 105 B(NC(L)) = 1*14159*ATAN(1(\text{ABS} \left(\text{CORJ} \left(\text{NC} \left(\text{L} \right) \right) - JS*1\right) - IS*1)) / \(1(\text{ABS} \left(\text{CORI} \left(\text{NC} \left(\text{L} \right) \right) - IS*1\right)\))
111 GO TO 28
112 106 B (NC (L)) = 1.57070 + ATAN ((ABS (CORI (NC (L)) - IS*J)) /  
   1 (ABS (CORJ (NC (L)) - IS*J))
113 GO TO 28
114 300 IF (CORI (NC (L)) - IS*J) # 301, 308, 302
115 301 D (NC (L)) = 3.14159
116 GO TO 28
117 302 D (NC (L)) = 0.0
118 GO TO 28
119 303 E (NC (L)) = 1.57080
120 GO TO 28
121 304 P (NC (L)) = 4.71238
122 28 IF (CORJ (NC (L)) - JS*J) = 201, 310, 202
123 201 IF (CORI (NC (L)) - JS*J) = 203, 313, 204
124 202 IF (CORI (NC (L)) - JS*J) = 206, 314, 205
125 203 BI (NC (L)) = ATAN ((ABS (CORJ (NC (L)) - JS*J)) /  
   1 (ABS (CORI (NC (L)) - IS*J)))
126 GO TO 29
127 204 BI (NC (L)) = 1.57070 + ATAN ((ABS (CORI (NC (L)) - IS*J)) /  
   1 (ABS (CORJ (NC (L)) - JS*J)))
128 GO TO 29
129 205 BI (NC (L)) = 3.14159 + ATAN ((ABS (CORJ (NC (L)) - JS*J)) /  
   1 (ABS (CORI (NC (L)) - IS*J)))
130 GO TO 29
131 206 BI (NC (L)) = 4.71238 + ATAN ((ABS (CORI (NC (L)) - IS*J)) /  
   1 (ABS (CORJ (NC (L)) - JS*J)))
132 GO TO 29
133 310 IF (CORI (NC (L)) - IS*J) # 311, 309, 312
134 311 BI (NC (L)) = 0.0
135 GO TO 29
136 312 BI (NC (L)) = 3.14159
137 GO TO 29
138 313 BI (NC (L)) = 1.5708
139 GO TO 29
140 314 BI (NC (L)) = 4.71238
141 GO TO 29
142 308 WRITE (6, 39)
143 29 CONTINUE
144 309 FORMAT (' TILT')
145 DO 33 L = 1, LMAX
146 DO 33 T = 1, LMAX
147 IF (R (NC (L))) # 33, 13, 1120
148 1120 IF (R (NC (T))) # 33, 33, 121
149 121 IF (ABS (W (NC (T))) - n (NC (L))) - BEAR (T)) = 30, 33, 33
150 30 IF (R (NC (T)) - P (NC (L))) # 31, 33, 32
151 31 R (NC (L)) = 0
This glossary defines the FORTRAN variables of the Grade Prediction Program. The glossary is not complete; some variables used to facilitate programming are not included in the list. The format of the entries is defined in Section 5.*4 GLOSSARY
GLOSSARY

ASSAY₁(𝑖, 𝑗, 𝑘), ASSAY₂(𝑖, 𝑗, 𝑘), ..., ASSAYₙ(𝑖, 𝑗, 𝑘) - (R) the assay composites for '𝑛' recoverable minerals:

- i - grid-level
- j - bore-hole identification number
- k - mineralization zone identification number

ASSAY₁, ASSAY₂, ..., ASSAYₙ - (R) a variable used to transfer assay composites from external storage to internal storage for '𝑛' recoverable minerals.

𝐵(𝑖) and 𝐵¹(𝑖) - (R) the bearings from north and south, respectively, between grid-intersection and assay composites:

- i - bore-hole identification number

BEAR(𝑖) - (R) a search parameter which is the minimum allowable angular separation between assay composites measured in radians:

- i - mineralization zone identification number

CORI(𝑖) and CORJ(𝑖) - (R) the I and J co-ordinate locations of the bore-holes in feet:

- i - bore-hole identification number

GRAD₁, GRAD₂, ..., GRADₙ - (R) the point mineral densities at a grid-intersection for '𝑛' recoverable minerals.

IS and JS - (I) the scale factors for converting between actual distances and grid co-ordinates.

LOC - (I) the location of a member in a direct access data set.

MAXD(𝑖) - (R) a search parameter which is the maximum distance between grid-intersection and assay composites:

- i - mineralization zone identification number
MIND(i) - (8) a search parameter which is the minimum distance between assay composite and grid-intersection:

1 - mineralization zone identification number

NBH - (I) the total number of bore-holes

NC(i) - (1) a variable used to identify the assay composites to be included in any particular point mineral density estimate:

i - bore-hole identification number

NCLUS(i) - (1) a search parameter which equals the maximum number of assay composites to be included in each point mineral density estimate:

i - mineralization zone identification number

NL, NJ and NK - (I) the maximum i, J and K grid co-ordinates, respectively

NZ - (I) the total number of mineralization zones

P(i) - (?1) a search parameter which is the inverse proportional weighting factor:

i - mineralization zone identification number

R(i) - (?7) the radial distances between grid-intersection and its surrounding assay composites:

i - bore-hole identification number

TOPO(i,j,k) - (112) an indicator array used to simulate the deposit's surface:

i,j,k - the I, J and K grid co-ordinates, respectively
The theories described in this Chapter are used to work out how to conduct the open pit mining operations on the user's surface deposit. Economic, mathematical and mine design theories are combined with computer programming techniques to yield a pit planning system based on maximum total profit and present values. This system is executed with the Mine Plan Programme listed in Chapter 8. The MP Programme produces a set of schedules and plans that represent an optimum open pit mining solution for the user's deposit. The flow diagram in Table 6.1 illustrates the computational sequence of the MP Programme. Chapter 5 is organized more or less parallel to this diagram.

6.1 INPUT DATA

There are two types of input data required to initiate the MP Programme. These data types are distinguished by being variable or fixed. They are described generally in the following Subsections and are listed specifically in Chapter 8.

6.1.1 VARIABLE INPUT DATA

The variable input data are the information requiring decisions by the user. These data are collectively called the 'INPUT MINING PROPOSAL' in Table 6.1. This proposal is the selection of all conceivable plant production capacities of the pit, concentrator and refinery, and an open pit mining method.

PLANT PRODUCTION CAPACITIES:

A routine has been written into the MP Programme which chooses the plant production capacities yielding the greatest present value. This choice is made from the possibilities given by the user. The user selects any number of plant
**TABLE 6.1**

capacities, and give the investment capital and overall
recovery efficiency of each combination. The investment capital must be discounted to the first production year at the opportunity cost of investment capital. Table 6.2 illustrates an example of this input data.

---

### THE CAPITAL COST OF 27 PLANT CAPACITY COMBINATIONS

<table>
<thead>
<tr>
<th>RECOVERY</th>
<th>EFFICIENCY</th>
<th>CAPACITY</th>
<th>SITE</th>
<th>LOCATION</th>
</tr>
</thead>
<tbody>
<tr>
<td>95%</td>
<td>1.5</td>
<td>1.6</td>
<td>1.7</td>
<td></td>
</tr>
<tr>
<td>90%</td>
<td>1.5</td>
<td>1.6</td>
<td>1.7</td>
<td></td>
</tr>
<tr>
<td>85%</td>
<td>1.5</td>
<td>1.6</td>
<td>1.7</td>
<td></td>
</tr>
</tbody>
</table>

**PIT CAPACITY=7 MILLION TONS PER YEAR**

**CAPITAL COSTS IN MILLIONS OF CASH UNITS**

(T)=capacities in thousands of tons per year

(M)=capacities in millions of tons per year

**TABLE 6-2**

The plants are defined in the following manner according to production processes and costing methods rather than by location:

**A)** The **pit**: The pit production is given as the total output including ore and waste. The pit production processes are all those common to both ore and waste excavation.

**B)** The **concentrator**: The concentrator is that part of the production processes which distinguishes the ore from the raw material of the pit. As opposed to refining,
concentration is the group of processes that can be charged without regard to the ore's mineral content. For example, it will include the difference between the productive effort of ore and waste excavation, crushing, grinding, floating, leaching, control sampling, and so forth. However, if the cost of any of these component processes becomes a function of mineral grade, it becomes part of the refinery production.

C) The refinery: The refinery is the group of purification processes that are distinguished from the concentrator production by the manner in which the costs are charged. Refining includes the processes that depend on the ore's mineral content. For example, it will include smelting and marketing.

OPEN PIT MINING METHOD:

The second part of the 'INPUT MINING PROPOSAL' requires the following decisions from the user:

1) Pit layout:
   a) the minimum angles between vertical, and the safe and working pit walls;
   b) the height and width of the pit benches, and
c) the number of exposed ore benches and working faces per ore bench.

2) Ore Excavation Sequence

Safe wall-angles:

The safe wall-angles define the minimum allowable angles to the vertical that the final pit surface may have to prevent it from sliding as a result of the slope stability conditions. Any number of safe wall-angles may be specified for changing slope stability conditions through the user's deposit. For example, safe wall-angles can be defined for each region of the pit illustrated in Fig. . The three regions are delimited by their orientation with the weak layer strength direction of the deposit's bedding planes.

Working wall angles:

The working wall-angles define the allowable minimum angle
to the vertical that the pit wall must exceed until they intersect with the final surface of the pit. This specifies the overall angle of the intermediate operating slopes which are required to expose the ore benches. Any number of these working wall-slopes may be specified for different working conditions through the pit. The relationship between the safe and working pit walls is illustrated in Fig. 6.2.

Bench Height and Width:

The size of the excavating equipment planned for the pit will determine the bench height and width. These dimensions are specified to equal the excavating equipment's working space requirements. The computer simulation of the pit is considerably simpler, if the vertical and transverse grid-block dimensions are made equal to the bench height and width as recommended in Subsection 3.2. Otherwise, the grid-blocks have to be built up to equal the bench dimensions.

The pit walls formed by staggering the benches at an offset equal to their height and width produces a fixed wall-angle as shown in Fig. 6.3A. However, a fixed wall-angle does not permit any freedom for planning:

1) the stripping ratio of ore to waste and
2) different intermediate and final pit walls in the various parts of the pit.

Therefore, the MP Programme is written to allow the user to specify the bench dimensions independently of the wall-angles. The Programme designs the pit walls at the specified angles by grouping the benches as follows:

1) The benches are grouped horizontally, as illustrated in Fig. 6.3B, when the pit walls are flatter than the slope fixed by a single offset of the benches. The horizontal bench grouping does not create any operational problems. Its only effect is to increase the horizontal working space on the grouped benches.

2) The benches are grouped vertically, as illustrated in Fig. 6.3C, if the pit walls are steeper than the slope fixed by a single bench offset. The bench grouping causes local slope variations of the pit walls alterna-
ting between steeper and flatter wall-angles than that specified by the user. The locally alternating slopes result in the desired overall wall-angle of the pit surface as illustrated in Fig. 68-30. However, the user must be confident that the pit walls can stand at the local slope exaggerations, if he specifies a wall-angle less than the single bench offset.

The pit wall design function of the MP Program permits the following specifications of the 'INPUT MINING PROPOSAL' to be made independently:

1) the bench height and width as defined by the excavation equipments working space requirements,

2) the working wall-angles as determined from the desired stripping ratios and the slope stability conditions, and

3) the safe wall-angles as defined by the slope stability conditions.

This design capability frees the user from matching the working space requirements to the stripping ratios within the limitations of the slope stability conditions.

Number of Exposed Ore Benches and Working Faces per Exposed Ore Bench:

The number of exposed ore benches and ore working faces per exposed ore bench is interrelated with:

1) the production capacity of the ore excavating equipment,

2) the production capacity of the concentrator, and

3) the bench height and width.

The total number of ore working faces must be capable of providing the concentrator input production. For a particular concentrator, this number equals the concentrator capacity divided by the production capacity per ore working face. However, a large range of concentrators may have been selected for analysis by the MP Program. Consequently, it may be difficult to match the number of ore working faces to
the whole range of concentrators. If this is so, then:

1) assume the number of ore working faces,
2) execute the EP Programme to select the best concentrator capacity,
3) check this concentrator capacity against the number of ore working faces, and
4) if necessary, adjust the number of ore working faces and re-execute the EP Programme.

Ore Excavation Sequence:
The simulation of the pit production requires the following decisions:

1) where should ore excavation begin?
2) in which direction should the ore working faces advance from the starting point?
3) which grid-blocks are ore and which are waste?
4) in which order should the ore grid-blocks be removed?
5) what overburden should be removed to expose the ore?
6) how much ore and waste should be excavated in any given year? and
7) in which year should the grid-blocks be mined?

The last five decisions are made by the XP Programme to maximize the present value of the user's deposit. The first two decisions are made by the user and are part of the 'INPUT MINING PROPOSAL'. The starting point of the ore excavation should be the most advantageous in relation to the location of the waste dumps, plant facilities, property lines, and so forth. The ore working face advance should be parallel to the long axis of the pit. This will minimize the number of changes required for the haulage roads. The pit production simulation routine automatically changes the direction of the working face advance on alternate ore benches in both the vertical and horizontal directions.
minimizes the movements of the ore excavation equipment.

The "INPUT MINING PROPOSAL" for a particular deposit can be
developed with the MP Programme as a planning aid. The
unknown or undecided items can be assumed and an open pit
mining solution computed; then, the assumptions are changed
to compute another solution. The cycle of input assumption
-- computation -- input adjustment -- re-computation is
continued until the best open pit mining solution results.
This rapid execution facility of the Programme provides a
means for quickly assessing the effects of changing the
mining proposal.

6. 1. 2 FIXED INPUT DATA

The fixed input data is characterized by being constant for
any given deposit. There are the following three categories:

1) spatial mineral distribution,
2) economic data, and
3) physical data.

Spatial Mineral Distribution:
The spatial mineral distribution is supplied by the computer
programmes described in Chapters 4 and 5. Recall that the
Grade Prediction Programme stores the spatial mineral distri­
bution in a direct access data set. This data set is read
by the MP Programme as part of its input data.

Economic Data:
Elements of the deposit's cost structure must be supplied by
the user. The cost structure is simulated in the Programme with the standard costing method. This method is
the computation of each grid-block's net cash value to the
venture from standard costs and prices. These standard
are adjusted and totalled individually to give the total net
cash value of each grid-block according to the peculiar
conditions under which it is mined. The standard costs, and
prices, and the peculiar costing conditions are the cost
structure elements that must be supplied by the user.

The standard costs and prices are given by the user in unit
terms and adjusted according to each grid-block's:

1) time of excavation - for the time inflation of costs and prices,

2) location - for the difference in the excavation costs in various parts of the pit, and

3) mineral content - for the differences in the metallurgical characteristics of the ore.

The adjusted standards are used to compute the particular grid-block's net cash value along with its unit weight, mineral content and overall recovery efficiency. The net cash value computations are discussed in more detail in Section 6.5.

The 'ANNUAL UNIT COSTS' defined in Table 6.3 are the arithmetical means of all the adjusted 'STANDARD UNIT COSTS' assigned in each year of the mine's life span. The annual unit costs represent an estimate of the yearly pit, concentrator, and refinery costs. The standard and annual unit costs are defined differently for two separate applications in the KP Programme. Consequently, Table 6.3 is presented to detail the differences in their definitions to avoid any confusion.

Physical Data:

The physical data is the last category of the fixed input. This data includes the deposit's surface topography and any geologic or structural features that will interfere with the development of an open pit mining solution. The following computational routines of the KP Programme are the ones which could be affected:

1) costing — for example, by the location of the water table,

2) final pit surface design — for example, by variable slope stability conditions, and

3) pit production — for example, by rock movements along a fault plane.

The KP Programme is written to recognize any of these
features so that it can modify the open pit mining solution for their occurrences. The programming techniques described in Section 3.1 are used to simulate these structural features. Application of the structural simulation techniques is explained later when the various computational routines that use them are discussed.

6.2 OBJECTIVES OF THE OPEN PIT PLANNING SYSTEM

The objectives of the open pit planning system are:

1) to design the final pit surface that will yield the maximum total profit and
2) to furnish a solution for exploiting the pit that will maximize the mine's present value.

This planning system is a development following from the previous work in the field. The latest and most advanced system (1, 5, 7) designs the final pit surface with the maximum total profit. This thesis introduces the time dimension and the exploitation solution. In particular, the mine's cash flow and the time value of money are included in the open pit planning system.

The objectives can be described with the following analytical development:

Let \( C(x, y, z) \) be defined in a closed three-dimensional region \( P \) encompassing the user's deposit. The region \( P \) is subdivided into subregions, called parallelepipeds, by constructing a grid consisting of planes parallel to the \( xy \), \( yz \), and \( xz \) planes. The function \( C(x, y, z) \) denotes the net cash value of \((x, y, z)\), which is a point or grid-block in the three-dimensional region \( P \).

Let \( P(C(x, y, z), t) \) be defined as the net cash value density function \( C(x, y, z) \) extended to include the time period \( t \). The function \( P(C(x, y, z), t) \) denotes the net cash value of grid-block \((x, y, z)\) for the year \( t \) during which it is mined. The domain of the function \( P(C(x, y, z), t) \) is restricted:

1) to the family of pit surfaces whose walls are not steeper than the minimum safe wall-angle at any point,

2) to the path specified by the user for the sequence of ore excavation,

3) by the prevailing cost and price inflation rates and the opportunity cost of investment capital.

The objective is to find:

1) the pit volume \( V \) of the region \( P \), and

2) the mine production schedule \( t \) of the time period \( T \) that will maximize the mine's present value \( P \) of the following multiple integral with the preceding restrictions:
The three innermost integrals are evaluated first to find the final pit surface yielding a maximum total profit. Then, the remaining integrals are performed to find the exploitation solution that will maximize the mine's present value. Unfortunately, a single relationship does not exist in the function \( F(C(x, i, t)) \). Additionally, the limits for the entire period \( T \) are unknown, or in other words, the mine's life span cannot be defined before the integration is executed. Therefore, the multiple integral's solution is executed indirectly with numerical, economic, mathematical and mine design methods which were devised to make up the DP Programme.

The solution is executed with the converging iteration illustrated in Table 6-1. The iteration converges towards a definite open pit mining solution following from certain initial assumptions.

So to initiate the solution, it is assumed that the entire deposit is mined in the first year. This assumption is made for the data sets that are time dependent, namely the two time-space distributions of minerals and net cash values, respectively, and the annual unit production costs.

To avoid confusion, it is important to define these preceding data sets:

A) Time-space (T-S) distributions: The T-S distributions provide the variation of mineral quantities and net cash values by their spatial location and by the period in which they are mined. The time and space distributions are used to account for the differences in the mineral quantities and net cash values available for exploitation at any period in the pit's life span. The two T-S distributions are defined somewhat differently. Firstly, the T-S net cash value distribution is a four-dimensional density function. The point density is defined by its three-dimensional location in the deposit and by the production year of the material represented by that point. Recall that the T-S net cash value distribution was defined previously as \( F(C(x, i, t)) \). According to the T-S mineral distribution, the entire grade interval per time frame interval per size
The location of these tonnages is given by their association with the time in which they are mined. The two distributions are described more thoroughly in Sections 6.1 and 6.5.

B) Annual unit production costs: These costs are the annual excavation, concentration and refining costs for throughput tonnage of the respective production plants. These terms were defined earlier in Table 6.3. Normally, they are estimated from the T-S net cash value distribution as described in Subsection 6.1.2. However, the estimates cannot be made until the T-S net cash value distribution has been computed. These computations are the fourth step in the iterative cycle as illustrated in Table 6.1. Meanwhile, the unit production costs are required for the preceding cut-off grades, production targets and plant capacities selection computations. Consequently, they are assumed for the first cycle of the iterative solution until they are replaced in the next cycle.

The assumptions required to initiate the MP Programme for the preceding data sets are:

1) The time dimension is assumed to be the first production year for the T-S distributions.

2) The space dimensions of the pit are assumed to include the entire region encompassed by the grid system for the T-S distributions.

3) An estimate of the unit production costs for the first production year is given by the user.

The time assumption represents a pit life span of one year; therefore, the unit production costs assumption is made only for that year. The three assumptions are replaced with better estimates as the solution progresses through the iterations.

The whole process can be conceived as a closed loop in which all computations depend upon each other. Obviously, it is necessary to break into this loop to initiate the solution. In the end, the iteration converges as corrections to the assumptions become compatible with the final pit surface.
design and exploitation solutions. Table 6-1 illustrates the process qualitatively.

The routines illustrated there can be described by means of an analogy with the functions and operations implied by the multiple integral developed previously:

\[ PV = \int \int \int (C(x, y, z), t) \, dt \]

The assumptions, as listed above, are made for the limits of the outermost integral. Based on these assumptions, it is possible to compute the path T for the integral. This path is restricted by the ore excavation sequence given by the user. The remaining decisions required to formulate the path are computed by the following routines to maximize the mine's present value:

1) PLANT CAPACITIES SELECTION,

2) CUT-OFF GRADES COMPUTATIONS, and

3) PRODUCTION TARGET COMPUTATIONS.

The point density of the function \( P(C(x, y, z), t) \) are computed in the routine T-S X - CASH VALUE DISTRIBUTION COMPUTATIONS assuming the time dimension. Then, the operation

\[ \int \int \int (C(x, y, z), t) \, dt \]

is executed to find the profit yielding the maximum total profit over the region \( F \), i.e., the pit system, by the routine PIT PRODUCTION DESIGN. Thereafter, the operation

\[ f(T) \]

is performed by the routine PIT PRODUCTION SIMULATION along the path \( T \). As the solution progresses through the iteration, more realistic assumptions are replaced each cycle by better estimates. This is achieved:

A) with the time computed by the routine PIT PRODUCTION SIMULATION,
with the final pit surface (for the space dimensions) computed by the routine FINAL PIT SURFACE DESIGN, and

C) with the annual unit production costs estimated by the routine T-S NET CASH VALUE DISTRIBUTION COMPUTATIONS.

The solution converges when the results of the preceding iteration cycle equal the results of the ultimate cycle. These results are listed as the OUTPUT in Table 6.1.

6.3 TIME-SPACE DISTRIBUTIONS OF MINERALS

The FORTRAN data set for the T-S mineral distribution is named:

ORES(M,N)

where the subscript:

'M' identifies the mineral grade increment, and

'N' identifies the pit increment or the time period during which the minerals will be mined.

Each array member is the weight of pit material with a mineral content 'M' which will be mined in the Nth period or pit increment. 'M' is given by the joint densities of the spatial mineral distribution 'GRAD' computed by the Grade Prediction Program (listed in Chapter 5). 'M' is given by the excavation times computed by the routine PIT PRODUCTION SIMULATION (see Table 6.1). 'N' identifies a horizontal cross-section of the deposit when the ORES data set is assumed in order to start the iterative solution. Subsequently, 'N' identifies a period in the mine's life span.

6.3.1 INITIAL ASSUMPTION OF THE T-S MINERAL DISTRIBUTION

As noted previously, the subscript 'N' of the ORES data set is unknown at the start of the iterative solution. A pit surface has not been designed and its dimensions are unknown. Therefore, all the material within the grid system is included in ORES. The pit production has not been simulated and the time dimension is unknown. Therefore, this is estimated by totalling the mineral quantities of succes-
with the final pit surface (for the space dimensions) computed by the routine FINAL Pit SURFACE DESIGN, and

C) with the annual unit production costs estimated by the routine T-S NET CASH VALUE DISTRIBUTION COMPUTATIONS.

The solution converges when the results of the preceding iteration cycle equal the results of the ultimate cycle. These results are listed as the OUTPUT in Table 6.1.

6.1 TIME-SPACE DISTRIBUTIONS OF MINERALS

The FORTRAN data set for the T-S mineral distribution is named:

ORERES (M,N)

where the subscript:

'M' identifies the mineral grade increment, and

'N' identifies the pit increment or the time period during which the minerals will be mined.

Each array member is the weight of pit material with a mineral content 'M' which will be mined in the Nth period or pit increment. 'M' is given by the point densities of the spatial mineral distribution 'GRAD' computed by the Grade Prediction Program (listed in Chapter 5). 'N' is given by the excavation times computed by the routine PIT PRODUCTION SIMULATION (see Table 6.1). 'N' identifies a horizontal cross-section of the deposit when the 'ORERES' data set is assumed in order to start the iterative solution. Subsequently, 'N' identifies a period in the mine's life span.

6.3.1 INITIAL ASSUMPTION OF THE T-S MINERAL DISTRIBUTION

As noted previously, the subscript 'N' of the ORERES data set is unknown at the start of the iterative solution. A pit surface has not been designed and its dimensions are unknown. Therefore, all the material within the grid system is included in ORERES. The pit production has not been simulated and the time dimension is unknown. Therefore, this is estimated by totaling the mineral quantities of succes-
DO 11 N=1,MAXN
   K1=4*2
   K2=K1-3
   DO 10 K=K2,K1
   DO 10 J=1,NJ
   DO 10 I=1,N1
   IF (TCPO(I,J,K)) 10,10,5
   5 M=GRAD(I,J,K)*GS
   IF (M-1) 6,7,7
   6 M=1
   7 ORESN(M,N,K)=OREN(S(M,N,K)*TONF
   10 CONTINUE
   11 CONTINUE

where (in order of appearance):

'N' identifying the pit increments

'MAXN' is the total number of pit increments

'K1' and 'K2' identify the lower and upper grid-levels between which the mineral contents are totalled for pit increment 'N'

'I', 'J', and 'K' are the grid co-ordinates

'NI' and 'NJ' are the maximum grid co-ordinates in the 'I' and 'J' co-ordinate direction, respectively

'TCPO' is an indicator array for the deposit's surface

'GRAD' is the point density of the spatial mineral distribution at (I,J,K)

'GS' is a scale factor

'TONF' is the unit weight of each grid-block

DO loop eleven provides the grid-level parameters 'K1' and 'K2' for DO loop ten. Note that these statements were written to include the pit material of every four grid-levels in one pit increment 'N'.

Minor groups of grid-levels from the surface down. The subscript 'K' denotes the grid-level groups. The following routine illustrates how this assumption is made:
In the loop ten, the 'TCFD' indicator array is used to branch past the remaining computations, if the grid-intersection \((i,j)\) is above the deposit's surface. Otherwise, the point mineral density, 'GRAN', are multiplied by a scale factor, 'GS', to increment the mineral grades and to make their values suitable for the integer sub-script \(n\) of 'CHocks'.

This conversion of real to integer precision is a truncation. Therefore, the tonnages in each grade increment are represented by their lower boundary. For example, assume that one of the point densities of 'GRAN' is 0.36 and 'GS' equals 106. Then, the tonnage represented by this point density is added to grade increment \(n=1\).

The second IF statement checks the value of \(n\) to see if it is less than one. If so, statement number six reassings \(n\) a value of one to prevent 'ORESES' from being out of subscript range. Consequently, 'GS' must be defined small enough so that all array members where \(n=1\) can represent the pit material with a mineral content range of 0\(<\mathrm{v}\<2\).

The unit weight 'TONF' of the grid-blocks is programmed as a constant in the preceding routine. However, if this unit weight varies through the deposit, there are two possibilities for programming the variation: Firstly, an indicator array can be created to store the changing unit weights, as described in Section 3.1. The array is constructed by adding a subscript to 'TONF' for each co-ordinate direction in which it varies. For example, the constant used in the preceding routine becomes 'TONF(K)', if the unit grid-block weight varies with depth only. Each array member \(K\) contains the grid-block unit weight for the respective \(K\) grid-level.

The second technique for simulating the variation of the unit grid-block weights is the derivation of surfaces. These surfaces are used to simulate the boundaries between the unit weight variation. This simulation technique was explained in greater detail in subsection 3.1.2. As pointed out in that subsection, it has a computationally slow execution time and is difficult to devise for irregular surfaces. However, the technique is effective for simple surfaces and it does conserve internal computer storage space. The technique can be demonstrated with a deposit having two different unit weights. In this deposit, the rock density was found to be different on either side of grid-level five. The following routine will identify the appropriate unit...
grid-block weight:

```
IF (F-S) = 1
    1 NW=1
    GO TO 3
2 NW=2
```

10 ORESW (M, N) = ORESW (M, N) + TONI (MN)

The IF statement expression represents the boundary between the two unit weights. The 'TONI' array members contain the unit weight of the grid-blocks whose relative positions with grid-level five are given by their 'K' grid co-ordinates. The preceding routine is rather simple but becomes complicated quite rapidly with more boundaries and irregular surfaces. This simulation technique is demonstrated again in the routine listed in the later part of this Section.

6: 3: 2 EQUIVALENT MINERAL CONTENTS

Note in the preceding computational routine for 'ORESW' that there was only one recoverable mineral reported. A deposit containing several recoverable mineral presents a difficulty for computing 'ORESW'.

The net cash value of any particular grid-block depends on its total recoverable mineral content. The cut-off grades must be a function of the net cash value of all recoverable minerals occurring in a particular grid-block. This problem is overcome by expressing the recoverable minerals at each grid-intersection as an equivalent mineral grade, and then calculating the cut-off grades in terms of this equivalent. A relationship is devised to equate the secondary minerals with the primary mineral. This relationship is an equivalent mineral grade determined for each secondary mineral based on its selling price and recovery efficiency relative to the selling price and recovery efficiency of the primary mineral. The equivalent mineral grades of all secondary minerals are totalled with the primary mineral grade. This total equivalent mineral content is calculated at each grid-intersection except for those representing air space. For example, ensure that:

1) there are three recoverable minerals 'GRAD', 'GRADY',
and 'GRAD3';

2) their respective selling prices are 'S1', 'S2' and 'S3'; and

3) their respective recovery efficiencies are 'Y1', 'Y2' and 'Y3'.

Then, the total equivalent mineral content 'EQGRAD' for grid-intersection (I,J,K) will be:

\[
EQGRAD(I,J,K) = GRAD1(I,J,K) + ((S2/S1) * (Y2/Y1) * GRAD2(I,J,K)) + ((S3/S1) * (Y3/Y1) * GRAD3(I,J,K))
\]

'EQGRAD' is multiplied by the scale factor 'S5', and this is used as the subscript 'm' of 'ORERES'. This is demonstrated in the routine to follow.

6.3.3 RE-COMPUTATION OF THE T-S MINERAL DISTRIBUTION

The T-S mineral distribution is recalculated whenever the iterative solution goes through another cycle (See Section 6.2(*) routine PIT PRODUCTION SIMULATION). The excavation time of each grid-block is substituted for the initial assumptions of the T-S mineral distribution. This recalculation is executed with the following routine:

```
DO 10 J=1,NJ
   DO 10 I=1,NI
   LOC=N3 *NI* (K-1) +NI* (J-1) +I
   IF (TOPC(I,J,K)) 10, 10, 1
   IF (Y1*LOC) 10, 10, 2
   IF (Y2*LOC) 10, 10, 3
   IF (Y3*LOC) 10, 10, 4
   IF (K-5) 10, 10, 5
   GO TO 6
10 CONTINUE
```

E.GRA1(I,J,K) = GRAD1(I,J,K) + ((S2/S1) * (Y2/Y1) * GRAD2(I,J,K))

```
E.GRA2(I,J,K) = GRAD2(I,J,K) + ((S3/S1) * (Y3/Y1) * GRAD3(I,J,K))
```

or 'ORERES(V,PITINC) = ORERES(V,PITINC) + TONF(*PITINC)'

10 CONTINUE
Definitions (in order of appearance):

'I', 'J' and 'K' are the grid co-ordinates.

'HI', 'HJ' and 'HK' are the maximum co-ordinates in the 'I', 'J' and 'K' co-ordinate directions, respectively.

'LOC' is the unidimensional array member location specified by the grid co-ordinates.

'TCPO' is an indicator array for the deposit's surface.

'YR' is a unidimensional indicator array that contains the year in which each grid-block is mined.

'PITINC' is the pit increment or time dimension of the T-S mineral distribution.

'PERIOD' is the number of production years included in each pit increment.

'W' identifies the unit grid-block weight.

'GRAD1' and 'GRAD2' are the point densities of two recoverable minerals, respectively, at (I,J,K).

'S1' and 'S2' are the selling prices of 'GRAD1' and 'GRAD2', respectively.

'EGRAD' is the total equivalent mineral density, at (I,J,K).

'G1' is a scale factor.

'Z' is the integer value of 'EGRAD'.

'CRES' is the T-S mineral distribution.

'TCWF' is the unit grid-block weight.

'Y1' and 'Y2' are the recovery efficiencies of 'GRAD1' and 'GRAD2', respectively.

The time when each grid-block (I,J,K) is examined is stored in data set 'Y2'. By the routine PIT PRODUCTION SIMULATION statement number two establishes whether the grid-block named by the FC loop parameters (I,J,K) has been...
mined. If not, the routine branches past the remaining computations. A negative or zero array value for YN indicates that the given grid-block (I,J,K) is mined. Otherwise the routine branches to statement number three for the grid-blocks that have been mined. The unit weight of the given grid-block (I,J,K) is assigned its pit increment 'PITINC' by statement number three. Subsequently, the unit weight 'TCNF' of the given grid-block (I,J,K) is added to the appropriate array member of 'ORERES' in statement number eight.

'PERIOD' is the number of production years included in each pit increment. This number must be small enough to correspond with the broad changes in the deposit's mineral assemblage. For example, a massive deposit with a waste overburden layer must be divided into at least two pit increments to correspond with the waste and mineralization. However, more pit increments would yield greater accuracy because they would correspond more closely to the spatial changes in the mineral distribution. The minimum length of these pit increments is one year. The reason for this is the way which the cut-off grades and plant production targets are computed, as described in Subsection 6.4.2.

The third IF statement simulates the boundary between this particular deposit's two different unit grid-block weights 'TCNF', as described in Subsection 6.3.1.

The preceding routine for recalculating the T-S mineral distribution is executed at the end of each iteration cycle. The calculation is repeated to realign this dataset with the increasing accuracy of the time factor as provided by the iterative solution.

The potential value of any deposit depends on the policy choice of plant capacities, production targets and cut-off grades. A group of subroutines were included in the MP programme to select this economic policy so that it will yield an open-pit solution with the highest present value. The theory used to compute this policy was derived by K-F. Lane (4).
The traditional method of calculating this economic policy (9; page 141, 143 and 147) serves as a good comparison to highlight the advantages of Lane's method. The traditional method is computed in the following manner: Firstly, the pit is designed by fixing the final surface at the boundary beyond which the deposit's mineralization ceases to be profitable. The location of this boundary, or, in other words, the 'pit limits', is fixed with the conditions:

1) 'break-even cut-off grade',
2) 'break-even stripping ratio' and
3) a safe wall-angle.

The 'break-even cut-off grade' is that proportional mineral content of the ore such that the:

\[
\frac{\text{Ore Value/Ton}}{\text{Production Cost/Ton}} - \text{Waste Stripping Cost/Ton} = 0
\]

is equal to zero. The 'break-even stripping ratio' is the maximum ratio of waste to ore tonnage allowable at the final pit surface to maintain a profit. This ratio is computed from an ore mineral content equal to the 'break-even cut-off grade' with the formula:

\[
\text{Break-Even Stripping Ratio} = \frac{\text{Recoverable Value/Ore Ton} - \text{Production Cost/Ore Ton}}{\text{Stripping Cost/Waste Ton}}
\]

In this formula, 'Production Cost' is the total of all costs through to the refined metal, exclusive of stripping costs. The 'pit limits' are set by using the 'break-even cut-off grade' and the 'break-even stripping ratio' along with the safe wall-angles to generate the surface that satisfies these marginal conditions.

Once the 'pit limits' are set, various life spans and cut-off grades are assessed for the pit. In coordination, these assumptions fixes the capacity requirements of the
production facilities. Finally, that combination of cut-off grade and life span, furnishing the highest present value is chosen to exploit the deposit. This final cut-off grade is not necessarily equal to the 'break-even cut-off grade'.

There are a number of deficiencies in this method of determining how to operate on a deposit's resources. Primarily, it ignores the 

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I  resent value over the pit's life span and the time 

inflation of costs and prices. The economic environment 

provided by the changing circumstances demands that the 

cut-off grade vary accordingly, to maximize the pit's 

present value. Additionally, the traditional method makes no 

effort to combine or unite into one interdisciplinary function 

the computations for the pit design, cut-off grade and plant 

capacity as a result the final cut-off grade chosen to 

make the one or waste decision may differ from the 'break-

even cut-off grade' used to design the 'pit limits'. The 

difference between these cut-off grades in the outcomes of 

employing different criteria. A marginal profit criterion is 

used to design the pit, whereas a maximum present value 

criterion is used to choose the final cut-off grades and the 

pit's life span.

Lane's economic theory (4) incorporates the interdisciplinary functions of plant capacities, cut-off grades and production targets. They are included in the MP Program in order to furnish the economic decision that will maximize the pit's present value. This computational interdependence is an advantage of the MP Program because it is the unification of the open pit relationships that exist conditionally upon one another. Recall that these decisions provide the relationship between the function \( F(x,y,z,t) \) and the path of the multiplicity integral developed in Section 6.2.

\[ IV = \int_T \int_R \int_{x,y,z} F(x,y,z,t) \, dv \, dt \]

The MP Program can select the optimum:

1) cut-off grades, and

2) pit, concentrator and refinery production targets

for all the possible combinations of plant capacities. Then, it selects the combination with its corresponding
grades and production targets that will yield the greatest present value.

The cut-offs are equivalent mineral grades, if the deposit has more than one recoverable mineral. Therefore, they are compared with the total equivalent mineral densities for the ore and waste decision. The 'equivalent mineral grades' were explained in Subsection 6.3.2.

The distinction between 'plant capacities' and 'plant production targets' should be made clear. The 'plant capacities' are the potential ability of the mine's production facilities to produce the given amounts. The 'plant production targets' are the optimum yield of each facility depending on the T-S mineral distribution, and the unit production costs and selling prices. The 'plant capacities' are the upper limit of the 'plant production targets'.

6.4 2 OPTIMUM CUT-OFF GRADES AND PRODUCTION TARGETS

The optimum cut-off grades, in Lane's theory (4), are computed in the following manner for any particular combination of plant capacities:

Firstly, "three cut-off grade formulae are derived by supposing that each of the three plants (pit, concentrator and refinery) alone limits the total capacity of the operation. These grades are computed on an annual basis and may be called the limiting economic grades. They depend directly on price and costs but only indirectly" (Lane 812), through the present value, on the T-S mineral distribution.

The following profit expression is constructed, and from this the limiting economic cut-off grades derived:

\[
p = (s-r)*w - [c_1 + c_2] - [m_1 + m_2] - [f + T]
\]

Where:

- 'p' is the profit,
- 's' is the selling price per unit weight of product,
- 'r' is the excavation cost per unit weight of pit material.

The unit excavation costs are the expenses common to both
ore and waste excavation*

'C' is the concentration costs per unit weight of ore. "The unit concentration costs include all costs incurred by ore excepting those which depend upon its grade." (4, page 814)

'T' is the refining costs per unit weight of product. The unit refining costs are all those "which are affected by decisions about grade which vary the amount of mineral product without altering the ore throughput." (4, page 311)

'f' is the fixed or administrative costs per period.

'T' is the period length.

'Qr' is the refinery output.

'Qc' is the concentrator input.

'Qm' is the pit output (waste plus ore).

Given the following additional definitions:

1) $PV = \text{the maximum possible present value of profits from the deposit and}$

2) $W = \text{the maximum possible present value of future profits after the next } Q_c \text{ of material has been mined.}$

Then, from the definition of present value:

$$PV = \frac{(P+W)}{(1+d)}$$

where: $d$ is the interest rate for the opportunity cost of investment capital. (Note: the listing of this formula in the original text is a typographical error).

Since $T$ is short and $d$ is less than one, the expression $(1 + d)^T$ to the power $T$ may be approximated by the first two terms of the expansion, which are $(1 + TD)$. Substituting this approximation into the preceding present value equation yields:

$$PV - W = P - (1+VT)$$
Further, substituting:

1) $V$ for $(PV-W)$, the increase in present value from mining the nearest that of trial and
2) the profit expression

yields the basic present value equation:

$$V = (s-r)(Q) - [(c+m) - (f+PV)\cdot T]$$

The solution of this equation is made by trial and error because $PV$ is initially an unknown quantity.

Line (4, page 816) points out the following limitation on the preceding present value expression: "The construction of the expression is valid only if the present value $PV$ depends on the ore reserves but not upon the time. This is equivalent to assuming stable selling prices and costs. In conditions of instability, the present value of the ore reserves can change with time. As a result of this limitation, the inflation rates for costs and selling prices must be constant so that the present value $PV$ of any particular ore reserve is not a function of time.*

The constant inflation rates limit the EP Programme to the roles of planning and property evaluation; and therefore, prevent it from being used to evaluate market fluctuations.*

The three limiting economic cut-off grades are found from the basic present value expression by assuming that the pit, concentrator and refinery, in turn, limit the capacity of the total operation. For example, assume that the pit is limiting the production:

Then:

$$T = Qr/M$$

if $r$ is the maximum pit capacity; and

$$V_m = (s-r)(Q) - [(c+m) - (f+PV)\cdot T]$$

"Given $Q$, the cut-off grade affects only $V$ and $T$. Therefore the cut-off grade must be chosen to make
\[(s-r) \cdot Q_e \cdot [c \cdot Q_c]\]

as large as possible. This implies that every unit of material for which \(s-r\) \cdot mineral content exceeds the concentration cost \(c\) should be classified as ore. Therefore, the optimum cut-off grade \(G_m\) or break-even grade is given by:

\[G_m = c / (s - r) \cdot y\]

where:

\[y\] is the overall recovery efficiency of the equivalent mineral.

Similarly, if \(C\) and \(s\) are the maximum capacity per period for the concentrator and refinery, then:

\[T = \frac{Q_c}{C}\]

\[V_c = \frac{[ (s-r) \cdot y ] - [ (f+\delta+T)/c+c] \cdot (c+c) - [m+c]}{c+c}]\]

\[G_c = \frac{[ (s-r) \cdot y ] - [ (f+\delta+T)/c+c] \cdot (c+c) - [m+c]}{c+c}]\]

\[T = \frac{Q_r}{R}\]

\[V_r = \frac{[ (s-r) \cdot r ] - [ (f+\delta+T)/c+c] \cdot (c+c) - [m+c]}{c+c}]\]

The three limiting economic cut-off grades \(G_m, G_c,\) and \(G_r\) will be the ones which most increase \(V_m, V_c\) and \(V_r\) respectively.

"As can be seen, these three limiting economic cut-off grades depend directly on selling prices, costs, and capacities but only indirectly, through the present value 'FY' on the T-S mineral distribution."

"Next, three balancing cut-off grades are defined. These are the grades which just balance the capacities of each pair of plants. They are independent of economics altogether, being directly determined by the T-S mineral distribution... These are the cut-off grades which cause each pair of plants to be just in balance at their maximum capacities."

The balancing cut-off grades are computed from the T-S mineral distribution for each pit increment."Three ratios
are required as functions of the cut-off grade - ore: pit material, refined product: ore (4, page 317)*. Then, the balancing cut-off grades are those which will separate ore from waste in a proportion that will equalize the production ratios to the ratios of the plant capacities. In other words, these balancing cut-off grades are defined as:

\[
G_{or} \text{ is the cut-off where:}
\]

\[
\frac{\text{ORE}}{\text{CONCENTRATOR CAPACITY}} = \frac{\text{PIT MATERIAL}}{\text{PIT CAPACITY}}
\]

Similarly:

\[
\frac{\text{REFINED PRODUCT}}{\text{REFINERY CAPACITY}} = \frac{\text{PIT MATERIAL}}{\text{PIT CAPACITY}}
\]

\[
G_{rc} : \frac{\text{ORE}}{\text{CONCENTRATOR CAPACITY}}
\]

"Sometimes a balancing position may not exist within the permissible range of grades; in such cases the corresponding balancing cut-off grade should be defined at the appropriate extreme value."

"The balancing cut-off grades are independent of economic factors entirely. They are also dynamic in the sense that they depend upon the grade distribution of the material ahead and can vary widely through an irregular ore body."

"The annual and overall optimum cut-off grade is one of the six cut-off grades consisting of the three limiting economic grades and the three balancing grades. To see why this is so and how to discover the actual optimum in particular circumstances, it is best to consider each pair of stages in turn."

"First, take the mine and concentrator: In the paragraph on limiting economic cut-off grades, expressions were derived for the increase in present value of different throughputs assuming the mine and concentrator alone limited capacity. As the cut-off grade varies, \( G_r \) and \( G_c \) vary; hence \( V_r \) and
Vc * At low cut-off grades Vc is smaller than Vm. This has been shown graphically in Fig. 6.4A.

"The curves intersect at one point only and this corresponds to the balancing cut-off grade Gmc.*

"When the mine and concentrator limit capacity simultaneously, the increase in present grade is actually the lower of Vm and Vc. It is marked heavily in the diagram. As can be seen, in this case the maximum is at the vertex Gmc.*

"However, two other cases must be examined. These also are best illustrated graphically by Figs. 6.4A and C. As these graphs show, when the balancing cut-off grade Gmc is less than Gm, the mine is really the bottleneck in the operation and Gm is the optimum cut-off grade. On the other hand, when Gmc is greater than Gc, the concentrator is the bottleneck and Gc is the optimum cut-off grade."

Three optimum cut-off grades Gmc, GPC and GMC are found by comparing the limiting economic and balancing cut-off grades for each pair of plants as demonstrated with the mine and concentrator. The following rules may be used to find these optimum cut-off grades:

For the mine and concentrator:

$$G_{MC} = \begin{cases} G_m & \text{if } G_m \leq G_m \\ G_c & \text{if } G_m > G_c \\ G_{mc} & \text{otherwise} \end{cases}$$

For the mine and refinery:

$$G_{MR} = \begin{cases} G_m & \text{if } G_m \leq G_m \\ G_r & \text{if } G_m > G_r \\ G_{mr} & \text{otherwise} \end{cases}$$

For the concentrator and refinery:

$$G_{RC} = \begin{cases} G_r & \text{if } G_r \leq G_r \\ G_c & \text{if } G_r > G_c \\ G_{rc} & \text{otherwise} \end{cases}$$

The overall optimum cut-off grade is selected from Gmc, GPC and GMC, and always in the middle value of the three. This can be demonstrated with the three V-curves illustrated.
in Fig. 6‡: "The largest increase in present value that can be achieved at any cut-off grade, allowing for the capacity restrictions, is actually the least of $V_m$, $V_r$, and $V_c$. This is the curve $G_1$ shown heavily in the diagram. The overall optimum cut-off grade $G^*$ obviously corresponds to the highest point in this polygon and it can be shown that this always occurs at the middle value of $G_m$, $G_r$, and $G_c$. In the case illustrated, it is $G^*$. Further, the associated increase in present value is always the least of the three increases which are possible considering the plants in pairs." (4, page 321)

The annual plant production targets for any year are computed from the cut-off grade $G^*$ of that year and the 7-S mineral distribution "ORES": The annual cut-off grades $G^*$ are chosen to maximize the present value of the ore reserves exposed for mining in these particular years. The 7-S mineral distribution "ORES" provides the quantities of pit material, concentrates and refined product available at the cut-off grade $G^*$. This cut-off decision determines the pit, concentrator and refinery throughputs in the following manner: The three available quantities are divided by the respective plant capacities to get the minimum processing time required for each plant requiring the longest processing time is run at full capacity, and the two remaining throughputs are scaled down just enough to keep the restricting plant at full capacity. For the following year, the production targets are reassessed in terms of that year's cut-off grade $G^*$ and the remaining unprocessed ore reserves.

A schedule illustrating the results of the computations of the cut-off grades and production targets is shown in Table 79. The last illustrated in the Table is computed for each combination of plant capacities and their overall recovery efficiencies given by the user. The combination furnishing the greatest present value is chosen automatically to exploit the deposit. Alternatively, the data provided in Table 79 can be printed out for each combination and the user can select the one most consistent with his requirements.

The computations required to execute the theory described in this Subroutine can be followed in the subroutines C001, C002 and C010, listed in Sections 3.4 and 5.5.
The time-space (T-S) distribution of net cash values is the concentration and dispersion of monetary value through the user's deposit. This distribution is measured by computing the point density of net cash value at each grid-intersection. Each point density is the economic gain obtainable from the unit weight of pit material delimited by the grid-block associated with the particular grid-intersection. The point net cash values are a function of time because they depend on the inflation of costs and selling prices, and the annual cut-off grades. The excavation times are computed by the routine PIT PRODUCTION SIMULATION.

The FORTRAN data set used to store the T-S net cash value distribution is:

```
VAL (I,J,K)
```

The subscripts (I,J,K) are grid co-ordinates that link the array members with their location in the user's deposit. Each array member will be the net cash value gained from the material within the associated grid-block (I,J,K), if it is mined. The point net cash values are the total value of the respective grid-blocks to the mining venture. Consequently, they can be positive, zero or negative depending on their economic potential.

6.5.1 STANDARD COSTING METHOD

The point net cash values are computed with a standard costing method. This method is devised by appointing standard values for the various unit production costs (is defined in Table 5.2), the selling prices and recovery efficiencies. Then, these standards are adjusted according to the peculiar conditions under which the particular grid-blocks are mined. The point net cash values of each grid-block depend on:

1) its total equivalent mineral content compared with the annual cut-off grades;
2) the year in which it is mined;
3) its location in the deposit;
4) its position relative to the physical features that affect production costs;
5) the quantities, selling prices and overall recovery efficiency of the recoverable minerals;
6) the presence of any minerals that will interfere with the purification processes of the recoverable minerals; and
7) its share of the fixed and administrative cost burden.

Each grid-block is classified as ore or waste to branch the routine to the appropriate costing sequence. The cut-off decision is made by comparing the grid-block’s total equivalent mineral content with the annual cut-off grade of the year in which the grid-block is mined (see Subsection 6.3.2 for an explanation of equivalent mineral contents). Thus, the cut-off decision is made on the grid-block’s total recoverable mineral content rather than just on the primary mineral. Adjustments are executed on the standard unit production costs and selling prices, and recovery efficiencies to furnish the correct figures for the conditions listed above. These adjusted standards are employed to compute the point net cash value of each grid-block. The following routine illustrates how these computations are executed; only the adjustments are omitted. They are described in Subsection 6.5.2.

```
01 DO 38 T=1,2
   ...
02 DO 34 K=1,NX
03 DO 34 J=1,NJ
04 DO 34 I=1,NI
05 LOC=(J)*N(J-1)+N(I)*(J-1)+I
06 IF (TOPO (LOC)) 34,34,14
07 IF (Y3 (LOC)) 14,15,17
08 IF (M=1)
09 IF (T-K) 14,16,34
```
\[ IF \ ( 2 \ \text{GRAD}(I, J, K) - \text{AVG}) \ 19, 23, 23 \]
\[ IF \ (\text{YR}(\text{LOC}) - L) \ 18, 18, 261 \]
\[ M = 1 \]
\[ \text{GO TO 16} \]
\[ IF \ (-\text{YR}(\text{LOC})) \ 34, 262, 34 \]
\[ M = T \]
\[ IF \ (2 \ \text{GRAD}(I, J, K) - \text{CUTOFF}(T)) \ 19, 23, 23 \]
\[ \text{REV} = 0 \]

(standard cost adjustments)

\[ \text{COST} = \text{TONF} \ (Y + (F(M)/\text{MIN}(M))) \]
\[ \text{GO TO 33} \]
\[ 23 \]

(standard price adjustments)

\[ \text{REV} = \text{TONF} \ (\text{GRAD}1(I, J, K) + S1(M) \cdot \text{YIELD}1) + \text{GRAD2}(I, J, K) + S2(M) \cdot \text{YIELD}2 + \ldots + \text{GRADn}(I, J, K) + Sn(M) \cdot \text{YIELDn}) \]

(standard cost adjustments)

\[ \text{COST} = \text{TONF} \ (Y + Z + \gamma \ (F(M)/\text{MIN}(K))) \]
\[ \text{VAL}(I, J, K) = \text{REV} - \text{COST} \]
\[ 34 \]
\[ \text{CONTINUE} \]

Definitions (in order of appearance):

'\( T \)' is the production year

'\( L \)' is the life span of the mine in years

'I', 'J', and 'K' are the grid co-ordinates of the deposit's grid-blocks.

'\( Y1 \)', '\( Y2 \)', and '\( Y3 \)' are the maximum (I, J, K) grid-co-ordinates, respectively.

'\( \text{LOC} \)' is an expression for converting the three-dimensional co-ordinates (I, J, K) to a unidimensional subscript. This subscript locates the indicator array member associated with (I, J, K).
'TOPD' is an indicator array of the deposit's surface.

'Y3' is an indicator array that stores the production time of the grid-block 'LOC'.

'K' is a transfer variable used to expedite programming; it equals the production year.

'EGRAD' is the equivalent mineral content of grid-block (I,J,K).

'AVEG' is the average of the annual cut-off grades.

'CUTOFF' is the cut-off grade for production year (T).

'REV' is the revenue of grid-block (I,J,K).

'COST' is the production cost of grid-block (I,J,K).

'TONTF' is the total weight of grid-block (I,J,K). This constant is made into a variable as explained previously in Section 6.2, if the unit grid-block weight varies through the user's deposit.

'X' is the adjusted waste excavation cost per unit weight.

'F' is the fixed costs in year (M).

'MIN' is the total pit production in year (M).

'GRA1', 'GRA2', ..., 'GRADn' are the point mineral grades at grid-intersection (I,J,K) for 'n' recoverable minerals, respectively.

'P1', 'P2', ..., 'Pn' are the selling prices per unit weight for the 'n' recoverable minerals, respectively, in year (M).

'YIELD1', 'YIELD2', ..., 'YELDn' are the overall recovery efficiencies for the 'n' recoverable minerals, respectively.

'Y' is the adjusted ore excavation cost per unit weight.

'W' is the adjusted refining cost per unit weight.

'Z' is the adjusted milling cost per unit weight.
'VAL' is the joint net cash value of grid-block \((I, J, K)\).

The point net cash value 'VAL' is computed for each grid-block \((I, J, K)\) with the preceding routine. A verbal description of this routine follows:

**LINE NO.**

01) **DO** loop 33 names the production year for which the following computations are made.

02) **LO** loop 31 computes the joint net cash value of the grid-block named by the DO loop parameters.

06) The IF statement in Line 06 simulates the deposit's surface, as explained in Section 3.3.

07) The IF statement in Line 07 establishes whether the production time of grid-block \((I, J, K)\) has been computed by the routine \(\text{PRODUCTION SIMULATION}\).

08) The grid-block \((I, J, K)\) production is assumed to occur in the first year by Line 03, if its production has not been simulated.

09) The IF statement in Line 09 is included to branch past the remaining computations for unmined grid-blocks, when DO loop 33 cycles to the second year.

10) The annual cut-off grades 'CUTOFF' cannot be used if the production year 'YP' is unknown. Therefore, an average 'AVG' of the annual cut-off grades is used for the ore or waste decision in Line 10. The routine branches to statement 11, if the grid-block \((I, J, K)\) is waste or it branches to statement 23 if the grid-block is ore.

11) The routine branches to Line 11, if the production of grid-block \((I, J, K)\) has been simulated. This IF statement compares the average total life span 'L' with the production year 'YP'. The grid-block \((I, J, K)\) is assumed to be mined in the first year by Line 12, if the production year 'YP' is greater than the mine life 'L'.

14) The IF statement in Line 14 compares the production
year 'YR' with the 'F' parameter of DO loop 38. The routine branches past the remaining computations until
they are equal.

16) The IF statement in Line 16 compares the equivalent mineral grade 'ZGRID' of the grid-block (I,J,K) with
the annual cut-off grade 'CUTOFF' for the production year 'T'.

17) The revenue 'REV' and cost 'COST' of grid-block (I,J,K) are computed for the waste grid-blocks between Lines 17
and 19, and for one grid-blocks between Lines 20 and 23.

23) The point net cash value 'VAL' is computed for grid-
_block (I,J,K) at Line 23.

The preceding costing routine is necessarily complicated to
cover all the computational situations. The different situations occur as a result of the IP Programme's iterative
solution. The numerous IF statements branch the routine to
the correct computational sequence for the following situations:

A) The costing routine is executed before the routine PIP
PRODUCTION SIMULATION in the first cycle of the iter­
tive solution. Therefore, the grid-block production
years are not simulated for the first estimate of the
point net cash values.

B) The final pit surface fluctuates as the solution converges. This causes some grid-blocks, within the
final pit surface, to be left unsimulated by the preceding iteration cycle's pit production simulation.

C) The mine's life span is computed separately by the
cut-off grade computations and the pit production simulation. The life span computed by these routines
may differ until the solution converges. The duration is controlled in the costing routine with the life span 'L',
given by the cut-off grade computations. Consequently,
the IF statement in Line 11 is required to reassess the grid-block production years that exceed
the mine life 'L'.

The grid-blocks in the preceding situations cannot be
classified as ore or waste with the annual cut-off grade 'CUTOFF' because their production years 'YE' are unknown or exceed the life span 'L'. Therefore, the mean 'AVES' of the annual cut-off grade 'CUTOFF' is employed to make the ore or waste decision. Additionally, these grid-blocks are assumed to be mined in the first year. These assumptions are crude, but this is inconsequential. The final pit surface becomes static as the solution converges, and as a result, the assumptions become redundant.

The preceding assumptions never become completely redundant in the last year of long-term proposals. The reason is that the point net cash values outside the final pit surface are unrealistically inflated because of the assumptions. Therefore, the final pit surface oscillates at the point where the grid-blocks are affected. It could be possible to stop this oscillation if 'K = 1' were substituted with 'K = L' at Line 12 and Line 13 deleted for the last cycles of the iterative solution. However, this modification has not been tried nor the effects on the EP Program analyzed. In any case, the slight production oscillation in the last year of a long-term proposal is insignificant compared to the total production.

6:5:2 ADJUSTMENTS OF THE STANDARDS

The standard unit production costs and selling prices, and the standard recovery efficiencies are corrected by factors to furnish the following adjusted standards defined in the preceding Subsection:

Costs: P(X), W, X, Y, and Z;

Selling Prices: S1(X), S2(X), • • • Sn(X); and

Recovery efficiencies: YIELD1, YIELD2, • • • YIELDn;

where 'n' is the number of recoverable minerals.

There are any number of factors by which these standards can be adjusted. These factors are chosen by the user to simulate the peculiar mining conditions that will be encountered by him during the following list suggests some of the factors which can be applied to adjust the standards:

TYPE 1:
Time inflation rates are employed to adjust the standard costs and selling prices to the correct value for the production year in which each grid-block is processed. This adjustment is made by identifying the times when the grid-blocks are mined. For example, the standard costs could be multiplied by the factor:

\[(1 + \text{INFRC}) 
\]

where:

'INFRC' is the proportional time inflation rate for costs.

'Y' is the production year in which the particular grid-block is mined.

This factor multiplied by the standard costs will yield the adjusted standards for year 'Y'. A similar factor could be used for prices.

**TYPE 2:**

The surface simulation techniques, which are the indicator arrays and polynomial equations described in Section 31, can be used to locate standard cost adjustments that are a function of location. These standards can be adjusted by identifying their particular grid-block's relative position with the simulated surfaces. These adjustments are best explained through demonstration with the following examples:

Example 1:

The gradual increased cost of excavation with depth and/or transport distance can be accounted for by deriving a relationship between the additional cost and the grid-block location. For example, the following factor could be added to the standard ore excavation cost 'O1' to get the additional cost of ore excavation at depth:

\[(K\times\text{OBCI})\]

where:

'K' is the vertical grid co-ordinate of the particular grid-block being evaluated.
'CUCI' is the increased cost of ore excavation per grid-level per unit weight of each grid-block.*

This factor multiplied by the standard ore excavation cost 'OR' will furnish the adjusted standard for the grid-blocks on the 'F' grid-level. Similar factors can be devised for all the containing situations depending on depth and transport distances.*

Example 2:

A family of surfaces could be used to define the cost differences between zones of variable excavating difficulty. For example, there could be a fracture zone surrounding a fault plane which causes the waste and ore excavation in this area to be less expensive than the other areas. The following routine could be used to make this required adjustment of the standards:

```
IF (FAULT(I,J,K)) 30, 30, 31
30 NN=1
   GO TO 32
31 NN=2
32 X=CB+FCW(NN)
   Y=CF+FCO(NN)
```

where

'FAULT' is an array used to indicate whether the particular grid-block (I,J,K) is in the fracture zone or not.*

'FCW' and 'FCO' are the correcting factors for waste and ore excavation costs, respectively, in the fractured and unfractured areas.*

'CB' and 'CF' are the standard waste and ore excavation costs respectively.*

'X' and 'Y' are the adjusted waste and ore excavation costs, respectively.*

The indicator array 'FAULT' could be substituted by a polynomial equation, if the fault surface has few irregularities and the internal computer storage space is scarce. For example, the following IF statement will identify the grid-blocks within a distance of three grid co-ordinates
from the given fault plane 'PLANE':

\[ \text{PLANE}=0 \times 22 \times I - J - (N \times 11 \times K) \times 10 \]
\[ \text{IF (ABS(PLANE-30)) 30, 30, 31} \]

Dimensioned data sets will be preferable to the correcting factors 'FCL' and 'FCC' for certain costing situations. For example, the above routine could be revised to replace the correcting factors with standard cost arrays, as follows:

\[ \text{IF (FAULT(I,J,K)) 30, 30, 31} \]
\[ 30 \text{ NN}=1 \]
\[ 31 \text{ NN}=2 \]
\[ 32 \text{ X}=\text{OB}(\text{NN}) \]
\[ \text{Y}=\text{OR}(\text{NN}) \]

Similar correcting factors or standard cost arrays could be devised for variable powder factors, water pumping costs, drilling costs, and so forth.

**Example 3:**

The final pit surface of a new deposit will be limited by legal or economic restrictions. For example, the pit could encroach on the territory beyond a property line or man-made structure, if these features are near enough to the assay's deposit and there are no restrictions imposed on the pit design routine. These restrictions are made by giving the computer's largest negative value to the point net cash values along the legal or economic boundaries. The excessively low point net cash values will automatically prevent the pit design routine from extending the final pit surface beyond these boundaries. The following statements placed within the costing routine will perform this restriction for the given railway line identified by its relative position 'RAIL' in the grid system:

\[ \text{RAIL}=10 \times I - J \]
\[ \text{IF (RAIL) 10, 10, 11} \]
\[ 10 \text{ VAL}(I,J,K)=10 \times 62.75 \]
\[ 11 \text{ CONTINUE} \]

This routine assigns all the point net cash values 'VAL' from the rail line outward to have an exceptionally low value. The pit design routine will not include any of the
grid-blocks associated with these point net cash values within the final pit surface

**TYPE J:**

Production penalties or rewards can be charged to the grid-blocks by identifying their minerals. For example, these may be minerals present in portions of the user's deposit which interfere with production. The standard recovery efficiencies could be adjusted, if the impurities are not separated. On the other hand, the grid-blocks containing these minerals could be charged a metallurgical penalty, for the additional refining expense with the following routine:

```
IF (GRAD3(I,J,K) > 0.005) 11,11,10
10 W = REFC + CHARGE
GO TO 12
11 W = REFC
12 CONTINUE
```

where:

- "GRAD3" is the point density of the offending mineral at (I,J,K).
- "CHARGE" is the metallurgical penalty factor.
- "REFC" is the standard refining cost.
- "W" is the adjusted refining cost.

The IF statement checks the proportional mineral content of the mineral "GRAD3" that interferes with production. The grid-block (I,J,K) is given a metallurgical cost penalty, "CHARGE", if the proportional mineral content of "GRAD3" is above the "0.005" level. A similar correcting factor could be devised for the proceeds of a by-product leaching process. This, of course, would be a credit item instead of a debit, as demonstrated with the preceding metallurgical penalty.

The standard costing method is a flexible technique of computing the point net cash values. This flexibility allows the user to simulate the exact cost structure of his deposit. However, the cost structure of all deposits varies consequently, the adjustments of the standards described in
this subsection must be tailored to the peculiar cost conditions existing for the user's deposit. The correcting factors are used separately, and in combination to simulate the particular cost structure. These factors have been included as examples in the cost structure programmed in Chapter 8 for the deposit described in Chapter 7.

There is one last function executed by the costing routine that has not been demonstrated. The adjusted unit production costs 'u', 'u', 'u' and 'u' assigned in each production year are averaged to estimate their mean values. These annual means are substituted for the initial assumptions of the annual unit production costs 'u', 'u' and 'u' which were made to begin the iterative solution. The estimates are simple averages, therefore the programming requires no explanation. However, recall that the unit production costs are defined differently for the costing routine and the cut-off grade routine. This difference is accounted for when the adjusted unit production costs are averaged.

6*6 DESIGN OF THE FINAL PIT SURFACE

The final pit surface is designed with the Leach-Enterprise algorithm (5) outlined in Chapter 2. Recall from Section 6*2 that the objective is to find a pit volume 'v' of the deposit 'p' that yields the maximum total profit from the three inner constant integrals of the multiple integral:

\[ PV = \int_{T}^{R} \int_{V}^{T} P(C(x,y,z),t) \, dt \]

The joint densities of the function \( P(C(x,y,z),t) \) are given by the T-S net cash value distribution. The Leach-Enterprise algorithm is a numerical method which is substituted for the integration over the region 'p'. By means of this numerical method, the pit volume is found that will yield a maximum total profit.

6*6*1 A DIRECTED GRAPHS GE AS OPEN PIT MINE

The Leach-Enterprise algorithm (5) uses the directed graph defined in Chapter 2. The following defines a finite, directed graph that is specific application in this case.
1) Vertices: The vertices of the directed graph are the grid-intersections. The mass of each vertex is the associated point density of the T-S net cash value distribution.

2) Arcs: Arcs are drawn from a vertex to any adjacent vertex, if the excavation of that vertex is dependent on the removal of the adjacent vertex.

3) Closure: A closure of the directed graph is any set of vertices such that, if a vertex belongs to the closure and an arc exists from that vertex to another vertex, then the second vertex must also belong to the closure. A closure is a reasonable pit surface.

4) Maximum Closure: A maximum closure is the set of vertices yielding the maximum total of the point net cash values.

The arcs are depicted quite easily; however, programming then presents a difficulty. It has been suggested (1 and 5) that the arcs be defined through the construction of the grid system. For example, Fig. 6 illustrates three grid constructions which have different wall-angles as a result of the way in which the grid-blocks are stacked. Theoretically, a grid construction could be found to furnish any desired wall-angle. Unfortunately, these wall-angles would be fixed over the entire deposit, and as a result, they could not be varied. This difficult was overcome by formulating a method of defining the arcs with the upper half of a cone.

To elaborate -- the dependence of one vertex on another for removal is defined by the arcs. This dependence is simulated by a set of overlapping cones for the purpose of programming the algorithm. Cones are used to define the boundary between those vertices which must be removed and those which may remain in order to expose any particular vertex. A cross-section of the directed graph illustrated in Fig. 6 shows how a cone can be used to simulate the arc. Note that the cone is a closure because of the way the directed graph is constructed for this pit design application. Therefore, all the vertices that must be removed to expose the vertex at the base of the cone, are included within the cone.

The final pit surface is the set of overlapping cones that
FIG. 6.6 THREE GRID CONSTRUCTIONS
FIG. 6.7 USING A CONE TO SIMULATE ARGSO
arc a maximum closure of the directed graph. These cones are
programmed by deriving a FORTRAN expression from the analy­
tical equation of a cone, the grid co-ordinates of the
vertices and the pit's safe wall-angles. This expression
is:

\[ \text{CONE} = \text{SLOPE} \times ((A(K-Z)^2) - (B(J-Y)^2) - (C(I-X)^2) \]

where:

'SLOPE' is the tangent squared of the safe wall-angle
measured from vertical in radians.

'X', 'Y' and 'Z' are the grid co-ordinates of the vertex at
the base of the cone.

'I', 'J' and 'K' are the grid co-ordinates of the remaining
vertices.

'A', 'B' and 'C' are constants which define the relative
lengths of the cone axes.

The 'CONE' expression is limited by the pit design routine
to its upper half to form the semi-cone shape of the desired
closure. Otherwise, the routine would be using the hour-
glass shape of a complete cone. Any particular semi-cone is
defined by the safe wall-angle 'SLOPE', the base vertex
(X,Y,Z) and the constants (A,B,C). The vertices (I,J,K) are
within the semi-cone closure, if the 'CONE' expression is
positive or zero; otherwise, they are outside the semi-cone.

The safe wall-angle 'SLOPE' and the constants (A,B,C) can be
varied as often as required by the slope stability condi­
tions. These factors can be specified for any number of pit
zones through simulation with the indicator arrays
and polynomial equations described in Section 3.3.

The safe wall-angle establishes the degree of slope for the
semi-cones. Variable wall-angles are accounted for by con­
structing a set of semi-cones to simulate the desired closure.
This set is defined by the variable safe wall-angles and the
grid co-ordinates of the base vertex as illustrated in the
following example: Assume that there are two safe wall-
angles for either side of the fifth grid-level as shown in
the cross-section of the directed graph in Fig. 4.4. It is
required to identify the vertices within the closure shown.
FIG. 6.8 A CROSS-SECTION OF A SEMI-CONE SET WITH TWO WALL-ANGLES
to expose the base vertex \((X,Y,Z)\). The vertices below the fifth grid-level are within the closure, if they satisfy the above 'CON' expression using \(SLOPE = SLOPE(2)\). The vertices above the water table are within the closure, if they satisfy the 'CON' expression with the following adjustments:

1) \(SLOPE\) is set equal to \(SLOPE(1)\), and

2) the vertical grid co-ordinate \('Z'\) of the base vertex is replaced with \('D'\); \('D'\) is calculated with trigonometric relationships of the semi-cone set.

This example demonstrates how the semi-cone closures are used to imitate the arcs for the pit design application of the directed graphs. It also illustrates the flexibility obtained by using the semi-cone sets with variable wall angles to replace the arcs.

The cone constants \((A,P,C)\) establish the shape of the semi-cones. A normal circular shape results, if the constants are all equal. The definition of these constants is a handy technique for simulating deposits with directionally dependent slope stability conditions. For example, elliptical semi-cones can be constructed by weighting the constants. Then, they can be used to simulate the pit surfaces requiring safe wall-angles that vary as their orientation with the weak strength direction of a bedding plane.

In addition, the cone constants must include a factor to compensate for different unit interval distances between grid co-ordinates on the three axes. For example, they would be \(A = 1, B = 2\) and \(C = 1\) for a circular semi-cone, if the intervals between grid co-ordinates in the \(E, J,\) and \(L\) directions were 150 ft, 200 ft, and 160 ft, respectively.

Having dealt with the directed graph used to simulate an open pit, it is now possible to turn to the pit design algorithm.

6. 6. 2 THE LERCHS-GROSSMAN ALGORITHM.

Lerchs and Grossman (5) use a mechanical analogue to demonstrate their algorithm. An illustration of the analogue is shown in Fig. 6. It can be seen from the analogue that the grid-blocks have either an upward or downward force. The
arrows in each grid-block show which way the force is exerted. An upward force represents a positive net cash value of the particular grid-block; whereas, a downward force represents a negative net cash value. In order to find the optimum pit surface, the grid-blocks are allowed to move along the vertical axis. The movement of a free mechanical system maximizes the work done. Therefore, it is the grid-blocks that have moved which maximize the total net cash values.

The steps follow for finding, with the directed graph, the pit volume yielding a maximum total profit. These steps are applicable only to the directed graph described in the preceding Subsection. The directed graph was constructed to permit a simplified version of the original steps derived by Lorch and Grossman.

STEP 1:
The vertices with a positive net cash value are examined in turn.

STEP 2:
A semi-cone closure is constructed from each positive vertex. The semi-cone shape is fixed by its particular safe wall-angles 'SLOPE' and the constants \((A, b, C)\).

STEP 3:
The net cash values of the vertices that satisfy the 'CONE' expression are calculated. If this total is positive, the semi-cone closure becomes part of the maximum closure. Otherwise, it is excluded. Some of the vertices included in any particular semi-cone closure can be a part of another semi-cone already within the maximum closure. The sum of the net cash values within any overlapping semi-cones does not include those vertices already contained within the maximum closure.

STEP 4:
When the overlapping semi-cone closures of all positive vertices have been tested and either included or excluded from the maximum closure, the result is the final pit volume.
In an illustration of the algorithm, one can see in Fig. 6*10*, note that the directed graph is represented in arc form, rather than semi-circle form. In Fig. 6*10*, the first positive vertex has a value of +1. This is the base of a test closure formed by including all vertices pointing to it. Only the -4 vertex is connected to the +1 vertex to form the test closure. With a total negative value of -3. Therefore, this closure is not included within the maximum closure. The test closure shown in Fig. 6*10C and 9 are included in the maximum closure because they both have a total positive value. The test closure shown in Fig. 6*10D has three vertices (17, -1, -2) which already belong to the maximum closure. Consequently, only the +3 and -2 vertices are summed, and since the total is positive, both are included in the maximum closure.

The exclusion of any positive test closure within the maximum closure will subtract from its total value. Additionally, the inclusion of any negative test closure will also subtract from the maximum value. Therefore, it is obvious that the maximum closure of Fig. 6*10E has the highest value possible from the vertices with the constraints of the area.

6.7 PIT PRODUCTION SIMULATION

The routine PIT PRODUCTION SIMULATION performs the outermost integration of the multi-level integral:

\[ P^T = \iiint P(x, y) dx dy \] (see Section 6.7)

In order to calculate the mine’s present value, the path of this integration is computed by the routine:

1) PLANT LOCATION SELECTION,
2) CUT-OFF GRADE CONSIDERATIONS,
3) PRODUCTION RATE computations.

The output sequence determined by the simulation of the pit production is essentially an attempt to identify a digital computer model to initiate the excavation sequence of the pit material.
The simulation is programmed with a loop of computations that cycles yearly. The loop is divided into two parts: the first part simulates ore excavation by identifying the grid-blocks with a positive net cash value that will be mined in each production year. The second part identifies the grid-blocks required to expose the ore production of each year. The routine production targets computations provides the pit production simulation with the quantities which should be produced to maximize the mine's present value. However, these targets are superseded by the simulation, if the deposit's physical conditions make it impossible to meet the targets.

The Ore - and Waste- Subsections to follow describe the pit production simulation in greater detail.

6.7.1 Ore production simulation

The ore production simulation is programmed by defining rules which establish the ore excavation sequence. The rules direct the computer to scan the grid-blocks for the next one to be excavated. When this grid-block has been found, it is tagged with its production year. Then, its tonnage is summed with the other ore production of the same year. This cumulative ore production is compared with the ore production target of the given year after each ore grid-block is excavated. The ore production for each particular year is completed when its cumulative total equals the ore production target. The yearly cycle is completed by identifying the grid-blocks required to expose the ore production. This cycle continues until the excavation has been simulated for all grid-blocks within the final pit surface.

An ore production simulation can be demonstrated with a grid system whose maximum grid co-ordinates in the longitudinal, transverse, and vertical directions are J=30, I=26 and K=16, respectively. The ore production is to be excavated from one working face. The benches are equal to the row of grid-blocks between the pit limits in the J co-ordinate direction. The ore working face advances parallel to the J axis and changes direction on alternate benches. The network diagram of Table 6.7 and continued in Table 6.8 illustrates the rules used to program this ore excavation sequence. In the diagram, the blocks represent computations and the arrows indicate the computational sequence. The symbols in the diagram are defined as:
1) 'L' is the production point.

2) 'A', 'B', and 'SIGN' control the direction of advance.
for the ore working face; and

3) \( (I, J, K) \) is the co-ordinate location of the grid-blocks.

Note that this simulation only works for grid systems with an even number of grid co-ordinates along the I and J axes.

---

**Table 6.4**

<table>
<thead>
<tr>
<th>IS GRID-BLOCK ((I, J, K)) WITHIN THE PIT?</th>
<th>NO</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>YES</strong></td>
<td></td>
</tr>
<tr>
<td><strong>DOES ((I, J, K)) HAVE A POSITIVE NET CASH VALUE?</strong></td>
<td>NO</td>
</tr>
<tr>
<td><strong>YES</strong></td>
<td></td>
</tr>
<tr>
<td><strong>TAG ((I, J, K)) AS ORE MINED IN YEAR (L)</strong></td>
<td></td>
</tr>
<tr>
<td><strong>DOES CUMULATIVE ORE PRODUCTION EQUAL TARGET IN (L)?</strong></td>
<td>NO</td>
</tr>
<tr>
<td><strong>IDENTIFY GRID-BLOCKS REQUIRED TO EXPOSE ORE MINED IN (L)</strong></td>
<td></td>
</tr>
<tr>
<td><strong>(L-L+1)</strong></td>
<td></td>
</tr>
</tbody>
</table>

**Table 6.5**

The computations illustrated in Table 6.4 define the grid co-ordinates of the next grid-block that should be mined in the ore extraction sequence. Then, the grid-block is tested as illustrated in Table 6.5. First, it is determined whether the grid-block is within the final pit surface. Second, it checks the net cash value of the grid-block to determine whether it is positive. If the grid-block fails either test, the remaining computations are bypassed and
the routine returns to look for the next grid-block in the ore excavation sequence. However, if the grid-block passes both tests, it is suitable for ore production at that time (given by the production year \(L\)). The grid-block is tagged with its excavation time \(C\) by storing this time in the associated array member of an indicator array. As described at the beginning of this Subsection, the yearly cycle is completed when the cumulative tonnage of the ore grid-blocks equals the ore production target, and the overburden required to expose this ore production has been identified.

The ore production simulation is tailored to study alternate ore excavation sequences. The ore excavation sequence can be modified to simulate multiple working faces and ore benches; for example, Table 6.6 continued through Table 6.9 illustrates an ore excavation sequence with two working faces on the same bench. Each bench is equal to the row of grid-blocks between the pit limits in the J co-ordinate direction. The two working faces advance in opposite directions parallel to the J-axis and change direction on alternate benches. One working face operates from one extreme of the pit to the middle; the second working face operates from the middle to the other extreme.

The network diagram (Table 6.6 to Table 6.9) of this ore excavation sequence appears to be extremely complex; however, on closer examination, it can be seen that this complexity is superficial. The computations illustrated in Table 6.9 define the grid co-ordinates of the next ore bench to be mined. Additionally, the latter part illustrates the following computations:

1) the ore grid-blocks on the particular ore bench \(1, i, j\) are identified and their J co-ordinate is stored in the array 20\(^{th}\).

2) the routine branches to the computations in Table 6.7, if it finds that the working faces advance towards one another; it branches to Table 6.8, if it finds that the working faces advance away from each other.

Table 6.7, and 6.8 are similar but not identical in both Tables. The left-hand branch represents the computations for the benches with an odd number of ore grid-blocks. The right branch shows the computations for an even number.
The two above examples illustrate how the pit production
simulation can be tailored to fit the user's ore excavation sequence. A variety of ore excavation sequences can be
TABLE 6.8
In addition, the ore production can be designed to include the simulation of any physical features which affect the ore excavation sequence. These physical features are simulated with the indicator arrays or polynomial equations described in Section 3.1. The ore excavation sequence is programmed to recognize these features in order to take account of their occurrence for example, a large fault displacement could force a kink in the ore bench shape. This would be accounted for by using a simulation of the fault to organize the ore excavation sequence.

6.7.2 WASTE PRODUCTION SIMULATION

The pit's waste production is simulated by identifying the overburden that must be removed to expose the ore. This simulation is achieved, on an annual basis, with overlapping semi-cones similar to those used in the design of the final pit surface. A set of overlapping semi-cones is formed for each production year from the ore grid-blocks excavated in that year by means of the following expression:

\[ \text{CONE} = \text{WORKSL} \times ((A(K-Z))^2) - ((B(J-Y))^2) - ((C(I-X))^2) \]

where:

- WORKSL is the tangent squared of the working wall-angle measured from vertical in radians and it may not be less than the minimum working wall-angle.
- \((X,Y,Z)\) are the grid co-ordinates of the ore grid-blocks at the base of the semi-cones.
- \((I,J,K)\) are the grid co-ordinates of the remaining grid-blocks.
- \((A,B,C)\) are constants which define the relative lengths of the semi-cone axes.

Each semi-cone set is defined by:

1) those semi-cones whose bases \((X,Y,Z)\) are given by the ore grid-blocks excavated in each year.

2) the shape as given by the working wall-angles \(\text{WORKSL}\) and the same constants \((A,B,C)\) as described in subsection 6.4.1.
In addition, the ore production can be designed to include the simulation of any physical features which affect the ore excavation sequence. These physical features are simulated with the indicator arrays or polynomial equations described in Section 3.1. The ore excavation sequence is programmed to recognize these features in order to take account of their occurrence. For example, a large fault displacement could force a kink in the ore bench shape. This would be accounted for by using a simulation of the fault to organize the ore excavation sequence.

6.7.2 WASTE PRODUCTION SIMULATION

The pit's waste production is simulated by identifying the excavation that must be removed to expose the ore. This simulation is achieved, on an annual basis, with overlapping semi-cones similar to those used in the design of the final pit surface. A set of overlapping semi-cones is formed for each production year from the ore grid-blocks excavated in that year by means of the following expression:

\[ \text{CORE}=\text{WORKSL} \times [(A (X-2))^{*}2] - [(B (I-J))^{*}2] - [(C (I-H))^{*}2] \]

where:

\( \text{WORKSL} \) is the tangent squared of the working wall-angle measured from vertical in radians but he less than the minimum working wall-angle.

\((X, Y, Z)\) are the grid co-ordinates of the ore grid-blocks at the base of the semi-cones.

\((I, J, K)\) are the grid co-ordinates of the remaining grid-blocks.

\(A, B, C\) are constants which define the relative lengths of the semi-cone axes.

Each semi-cone set is defined by:

1) those semi-cones whose bases \((X, Y, Z)\) are given by the ore grid-blocks excavated in each year.

2) the shape is given by the working wall-angles \(\text{WORKSL}\) and the same constants \((A, B, C)\) as described in Subsection 6.7.1.
The grid-blocks \((I,J,K)\) that satisfy the 'CONE' expression are those required to be removed to expose the annual ore production. The expression is satisfied when 'CONE' is zero or positive. The semi-cones in any set are programmed to overlap so that any material common to two or more semi-cones is not accounted for more than once. Additionally, the semi-cone set is programmed to exclude any material excavated in a previous year. The grid-blocks within this semi-cone set are tagged with the production year of the ore grid-blocks at the base of the set. These production years are stored in the array member associated with the particular grid-blocks of the indicator array ['Y']. This array is used to update the time dimension for the iterative solution.

The total tonnage of the semi-cone set is compared with the waste production target for the particular year and the pit capacity computed in preceding routines. As a result of this comparison, there are the following possibilities:

1) The waste production could be less than the particular annual target. In this case, the semi-cone set is reconstructed by increasing the working wall-angle 'W0RED' in increments until the set includes enough material to equal the target.

2) The waste production could be greater than the particular annual target. In this case, the waste material required to expose the ore production is greater than anticipated by the target computations. The working wall-angle in the minimum allowable to maintain the working space requirements. Therefore, the waste production simulation is programmed to override the target.

3) The waste production plus the ore production could exceed the total pit capacity. In this case, the pit capacity is insufficient to expose the ore production. This solution is invalid because the production targets were computed from an inadequate pit capacity. Consequently, the solution must be re-executed with a greater pit capacity and/or a steeper working wall-angle. An example of a solution which is invalid for this reason has been included in Section 7a.

The waste production simulation routine can be used for any surface deposit including those with variable working wall...
For example, a deposit which has different stripping equipment for an upper unconsolidated rock layer and a lower stronger rock layer may require two different working wall-angles. The variable working wall-angles are simulated in the same manner as the variable safe wall-angles described in Subsection 6.6.1.

Finally, the excavation times of the grid-blocks are used to compute production schedules. The schedules are calculated for the total annual production of ore and waste, and the total annual revenue and overburden stripping costs.
CHAPTER 7 AN OPEN PIT MINING SOLUTION

The surface deposit described broadly in Section 7.1 is used as an example to illustrate an application of the MP Program. This example does not necessarily duplicate any existing mining conditions. It was devised to demonstrate the simulation techniques. The Program's output forms the open pit mining solution which also is evaluated economically. Finally, an example of a solution is given that is invalid for insufficient pit capacity.

7.1 THE EXAMPLE DEPOSIT

The example deposit is a massive surface type. Two minerals with a potential value were discovered in the deposit. The selling price of the primary and secondary minerals are $500,000 and $300,000 per ton of end product. It is anticipated that these selling prices will inflate at a 1% interest rate over the pit's life span. The distribution of both minerals is such that they have economic break-even values as opposed to geologic cut-off points. The mineralization occurs in an area approximately 3000 ft by 2500 ft and is found from a depth of 600 ft. There is up to 250 ft of overburden in some places.

A topographic map of the deposit showing the exploration bore-holes is illustrated in Fig. 7.1. The exploration drilling was done with vertical bore-holes located randomly within the 100 ft squares of the grid. The drilling progressed from the center of the deposit out to the limit of mineralization. The outer mineralization limit was found on further bore-holes were drilled.

The vertical cross-section shown in Fig. 7.2 was drawn from the exploration drilling. The primary mineral occurs within the outline shown by the dashed line. The secondary mineral occurs below the past levels of the water table or all the geologic features reported from the bore-holes, only one
FIG. 7.1 PLAN MAP AND EXPLORATION DRILLING PATTERN OF THE EXAMPLE DEPOSIT
fault and the present water table level are significant to the open pit mining solution. Only these features have any effect on the solution.

The fault and water table divide the deposit into four zones that are necessary to delimit the areas in which the geologic conditions vary sufficiently to influence grade prediction. The four mineralization zones are identified by the Grade Prediction Programme to prevent the estimation of point mineral values from assays occurring in different zones. This identification was described previously in Subsection 3.4.1. The spatial mineral distributions of the two recoverable minerals were computed by the Grade Prediction Programme and passed on to the MP Programme through magnetic disk storage.

The fault divides the deposit into two zones with different safe wall-angles. The slope stability conditions on the footwall side of the fault are better than those on the hangingwall side. The minimum safe wall-angles as measured from vertical are 0.096 radians and 0.726 radians for the footwall and hangingwall sides, respectively. Additionally, a metallurgical penalty will have to be imposed on the ore coming from a 0.45 m zone centered on the fault. The heat, pressure, and intrusions along the fault zone changed the mineral composition of the rocks in this area; consequently, the concentrates originating from this area require an additional refining expense. The standard refining cost is $90.60 cash units per ton of end product; whereas, the adjusted refining cost is $90.60 cash units per ton of end product for the concentrates originating from the metamorphosed zone.

The water table divides the deposit into two zones for which there are different tonnage factors. The tonnage factors are 3,526 tons and 1,750 tons per grid-block for the upper and lower sides of the water table, respectively. (The grid system used for this deposit is described later in this Section.)

Additionally, the water table divides the deposit into two zones for which it is convenient to have two types of overburden excavation equipment. The two equipment types have different mixture working space requirements above the water table. The equipment requires a minimum vertical wall angle of 0.106 radians measured from vertical and the
equipment used below the water table requires a minimum working wall-angle of 1.045 radians. The switch in equipment types is planned as a result of:

1) anticipated changes in the annual quantities of overburden removal,

2) changes in the digging characteristics of the pit material, and

3) required replacements for deteriorated excavation equipment.

With the change of equipment type, the standard waste excavation cost alters from 0.15/ton above the water table to 0.18/ton below. The difference in these standards includes the additional water pumping charges for material mined below the water table. The increased cost of excavation with depth as a result of greater transport distances and greater rock breaking strength is accounted for with the following adjustments of the standards: The overburden excavation cost on any particular mid-level is equal to 0.0024$ per ton above the water table or 0.0054$ per ton below, multiplied by the mid-level number plus the respective standard excavation cost.

The new excavation equipment planned for the pit is different than the overburden equipment and it will be the same type throughout the pit's life span. The new excavation costs are charged in a similar manner to the overburden costs as a result of the water pumping expenses below the water table and the changing digging characteristics. Therefore, the new excavation costs are 0.49$ per ton plus 0.005$ per ton per mid-level above the water table and 0.50$ per ton plus 0.005$ per ton per mid-level below.

The milling cost at the ore varies depending on whether it contains the secondary mineral. It is 0.05$ per ton of mill input for ore without the secondary mineral and 0.03$ per ton of mill input for ore with the secondary mineral.

The administrative or fixed costs are anticipated to be 500,000 cash units per year.

It is anticipated that all the above unit production costs will inflate at a 2.5% interest rate over the pit's life span.
The possible plant production capacities proposed for the mine are 7 million tons per year for the pit and the twenty seven combinations of concentrator, refinery and overall recovery efficiencies shown in Table 7.1. The entries under every combination are their capital costs. These costs represent the total capital required to finance the development of each combination. They are the total of the annual requirements discounted to the first production year. The discount interest rate is 8% which is the opportunity cost of investment capital.

| TABLE 7.1 |

---

The grid system used to simulate the deposit has rectangular axes which have regularly spaced grid coordinates. The distance between grid coordinates in the longitudinal direction is 100 ft. The traverse and vertical interval...
distances are equal to the bench width and height which are 100 ft and 50 ft, respectively. Ore excavation occurs from one working face that advances parallel to the longitudinal co-ordinate axis and changes the direction of its advance on alternate benches.

Finally, there are two further pieces of input data required to initiate the DP Programme. These are the initial assumptions for the T-S mineral distribution and the average unit production costs for the first year. The pit increments ‘X’ of the T-S mineral distribution’s data set ‘ORE5E5’ are assumed to include all material in every four grid-levels from the top of the deposit down. (See Subsection 6.3.1 for a detailed description of this assumption.) The average unit production costs for the first year were assumed to be:

- Pit: $Z = 0.83$ per output ton
- Concentrator: $C = 0.80$ per input ton
- Refinery: $R = 50.00$ cash units per output ton

Recall from Section 6.2 that these assumptions are replaced as the solution converges.

### 7.2 THE SOLUTION

A solution to exploit the deposit described in the preceding section was computed with the Model. The output of the Model’s DP Programme is presented in this section. Some of this information is used to monitor the solution — which can be terminated when these results are equal for two consecutive cycles. The output printed at the completion of each iteration cycle is illustrated in Tables 7.2 through 7.17. Every four tables are similar. The data contained in the similar tables represent the results for the iteration cycles one through four, respectively. Note that the results of the third and fourth iteration cycles are the same; therefore, this particular solution was terminated after the fourth cycle.

The cut-off grades and production targets are computed for each plant combination to maximize its present value. The present value of each plant combination is the anticipated annual profit from its production targets discounted to the first production year plus the capital costs illustrated in
THE NET PRESENT VALUE OF 27 PLANT CAPACITY COMBINATIONS
(FIRST ITERATION CYCLE)

<table>
<thead>
<tr>
<th></th>
<th>(T)</th>
<th>(M)</th>
<th>(1.5)</th>
<th>(1.6)</th>
<th>(1.7)</th>
</tr>
</thead>
<tbody>
<tr>
<td>85%</td>
<td>20.0</td>
<td>-</td>
<td>-23.55</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>90%</td>
<td>22.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>95%</td>
<td>25.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

PIT CAPACITY = 7 MILLION TONS PER YEAR
PRESENT VALUES IN MILLIONS OF CASH UNITS
(T) = capacities in thousands of tons per year
(M) = capacities in millions of tons per year

TABLE 7.2

*These present values are presented in Table 7.2 through Table 7.9. Table 7.2 only shows the present value of one plant combination. Only this present value is computed because the computations of the first iteration cycle are crude enough that it is immaterial which combination is selected. Tables 7.6 through 7.9 contain the cut-off grades and production targets for the plant combination selected to exploit the miner's deposit. The mine's life span, as presented in these tables, is required as input data for the following iteration cycle.*
## The Net Present Value of 27 Plant Capacity Combinations (Second Iteration Cycle)

<table>
<thead>
<tr>
<th>Overall Refinery Concentrator Capacity (M)</th>
<th>Recovery Capacity 1+5</th>
<th>1+6</th>
<th>1+7</th>
</tr>
</thead>
<tbody>
<tr>
<td>Overall Efficiency (T)</td>
<td>20+0</td>
<td>-16+11</td>
<td>-8+41</td>
</tr>
<tr>
<td>85%</td>
<td>22+5</td>
<td>-10+81</td>
<td>-2+01</td>
</tr>
<tr>
<td>90%</td>
<td>25+0</td>
<td>-6+74</td>
<td>-3+21</td>
</tr>
<tr>
<td>95%</td>
<td>20+0</td>
<td>-16+13</td>
<td>-3+50</td>
</tr>
<tr>
<td>97%</td>
<td>22+5</td>
<td>-17+04</td>
<td>-17+86</td>
</tr>
<tr>
<td>99%</td>
<td>25+0</td>
<td>-13+26</td>
<td>-16+80</td>
</tr>
<tr>
<td>Pit Capacity = 7 Million Tons Per Year</td>
<td>20+0</td>
<td>-9+52</td>
<td>-16+83</td>
</tr>
<tr>
<td>22+5</td>
<td>-4+48</td>
<td>-17+92</td>
<td>-12+72</td>
</tr>
<tr>
<td>25+0</td>
<td>-6+98</td>
<td>-2+17</td>
<td>-9+96</td>
</tr>
</tbody>
</table>

Table 7: 3
# The Net Present Value of 27 Plant Capacity Combinations (Third Iteration Cycle)

<table>
<thead>
<tr>
<th>Efficiency</th>
<th>Recovery Capacity (T)</th>
<th>Refinery Concentrator Capacity (T)</th>
<th>Overall Efficiency</th>
</tr>
</thead>
<tbody>
<tr>
<td>85%</td>
<td>20:0</td>
<td>-10:11</td>
<td>-2:56</td>
</tr>
<tr>
<td>85%</td>
<td>25:0</td>
<td>-1:20</td>
<td>2:16</td>
</tr>
<tr>
<td>90%</td>
<td>20:0</td>
<td>-10:13</td>
<td>2:35</td>
</tr>
<tr>
<td>90%</td>
<td>25:0</td>
<td>-7:58</td>
<td>11:35</td>
</tr>
<tr>
<td>95%</td>
<td>20:0</td>
<td>-3:60</td>
<td>-11:03</td>
</tr>
<tr>
<td>95%</td>
<td>25:0</td>
<td>-1:13</td>
<td>3:33</td>
</tr>
</tbody>
</table>

**CAPACITY=7 MILLION TONS PER YEAR**

Table 7-4

Next in each iteration cycle, the T-S net cash value distribution is computed with the current values of the cut-off grades and grid-block excavation times. The unit costs assigned to the grid-blocks are averaged for each production year. These annual production costs are printed out for each iteration cycle except the first. As for the present values in Table 7-2, the initial estimates of the unit costs are crude. Therefore, what would have been Table 7-10 is omitted from the print out.
<table>
<thead>
<tr>
<th>Recovery Capacity (t)</th>
<th>Efficiency (T)</th>
<th>Overall Refinery Concentrator Capacity (k)</th>
</tr>
</thead>
<tbody>
<tr>
<td>20 0</td>
<td>-10 20</td>
<td>1 6</td>
</tr>
<tr>
<td>22 5</td>
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Pit Capacity = 7 Million Tons Per Year

Table 7.5
CUT-OFF GRADES & TARGETS FOR THE PLANT COMBINATION:
(FIRST ITERATION CYCLE)

PIT CAPACITY = 7.0 MILLION TONS/YEAR
CONCENTRATOR CAPACITY (CON) = 1.6 MILLION TONS/YEAR
REFINERY CAPACITY (REF) = 20.0 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 85%

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PRESENT VALUE OF PROFITS LESS CAPITAL COST = 17.05 MILLION CASH UNITS

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**PRESENT VALUE OF PROFITS LESS CAPITAL COST = -0.49 MILLION CASH UNITS**
CUT-OFF GRADES & TARGETS FOR OPTIMUM PLANT COMBINATION:
(THIRD ITERATION CYCLE)

PIT CAPACITY = 70 MILLION TONS/YEAR
CONCENTRATOR CAPACITY (CON) = 1.7 MILLION TONS/YEAR
REFINERY CAPACITY (REF) = 200 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 95%

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PRESENT VALUE OF PROFITS LESS CAPITAL COST = 5.40 MILLION CASH UNITS

TABLE 78

The final pit surface is redesigned from the T-S net cash.
CUT-OFF GRADES & TARGETS FOR OPTIMUM PLANT COMBINATION:
(FOURTH ITERATION CYCLE)

PIT CAPACITY = 7.0 MILLION TONS/YEAR
CONCENTRATOR CAPACITY (CON) = 1.7 MILLION TONS/YEAR
REFINERY CAPACITY (REF) = 20.0 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 95%

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PRESENT VALUE OF PROFITS LESS CAPITAL COST = 5.34 MILLION CASH UNITS

TABLE 7-9

value distribution whenever the distribution is re
### AVERAGE UNIT PRODUCTION COSTS

(Second Iteration Cycle)

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<th>REFINERY (3)</th>
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**Table 7.11**

- Evaluated: These pit surfaces for the four iteration cycles.
### Table 7.12

**Average Unit Production Costs**

**Third Iteration Cycle**

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<td>Cash Units/ Input Ton</td>
<td>Cash Units/ Output Ton</td>
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*Note: The figures in this table are illustrated in Fig. 7.3. Each figure on the page is the*
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<th>Year</th>
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<th>Concentrator (C)</th>
<th>Refinery (R)</th>
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<td>63.357</td>
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</table>

**Table 7.13**

elevation of the final pit surface at that point. The
elevations are the distances in grid-levels from a horizontal datum level above the deposit down to the pit surface. These elevations represent the intercept of the pit surface with the grid system. However, the intercept at every grid-intersection are not shown here. They were deleted to speed up the solution. The contours on the map were drawn in by hand at 100 ft. intervals.

The grid system was not made large enough for this solution. The rim of the pit went past the edges of the grid system. This example illustrates the care that must be taken to make the grid large enough to encompass the entire pit. Otherwise, the grid system does not contain all the overburden required to expose the deposit, and the pit will be larger than it should be to maximize the profits. However, if the limits of the grid system are socio-political boundaries, then those boundaries must be simulated as described in Example 3, Subsection 6.4.2, in which case, the pit design routine will insure that the pit cases entirely within the boundaries.

A schedule of the simulated production is computed for the pit material after each design of the final pit surface. These schedules are illustrated in Tables 7.16 - 7.17. The annual overburden stripping costs listed in these tables are the expenses of exposing the ore. The annual revenue is the gross profit earned from the ore less the production costs. The pit production simulation gives the information in Tables 7.16 - 7.17 as targets. However, if the mining conditions are such that the targets cannot be met, they are then superseded, as explained in subsection 6.4.2. The refinery output is not listed in these production schedules because it is not significant information for monitoring the solution. The refinery output is fixed by the cut-off grades and concentrator input - information already presented.

Finally, there are two last pieces of information used to monitor the iterative solution that are printed out with the production schedules. First, there is the number of pit increments in the 50 annual distributions. This shows, in fact, of the input data for the following iteration cycle. Secondly, there is the total of the annual revenue.

When the iterative solution is completed, there is a large amount of additional information that can be printed out. This information consists of:
FIG. 7.3A  THE FINAL SURFACE OF THE PIT (FIRST ITERATION CYCLE)
FIG. 7.3B THE FINAL SURFACE OF THE PIT (SECOND ITERATION CYCLE)
OPTIMUM PIT SURFACE

FIG. 7.3C THE FINAL SURFACE OF THE PIT (THIRD ITERATION CYCLE)
OPTIMUM PIT SURFACE

FIG. 7. 3D: THE FINAL SURFACE OF THE PIT (FOURTH ITERATION CYCLE)
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TOTAL REVENUE = 109.11 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREASES = 11

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TOTAL REVENUE = 148.05 MILLION CASH UNITS

NUMBER OF PIT INCREMENTS = 9

TABLE 7.15

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TOTAL REVENUE = 14,510 MILLION CASH UNITS
TOTAL RANGE OF PIT INCREASES = 9

TABLE 7-10
### Table 7.17

**SIMULATED PIT PRODUCTION SCHEDULE (FOURTH ITERATION CYCLE)**

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**Notes:**

1. Total revenue = 142.50 million cash units
2. Instantaneous and overall stripping ratios,
3) the annual refinery output,
4) a list of the T-S net cash value distribution,
5) a list of the year and order of excavation for each grid-block mined, and so forth.

This data and any of the other data generated during the solution can be printed out by merely including the appropriate FORTRAN statements into the EP Program.

Finally, it should be pointed out that the EP Program can be used to study constraints in the production. These constraints are identified in the output illustrated in Table 7-10. In this example, note that the pit constraints production in the first four years and then the concentrator for the rest of the pit's life span. Additionally, the refinery capacity is a great deal larger than required to process the concentrates. It is possible to smooth out these constraints by specifying a more compatible combination of plant capacity and pit production schedule. Similarly, various rates of stripping overburden can be studied by measuring the effect on profits of changing the pit production capacity.

7.3 ECONOMIC EVALUATION OF THE SOLUTION

The open pit mining solution in the preceding Section is evaluated economically to assess its profitability. The revenue (R) and operating stripping costs (C) of Table 7-10 are analyzed in the cash flow of Table 7-21. The cash flow out to the equity capital (E) is the cash flow (C) in less the revenue from the operating stripping costs, the tax, and the interest (I) on total capital (K). The interest rate on long capital is taken as the opportunity cost of investment capital (I). The profitability of the total cash flow (K) is measured by computing the present value (PV) and discounted cash flow yields. The tax entries are the present value in the given year of the entire profit. The cash flow calculation has no entry for fixed or capital-intensive costs because they have already been accounted for in the revenue and overburden stripping costs by the standard costing method used in the EP Program. The taxes are computed according to the small-volume tax structure.
FIG. 7.4 FIVE YEARLY PLAN MAPS OF THE PIT
FIG. 7.4 (CONTINUED)
TOPOGRAPHIC MAP OF PIT IN YEAR 11

FIG. 7.4 (CONTINUED)
TOPOGRAPHIC MAP OF PIT IN YEAR 16

FIG. 7.4 (CONTINUED)
FIG. 7.4 (CONTINUED)
FIG. 7.4 (CONTINUED)
TOPOGRAPHIC MAP OF PIT IN YEAR 31

FIG. 7.4 (CONTINUED)
FIG. 7.4 (CONTINUED)
FIG. 7.4 (CONTINUED)
The capital requirements (E) listed in the cash flow are the development expenditures for the construction of plant facilities derived from Table 7.1 to exploit the example deposit. Recall that these capital requirements were discounted to the first year to make up the entry: pit = 7.4 tons/yr, concentrator = 1.74 tons/yr, refinery = 20 tons/yr and overall recovery efficiency = 95%.

The last column lists the loan capital repayments (L). All entries are in millions of cash units.

The total present value (-5.19) of the cash flow in Table 7.18 is not identical with the corresponding present value of the production targets for the preceding plant combination (See Table 7.9). Obviously, this must be so, if the mining conditions are such that the production targets cannot be met.

The cash flow analysis is the ultimate measure of any open pit mining operation's worth. That solution resulting from the input mining proposal which yields the greatest profitability, is the obvious choice for exploiting the owner's deposit. Note that this solution for the example deposit is unprofitable with a negative total present value and a discounted cash flow yield less than the opportunity cost of investment capital. However, it is possible that the deposit could be made profitable with a different mining proposal and better scheduling and proportioning of the loan and equity capital.

7.6 AN INVALID SOLUTION

An invalid solution of the PP programme is illustrated in Table 7.16. This table was generated by the programme for the example deposit with the following plant capacities:

1) Pit = 5.67m tons per year.
2) Concentrator = 294 tons per year, and
3) Refinery = 20 tons per year.

Recall from Subsection 6.7.2 that an invalid solution arises when the simulated waste volume production exceeds the
<table>
<thead>
<tr>
<th>YR REV</th>
<th>INT TAX OB</th>
<th>CFI ECAF CF PV LC CR LR</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.59 0.0 0 0 2.61 2.02 11 0 13 02 -5.39 0.0 7.1 0 0.0</td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>0.56 0.0 0.4 2.53 -1.97 4.2 -6.22 7.19 0.0 5.6 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>3</td>
<td>0.63 0.0 0 0 2.76 -2.10 0.0 -2.10 13.99 0.0 5.5 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>4</td>
<td>2.14 0.0 0 0 3.33 -1.75 0.0 -1.75 17.21 7.5 6.5 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>5</td>
<td>2.93 0.0 0 0 0.5 0.0 0.0 0.3 20.34 7.1 8.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>6</td>
<td>2.26 0.0 0 0 1.42 0.0 0.0 0.3 21.43 0.0 7.5 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>7</td>
<td>2.08 0.0 0 0 1.51 -0.24 0.0 -0.24 23.11 0.0 0.0 2.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>8</td>
<td>2.31 0.0 0 0 1.26 0.0 0.0 0.3 25.20 0.0 0.0 2.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>9</td>
<td>3.31 0.0 0 0 1.49 1.33 0.0 1.33 26.84 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>10</td>
<td>4.89 0.0 0 0 1.33 0.0 0.0 0.3 27.43 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>11</td>
<td>3.52 0.0 0 0 0.6 0.0 2.0 0.3 29.15 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>12</td>
<td>3.26 0.0 0 0 1.32 0.0 0.0 0.3 27.43 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>13</td>
<td>3.67 0.0 0 0 1.74 0.0 0.0 0.3 30.54 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>14</td>
<td>4.02 0.0 0 0 1.3 1.43 0.0 1.43 32.05 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>15</td>
<td>3.68 0.0 0 0 0.3 0.0 2.3 0.0 33.19 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
<tr>
<td>16</td>
<td>6.51 0.0 0 0 0.2 4.81 0.0 4.81 33.54 0.0 0.0 0.0 0.0 0.0</td>
<td></td>
</tr>
</tbody>
</table>

**Discounted Cash Flow Yield = 6.24%**

**Table 7.18**

Total pit capacity, as it has in the third, fourth, fifth years, etc. This is due to the fact that the production targets have been computed from an inadequate pit capacity.
<table>
<thead>
<tr>
<th>YR</th>
<th>M TONS</th>
<th>M CASH UNITS</th>
<th>STRIPPING COSTS</th>
<th>ORE</th>
<th>M TONS</th>
<th>M CASH UNITS</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
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<td>1.27</td>
<td>1.66</td>
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<tr>
<td>5</td>
<td>2.25</td>
<td>1.12</td>
<td>1.31</td>
<td></td>
<td>2.20</td>
<td>2.30</td>
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<td></td>
<td>2.30</td>
<td>3.19</td>
</tr>
<tr>
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<td></td>
<td>2.30</td>
<td>5.07</td>
</tr>
<tr>
<td>10</td>
<td>5.41</td>
<td>2.65</td>
<td>2.11</td>
<td></td>
<td>3.05</td>
<td></td>
</tr>
<tr>
<td>11</td>
<td>6.62</td>
<td>3.48</td>
<td>1.97</td>
<td></td>
<td>1.84</td>
<td></td>
</tr>
<tr>
<td>12</td>
<td>5.43</td>
<td>2.35</td>
<td>1.97</td>
<td></td>
<td>3.09</td>
<td></td>
</tr>
<tr>
<td>13</td>
<td>5.15</td>
<td>2.90</td>
<td>1.97</td>
<td></td>
<td>2.56</td>
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</tr>
<tr>
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<td>0.15</td>
<td>1.97</td>
<td></td>
<td>4.53</td>
<td></td>
</tr>
<tr>
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<td>1.69</td>
<td>1.02</td>
<td>1.97</td>
<td></td>
<td>7.26</td>
<td></td>
</tr>
<tr>
<td>16</td>
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<td>0.53</td>
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<td></td>
<td>11.35</td>
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<tr>
<td>17</td>
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<td>0.20</td>
<td>2.96</td>
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<td>8.01</td>
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<td>18</td>
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<td></td>
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<tr>
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<td>0.40</td>
<td>2.96</td>
<td></td>
<td>7.05</td>
<td></td>
</tr>
<tr>
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<td>4.22</td>
<td>2.85</td>
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<td>1.22</td>
<td>2.96</td>
<td></td>
<td>6.50</td>
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<tr>
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<td>0.00</td>
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<td></td>
<td>6.61</td>
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</tr>
<tr>
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<td>0.00</td>
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<td></td>
<td>8.33</td>
<td></td>
</tr>
<tr>
<td>26</td>
<td>0.28</td>
<td>0.18</td>
<td>2.96</td>
<td></td>
<td>7.61</td>
<td></td>
</tr>
<tr>
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<td>0.27</td>
<td>2.96</td>
<td></td>
<td>5.47</td>
<td></td>
</tr>
<tr>
<td>28</td>
<td>2.12</td>
<td>1.07</td>
<td>2.96</td>
<td></td>
<td>7.88</td>
<td></td>
</tr>
<tr>
<td>29</td>
<td>2.95</td>
<td>3.26</td>
<td>1.00</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

TOTAL REVENUE = 154.24 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREMENTS = 6

TABLE 7.19
The MP Programme is made up of three modules which are needed for the different computations in the various iteration cycles of the open pit mining solution. The computations within the iterative solution are varied by calling different modules. The construction of these modules is illustrated in Table 8.1. Each module is created by including the appropriate calling programme and four subroutines. The calling programmes and subroutines are stored individually in a partitioned data set. There is one module per cycle for the execution of the first two cycles of the iterative solution. The third module is used to execute all additional cycles.

Having divided the MP Programme in this way, it became possible to exclude some time consuming operations in the first two iteration cycles. The excluded operations are not needed because they are refinements unnecessary to the initial accuracy. The large computer time per cycle also made it advantageous to pay scant the computations for increased control of the solution and to achieve a minimum time loss from computer failures. The results of each iteration cycle are stored on a magnetic disk. If there is a failure in the solution, it is unnecessary to restart the Programme from the beginning. The user merely is required to retrieve the results of the previous cycle and proceed.

8.1 JOB CONTROL LANGUAGE

The MP Programme is executed in the three job steps illustrated in Fig. 8.1 through 8.3. Note that certain entries in the figures are written in lower case letters. These entries mean that which is to be replaced in their position. The label of the disk pack on which the partitioned direct access and sequential data sets should be substituted for 'disk pack label'. The position of the source deck and data sets within the control cards is
indicated with explanatory phrases. The remaining substitutions are explained in the following description of the job steps:

JOB STEP 1:

The calling programmes and subroutines are compiled, link-edited and stored individually in a partitioned data set. The control cards illustrated in Figs. 8*1 are used repetitively to load the source deck of cards for each member of the partitioned data set. The names of the calling programmes and subroutines, as listed in Table 8*1, are substituted, in turn, for the internal name of the LKED-SYSLMOD card's DSNAME parameter.

<table>
<thead>
<tr>
<th>MP PROGRAMME</th>
</tr>
</thead>
<tbody>
<tr>
<td>MODULE 1</td>
</tr>
<tr>
<td>CALLING PROGRAMMES</td>
</tr>
</tbody>
</table>
| SUBROUTINES | CCGB | COG2 | ---->
| CGJSUB | ----> | ---->
| PD | ----> | ----> | ---->
| SUBPD | ----> | ----> | ---->

TABLE 8*1

JOB STEP 2:

The second job step creates one of the three modules (MP1, MP2 or MP3) by including the appropriate members of the partitioned data set. The modules are created by executing the control card list in Figs. 8*2 with the following substitutions:

1) 'M' and 'MP' are replaced with '1' to create the module MP1 which completes the first iteration cycle,

2) 'M' and 'MP' are replaced with '2' to create the module
The macros SYSLIB and SYSDATA are used to load the first member of the partial data set only. The 'LINK,SYSLIB' card must be replaced by the following card to compile the remaining data set members or to recompile any of these members:

```
//LINK,SYSLIB DD DSELECTION('h170'),DISCARD,STORAGE=43,MNAME=(AAAAAAA)
//```

**Figure 3.1** Compilation and Link-Post Job Step
(job anl)

// EXEC PT=IMP1,PARM="WAT",OPTION=150K
//SYSLIN DD SYSIN=A
//SYSLIN DD UNIT=SYYDA,VOLUME=GEN认'ask pack label,SPACE=(1K,(10,10))
//SYSLIN DD CBLN=TYY1,STGL,DISP=SHR
//SYSLIN DD SYSLIN=SYS1(1M),DISP=(1K,1K,1K),UNIT=SYYDA
//DD (REDEF=U,REXX=3625)
//A DD CLASS=SYS1IP,DISP=SH,UNIT=SYYDA,DISP=HASK pack label,UNIT=SYYDA
//SYSLIN DD *,DISP=SH,REXX=60
EVER 'WIN
INCLUDE A(MAI1NS)
INCLUDE A(CO)
INCLUDE A(COSSUB)
INCLUDE A(PD)
INCLUDE A(SUBPD)
*
//

The 'DISP' parameter must be equal to 'OLD', with the 'CEE' and 'SHARE' parameters deleted on the 'SYSLIN' card for any executions of this job step after the first, as follows:

//SYSLIN DD CBLN=TYY1(1M),DISP=(1K,1K,1K),STGL,UNIT=SYYDA,DISP=HASK pack label

PICT. 8A. CREATION OF MODULES JOE ANL
MP2 which computes the second iteration cycle, and

3) 'n' is replaced by '3' and 'nn' by '2' to create the module MP3 which computes all iteration cycles from the third one on.

JOB STEP 3:

The module created in the second job step is executed by the job control cards listed in Table 8.4. The module number 'n' is replaced by the one created in the second step.

Each module required an average of thirty three minutes to execute a solution for a grid system with 1,300 grid intersections on the International Business Machines Corporation's IBM 360/50 computer.

2 PREPARATION OF THE PROGRAMME FOR EXECUTION

The steps needed to implement the FORTRAN programme are listed below. The Program can be prepared for execution by using these steps to locate the comments within the Programme listing which describe any required modifications.

STEP 1:

The dimensions must be specified for the FORTRAN arrays, sequential data sets and direct access data sets. These dimensions are

A) defined in Section 8.8 for the specification statements of the arrays,
B) given for the sequential data sets in Section 8.1, and
C) given for the direct access data sets in Section 8.1 for the DECLARE DATASETS and in Subsection 8.7.2 for the DEFINE FILE statement.

STEP 2:

The grid system is already defined for the preceding Programmes in the Model. However, the maximum number of grid co-ordinates X, Y and Z in the I, J and K co-ordinate
The sequential and linear source list acts remain for the three modules (P11, P12, and P13) are:

1) For the first module (P11):

```
//GO.PT11FO01 DD DECKED=NOHANDLE,DISP=OLD, MODE=3311,UNIT=3311,DISP=NEWFILE
//GO.PT11FO01 DD UNIT=3311,DISP=(NEW,FULL),UNIT=3311,
// VOLUME=NOVOL, FILE=MODC11.311, LIST=(311,311)),
// DCP=(NOCP=NEW,UNIT=3311,DISP=3311)
//GO.PT12FO01 DD DECKED=NOHANDLE,DISP=OLD,MODE=3311,UNIT=3311,
// VOLUME=NOVOL, FILE=MODC12.311, LIST=(311,311)),
// DCP=(NOCP=NEW,UNIT=3311,DISP=3311)
//GO.PT13FO01 DD DECKED=NOHANDLE,DISP=OLD,MODE=3311,UNIT=3311,
// VOLUME=NOVOL, FILE=MODC13.311, LIST=(311,311)),
// DCP=(NOCP=NEW,UNIT=3311,DISP=3311)
```

2) For the second module (P12):

```
//GO.PT11FO01 DD DECKED=NOHANDLE,DISP=OLD,MODE=3311,UNIT=3311,
//GO.PT12FO01 DD DECKED=NOHANDLE,DISP=OLD,MODE=3311,UNIT=3311,
//GO.PT13FO01 DD DECKED=NOHANDLE,DISP=OLD,MODE=3311,UNIT=3311,
```

(continued on page)

---

**FIG. 8.3** ENVIRONMENT FILE
3) For the third module (W3):

```c
//GO.PT14FOOL DD VOLUME=UPDATE2,UNIT=2311
// VOLUME='IR', H=3, L=4, X=(CYL,(2,1)),
// DCI=(RECP:V,1,0,=12,33(40,35))
//GO.PT15FOOL DD VOLUME='IR', UNIT=2311
// VOLUME=SER=Ishro, LABEL, SPAC=( possessions, (2,1)),
// DCI=(HREM=V,1,0,=12,33(40,35))
//GO.PT16FOOL DD VOLUME='IR', UNIT=2311
// VOLUME=SER=Ishro, LABEL, SPAC=( possessions, (2,1)),
// DCI=(HREM=V,1,0,=12,33(40,35))
```

FIG. 3.5 (CONT.)
revisions, respectively, are required as part of the HP Program's input data.

STEP 3:

The appropriate number of FORTRAN variables must be created for each recoverable mineral reported by the Grade Prediction Program*. This modification only is required for the calling program*. The variables and their location in the program are described in the comments of Subsection 8.3.2.

STEP 4:

A factor 'GS' is selected to scale the mineral grades up to integer values. Recall from Section 6.3 that the integer mineral values are used for the subscript 'M' of 'GAMMA'. Additionally, a mineral grade increment 'GI' is selected to limit the size of subscript 'M'. This increment groups all minerals occurring in the same grade interval into one class. A balance must be made between accuracy and the ORGAN array size. The scale factor 'GS' must be large enough to avoid INTEGER truncation. In contrast, the grade increment 'GI' must be small enough that subscript 'M' of ORGAN does not become excessively large. 'GI' and 'GS' are input data to the HP Program.*

STEP 5:

The initial subscript for the subscript 'M' of 'ORGAN' is given by the user (Subsection 8.3.1) for a description of how this is done. This assumption is located in the calling program MAIN (Subsection 8.3.2).

STEP 6:

Any geologic or physical features, legal boundaries and costing situations which may affect the following operations or variables must be simulated:

A) T-3 not cull value distribution,
B) design of the final pit surface,
C) the tonnage factors, and
D; pit production simulation.

There is a comment at those places in the Program lists where the simulation of these features is required.

STEP 7:

The variable 'D' in the cone expressions for the final pit surface design and waste mining simulation routines must be dependent on the pit wall-angles, if these angles vary. The relationship between 'D' and the pit wall-angles is explained further in the comments in the lists of subroutines PD and SUDP.

STEP 8:

The ore excavation sequence programmed in subroutine PD must be tailored to the user's requirements. The desired number of ore working faces per bench and the number of exposed ore benches, and the direction of their advance is simulated as explained in Subsection 6.7.1.

8• 3 CALLING PROGRAMMES

A verbal description of the first calling programme's computational sequence is presented in Subsection 8•3•1. It is then listed in the next Subsection with comments to explain some of its features. The last two Subsections contain the lists of the other calling programmes with comments explaining how they differ from those preceding them.

8• 3• 1 COMPUTATIONAL SEQUENCE OF KAIN

LINES 001 - 007
Specification statements.
LINES 008 - 054
Input statements.
LINES 059 - 080
DO loop 11 computes:

A) an equivalent mineral grade 'EQPAD' to represent all the deposit's recoverable minerals, and

B) the initial assumption for the T-S mineral distribution.

LINES 083 - 091

The first estimate of the unit production costs is computed.

LINE 094

The following values are supplied by the nested subroutines CSG1 and COGUSD for the selected plant combination:

A) the annual cut-off grades 'CUTOFF' and production targets "I", "CON", (REF);

B) the overall recovery efficiency 'YIELD', and

C) an average 'AVG' of the annual cut-off grades 'CUTOFF'.

LINES 095 - 158

These computations execute the first estimate of the T-S net cash value distribution 'VAL' and the second estimate of the average unit production costs 'E, C, N'. At this point in the solution, it is not known when the ore-blocks are mined; consequently, it is assumed that they are mined in the first year 'T' and, in addition, their equivalent mineral contents are compared with the average cut-off grade 'AVG'.

LINE 151

The following computations are executed by the nested subroutines PD and SUU:

A) the final pit surface is designed and

B) the pit production is simulated.

LINES 162 - 167
The annual revenue is totalled 'TOTAL' and the number of pit increments computed 'MAXI'. These results are printed out to assist the user in monitoring the solution.

**LINES 160 - 179**

The second estimate of the T-S mineral distribution 'ORERES' is computed with the grid-block excavation times 'YR' from the pit production simulation in subroutine 'PD'.

**LINES 180 - 185**

The results of the first module 'MY' are stored on a disk pack for future use. The T-S mineral distribution 'ORERES' and the grid-block excavation times 'YR' are used to update the iterative solution. The remaining data (TOPO, IGRAD1, IGRAD2) are stored for rapid access rather than being recalculated in the other modules.

**LINES 186 - 201**

Yearly topographic maps are printed to illustrate the pit production simulation.

8 * 3 2 A LIST OF MAIN1

---------------------------------------------------------------------------------------------------

The example deposit has two recoverable minerals 'GRAVI' and 'GRAD2' whose present selling prices are 'S1' and 'S2', respectively. The following lines must be modified if the user's deposit contains different minerals:

**MAIN1** : LINES 004, 005, 021-022, 036-038, 047, 071-072, 081-082, 124-126, 142 and 144;

**MAIN2** : LINES 004, 005, 016-017, 040, 049-050, 090-092, 100 and 117;

**MAIN3** : LINES 004, 005, 016-017, 046-047, 049, 052, 091-093, 100 and 117

---------------------------------------------------------------------------------------------------

001 DOUBLE PRECISION IN(1, 1, 1), COMCAP(2), REFCAP(3), NINSAI(97, 20), KIN(12), 20(20), 207(65, 20), 20(65, 20), 209(45, 20), 310(45, 20).
ii = HK*NN+NN; jj = 4*(number of recoverable minerals)

The scale factor 'ii' and junk increment 'jj' (Line 011) are given by the user to set up the subscript 'ii' of 'OBERNA' in Lines 037, 076 and 177 as explained in Step 4, Section 8.2.
The initial auburnv-...
The example deposit has two tonnage factors 'TONF' one for each side of the tenth mid-level. The following four statements select the appropriate tonnage factor for the 'ORETIES' computations. These statements are modified as explained in Subsection 6.1.1, if the user's deposit requires different tonnage factors. Additionally, the modification must be repeated each time tonnage is involved in a computation.

(SEE: MAIN1: LINES 110, 120 & 173; MAIN2: LINES 076, 096 & 144; MAIN3: LINES 077, 087 & 145; PC: LINES 118 & 143; AND SUPPORT: LINES 050+)

The following two statements must be modified to calculate an equivalent mineral grade for each secondary mineral reported in the user's deposit, as explained in Subsection 6.3.2.

```
060 DO 11 N=1,MAXN
061 K1=N*K
062 DO 11 K=1,MAXK
063 DO 11 J=1,MAXJ
064 DO 11 I=1,MAXI
065 LOC=J*NI*(K-ONE)+I*(J-ONE)*I
066 IF (TOPO(LOC), 11,11,5

-------------------COMMENT-------------------
The example deposit has two tonnage factors 'TONF' one for each side of the tenth mid-level. The following four statements select the appropriate tonnage factor for the 'ORETIES' computations. These statements are modified as explained in Subsection 6.1.1, if the user's deposit requires different tonnage factors. Additionally, the modification must be repeated each time tonnage is involved in a computation.

(SEE: MAIN1: LINES 110, 120 & 173; MAIN2: LINES 076, 096 & 144; MAIN3: LINES 077, 087 & 145; PC: LINES 118 & 143; AND SUPPORT: LINES 050+)

The following two statements must be modified to calculate an equivalent mineral grade for each secondary mineral reported in the user's deposit, as explained in Subsection 6.3.2.

```

```
071 IF (K-10) 6,6,7
072 MM=ONE
073 GO TO 8
070 MM=2

-------------------COMMENT-------------------

The following two statements must be modified to calculate an equivalent mineral grade for each secondary mineral reported in the user's deposit, as explained in Subsection 6.3.2.

```

```
074 8 y=(234(J)/21.0)*CRIMEAU(J,X,F)
075 X=MAX(J,X)*EXPAND(J,X,F)*TIX(X)
076 IT (EXPAND(J,X,F)*TIX(X)) 9,9,10
077 9 MM=ONE
078 GO TO 70
079 10 MM=MAX(J,X)/FIT
077 11 IT ("ADD=X") 40,40,70
078 40 RAXG=X

-------------------COMMENT-------------------

The following two statements must be modified to calculate an equivalent mineral grade for each secondary mineral reported in the user's deposit, as explained in Subsection 6.3.2.

```

```
079  "0 ORDERES (X, N) = ORDERES (L, N) + TONE (MN)
080  11 CONTINUE
081  WRITE (6,22) MAXG
082  226 FORMAT (1X, 'SAMAXG=',I4)
083  LC 13 N=1,99
084  T=n
085  X=(1+INFFC)**T
086  Y=(1+INFRP)**T
087  X(K)=X*WNC
088  C(K)=X*COMC
089  R(N)=X*P2FC (1)
090  F(N)=X*F1XC
091  S2(N)=Y*S2(1)
092  S1(J)=Y*S1 (1)
093  13 CONTINUE
094  CALL COS (AVG, YIELD)
095  T=ONE
096  LC=0
097  MC=G
098  SUM=E=0
099  SUXC=0
100  SUMX=0

---COMMENT---

This loop 74 computes the T-S not cash value distribution.
This routine, and its duplicates in MAIN1 and MAIN2, must be
tailored to the user's desired cost structure, as explained
in Subsection 6.5.2.

101  DO 34 K=1,NK
102  DO 34 J=1,NJ
103  DO 34 I=1,N1
104  LOC=LOC*1* (K-O'H1)+N1+(J-O'12)+1
105  IF (TOPO (LOC)) 34,34,14
106  34 N=05E
107  16 IT (EC, PAD (L, J, K)-AVG) 14,27,21
108  13 LC=LC*ONE
109  REV=0*0
110  IE (K-10) 20,21,21
111  20 N=G3L
112  GO TO 22
113  21 N=TWN
114  22 Y=(OUC (N)+(K*O8CI (N))+(1+INFR)+1)
115  SUM=SUM+X
116  COST=(X*IC1M1)/XEC (M1)*TONE (N)
GO TO 33
118 LC=LC+ONE
119 KC=KC+ONE
120 IF (K-10) 24, 24, 25
121 24 S=0.0
122 GO TO 26
123 25 N=K
124 26 I=IGRAD1(I,J,K)
125 I=IGRAD2(I,J,K)
126 REV=((FLOAT(I1)/GS*S1(M))+((FLOAT(I2)/GS*S2(M)))
127 1TonF(M)+1M
128 X=(OJ+(OJ+1.0)/0.5)
129 Y=(OJ+(OJ+1.0)/0.5)
130 IF (IGRAD1(J,K)) 27, 27, 27
131 27 XX=ONE
132 GO TO 29
133 28 IF T=0
134 29 Z=KILC(X))=((1*INFR)**M)
135 SUM=SUM+Y+Z
136 IF (ABS((C*3.1)-(27/5))-37-0.525)+3274*0.071
1/280)-300.30, 30, 31
137 30 T=ONE
138 GO TO 12
139 31 H=ONE
140 32 W=(KILC(X2))=((1*INFR)**M)
141 SUM=SUM+X
142 COST=Y*(2*YIELD+((FLOAT(I1+I2))/GG)+F(M)/
143 1MIN(Y)))*TONF(M)
144 33 VAL(LOC)=?E7-COST
145 34 CONTINUE
146 IF (T=0) 36, 36, 35
147 35 E(T)=SUM/LC
148 36 IF (SMK=33, 33, 37
149 37 C(T)=SUM/XC
150 R(T)=SUM/YC
151 38 CONTINUE
152 CALL LD
153 Ph.rk=-N
154 DO 39 X=1, L
155 39 PM(X)=PM(X)+PNEW
156 222 N=K+1,TOTAL=0.0
157 223 N=K+1,TOTAL=TOTAL+P(X)*X
158 224 N=K+1,TOTAL=TOTAL+P(X)*X
159 225 N=K+1,TOTAL=TOTAL+P(X)*X
DO 41 M = 1, 99
DO 41 N = 1, 20
41 ONERES(K, M) = 0.0
DO 43 K = 1, M
DO 44 J = 1, M
44 CONTINUE
LOC = N * M + K
IF (TOTO(LOC)) 45, 48, 42
IF (Y9(LOC)) 48, 49, 51
PITINC = (X(LC) - 0.5)/(S*0.5)
IF (K < 10) 43, 44, 45
43 M = ONE
GO TO 45
M = 7.0
IF (ECGRAD(I, J, K) - G1) 46, 46, 71
46 M = ONE
GO TO 47
WRITE (11) (TOTO(N), Y9(N), N=1, NULKS)
47 CONTINUE
END FILE 11
WRITE (12) ((TFRS(N, N), N=1, MAX), N=1, MAX)
END FILE 12
WRITE (13) ((ECGRAD(I, J, K), TGRAD(I, J, K))
11 TOPOG(I, J, K), I=1, 12, J=1, 40)
END FILE 13
50 WRITE (6, 261)
201 FORMAT (" ANNUAL PIT OUTLINE")
DO 211 N = 1, 99
211 SURF(N) = 0
DO 210 N = 1, L
210 WRITE (102) N
202 FORMAT (" A TOPOGRAPHIC MAP OF MINE IN YEAR")
DO 210 L = 1, 99
210 M = 0
K = ONE
217 IF (K < 30) 190, 154, 174
300 IF (ECGRAD(I, J, K) = 0.0) 210, 212, 171
190 IF (TOTO(LOC)) 174, 174, 175
212 M = F
300 K = K + 1
DO TO 203
300
The comments within the ' ' describe how MAIN2 differs from MAIN1:

```plaintext
001  DOUBLE PRECISION CAP(3,3,3), CONCAP(3), REICAP(3),
     1 MISCAP, CIERRA(93,25), CIER(93), CI(93),
     2 CIOR(3,25,20), CI(65,20), CI(65,20), CI(65,20),
     3 CI(65,20), CI(65,20), CI(3,3,3), CI(49,99), POLE, PNEW, PANGE, PIAC,
     4 CIOR(3,25,20)
002  REAL XIC(2), INFAC, INFRD, MISC
003  INTEGER XIC(2), INFAC, INFRD, MISC
004  INTEGER XIC(2), INFAC, INFRD, MISC
005  DIMENSION CIOR(3,25,20), CIERRA(93,25), CIERRA(93,25),
     1 CIERRA(93,25), CIERRA(93,25), CIERRA(93,25), CIERRA(93,25),
     2 CIERRA(93,25), CIERRA(93,25), CIERRA(93,25), CIERRA(93,25),
     3 CIERRA(93,25), CIERRA(93,25), CIERRA(93,25), CIERRA(93,25),
     4 CIERRA(93,25), CIERRA(93,25), CIERRA(93,25), CIERRA(93,25),

The input data of line 007 are required from the results of the first module EP1:

A) the analytic equivalent annual grade 'MAXO' required in the model deposit at shift 04
B) the mine's life span 'L' reported by the subroutine COG1
```
The total number of pit increments \( \text{MAXN} \) reported by the preceding iteration cycle is required to modify the input of the following 'READ' statement:

```plaintext
READ (5,6) MAXN,L
6 FORMAT (2I3)
READ (5,4) SJ,NK,GS
NLKN=NI*K=KK
4 FORMAT (3I1,H1*)
GI=5
DO 260 K=1,NK
DO 260 J=1,K
DO 260 I=1,NI
160 IGRAD1(I,J,K)=0
170 IGRAD2(I,J,K)=0
260 FORCGRAD(I,J,K)=C
DO 210 LOC=1,NI*K
210 VAL(LOC)=6.0
190 YR(LOC)=0
210 CONTINUE
READ (5,6) (((CAP(I,J,K),I=1,3),J=1,3),K=1,3)
60 FORMAT (1F12.1)
READ (5,4) (CS(I),OSCL(I),OBSL(I),0*OHQ(I),PIEC(I),
1EFQ(I),PIEC(I),K00N(I),1=1,3)
51 FORMAT (5F*3,17*1,2F*4)
DO 27 SLOPE(I)=(2*2(XITANH(I))/COS(PITANH(I)))**2*
63 WSLOPE(I)=(1.0*THAND(I))/COS(2*THAND(I)))**2*
```

The two data sets 'MAX' and 'Ndek' are used to update the iterative solutions. The data entered in the format 'Maxn', 'Di-
PAD2', and 'Ndek' are used by the data block where they are
stored rather than being recalculated.

READ (11) (TOPO(X), YR(N), N=1, M3LKS)
READ (12) (DUCE5(Y, N), X=1, MAXN), N=1, M3X)
READ (13) ((IYR(i, j, k), I=1, M1), J=1, SJ), K=1, MK)

DO 13 N=1, 99
T=H
X=(1+1.3FRC)*7
Y=(1+1.3FRC)*7
E(i)=VX3,3
C(i)=VX3,3
R(i)=VX3,3
F(i)=VX3,3
S2(i)=VX3,3
S1(i)=VX3,3

13 CONTINUE
CALL COG (AVEG, YIELD)

DO 38 has been added to this module to make the first estimate of the time dimension for the net cash value distribution 'Y'. It and the average unit production costs 'L' and 'U'.

053 DO 34 T=1, L
054 LC=0
055 X=0
056 SUX=0
057 SUY=0
058 SUZ=0
059 DO 34 K=1, SK
060 DO 34 J=1, S3
061 DO 34 I=1, S1
062 LOC=X+Y+Z+(F-CXF)+NI+(J-CJF)*I
063 IF (TOPO(LC)) 10, 34, 10

The time dimension 'L' and net block excavation time 'Y' were added to COG COG with the statement in line 999, 999. The number of blocks was assumed to be the first year of net block production for the net block production has not been calculated in the first module 'Y'. This equation eventually becomes relevant as the solution converges.
14 IF (YR(LOC)) 15, 15, 17
15 N=ONE
16 IF (I X) 16, 16, 34
17 IF (CUTPAD(I,J,K) - APPLICATIONS) 19, 23, 23
18 IF (20 (LOC) - L) 16, 16, 261
261 N=ONE
262 GO TC 16
263 IF (? Y (LOC)) 14, 262, 34
262 M=T
263 IF (CUTPAD(I,J,K) - CUTOFF(T)) 19, 23, 23
52, 56, 56
56 OPT=
57 IF (X-10) 20, 20, 21
58 N=ONE
59 GO TC 22
60 N=T C
61 X=(B(I) * (C*OBCE(R))) * ((T*INFRG) * X)
62 SUM=SUM+X
63 COST=(X * (F(R) / MIN(R))) * TONE(N)
64 GO TC 33
65 LC=LC+ONE
66 IF (X-10) 24, 24, 25
67 N=ONE
68 GO TC 26
69 N=T C
70 I=IAD2(I,J,K)
71 I2=IAD2(I,J,K)
72 REV=((FLOAT (T) / GS*SQL) / (FLOAT(I2) / GS*SQL)) *
73 TONE(I) * TONE(1)
74 K=OB(R) * (C*OBCE(R)) * ((T*INFRG) * X)
75 Y=OB(R) * (C*OBCE(R)) * ((T*INFRG) * X)
76 SUM=SUM+X
77 IF (IAD2(I,J,K)) 27, 27, 30
78 N=ONE
79 GO TC 29
80 V=W
81 X=(CILCE(R)) * (T*INFRG) * X
82 SUM=SUM+X
83 IF (SUM((7*0.1) - (7*0.1)) = (7*0.1) * (7*0.1))
84 TONE(I) * TONE(1)
85 IF (X=0) 30, 30, 11
86 IF (X=0) 30, 30, 11
87 N=SORT
88 GO TC 12
89 N=SORT
The re-evaluated unit production costs are stored on a disk pack by the following statement to replace previous estimates:

```
177 WRITE (16) (E(T), C(T), R(T), S2(T), S1(T), F(T), T=1,L)
178 END FILE 16
179 WRITE (16)
180 FORMAT (*14, 12, 11)
181 WRITE (6, 187) (T, E(T), C(T), R(T), S1(T), S2(T), F(T), T=1,L)
182 FORMAT (*14, 12, 11)
183 CALL PD
184 PNF = PNF + P
185 DO 187 K = 1, N
186 IF (TOPK (LOC)) K = 1, N
187 DO 187 L = 1, N
188 LOC = LOC + K - N + 1
189 IF (TOP (LOC)) K = 1, N
190 IF (RF (LOC)) K = 1, N
191 IF (RF (LOC)) K = 1, N
192 IF (RF (LOC)) K = 1, N
193```

Additional external data sets are created by the following four statements to store the updated variables 'YQ' and 'KQ', 'KQ+1', for the next iteration cycle. The T-6 net cash value distribution 'VAL' is stored for informational purposes only. The 'TQ' data set is stored rather than being recalculated for each solution.

```
154 WRITE (14) (TOPO(X),YR(N),VAL(N),N=1,NJLKS)
155 END FILE 14
156 PRINT (15) ((CRRES(X,N),N=1,NAXS),N=1,NAXS)
157 END FILE 15
158 GO TO (6,201)
159 STOP FORMAT ("*\14\13\12\11\10\9\8\7\6\5\4\3\2\1",1)年,13年)
160 DO 211 J=1,NJ
161 211 SURFC(N)=0
162 DO 100 K=1,L
163 WRITE (6,202) N
164 202 FORMAT ("*",1,1)*, TOPOGRAPHIC MAP OF RIME IN YEAR*1,13
165 DO 100 I=1,M
166 100 M=0
167 M=0
168 K=ONE
169 IF (K=ONE) 154,154,175
170 154 LOC=N+K*T (K=ONE) +N* (J-ONE) +1
171 IF (TOPO(LOC)) 152,152,105
172 105 IF (YR(LOC)) 175,175,175
173 170 IF (Q(LOC)-N) 152,152,175
174 152 K=ONE
175 K=ONE
176 GO TO 154
177 175 SURFC(N)=N
178 104 CONTINUE
179 WRITE (6,203) (SURFC(LEVEL),LEVEL=1,NJ)
```
After the first two iteration cycles, only one module 'M' is used to compute the remaining cycles of the iterative solution. The comments listed in this subsection describe how MAIN2 was modified to form MAIN3 for these computations.

## Comment

The mine's life span 'L' reported by the results of the preceding iteration cycle's subroutine CO2 is required to modify the input of the following statement.
The total number of fit increments 'MAXN' reported by the
preceding iteration cycle is required to modify the input of
the following 'READ' Statement:

```
030  READ (5,1) MAXN, S1(1), S2(1), D, INFR, INRP, TOMP(1),
      TCHR(2)
031  1 FORMAT (5,2), CAP(1), J=1,3), K=1,3)
032  READ (5,2), CORC(I), J=1,3), K=1,3)
033  2 FORMAT (F111/1/5F8*2)
034  REAL (5,3), MHCAP
035  3 FORMAT (FI111)
036  READ (5,12) LINC, CONC, RLC
037  12 FORMAT (5F12*1)
```

The unit production rate and selling prices evaluated by the
loop 11 are replaced by overwriting them with the new
accumulated ones read in from the preceding iteration cycle
(line 027). However, the values read in from this set to
have only been adjusted for the single-life span 't' of the
preceding iteration cycle. Therefore, the log-likelihood is
required to estimate those units for any years preceding the
previous cycle's since life 't'. If the single-life of the
current cycle exceeds that of the previous cycle this
estimate becomes redundant as the solution converges.

The following external data sets (13, 14, 15, and 16) are read into internal storage. The unit production costs and prices, excavation times, and the T-S mineral distribution from the preceding iteration cycle are used to update the solution. The remaining data sets are constant for all iteration cycles.

```
049 READ (13) ((NICEAR(I,J,K),1=1,MAX),I=1,NN),J=1,NN),K=1,NN)
050 READ (14) ((LOWINS(I),I=1,MAX),I=1,MAX)
051 READ (15) ((PRES(I,N),I=1,MAX),N=1,MAX)
052 READ (16) ((E(I),I=1,MAX),C(I),C(I),S1(I),S2(I),S1(I),S2(I),Y(I),Z(I))
053 CALL COG (AVERG,YIELD)
054 92 DO 10 T=1,L
055 LC=0
056 KC=0
057 SUXY=0
058 SUXC=0
059 SUXR=0
060 DO 34 K=1,NN
061 DO 33 J=1,NN
062 DO 34 I=1,NN
063 LOC=1*(LOC-(1-0.1)*L)+1*(1-0.13)+1
064 IF (TODO(LOC)) 34,34,14
065 IF (Y*(LOC<15,15,17
066 15 X=0.01
067 IF (Y*(LOC<18,18,24
068 16 IF (Y*(LOC<19,19,33)
069 17 IF (Y*(LOC<20,20,16)
```
070 261 X=ONE
071 GO TO 16
072 18 IF (I=VA(LG)) 34, 262, 34
073 262 X=T
074 IF (NORM((I,J,K)-CUTOFF(T))) 19, 23, 23
075 19 LC=LC+ONE
076 REV=0*0
077 IF (K=10) 20, 20, 21
078 20 Y=ONE
079 GO TO 22
080 21 N=THC
081 22 X=(OB(N)*(V*ORCI(N)))*((1+INFRC)**k)
082 SUM=SUM*X
083 COST=(X*F(X)/M(N))*TONF(N)
084 GO TO 31
085 21 LC=LC+ONE
086 MC=MC+ONE
087 IF (K=10) 24, 24, 25
088 24 Y=ONE
089 GO TO 26
090 25 N=TMC
091 26 I=ISAD1(I,J,K)
092 T=I+M12(I,J,K)
093 REV=((FLOAT(I)/GS*S1(I))+(FLOAT(J)/GS*S2(J)))*
094 TONF(N)*YIELD
095 X=(OB(N)*(V*ORCI(N)))*((1+INFRC)**k)
096 Y=(OC(N)*(V*ORCI(N)))*((1+INFRC)**k)
097 IF (I=RAD2(I,J,K)) 27, 27, 28
098 27 X=ONE
099 GO TO 29
100 29 N=THN
101 GO TO 30
102 29 Z=(YL*(N))+(1+INFRC)**K
103 SUM=SUM+Y*Z
104 IF (ANG((60*37)-(277*50)-(37*625*K)))+(37*5075)
105 GO TO 30
106 30 N=ONE
107 GO TO 32
108 31 N=THN
109 32 X=(TTC(N))+(1+INFRC)**K
110 SUM=SUM+Y
111 33 VAL(DEC)+REV-COST
112 34 CONTINUE
The external data (or V baff, only of 14, 15 and 16 (Lines 118 and 155 - 150) — used to update the solution are the same for all iteration cycles from the third one. Therefore, the obsolete data is lost when the current updating information is over-written on the same data sets. The 'VAR' data set is updated for informational purposes only. The 'TOPO' data set is stored rather than being recalculated for each iteration cycle.

```
WRITE (16) (E(T),C(T),R(T),S2(T),S1(T),F(T),T=1,L)
END FILE 16
WRITE (6,240)
240 FORMAT (11, 'AVERAGE UNIT PRODUCTION COSTS')
WRITE (6,241)
241 FORMAT (10,1X, 'E', 1X, 'C', 2X, 'T', 4X, 'F', 1X, 'VAR')
WRITE (6,242) (T, C(T), R(T), T=1,L)
242 FORMAT (11,25*2,7*)
CALL PD
DO 49 I=1,L
49       PN2=F-P30FIT(I)→PNFV
WRITE (6,222) PNFV
222 FORMAT (6, 'AVERAGE UNIT PRODUCTION COST')
WRITE (6,223) MAXV,
223 FORMAT (11, 'TOTAL REVENUE', 14, 'L=',14)
DO 39 J=1,7
39       J=J+1
DO 48 J=1,8,20
48       ONES(J,I)=I
DO 46 K=1,N
46       ONES(J,K)=J
DO 41 M=1,51
41       LOC=M+((J-ONE)*M+51)*I
42 IF (TOP0(LOC)) 44,49
44       IF (T-10) 41,43,44
43 IF (K(J(M))) 41,43,44
45 IF (K(J(M))) 41,43,44
46 DO 52 M=1,51
52       TOP0(LOC)=TOP0(LOC)+ONES(J,M)*T
48 IF (TOP0(LOC)) 44,49
49       IF (T-10) 41,43,44
```
146 43 MM=ONE
147 GO TO 45
148 44 MM=THO
149 45 IF (ECGNAD(I,J,K)-GI) 46,46,71
150 46 MM=ONE
151 GO TO 47
152 71 MM=EUCNAD(I,J,K)/GI
153 47 ORENCS(J,PITINC)=ORENC(J,PITINC)+TONF(MM)
154 48 CONTINUE
155 WRITE (14) (TOPO(N),YP(M),VAL(N),N=1,NULKS)
156 END FILE 14
157 WRITE (15) ((ORENC(Y,N),K=1,NAXG),N=1,NAXN)
158 END FILE 15
159 WRITE (6,201)
160 201 FORMAT (" ANNUAL PIT OUTLINE")
161 DO 211 N=1,NJ
162 211 SURFC(N)=.0
163 DO 100 I=1,LI
164 WRITE (6,202) I
165 202 FORMAT ("1", " TOPOGRAPHIC MAP OF MINE IN YEAR", I)
166 DO 100 J=1,LJ
167 DO 100 J=1,LI
168 I=0
169 K=ONE
170 151 IF (K-NF) 154,154,175
171 154 LOC=N*16*(K-ONE)+16*(J-ONE)+I
172 IF (TOPO(LOC)) 152,152,109
173 105 IF (YR(LOC)) 177,177,170
174 170 IF (YR(LOC)-N) 152,152,175
175 152 X=K
176 K=K+ONE
177 GO TO 151
178 175 SURFC(J)=X
179 100 CONTINUE
180 WRITE (6,203) (SURFC(LEVEL),LEVEL=1,3J)
181 100 CONTINUE
182 203 FORMAT ("1",3013)
183 99 STOP
184 DEBUG SURTRACE,SUBCK
185 END
A verbal description of subroutine COG1's computational sequence follows in Subsection 8.4.1. Then, it is listed in the next Subsection. The last Subsection contains a list of COG2 with comments to explain how it differs from COG1.

8.4.1 COMPUTATIONAL SEQUENCE OF COG1

LINES 001 - 008
Specification and input statements:

LINES 009 - 052
Ratios of ore : pit material 'R1', refined product : pit material 'R2' and refined product : ore 'R3' are calculated for each grade increment 'M' and pit increment 'N' of 'ORES'.

LINES 053 - 056
Subroutine 'COGSUB' is called to compute the following information for the combination of plant capacities (MINCAP, CONCAP, REFSCAP, etc) named by (i,j,k):

A) the annual cut-off grades 'CUTOFF';
B) the annual production targets 'MIN', 'COM' and 'QREF';
C) the overall recovery efficiency 'YIELD'.

It is not necessary to compute the plant combination yielding the maximum present value for the rough computations of the first iteration cycle. Therefore, any plant combination (i,j,k) is selected. The plant combination yielding the highest present value is selected later by replacing this subroutine with subroutine 'COG2' in subsequent iteration cycles.

LINES 057 - 060
These statements execute printer output used to monitor the
solution as described and illustrated in Section 7.2.

LINE 070 - 071

An average 'Yield' of the annual cut-off grade 'CUTOFF' is computed.

8 4 2 A LIST OF COG1

001 SUBROUTINE COG (AVG, YIELD)
002 DOUBLE PRECISION MAT(20), OBJRES(19, 20),
003 WASTE(d4, 20), DRE(d6, 20), NDF(20), ODFAT, PROD(d9, 20),
004 P11(4, 20), P12(4, 20), P13(6, 20), P14, P15, P16, P17(3, 3, 1),
005 P18, P19, P20, P21, P22, P23, P24, P25, P26, P27, P28, P29, P30,
006 P31(1), P32(1), P33(1), P34(1), P35(1), P36(1), P37(1), P38(1), P39(1),
007 P40(1), P41, P42, P43, P44, P45, P46, P47, P48, P49, P50, P51, P52, P53,
008 P54, P55, P56, P57, P58, P59, P60, P61, P62, P63, P64, P65, P66,
009 R(20), S(3, 1), T(3, 1), U(3, 1), V(3, 1), W(3, 1), X(3, 1), Y(3, 1), Z(3, 1),
010 L0 = 0

'GI' is given by the user and is equal to 'GI' in Line 011 of 'MAIN1'.

008 GI = 5
009 DO 151 X = 1, 20
010 MAT(X) = 0
011 REN(X) = 0.0
012 DO 151 X = 1, 60
013 PDIC(Y, 1) = 1.0
014 WASTE(X, 1) = 0.0
015 ORG(X, Z) = 0.0
016 R11" = 0.0
017 R2" = 0.0
018 R3" = 0.0
019 351 CONTINUE
020 DO 160 X = 1, MAXG
021 DO 160 X = 1, MAXG
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```
022 308 MAT (V) = MAT (V) * ORESES (K, M)
023  DO 309 N=1,K AXN
024  WASTE (1, M) = ORESES (1, M)
025  IF (MAT (N) - K ACT (1, M)) /= 2,2,3
026  ORE (1, M) = ORESES (1, M)
027  GO TO 4
028  ORE (1, M) = MAT (V) - WASTE (1, M)
029  DO 309 N=2,K AXN
030  WASTE (N, M) = WASTE (N-1, M) * ORESES (K, M)
031  ORE (N, M) = MAT (V) - WASTE (K, M)
032  DO 310 N=1,K AXN
033  REF (N) = ORESES (1, N) * (GL/GS)
034  DO 310 N=1,K AXN
035  REF (N) = REF (N) * (DFLOAT (N+1) /GS) * ORESES (K, M)
036  DO 311 N=1,K AXN
037  PPRO (1, K) = REF (N)
038  DO 311 N=1,K AXN
039  IF (CRE (N, N)) = 111,111,1
040  1 PROD (N, V) = PROD (N-1, V) - (DFLOAT (M+1) /GS) * ORESES (K, M)
041  CONTINUE
042  DO 311 N=1,K AXN
043  DO 312 N=1,K AXN
044  R1 (N, V) = ORE (N, V) / MAT (V)
045  DO 312 N=1,K AXN
046  R2 (N, V) = PROD (N, V) / MAT (V)
047  DO 313 N=1,K AXN
048  IF (CRE (N, N)) = 151,151,151
049  354 R3 (N, V) = 0
050  GO TO 153
051  355 R3 (N, V) = PROD (N, V) / ORE (V, V)
052  CONTINUE
053  I=ONE
054  J=ONE
055  K=ONE
056  CALL COSUBS (I,J,K,P)
057  WRITE (L,6)
058  6 FORMAT ('11,'DISCOUNTED PROFITS FROM THE PRODUCTION
059  TARGETS OF EACH PLANT COMBINATION'
060  WRITE (L,5) ((FY (1,4,N), N=1,4), J=1,3), K=1,4)
061  5 FORMAT (11,13,1)
062  YIELD=SEC (K)
063  WRITE (L,7)
064  7 FORMAT ('X-COSTS & YIELD U-SHAPED U-FATIGUE FOR THE PLANT
065  CONFIGURATION YIELDING THE GREATEST VALUES OF PRESENT
066  VALUE (OSA CAPITAL COST)'
067  WRITE (L,8) (10,1,3), (3,4,1), (2,1,4)
```

065 8 FORMAT ('D', 'PRESENT VALUE=', F13.1, 'CONCENTRATOR CA-
1 PACITY=', F11.1, 'SOLVENT CAPACITY=', F9.1, 'OVERALL R
2 COPY EFFICIENCY=', F5.2)
066 WRITE (6, 7)
067 9 FORMAT ('D', 'YR COG', 'R', 'MIN', 'BX', 'COG', 'BX', 'REF',
1 'PROFIT')
068 WRITE (6, 10) (N, CUTOFF (Y), MIN (Y), COG (N), REF (Y),
1 (N), N, 1, L)
069 10 FORMAT ('D', 15, 31.10*1, F13.1)
070 SUM = 0
071 DO 151 N = 1, L
072 151 SUM = SUM + CUTOFF (N)
073 AVERAGE = SUM / L
074 RETURN
075 DEBUG SUBTRACE, SUBCHK
076 END
08. 4 3 A LIST OF COG2

001 SUBROUTINE COG (AVER, YIELD)
002 COMMON PRECISION MAT (20), DURES (39, 20),
003 WASTE (65, 20), D (65, 20), WC (20), MIG. 1, PROD (65, 20),
004 2 (1, 2), 1 (65, 20), 83 (65, 20), PYREX, P7 (3, 1, 1),
005 ICAP (1, 1, 1), 2 (3), 3 (3), 94, 05 (39), MIR (39), COG (39),
006 PROF (39), P (39)
007 INTEGER CUTOFF (39), OIL/1/SUM1, SUM, GI
008 INTEGER, 2 AVER, 13 (1500), 14 (1500), 15 (1500),
009 1037 (10), 13500 (13500), 1240 (13500),
010 DIMENSION REC (3), 16 (2), 17 (2), 24 (2), 19 (2),
011 122 (9), 222 (9), 24 (95), 224 (9), 25 (2), 36 (94),
012 PROF (97), 64 (97), 13 (97), 1500 (97)
013 DO 151 M = 1, 20
014 MAT (M) = 0
015 DO 151 M = 1, 16
016 YIELD (M) = 0
017 YIELD (M) = 0
selected subroutine

The annual cut-off values 'CUTOFF' and production targets 'PRED' are calculated for all plant combinations (L,M,F) rather than just the single trial combination selected in subroutine 'COST1'.
The plant combination yielding the greatest present value of the profits that would result from its production targets less its capital cost 'CAP' is chosen to exploit the given deposit between Lines 061 and 070.

The cut-off grades 'CUTOFF' and production targets 'MIN', 'CON', and 'REF' are recalculated (Line 071) for the plant combination yielding the maximum present value.

```plaintext
071 CALL COSUB (I, J, K)
072 YIELD=SEC(K)
073 WRITE (6,7)
076 IF (YIELD > CUTOFF) READ (13, 1) L, M, N, T, I, J, K
077 WRITE (5,1)  
078 WRITE (4,1)  
079 WRITE (3,1)  
080 IF (L > I) IF (L > J) IF (L > K)
```

The plant combination yielding the greatest present value of the profits that would result from its production targets less its capital cost 'CAP' is chosen to exploit the given deposit between Lines 061 and 070.

The cut-off grades 'CUTOFF' and production targets 'MIN', 'CON', and 'REF' are recalculated (Line 071) for the plant combination yielding the maximum present value.

```plaintext
061 PVMAX=PV(1,1,1)-CAP(1,1,1)
062 DO 162 LL=1,3
063 DO 162 M=1,3
064 DO 162 N=1,3
065 IF (PVMAX<PV(LL, M, N)*CAP(LL, M, N)) L=1,162,162
066 PVMAX=PV(LL, M, N)-CAP(LL, M, N)
067 J=LL
068 K=M
069 R=N
070 162 CONTINUE
```

The cut-off grades 'CUTOFF' and production targets 'MIN', 'CON', and 'REF' are recalculated (Line 071) for the plant combination yielding the maximum present value.

```plaintext
071 CALL COSUB (I, J, K)
072 YIELD=SEC(K)
073 WRITE (6,7)
076 IF (YIELD > CUTOFF) READ (13, 1) L, M, N, T, I, J, K
077 WRITE (5,1)  
078 WRITE (4,1)  
079 WRITE (3,1)  
080 IF (L > I) IF (L > J) IF (L > K)
```

The plant combination yielding the greatest present value of the profits that would result from its production targets less its capital cost 'CAP' is chosen to exploit the given deposit between Lines 061 and 070.

The cut-off grades 'CUTOFF' and production targets 'MIN', 'CON', and 'REF' are recalculated (Line 071) for the plant combination yielding the maximum present value.

```plaintext
061 PVMAX=PV(1,1,1)-CAP(1,1,1)
062 DO 162 LL=1,3
063 DO 162 M=1,3
064 DO 162 N=1,3
065 IF (PVMAX<PV(LL, M, N)*CAP(LL, M, N)) L=1,162,162
066 PVMAX=PV(LL, M, N)-CAP(LL, M, N)
067 J=LL
068 K=M
069 R=N
070 162 CONTINUE
```
8. 5  SUBRCUTIN2 CGSUB

8. 5* 1 COMPUTATIONAL SEQUENCE OF CGSUB

LINES 001 - 007

Specification and input statement.

LINES 008 - 027

Computation of the balancing cut-off grades 'GMC', 'GBX', and 'GRC'.

LINES 030 - 234

An implicit loop in which the optimum cut-off grades 'CUTOFF' and production targets 'X1', 'CON' and 'NAP' are computed. This is a trial and error computation because the present value 'V' is unknown in the economic cut-off grade expressions (LINES 044 - 046). These computations converge on 'V'.

The implicit loop is divided into two parts, LINES 047 - 111 and LINES 014 - 237, in both of which many of the computations are duplicated for two different situations. The first part computes the cut-off grades and targets for annual production coming entirely from the material within a single pit increment 'N' of 'OPPER'. The second part performs the same computations for annual production including two pit increments.

LINES 044 - 046

The economic cut-off grades 'GM', 'GC' and 'G3' are computed for the year 'L'.

080 10 FORMAT (13,15,3F10*1,F13*1)
081 SUM=0
082 DO 165 K=1,L
083 165 SUM=SUM+CIUTOFF(K)
084 AVE3=SUM/L
085 RETURN
086 DEBUG SUBTRACE,SUBCHK
087 END
080 10  FORMAT (13,I5,3F10*1,F13*1)
081      SUM=0
082      DO 165 K=1,L
083  165  SUM=SUM+SUM(K)
084      AVE3=SUM/L
085      RETURN
086      DEBUG SUBTRACE, SUBCHK
087      END

8* 5 SUBROUTINE COGSUB

8* 5* 1 COMPUTATIONAL SEQUENCE OF COGSUB

LINES 001 - 007

Specification and input statements*

LINES 008 - 029

Computation of the balancing cut-off grades 'GMC', 'G3Y' and 'GRC'.

LINES 030 - 234

An implicit loop in which the optimum cut-off grades 'CUTOFF' and production targets 'GMC', 'G3Y' and 'GRC' are computed. This is a trial and error computation because the present value 'Y' is unknown in the economic cut-off grade expressions (LINES 044 - 046). These computations converge on 'Y'.

The implicit loop is divided into two parts, LINES 047 - 113 and LINES 014 - 207, in both of which many of the computations are duplicated for two different situations. The first part computes the cut-off grades and targets for annual production coming entirely from the material within a single pit increment 'N' of 'G3Y'. The second part performs the same computations for annual production attending two pit increments.

LINES 044 - 046

The economic cut-off grades 'G3Y', 'G3C' and 'G3Z' are computed for the year 'L'.

---

The above text is a part of a computational sequence for determining certain economic cut-off grades and production targets. The sequence involves an implicit loop that is divided into two parts for different scenarios. The loop is designed to handle cases where the present value 'Y' is unknown, requiring a trial and error approach to converge on the correct values.
The optimum cut-off grade 'G' is chosen from the balancing cut-off grades (GNC, GRK, GRC) and the economic cut-off grades (GX, JC, JG) for the year 'L'.

The plant production targets 'QM', 'QC' and 'QR' are computed for the year 'L'.

The year 'L' and pit increment 'N' are controlled with these statements by comparing the cumulative quantity of materials 'SUMX' produced from the current pit increment 'N' to the total material 'PAT' contained in that pit increment.

The production targets (QM, QC, QR) are adjusted for the first part of production year 'L' that straddles two pit increments 'N'.

The plant capacities (CONCAP, SUMCAP, MAXCAP) are scaled down to compute the cut-off grades and production targets for the remaining fractional part of the production year 'L' from the next pit increment 'N'.

The cut-off grades 'GK' and production targets 'QK', 'QC', and 'QR' are calculated for the second part of production year 'L' that straddles two pit increments 'N'. These computations are executed in a similar manner to those between Lines 047 and 111.

The year 'L' and pit increment 'N' are controlled with these statements.
The subscript 'i' is deleted from the cut-off grade 'G' and production targets 'E(i)', 'G(i)' and 'Q(i)'. These data sets (CUTOFF, E(i), G(i), Q(i)) are supplied to the subroutine through the COMMON statement.

LINES 225 - 230

The annual profit 'i' and its present value 'SU.PV' are computed.

LINE 231

The trial and error computations for the cut-off grades and production targets are controlled with this statement by testing the convergence of the present value 'SU.PV' in successive cycles. If the computations have not converged, the subroutine returns for another trial; otherwise it exits.

8° 5° 2 A LIST OF COGSUB

001 SUBROUTINE COGSUB (I,J,K,P)
002 DOUBLE PRECISION CONCAP(3), EPECAP(3), XCNCAP, V(J),
1 SU(22), R(65,20), R2(65,20), R3(63,2), P(99),
2 F(99,20), F(99,20), F2(99,20), F(99), N(20),
3 E(93,20), E(91,19), E(91,19), E(91), E(91),
4 SU(22), P(99,1,3), S1, S2, S3, U1, U2, U3, US, US2, PROD.
002 DOUBLE PRECISION H(2), FAH, DFLCAK, T(99,20),
1215 (3,3,3),
003 IN2, IN3, IN4, IN5, IN6,
004 COMMON 21, CONCAP, EPECAP, XCNCAP, F, MAT, OSE, PROD, N1,
132, P7, W(10), CON, LEPF, L(15,4,16,17,18,19,20,20,20,20),
3 O, S2, S3, S4, S5, S6, S7, S8, S9, S10, S11, S12, S13, S14, S15, S16, S17, S18, S19, S20, S21, S22, S23, S24, S25, S26, S27, S28, S29, S30, S31, S32, S33, S34, S35, S36, S37, S38, S39, S40

'GI' is given by the user and is equal to 'GI' in Line 911 of 'MANS'.
007 088 GI=5
009 DO 160 L1=1,99
010 160 V(L1)=0*0
010 900 S1=CCAP(I)/XCAP
011 S2=FFCAP(J)/XCAP
012 S3=FFCAP(J)/CONCAP(I)
013 DO 6 N=1,99
014 SUBCOM(N)=0
015 GSC(N)=0
016 DO 2 *1,99
017 IF (ABS(R1(M,N)-S1)*G) 1,1,2
018 1 GSC(N)=Y*G1
019 2 CONTINUE
020 GSC(N)=0
021 DO 4 *1,99
022 IF (ABS(R2(M,N)-S2)*G) 1,3,4
023 3 GSC(N)=Y*G1
024 4 CONTINUE
025 GSC(N)=0
026 DO 5 *1,99
027 IF (ABS(S1(M,N)-S3)*G) 5,5,6
028 5 GSC(N)=Y*G1
029 6 CONTINUE
030 52 DO 151 L1=1,99
031 CUTOFF(L1)=0
032 P(L1)=0*0
033 DO 151 *1,20
034 M(L1,G)=0*0
035 GC(L1,R)=0*0
036 UP(L1,R)=0*0
037 G(L1,N)=0
038 151 CONTINUE
039 L=G
040 N=0
041 999 N=N+1
042 SUBCOM(N)=0
043 10 L=L+1
044 B(L)=C(L)/[1.0-C(L)-D(L)*FCAP(I)]*2
045 G(L)={1.0-D(L)+[2.0*G(L)]/CONCAP(I)}/[1.0-D(L)
046 V(L)={3.0*G(L)}*G5
047 IF (S(L)-GSC(N)) 12,11,11
048 11 GSC1(L)=G(L)
049 12 G0 TO 70
096      GO TO 117
097  115  OREG=OGE(K,N)+((DFLOAT(K*G1)-DFLOAT(G(L,N)))/G1)
098  1* (OGE(W-1,N)-OGE(K,N))
099  117  T2=DFRCT/RECFAE(L)
100      T3=DFRCT/RECFCAP(L)
101      IF (T1-T2) 501,502,502
102      501 IF (T2-T3) 506,503,505
103      502 IF (T1-T3) 506,504,504
104      504 T=T1
105      GO TO 74
106      505 T=T2
107      GO TO 74
108      506 T=T3
109      74 X(L,N)=XAT(X)
110      G(L,N)=GTA/L
111      V(L,N)=APV/L
112      SUMX_(X)=SUMX(L)*QX(L,N)
113      IF (XAT(X)-SUNX(L)) 23,996,16
114      23 QM2=UM(L,N) + (XAT(X)-SUNX(L))
115      CC(L,'Y') = (X*2/CC(L,N) + CC(L,'I'))
116      OP('X',X') = I2X2/CC(L,N)
117      OM(L,N) = X2
118      2*KINCAP-X(L,N)
119      C2=CCVPAP(I) - L(L,N)
120      P2=RECFCAP(I) - R(L,N)
121      U1=C2/E2
122      U2=P2/E2
123      U3=P2/C2
124      N=1
125      IF (MAX-M) 43,130,110
126      130 GXC2(X) = 0
127      43 U =1,MAX
128      IF (MAX(MAX-1) -U1) = 02) 26,24,25
129      24 GXC2(X) = MAX
130      25 CONTINUE
131      26 GXC2(X) = 0
132      27 CONTINUE
133      28 GXC2(X) = X2
134      29 CONTINUE
135      30 GXC2(X) = 0
136      31 CONTINUE
137      32 IF (N=1.5377
138      33 IF (MAX(X) - 03) = 00019) 25,27,27
139      26 GXC2(X) = X2
140      27 CONTINUE
141      28 GXC2(X) = X2
GO TO 217

IF (G(L,1) - GI) 218, 214, 216

104 ORG = CREF (1, 1)

GO TO 217

215 ORG = ORG (X, Y) * (((DFLOAT (X+GI) - DFLOAT (G(L, 3)))/GI))

216 IF (G(L, 3) - GI) 214, 217

217 T2 = T2/G

218 IF (T2 = T1) 210, 201

219 IF (T2 = T3) 210, 201

202 T2 = T2/G

203 N = (N/L) = MAX (N)/T

204 CC (N, N) = OP = CC/T

205 SUM (N) = SUM (L, N)

206 IF (N = MAX (N)) 23, 99, 10

208 906 IF (MAX (N) - 43, 43, 400

209 43 DO 45 X = 1, L

210 MIN (X) = 0

211 CONTINUE

212 DO 99 Y = 0

213 SUM = 0

214 IF (X = 1, Y) 23, 44

215 SUM = SUM + V(X)

216 MIN (Y) = MIN (Y) + W (Y)

217 CONTINUE

218 MAX = MAX (Y) + W (Y)

219 MIN (X) = MIN (X) + W (X)

220 CONTINUE

221 CONTINUE

222 CONTINUE

223 CONTINUE

224 CONTINUE

225 DO 46 X = 1, 4

226 46 IF (X = 1, 2) 227, 228

227 DO 47 X = 1, L

228 SUM (X) = 0
DO 47 N = N, L
230 47 SUMV(N) = SUMV(N) + F(Y)/(1*D)**(Y-M+1)
231 IF (CAUS(SUMV(Y) - V(Y)) - 1**000000.0) 53, 48, 43
232 48 DO 51 I = 1, L
233 51 V(I) = SUMV(I)
234 GO TO 52
235 53 IF (L, J, K) = Y (1)
236 RETURN
237 DEBugging SUBTRACT
238 END

8. 6 SUBROUTINE PD

8. 6 1 COMPUTATIONAL SEQUENCE OF PD

LINES 001 - 006

S pecification and input statements.

LINES 067 - 067

Design of the final pit surface.

LINE 013

The grid-blocks (X, Y, Z) with a positive net cash value 'VAL' are detected, in turn, to form the base of the semi-cone closures of the barchan algorithm (6).

LINES 020 - 057

The grid-blocks (X, Y, Z) included to expand the base grid-block (X, Y, Z) are listed by storing their location in the array 'LIE'. Any grid-blocks (X, Y, Z) included in the pit 'IP' or a previous test are not included.

LINES 058 - 067

The net cash values 'VAL' of the grid-blocks within the semi-cone closure 'LIE' are totaled. If this total 'SUM' is positive, the semi-cone closure 'LIE' is included in the pit 'IP'; otherwise, it is neglected.

LINES 068 - 088

The grid-blocks (X, Y, Z) with a positive net cash value 'VAL' are detected, in turn, to form the base of the semi-cone closures of the barchan algorithm (6).

LINES 020 - 057

The grid-blocks (X, Y, Z) included to expand the base grid-block (X, Y, Z) are listed by storing their location in the array 'LIE'. Any grid-blocks (X, Y, Z) included in the pit 'IP' or a previous test are not included.

LINES 058 - 067

The net cash values 'VAL' of the grid-blocks within the semi-cone closure 'LIE' are totaled. If this total 'SUM' is positive, the semi-cone closure 'LIE' is included in the pit 'IP'; otherwise, it is neglected.

LINES 068 - 088
A topographic map of the final pit surface is printed by these statements.

LINES 087 - 136

An implicit loop which simulates ore production from the pit.

LINES 138 & 139

Subroutine SUBRO is called to simulate the pit's waste production.

LINES 137 - 169

Schedules for the annual revenue, costs, ore production and waste production are computed and printed. Additionally the excavation times for ore 'IOR' and waste 'IFW' are combined into one array 'ITP'.

8. 6. 2 A LIST OF PDS

001 SUBROUTINE PROD
002   DO 2, N=1, NPS
003    D 1.7, 5.5, 1.9, 1.8
004    D 2.0, 1.7, 5.5, 1.9
005    D 1.7, 5.5, 1.8, 1.9
006    D 2.0, 1.7, 5.5, 1.8
007    D 1.7, 5.5, 1.8, 1.9
008    D 2.0, 1.7, 5.5, 1.8
009    D 1.7, 5.5, 1.8, 1.9
010    D 2.0, 1.7, 5.5, 1.8
011    D 1.7, 5.5, 1.8, 1.9
012    D 2.0, 1.7, 5.5, 1.8
013    D 1.7, 5.5, 1.8, 1.9
014    D 2.0, 1.7, 5.5, 1.8

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The constants (X,Y,Z) of the same expression in line 51 are given by the grid coordinates of the last gold-block (X,Y,Z). However, the constant 'Z' must correspond to two grid safe well-angles. For example, the given deposit (see Section 7.1) is divided into two zones by a fault (Line 87), which has a different safe well-angle 'HTAB'. The constant 'Z' is adjusted to construct the overlapping zones required for the casing. The following statements (Lines 228 - 236 and 291 - 299), along with Fig. 9-1, demonstrate how this is accomplished for the example deposit. These statements must be referred to the user's requirements.

```plaintext
020  7 PAULT = (X+1)+ (X+9)- (173+65)-(177+75)-(127+85)
021    =80
022  DO 20 X=1,90
023  DO 20 Y=1,90
024  DO 20 Z=1,90
025  LOC=488+(2-015)-878-(7-085)+7
026  IF (488+085) 20,20,2
```
A. A HANG-UP BLOCK ABOVE THE FAULT

B. A HANG-UP BLOCK BELOW THE FAULT

FIG. 8.4. CROSS-SECTION OF OVERLAPPING BLOCKS USED AS closure
044 105 D=Z
045 106 GO TO 108
046 107 XOC=PICTANG (2)
047 108 IF (D) 14, 14, 108
048 109 D=Z-C0
050 107 IF (D) 14, 14, 108
051 108 CONE=SOLVE (IS) * (K-D) * (K-D) - ((2*J-2*Y) * * 2) - ((I-X)
1*2)
052 12 IF (CONT) 14, 12, 12
053 13 IF (IP()) 13, 13, 14
054 14 ID=19+085
055 15 ICL(IS)=5
056 16 CONTINUE
057 17 DO 15 M=1,1
058 18 IP (ICL (I)) = ICL (N)
059 19 DO 13 X=1, 1
060 20 DO 19 Y=1, 1
061 21 WRITE (6,99)
062 99 FORMAT ('1', ' OPTIMUM PIT SURFACE')
063 22 DO 220 X=1, 1
064 23 DO 20 Y=1, 1
065 24 WRITE (6,99)
066 99 FORMAT ('1', ' OPTIMUM PIT SURFACE')
067 25 DO 220 X=1, 1
068 26 DO 20 Y=1, 1
069 27 WRITE (6,99)
070 28 WRITE (6,99)
071 29 WRITE (6,99)
072 30 WRITE (6,99)
073 31 WRITE (6,99)
074 32 WRITE (6,99)
075 33 WRITE (6,99)
076 34 WRITE (6,99)
077 35 WRITE (6,99)
078 36 WRITE (6,99)
079 37 WRITE (6,99)
080 38 WRITE (6,99)
081 39 WRITE (6,99)
082 40 WRITE (6,99)
083 41 WRITE (6,99)
084 42 WRITE (6,99)
085 43 WRITE (6,99)
086 44 WRITE (6,99)
The following one excavation sequence (Lines 088 - 114) illustrates a simulation for one working face that advances in reorganized directions or alternate benches parallel to the 'X' co-ordinate axis. This simulation must be tailored to the user's requirements, as described in Subsection 6.7.1.

```
087 26 L=ONE
088  J=0
089  I=0
090  K=0
091  SOR=0
092  SWASTE=0
093  31 ISIGN=ONE
094  GO TC 33
095  32 ISIGN=-ONE
096  33 K=F+ONE
097  IF (K=11) 34, 35, 160
098  34 X=K-((L/2)*2)+ONE
099  GO TC (30, 36, 8)
100  35 B=-ONE
101  GO TC 47
102  36 D=ONE
103  37 J=J+1
104  IF (J-N1) 38, 39, 32
105  38 IF (J-ONE) 31, 39, 39
106  39 H=J-((J/2)*2)+ONE
107  GO TC (40, 41, 8)
108  40 A=(-CY2)*ISIGN
109  GO TC 42
110  41 A=ISIGN
111  42 A=ONE
112  IF (I-ONE) 43, 43, 37
113  43 IF (I-ONE) 37, 44, 44
114  44 TC=7.5*e^((P-O/F)+10*(J-ONE)+1
115  IF (FOR(ONE) 13J, 11C, 42
116  110 IF (K=(N)) 42, 43, 110
117  111 IF (V(L)(C)) 42, 42, 45
118  45 IF (K-1) 46, 46, 49
119  46 ==ONE
120  GO TC 48
121  47 K=2
122  96 (OCT)+20*F(X)
123  IF (C=0)(-800F) 30, 50, 50
124  48 TC=ONE
125  50 TC 52
```
126 50 IF (SCEF-0.0E(L)- (TCMF(M)/2.)) 51,51,60
127 51 IOR(I)=L
128 CALL SUBPD (SCEF,L,M)
129 SCEF=0
130 L=L+1
131 GO TO 42
132 60 CALL SUBPD (SCEF,K,Y)
133 L=L+CPE
134 SCEF=TONF(K)
135 IOR(I)=L
136 GO TO 42
137 160 DO 69 K=1,99
138 PROFIT(I)=0.0
139 CAPOD(I)=0.0
140 OPE(I)=0.0
141 69 WASTE(I)=0.0
142 DO 251 T=1,48
143 IF (T-10) 71,71,72
144 71 IF =OKE
145 GO TO 71
146 72 IF =TWO
147 73 DO 76 Y=1,51
148 76 CONTINUE
149 M=M*1.0E(L)*RI4(Y-OKE)+X
150 IF (TCPO(M)) 86,86,260
151 260 IF (IIP(M)) 166,166,166
152 180 IF (200(M)) 181,181,181
153 171 IF (100(M)) 172,172,172
154 152 IMA(N(K))=L
155 34 YP(T)=1.0E(K)
156 CAPOD(IMA(N))-CAPOD(IMA(N))+VAL(N)
157 WASTE(IMA(N))=WASTE(IMA(N))+TONF(N)
158 GO TO 35
159 53 YP(T)=100(K)
160 PROFIT(IOR(I))=PROFIT(IOR(I))+VAL(M)
161 OPE(IOR(I))=OPE(IOR(M))+TONF(I)
162 26 CONTINUE
163 251 CONTINUE
164 WRITE (6,250)
165 250 FORMAT (*11, SIMULATED PIT PRODUCTION SCHEDULE*)
166 WRITE (6,251)
167 251 FORMAT (*11, ' IF OVERHEAD STRIPPING COSTS OVER
110
168 WRITE (6,252)
169 252 FORMAT (13,41,110)
The grid-blocks required to expose the ore production of year 'L' are identified with the semi-cone set (Lines 024 - 061) and labelled by storing their location in the array 'IMA'.

The appropriate minimum working wall-angle 'WSLOPE' is selected.

DC loop 56 increases the working wall-angle 'WSLOPE' is increased between its minimum 'WSLOPE' (Line 16) and its maximum 'WSLOPE' (Line 17) until the waste closure approximates the waste production target 'MAT - ORIG'. Then, the accuracy of the working wall-angle 'WSLOPE' is increased by setting the increment 'STEP' and repeating the computations within the realistic tolerance. The computations are terminated when the overburden within the waste closure equals the waste production target. The termination can also occur if the working wall-angle 'WSLOPE' reaches the maximum or minimum limit without satisfying the waste production target. The minimum working wall-angle 'SLOPE-MIN' in Line 15 is a practical upper limit imposed to prevent the computations from going to infinity.

Lines 024 - 061

DO loop 56 identifies the active grid-blocks 'IC' that form
140 29 CONTINUE
141 30 IF (SM(L) - GMC2(X)) 31, 30, 30
142 31 GMC1(L) = IY(L)
143 32 GO TO 75
144 33 IF (SC(L) - GMC2(X)) 32, 32, 33
145 34 GMC1(L) = SC(L)
146 35 GO TO 75
147 36 GMC1(L) = GMC2(X)
148 37 IF (GM(L) - GMC2(X)) 35, 35, 34
149 38 GMC1(L) = GM(L)
150 39 GO TO 76
151 40 IF (SH(L) - GMC2(X)) 36, 36, 37
152 41 GMC1(L) = SH(L)
153 42 GO TO 76
154 43 GMC1(L) = GMC2(X)
155 44 IF (GJ(L) - GMC2(X)) 38, 38, 36
156 45 GMC1(L) = GJ(L)
157 46 GO TO 77
158 47 IF (SC(L) - GMC2(X)) 40, 40, 41
159 48 GMC1(L) = SC(L)
160 49 GO TO 77
161 50 GMC1(L) = GMC2(X)
162 51 IF (GMC1(L) - GMC1(L)) 202, 205, 201
163 52 IF (GMC1(L) - GMC1(L)) 204, 205, 203
164 53 IF (GMC1(L) - GMC1(L)) 205, 203, 202
165 54 IF (GMC1(L) - GMC1(L)) 205, 203, 202
166 55 IF (GMC1(L) - GMC1(L)) 206, 207, 204
167 56 GMC1(L) = GMC1(L)
168 57 GO TO 70
169 58 GMC1(L) = GMC1(L)
170 59 GO TO 70
171 60 GMC1(L) = GMC1(L)
172 61 T1 = SAT(V) / E2
173 62 IF (G(L,X)) 555, 552, 155
174 63 ORGL = ORG(1, 1)
175 64 PPGO = OPD(1, 1)
176 65 GO TO 217
177 66 IF (GP(X,Y) = GMC1(L)) 212, 214, 216
178 67 CONTINUE
179 68 GO TO 7
180 69 CONTINUE
181 70 OPCO = 0
182 71 OPCO = 0
183 72 GO TO 217
184 73 OPCO = 0
185 74 OPCO = OPCO(1, 1) / GI, 1
186 75 CONTINUE
the zone of the semi-cone set. Do loops 19, 22 and 9 identify the grid-blocks that satisfy the semi-cone closure.

LINE 052

Note that any other grid-blocks are excavated as well, if they satisfy the semi-cone set.

8.7 2 A LIST OF SUBLIST

001 SUBROUTINE SUBLIST (SORE, M, K)
002 COUPLE PRECISION SOR$, SOR$, SOR$, G1 (2, 20), G2 (3, 3),
003 G3 (2, 20), G4 (2, 20), G5 (2, 20), G6 (2, 20), G7 (2, 20),
004 G8 (2, 20), G9 (2, 20), G10 (2, 20), G11 (2, 20), G12 (2, 20),
005 G13 (2, 20), G14 (2, 20), G15 (2, 20), G16 (2, 20), G17 (2, 20),
006 G18 (2, 20), G19 (2, 20), G20 (2, 20), G21 (2, 20), G22 (2, 20),
007 G23 (2, 20), G24 (2, 20), G25 (2, 20), G26 (2, 20), G27 (2, 20),
008 G28 (2, 20), G29 (2, 20), G30 (2, 20), G31 (2, 20), G32 (2, 20),
009 G33 (2, 20), G34 (2, 20), G35 (2, 20), G36 (2, 20), G37 (2, 20),
010 G38 (2, 20), G39 (2, 20), G40 (2, 20), G41 (2, 20), G42 (2, 20),
011 G43 (2, 20), G44 (2, 20), G45 (2, 20), G46 (2, 20), G47 (2, 20),
012 G48 (2, 20), G49 (2, 20), G50 (2, 20), G51 (2, 20), G52 (2, 20),
013 G53 (2, 20), G54 (2, 20), G55 (2, 20), G56 (2, 20), G57 (2, 20),
014 G58 (2, 20), G59 (2, 20), G60 (2, 20), G61 (2, 20), G62 (2, 20),
015 G63 (2, 20), G64 (2, 20), G65 (2, 20), G66 (2, 20), G67 (2, 20),
016 G68 (2, 20), G69 (2, 20), G70 (2, 20), G71 (2, 20), G72 (2, 20),
017 G73 (2, 20), G74 (2, 20), G75 (2, 20), G76 (2, 20), G77 (2, 20),
018 G78 (2, 20), G79 (2, 20), G80 (2, 20), G81 (2, 20), G82 (2, 20),
019 G83 (2, 20), G84 (2, 20), G85 (2, 20), G86 (2, 20), G87 (2, 20),
020 G88 (2, 20), G89 (2, 20), G90 (2, 20), G91 (2, 20), G92 (2, 20),
021 G93 (2, 20), G94 (2, 20), G95 (2, 20), G96 (2, 20), G97 (2, 20),
022 G98 (2, 20), G99 (2, 20), G100 (2, 20), G101 (2, 20), G102 (2, 20),
023 G103 (2, 20), G104 (2, 20), G105 (2, 20), G106 (2, 20), G107 (2, 20),
024 G108 (2, 20), G109 (2, 20), G110 (2, 20), G111 (2, 20), G112 (2, 20),
025 G113 (2, 20), G114 (2, 20), G115 (2, 20), G116 (2, 20), G117 (2, 20),
026 G118 (2, 20), G119 (2, 20), G120 (2, 20), G121 (2, 20), G122 (2, 20),
027 ...
The constants \((X,Y,D)\) of the con expression in Line 47 are given by the grid co-ordinates of the base grid-blocks \((X,Y,D)\).* However, the constant 'D' must respond to variable working wall-angles. For example, the given deposit (see Section 7.1) is divided into two zones by the tenth grid-level (Line 149), either side of which has a different minimum working wall-angle \(\theta_{min}\). The constant 'D' is adjusted to construct the overlapping concept used for the closure. The following statements (Lines 679 - 681), along with Fig. 49, demonstrate how this is accomplished for the example deposits. These statements must be followed to the user's requirements.

* Here the \(X,Y,D\) are the grid co-ordinates of the deposit at the base grid-block.
The format of this glossary follows the same pattern as those of Chapters 6 and 7, wherever a variable name appears twice in the glossary, it is in the result of using the same name for two different applications in the EN program. The brackets which follow their definitions list the subroutines in which the given definition is applicable. Note also that any variables appearing in the specification statements of the form: \texttt{FO}, where \texttt{FO} is an identifying number, are only necessary to catch the \texttt{COMMON} statements. For this reason, they do not appear in the glossary. The glossary is not complete; it does not contain the variables used to facilitate programming.

\textbf{AVG} \quad (I+2) \text{ the average cut-off grade over the pit's lifespan}.

\textbf{C(I)} \quad (I) \text{ the annual concentration costs per input ton. See Subsection 6.1.2 for a rigorous definition.}

1 \quad \text{year}.

\textbf{CAP(I,J,K)} \quad (IJK) \text{ the total capital cost of each possible plant combination discounted to the first year at the opportunity cost of investment capital (IJK)}

1 \quad \text{concentrator identification number}

j \quad \text{retinary identification number}

k \quad \text{overall recovery efficiency identification number}.
CAP03(i) - (R) the annual cost of stripping overburden; i - year

CPR(i) - (R) the annual production target for the concentrator in input tons; i - year

CAPCA(i) - (D) the plant capacity of all possible concentrators in input tons; i - concentrator identification number

CUTOFF(i) - (I) the annual cut-off grades scaled up by the grade scale factor 'P' to integer value; i - year

D' - (E) the opportunity cost of investment capital ('MAIN-1', 'CCG-2' and 'CCG-3')

D - (R) a constant used to define the semi-cone closures ('E' & 'SUBP')

Ein(i) - (R) the annual extraction grade (or ton of pit output). See Subsection 6.1.2 for a rigorous definition; i - year

MINN(i, j, k) - (I*2) the total equivalent mineral content at each grid-intersection;

l, m, n - 1, 2 and 3 grid co-ordinates, respectively

F(i) - (D) the fixed plus administrative costs per year; i - year

FIXC - (E) a variable used to transfer the fixed costs 'F' from the civil reader to internal storage

G(i, j, k) - (R) the annual cut-off grade to each pit increment. They are scaled up to integer values with the grade scale factor 'N';
Gm(i), Gc(i), G3(i) - (i=2) the annual economic cut-off grades for the pit, concentrator and refinery, respectively. They are scaled up to integer values with the grade scale factor 'G5':

i - year

Gc(i), Gm(i), G3(i) - (i=2) the balancing cut-off grades for each pit increment between mine and concentrator, refinery and mine, and refinery and concentrator, respectively. They are scaled up to integer values with the grade scale factor 'G5':

i - pit increment

G5 - (i) the class interval for the increments in the T-S mineral distribution 'GRASCS'

G5 = [ ] the scale factor used for converting the proportional mineral grades of the T-S mineral reported in the market's program from external to internal storage

G5 - (i) the scale factor used for converting proportional mineral grades in increments so that they can be used as subscripts in arrays

ICL(i) = (i=2) a prefix used to indicate the grid-blocks in a total of one of the pseudo-random data algorithm

i - grid-block identification number

IGRADE(i,j,k), IGRAD(i,j,k) = i + 1G5(i,j,k) - (i=2) the mineral grades of the 'i'th mineral reported in the market's program at each grid-intersection. They are scaled up to integer values with the grade scale factor 'G5':

i,j,k - 3-d and k grid coordinates, respectively

ICM(i), ICN(i) - (i) the interest rate for the time inflation of costs and selling prices, respectively

YEAR(i) - (i=2) the evaluation year of the i-th grid-blocks
i - grid-block identification number.

IP(i) - (I*2) a pointer for the grid-blocks within the final pit surface;

i - grid-block identification number.

IWA(i) - (I*2) the excavation year of the overburden grid-blocks;

i - grid-block identification number.

L - (I) the mine's life span.

LOC - (I) the subscript for certain indicator arrays. It is computed to locate specific members of uni-dimensional arrays used to store three-dimensional data sets.

MAT(i) - (D) the total material in each pit increment measured in tons (CO-1 and SUBPD);

i - pit increment.

MAT(i) - (I) the annual pit production target, ore plus overburden, in tons (I1 and SUBPD);

i - year.

MAXG - (I) the maximum equivalent mineral grade, reported in the user's deposit.

MAXN - (I) the total number of pit increments.

MILC(i) - (D) the milling cost per unit ton for each recoverable mineral of the example deposit (see Section 7.11);

i - mineral identification number.

MIN(i) - (I) the annual pit production target, ore plus overburden, in tons;

i - year.

MINC - (D) a variable used to transfer the initial assumption of the excavation cost per unit ton from the card reader to internal storage.
MINCAP - (D) the pit capacity, ore plus overburden, per year measured in tons.

NHLS - (I) the total number of grid-intersections or grid-blocks.

MI, NJ, NY - (I) the maximum I, J and K grid co-ordinates, respectively.

O2(i) - (E) the overburden stripping cost per ton on each side of the tenth grid-level of the example deposit. See Section 7.1.

i - pit zone identification number.

OIC(i) - (F) the increased cost of overburden stripping grid-level per ton on each side of the tenth grid-level of the example deposit. See Section 7.1.

i - pit zone identification number.

OR(i) - (G) the ore excavation cost per ton on each side of the tenth grid-level of the example deposit. See Section 7.1.

i - pit zone identification number.

OEC(i) - (H) the increased cost of ore excavation per grid-level per ton on each side of the tenth grid-level of the example deposit. See Section 7.1.

i - pit zone identification number.

ORE(i,j) - (I) the available ore tonnage, in gig increments, for the range of cut-off grade from zero to the maximum reported mineral grade (MCAD, MCAQ, MCAQW, etc) - 2 and 5 in OEC(i).

i - integer cut-off grade increment
j - gig increment.

ORE(i) - (I) this variable has two different definitions in the PD subroutine. First, it is used as the production target for the concentrator. Thus, it becomes the total simulated annual production in each year, with data sets are measured in tons of concentrate input (OS).
ORES(i,j) the data set for the Z-S mineral distribution;

i - integer mineral grade increment
j - pit increment

P(i) - (D) the annual profits estimated from the production targets 'MIN', 'CON' and 'WRAP';

i - year

PIITANG(i) - (F) the safe pit wall-angle, on each side of the fault given for the example deposit. See section 7.1. The angles are measured in radians from vertical.

i - pit zone identification number

PITINC - (1-2) the pit increment

PNEW - (E) the total revenue over the mine's life span.

PFCR(i,j) - (U) the tonnage, in each pit increment, of refined product available at any cut-off grade.

i - integer cut-off grade increment
j - pit increment

PROFIT(i) - (D) the annual revenue from the simulated pit production;

i - year

PV(i,j,k,n) - the total present value of the annual profits (P) estimated from the production targets 'MIN', 'CON' and 'WRAP' for each combination of the given plant capacities;

i - concentrator identification number
j - smelter identification number
k - overall recovery efficiency identification number
n - plant capacity - identification number

PVAT - (V) the present value of the combination of plant capacities yielding the maximum present value.

QF(i,j), QG(i,j), QH(i,j) - (D) the annual plant production targets for each pit increment in tons out the concentrator.
lit and refineries, respectively.

\[ C(i, j) = \begin{array}{ll}
& C(i, j) - (C) \text{ the annual production target for the refinery;}
& i - \text{year}
& j - \text{pit increment}
& \end{array} \]

\[ B(i) = (B) \text{ the annual cost of refining per ton of product;}
& i - \text{year}
& \]

\[ P(i, j), \ P2(i, j), \ P3(i, j) - (P) \text{ the ratios, in each pit increment, of ore 'ORE' to pit material 'MAT', refined product 'PROD', and refined product to ore, respectively, for the range of cut-off grades from zero to the maximum reported mineral grade 'MAX';}
& i - \text{integer cut-off grade increment}
& j - \text{pit increment}
& \]

\[ R(i) = (R) \text{ the proportional overall recovery efficiency for each estimation of plant capacities;}
& i - \text{overall recovery, efficiency identification number}
& \]

\[ R(i) = (R) \text{ the refining cost per ton of product for the concentrate originating from either side of a zone surrounding the fault given for the example deposit. See section 7.1;}
& i - \text{pit zone identification number}
& \]

\[ R(i) = (R) \text{ the plant capacity of the refinery in output tons per year;}
& i - \text{refinery identification number}
& \]

\[ S(i) = (S) \text{ the annual selling price per ton of the primary material;}
& i - \text{year}
& \]

\[ S(i), \ R(i) - (S) \text{ the annual selling price for production of the 'i' th primary material in the model;}
& i - \text{year}
& \]
deposit (MAIN1 - 1, PD, S\textsuperscript{2}SPD);
- \( i \) - year;

\( s_1, s_2, s_3 \) - (\( \sigma \)) the ratio of concentrator capacity \( '\text{CONCAP}' \),
\( \text{to the pit capacity} \ '\text{PITCAP}' \),
refinery capacity \( '\text{REFCAP}' \),
to pit capacity and refinery capacity to concentrator capacity,
respectively, \( '\text{COGSUH)' \).

SLOPE\( (i) \) - (\( \gamma \)) this variable is used to define the semi-cone
angle of the final pit surface design routine; it is
equal to the tangent squared of \( '\text{PITANG}' \).

\( i \) - pit zone identification number.

\( \text{PROD} \) - (1) cumulative total of the simulated ore production
in any given year and measured in tons.

S\text{SUM}(i) - (\( \gamma \)) the present value of the future profits \( '\text{P} \)
in each year;

\( i \) - year.

S\text{SUM}(i) - (2) the cumulative total of the annual pit
production through \( i \) in each pit increment;

\( i \) - pit increment.

SWASTE - (2) the cumulative total of the simulated overburden
production in any given year.

TV, TV\text{2}, TV\text{3}, TV\text{4} - (\( \gamma \)) the time period required to process a
given amount of the material at the optimum cut-off grade,
for the pit, concentrator and refinery, and the maximum
of the three, respectively.

T\text{CF}(i) - (\( \gamma \)) the total area of the grid-blocks on each
guide of the ten grid-lines or the example deposit; see
Section 7.1.

\( i \) - pit zone identification number.

T\text{CF}(i) - (1) grid-intersection identification number;

\( i \) - grid-intersection identification number.
$V(i) = (D) \text{ the present value of the future profits in each year; }$

1 - year

$VAL(i) = (Q) \text{ the T-S net cash value distribution; }$

1 - grid-block identification number

$WASTE(i,j) = (Q) \text{ the thickness of overburden in each pit increment for the range of cut-off grades from zero to the maximum reported mineral grade 'MAX'; }$

1 - integer cut-off grade increment

j - pit increment

$WASTE(i) = (Q) \text{ the total simulated overburden production in tons for each year (WASTE - 3, DD, SUBPD); }$

1 - year

$WMAX - (Q) \text{ this variable is used to define the semi-cone base of the actual pit production simulation. It is the current value of the working wall-angle; }$

$WMAXANG(i) = (Q) \text{ the maximum pit wall-angle at which the excavation equipment can work on each side of the tenth grid-layer of the example deposit. See Section 7.1; }$

1 - pit zone identification number

$WANG(i) = (Q) \text{ this variable is equal to the tangent squared of 'WMAXANG'; }$

1 - pit zone identification number

$Y(i) = (Q) \text{ the extraction year of each mined grid-block; }$

1 - grid-block identification number

$YIELD = (Q) \text{ the overall recovery efficiency of the operation of plant capacities yielding the maximum present value; }$
CHAPTER 9 CONCLUSIONS AND RECOMMENDATIONS

The economic model of an open-pit mine described in this thesis can be succinctly illustrated with the following diagram:

**MINE PLANNING**

† +TIME FACTORS-OPEN PIT PLANNING SYSTEM

**FINANCIAL PLANNING**

This model can, in fact, be considered as a "total systems" planning tool for the evaluation of open-pit mining ventures. It can be adapted to the study of any proposed open pit deposit by altering the computer program. The computer program thus integrates:

1) the physical and geologic conditions of the deposit,
2) the particular pit production method, and
3) the venture's cost structure.

Previous contributions to the field employed in this model are:

1) the estimation of mineral grades by Weaver (11),
2) the design of an optimum pit surface by Fercha and Green in (4), and
3) the economic planning decisions derived by Same (4).

This thesis introduces simulation techniques for pit production, physical features and costs. Most importantly, it also introduces the time factor into planning and evaluation. The incorporation and synthesis of these innovations, as well as the preceding contributions are the model on
improvement over previous open pit planning systems. It does, in fact, represent a significant development in the field.

This development ensured that there is no loss of accuracy as a result of the model being a general planning tool. This is because the computer program was tailored to calculate the physico-geological differences between deposits. Additionally, equivalent cut-off grades are compared with the total mineral content of all recoverable minerals at each point in the deposit, instead of comparing the traditional predetermined cut-off grade with the primary mineral only. The interpolation method used in the model to estimate mineral grades is more accurate than the other methods used for massive deposits. This is due to the search parameters which are redefined for each deposit.

In conclusion, certain recommendations for the further development and understanding of the model can be made. These would be:

A) An analysis of the model to measure the sensitivity of its variations. A sensitivity analysis also would reveal any pathological situations not encountered in the testing of the computer program.

B) A study of the model to establish whether its lengthy execution time can be cut down. For example, the grade prediction program could be made more efficient by eliminating some of the repetitive computations. The computations for the X-Y net cash value distribution can be improved by eliminating its outermost DO loop which cycles with time. The time factor could be obtained directly from the indicator array. DO instead of from both the array and the DO loop parameter. This would eliminate the need to search through all the blocks for each particular year.

C) Finally, the model could be further developed to include all wanted situations. Then, rather than altering the program, the user would merely select those situations relevant to his needs.
CHAPTER 10 REFERENCES


