

https://dspace.org/

Lives of the Fellows of the Royal College of Surgeons of England,

Robinson, R.H.O.B

2015
http://hdl.handle.net/10539/19217

Downloaded from a DSpace repository. https://dspace.org/

AN [illegible] 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 -

ROBERT HOWARD ROBINSON

June, 1959.

E R R A T A

Page 33: A = ((Y• XZ) - (Y•Y)) / ((NS••Z) - (X••Z)) should be:
A = (((Y•XZ) - ((X) • (X•Y))) / ((NS••Z) - (X••Z))

Page 50: First line - 41000 should read 48000.

Page 85: 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 89: GNC = ... should read:

GNC = Gs if Gmc < Gs
= Gc if Gmc > Gc
= Gmc otherwise

The phrase "different than" is used in pages 21 and 120.

Page 118: Second last line: "past" should read "last".

I would like to thank my [illegible], Professor [illegible] and [illegible] of the [illegible] Department for their guidance and duly with this work throughout.

I am also indebted to Dr. [illegible] of the Statistics Department who allowed us the use of his editing programs in the preparation of this thesis, and never failed to assist us with [illegible].

Finally, my thanks to all of the [illegible] [illegible] [illegible] [illegible] [illegible] [illegible].

TABLE OF CONTENTS

		PAGE NO.
CHAPTER 1	INTRODUCTION	1
CHAPTER 2	LITERATURE SURVEY	5
CHAPTER 3	SIMULATION OF THE SURFACE MINERAL DEPOSIT	13
3.1	OPENING DISCUSSION	13
3.2	SELECTION OF A GRID SYSTEM	14
3.2.1	GRID SYSTEM TYPES	15
3.2.2	DIMENSIONS OF THE GRID SYSTEMS	17
3.2.3	SPATIAL DATA BASE	18
3.2.4	SOME PROGRAMMING CONSIDERATIONS	20
3.3	SIMULATION OF THE DEPOSIT'S STRUCTURAL FEATURES	20
3.3.1	INDICATOR ARRAYS	21
3.3.2	POLYNOMIAL RELATIONS	23
3.3.3	SOME PROGRAMMING CONSIDERATIONS	24
3.4	SPATIAL DENSITY OF THE DEPOSIT'S MINERAL VALUES	25
3.4.1	COMPUTATION OF THE ASSAY COMPOSITES	25
3.4.2	INTERPOLATION OF THE SPATIAL MINERAL DISTRIBUTION	30
CHAPTER 4	ASSAY COMPOSITES PROGRAMME	37
4.1	PREPARATION OF THE PROGRAMME FOR EXECUTION	37
4.2	SEQUENCE OF PROGRAMME COMPUTATIONS	40
4.3	A LIST OF THE ASSAY COMPOSITES PROGRAMME	41
4.4	GLOSSARY	45
CHAPTER 5	GRADE PREDICTION PROGRAMME	48
5.1	PREPARATION OF THE PROGRAMME FOR EXECUTION	48
5.2	SEQUENCE OF PROGRAMME COMPUTATIONS	51
5.3	A LIST OF THE GRADE PREDICTION PROGRAMME	52
5.4	GLOSSARY	56
CHAPTER 6	OPEN PIT PLANNING SYSTEM	61
6.1	INPUT DATA	64
6.1.1	VARIABLE INPUT DATA	61

		PAGE NO.
6.1.2	FIXED INPUT DATA	68
6.2	OBJECTIVES OF THE OPEN PIT PLANNING SYSTEM	70
6.3	TIME-SPACE DISTRIBUTION OF MINERALS	75
6.3.1	INITIAL ASSUMPTION OF THE T-S MINERAL DISTRIBUTION	75
6.3.2	EQUIVALENT MINERAL CONTENTS	78
6.3.3	RE-COMPUTATION OF THE T-S MINERAL DISTRIBUTION	79
6.4	ECONOMIC MANAGEMENT OF THE OPEN PIT PROPOSAL	81
6.4.1	TRADITIONAL METHOD OF ECONOMIC POLICY DECISIONS	82
6.4.2	OPTIMUM CUT-OFF GRADES AND PRODUCTION TARGETS	84
6.5	TIME-SPACE DISTRIBUTION OF NET CASH VALUES	91
6.5.1	STANDARD COSTING METHOD	91
6.5.2	ASSESSMENTS OF THE STANDARDS	97
6.6	DESIGN OF THE FINAL PIT SURFACE	102
6.6.1	A PARAMETER GRAPH OF AN OPEN PIT MINE	102
6.6.2	THE LAPONE-GROSSMAN ALGORITHM	105
6.7	PIT PRODUCTION SIMULATION	107
6.7.1	ORE PRODUCTION SIMULATION	108
6.7.2	WASTE PRODUCTION SIMULATION	11[illegible]
CHAPTER 7	AN OPEN PIT MINING SOLUTION	118
7.1	THE EXAMPLE DEPOSIT	116
7.2	THE SOLUTION	122
7.3	ECONOMIC EVALUATION OF THE SOLUTION	139
7.4	AN INVALID SOLUTION	140
CHAPTER 8	THE MINE PLAN PROGRAMME	144
8.1	JOB CONTROL LANGUAGE	144
8.2	PREPARATION OF THE PROGRAMME FOR EXECUTION	146
8.3	CALLING PROGRAMMES	14[illegible]
8.3.1	COMPUTATIONAL SEQUENCE OF MAIN1	148
8.3.2	A LIST OF MAIN1	150
8.3.3	A LIST OF MAIN2	157
8.3.4	A LIST OF MAIN3	163
8.4	SUBROUTINES COG1 AND COG2	169
8.4.1	COMPUTATIONAL SEQUENCE OF COG1	169
8.4.2	A LIST OF COG1	170
8.4.3	A LIST OF COG2	172
8.5	SUBROUTINE CONSUM	17[illegible]

		PAGE NO.
8.5.1	COMPUTATIONAL SEQUENCE OF COSSUB	175
8.5.2	A LIST OF COSSUB	177
8.6	[illegible]	183
8.6.1	COMPUTATIONAL SEQUENCE OF FD	183
8.6.2	A LIST OF FD	184
8.7	[illegible]	189
8.7.1	COMPUTATIONAL SEQUENCE OF SUBFD	190
8.8	GLOSSARY	192
CHAPTER 9	CONCLUSIONS AND RECOMMENDATIONS	201
CHAPTER 10	REFERENCES	203

CHAPTER 1 INTRODUCTION

The model described in this manual is an economic planning tool designed to analyse alternative methods for the valuation of any mineral deposit amenable to exploitation by open pit mining methods.

The model devises a plan whereby some physical unit of the given mineral deposit according to a specified proposal. The proposal is selected by the user and is the projection of:

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

2) the open pit production method

   a. the sequence of excavation
   b. the dimensions of the pit benches,
   c. the angles of the safe and working wall-slopes,
   d. the number of ore working faces and waste benches required to supply the plant listed in sub-paragraph one;

3) the scheduling and proportioning of equity and loan capital to meet the capital requirements.

This proposal directs the model to find a course of action devised to exploit the given deposit. The next proposal is enforced by repeating the calculations to find the combination which yields the greatest return on investor capital. In this way, the model establishes the course of action which will generate the optimum return for the mine.

In addition to the proposal, the following information on the user's deposit is required to execute the model:

1) mineral reserves,

2) topography,

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 systems" analogue of an open pit mine. The 'total systems" analogue is a duplication of the interdependent economic, mathematical, mining and design concepts forming the integral open pit mining concept. In this way, the dependence 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 up 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 mineral content of the material exposed for mining at any given time and the current value of costs and selling prices 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 out 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

this solution's profitability. 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, [illegible], do hereby declare that this [illegible] is my [illegible] and has not been presented to any other university for the purpose of obtaining a degree.

[illegible]

grid-block values include all costs and revenue accrued by the production of the minerals occuring 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 excavation. A record is kept of each grid-block excavation time. The annual production schedules and topographic maps are computed with this time record.

The simulation of the deposit's physical structure and pit production method tailors the Model to the user's deposit. Only simple modifications are required to adapt the simulations to a different deposit or pit production method. This flexibility was an objective in the design of the Model to insure 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 repetitatively 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 programmes. These programme listings follow the

chapters in which their computations are discussed. An example of an application of the Model is presented in Chapter 7. Chapter 8 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 programmes used in this thesis are described in "IBM System/360 Operating System Job Control Language" (3). The job control language is the method of commanding the computer to execute the action required on a particular programme. The Chapters 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.

CHAPTER 2 LITERATURE SURVEY

A broad view of the customary mine planning and pit design methods is presented in "Surface Mining" (3). 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 S. W. Sundeen, page 54, 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.

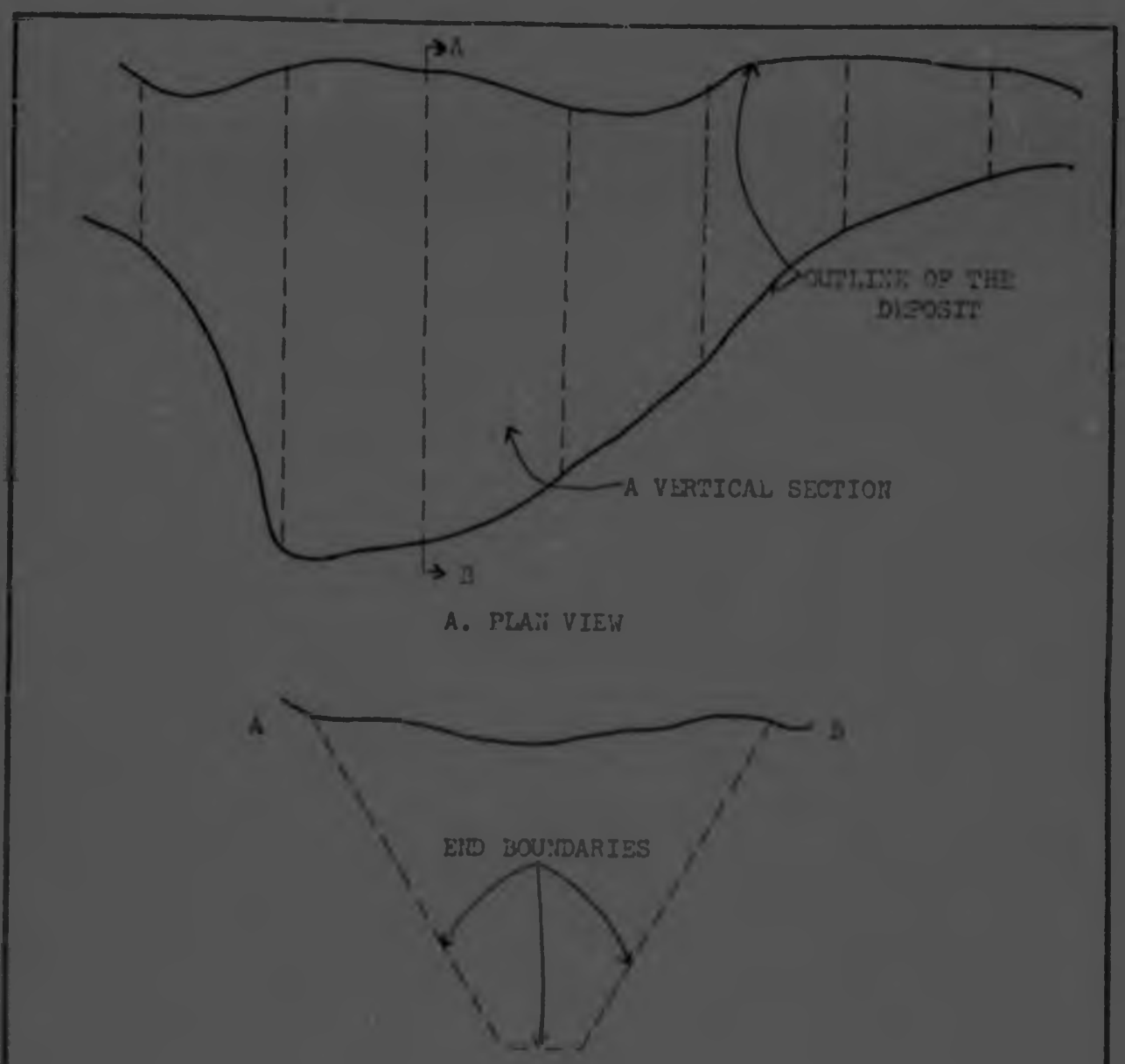
The first five items in Sundeen's list are used by the Model

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 analyse the daily data of the continuous evaluation, which is predominantly socio-political. Therefore, the last two items in the list are included in the model's evaluation indirectly, in 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 section's end boundaries conform generally to the final pit shape. The side 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. These 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-angle. 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 limits' 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 section's large size and the two-dimensional assumption for their magnitude over-simplifies 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 occuring 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. 2.7. 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 marginal 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.

M. 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 series 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: Firstly, 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. H. Lerchs and I. F. 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 Ore (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 consisted 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 simple 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."

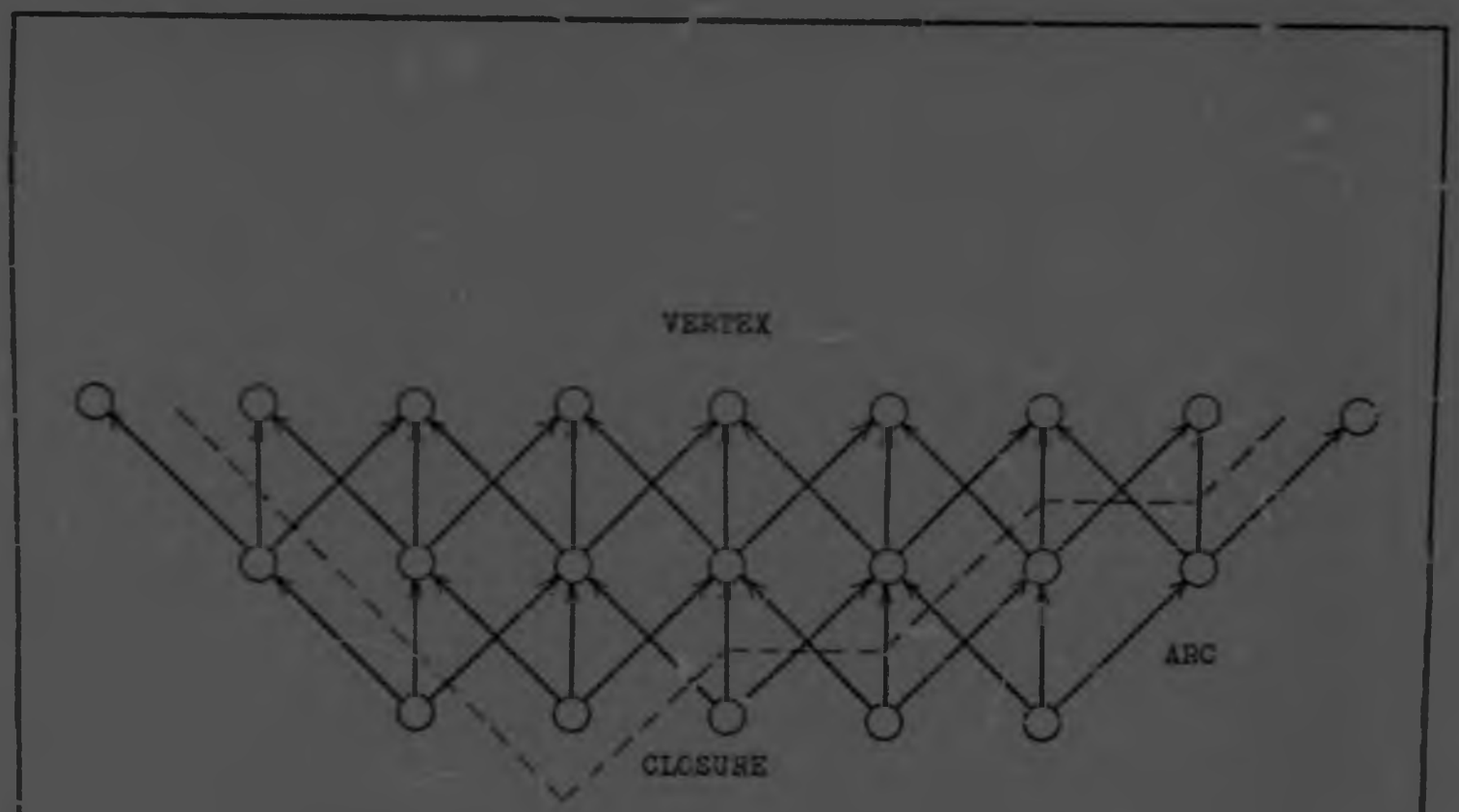


FIG. 2.2 A DIRECTED GRAPH

Lerchs 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. 2.2.)

This "maximum" closure is found by following a set of rules which identify the vertices or 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. This given information represents an assumption of the economic circumstances under which the deposit exists. Lerchs and Grossman (5) observe, however, that "besides pit design, planning has been 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 knowhow."

This deficiency in mathematical knowledge is overcome by a 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. Those conditions, which are variable, are changed one at a time to find the ideal solution for exploiting the user's deposit. The ideal solution is chosen by 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

mentioned. D. J. Hartman and C. G. Verma (1) have written a paper describing an open pit design model which also is based on the Lerchs - Grossman algorithm. Their model performs only the design function for the final pit surface. Additionally, they made some suggestions for the methods of grade prediction and costing that are employed in the Model described by this Thesis.

CHAPTER 3 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 of the grid system's fixed network of lines. For example, the point 500ft south, 400ft east and 200ft deep of the grid system illustrated in Fig. 3.1 would furnish the subscripts (5,4,4) of the array address (I,J,K). The grid systems are discussed 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 3.3.

Every new deposit evaluated with the Model will have a different configuration of structural, geologic and physical features. Therefore, the Model's computer programmes have to be tailored to the user's deposit. The structural simulation techniques and computer programmes are discussed in detail to explain how the programmes 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 programmes discussed in Section 1.4 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 co-ordinate 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.

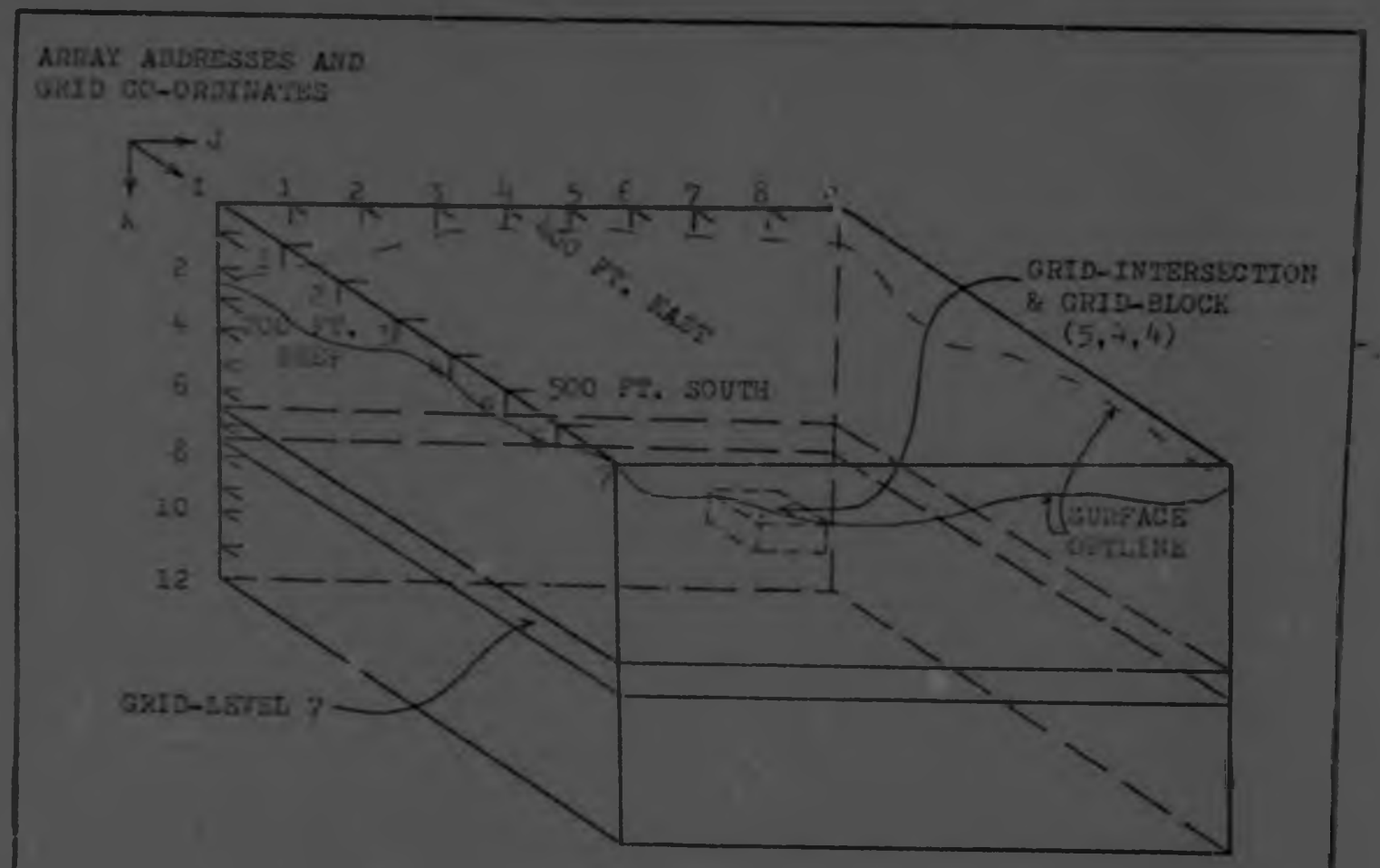


FIG. 3.1 A RECTANGULAR AND REGULARLY SPACED GRID SYSTEM

3) Grid-blocks: The grid-block boundaries are the planes intersecting the midway 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 or nearly 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 and rectangular grid systems. The zero grid-level is designed to insure 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 quantites in an

particular grid-level solely from the assays occuring 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 Rausch (1, 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 Rausch 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 still exist, except in a different dimension.

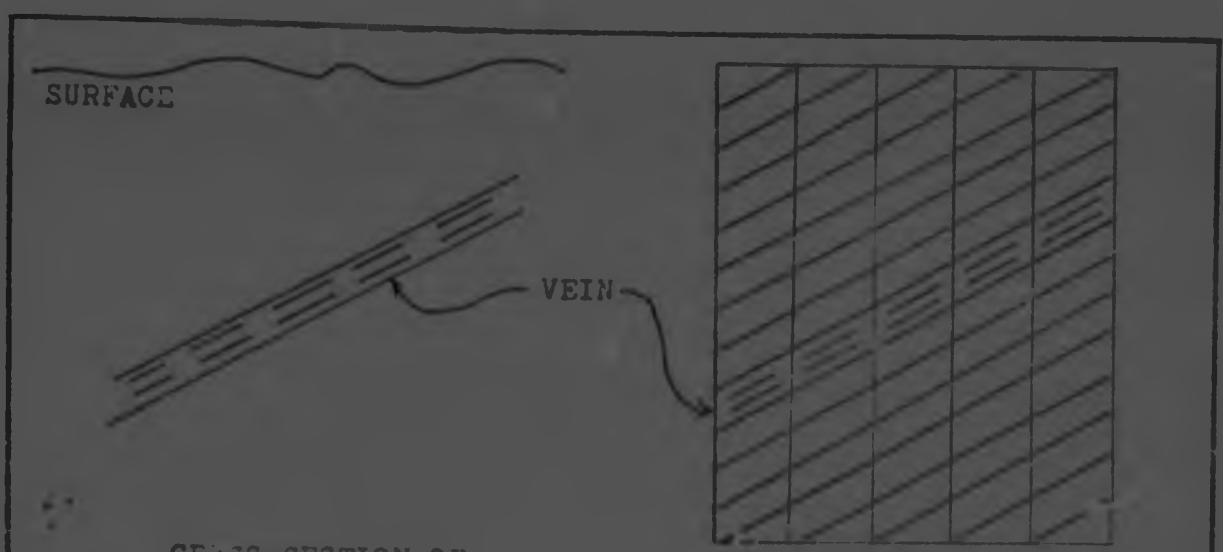
Another solution to this representation problem can be formulated by constructing grid systems with different configurations from the traditional regularly spaced and rectangular grid system. The answer is to design the grid system to represent the mineral bearing rock structures. These grid systems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

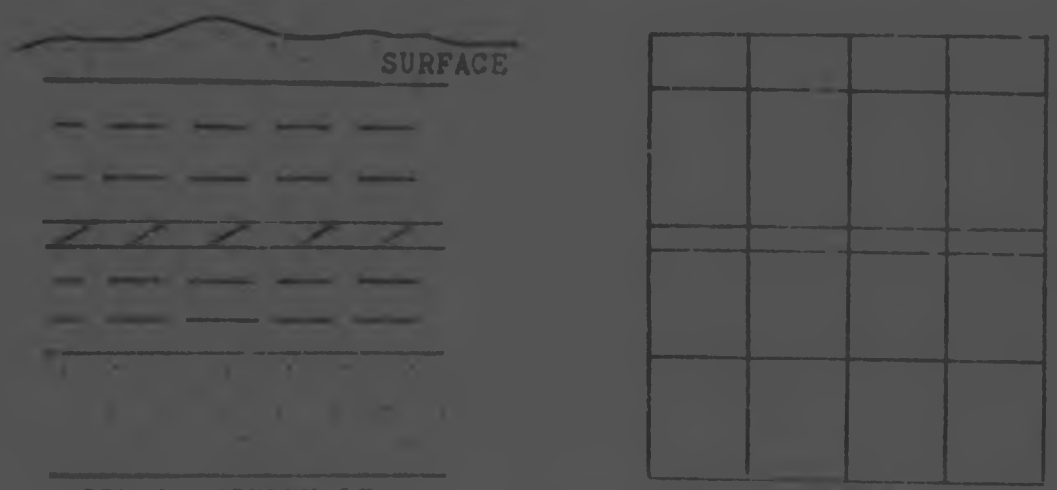
The vein deposit and grid system illustrated by the cross-section view in Fig. 3.2A. The grade prediction of minerals is calculated individually for each grid-level parallel to



CROSS-SECTION OF
A VEIN DEPOSIT

CROSS-SECTION OF A
NON-RECTANGULAR GRID SYSTEM

FIG. 3.2A A NON-RECTANGULAR GRID SYSTEM
USED TO SIMULATE A VEIN DEPOSIT



CROSS-SECTION OF
A LAYERED DEPOSIT

CROSS-SECTION OF AN
IRREGULARLY SPACED GRID SYSTEM

FIG. 3.2B AN IRREGULARLY SPACED GRID SYSTEM
USED TO SIMULATE A LAYERED DEPOSIT

the veins. 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 below 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 capacity 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 210,000 bytes for the programme and remaining data sets. These capacity estimates are slightly generous to allow for uncertain events. The total capacity could be trimmed 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 these unit tonnages and volumes can accurately measure the mine production and simulate the pit surface. Therefore, the vertical and transverse unit interval distances should be an integer divisor of the bench height and width as put forward for solution by the given mining proposal. 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 18) For example, consider the array named 'VALUE' which has three subscripts corresponding to the three co-ordinate directions of the grid system illustrated in Fig. 3.1. If it is desired to address the point density of 'VALUE' at grid co-ordinate I = 1, J = 3 and K = 2, this would be:

VALUE(1,3,2).

"Ordinary mathematical notation might use VALUE i,j,k to represent any element of the array VALUE." (2, page 19) In

FORTRAN, this is written as VALUE(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 disk, 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 53) The FORTRAN statements 'READ' and 'WRITE' "cause transfer of data into or from internal storage." (2, page 53) 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 = 3, the record location 'LOC' is found with the following equation:

```
      LOC=NI*NJ*(K-1)+NI*(J-1)+I
```

Where: 'NI' and 'NJ' 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*9*(3-1)+7*(4-1)+2
      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 set 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 programme 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 programmes 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 conveinently for the Grade Prediction Programme, and then substituted by a regular grid for the remaining computations of the Model. This change is made easily by transfering 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 programmes 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' is 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.

Surface Intercepts			Input	
I	J	K	N	INTERCEPT
1	1	2	SCORK(1)	= 2
2	1	2	SCORK(2)	= 2
3	1	1	SCORK(3)	= 1
4	1	1	SCORK(4)	= 1
5	1	1	SCORK(5)	= 1
6	1	1	SCORK(6)	= 1
7	1	0	SCORK(7)	= 0
1	2	2	SCORK(8)	= 2
2	2	2	SCORK(9)	= 2
3	2	2	SCORK(10)	= 2
.	.	.		
.	.	.		
.	.	.		
7	8	1	SCORK(56)	= 1
1	9	2	SCORK(57)	= 2
2	9	2	SCORK(58)	= 2
3	9	1	SCORK(59)	= 1
4	9	1	SCORK(60)	= 1
5	9	1	SCORK(61)	= 1
6	9	1	SCORK(62)	= 1
7	9	1	SCORK(63)	= 1

where:

N = NI * (J - 1) + I

'NI' is the maximum value of 'I'

TABLE 3.1

The indicator array 'TOPO' is created from 'SCORK' by

executing the following FORTRAN statements:

```
      READ(5,1) NI,NJ,NK
    1 FORMAT (3I4)
      NN=NI*NJ
      READ (5,10) (SCORK(N),N=1,NN)
   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
  105 II=I
      GO TO 104
  103 II=0
  104 TOPO(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 prediction computations 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.

Bore-hole Intercepts with the Fault Plane.

Bore-hole No.	Grid Co-ordinate Intercept		
	I	J	K
1	29.50	18.25	4.40
26	17.50	14.00	14.60
38	3.00	12.00	3.80

Matrix of the Bore-hole Intercepts:

I	J	K	?	
29.50	18.25	4.40	1	= 0
17.50	14.00	14.60	1	
3.00	12.00	3.80	1	

Matrix Solution of the Fault plane:

66.3*I - 277.5*J - 37.625*K + 3274.075 = 0

TABLE 3. 2

The grid co-ordinate location of the bore-hole samples compared with the fault plane equation. Then, the mineral quantites 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. 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 32,[illegible]. 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 subsections 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 Programme.

The mineral samples from the exploration bore-holes are input data to the Assay Composites Programme. The format for these data consists of the proportional mineral content per bore-hole length for each recoverable mineral. Table 3.3 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.

An Example of the Input Data from One Bore-hole

Sample No.	Sample Length	Proportional Mineral Content	
		Mineral 1	Mineral 2
1	340.6	0.0	0.0
2	87.2	0.0025	0.0
3	20.4	0.0028	0.0
4	50.2	0.0041	0.0
5	39.8	0.0038	0.0
6	47.6	0.0049	0.0
7	74.8	0.0052	0.0
8	249.2	0.0053	0.0
9	136.4	0.0049	0.0
10	158.6	0.0036	0.0010
11	285.8	0.0075	0.0026
12	62.8	0.0041	0.0030
13	35.8	0.0036	0.0012
14	18.0	0.0005	0.0001

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 53.2 ft. from the zero grid-level. Consequently, only the bottom 27.8 ft. of Sample No. 2 and the top 1.6 ft. of Sample No. 5 are included in the average for the fifth grid-level's assay composite.

Example Calculations for an Assay Composite

Sample No.	Mineral Content	*	Sample Length	=Product
2	0.0025	*	27.8	= 0.0695
3	0.0028	*	20.4	= 0.0571
4	0.0041	*	50.2	= 0.2058
5	0.0038	*	1.6	= 0.0060
		Total	100.0	0.3385

$$\text{Assay Composite} = \frac{\text{Total Product}}{\text{Total Sample Length}}$$

$$= \frac{0.33852}{100.0} = 0.0034$$

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 pointed

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:

```
      I=0
      DO 10 N=1,NBH
      DO 10 K=1,NK
      I=I+1
      IF (3274•1+0•663*CORI(N)-1•3875*CORJ(N)-37•625*7) 6,7,7
    6 IF (K-10) 1,1,2
    7 IF (K-10) 8,8,9
    8 M=1
      GO TO 3
    9 M=2
      GO TO 3
    1 M=3
      GO TO 3
    2 M=4
    3 ASAY1(I,N,M)=SACO(K,1,N)
      ASAY2(K,N,M)=SACO(K,2,N)
      WRITE(11'I) ASAY1(K,N,M),ASAY2(K,N,M),M
   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 co-ordinates in feet•

'M' 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 'K' 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. 'CORI' and 'CORJ' 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 the 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:

```
FAULT(CORI(N),CORJ(N),K)
```

where 'CORI' and 'CORJ' 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 Programme.

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 Programme listed in Chapter 5.

The interpolation method was suggested by R. C. Weaver (11), and it was chosen over the trigonal 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 trigonal 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:

```
      X=0
      Y=0
      DO 10 N=1,NAC
      X=X+(G(N)/(D(N)**P))
   10 Y=Y+(1/(D(N)**P))
      GRAD(I,J,K)=X/Y
```

where:

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

'NAC' is the total number of assay composites.

'G(N)' is the proportional mineral content of assay compo-

site 'N'.

'D(N)' is the distance between assay composite 'N' and grid-intersection (I,J,K).

'P' is a weighting factor.

'GRAD(I,J,K)' is the estimated point mineral density at grid-intersection (I,J,K).

The preceding routine is executed at each grid-intersection (I,J,K) to estimate the point densities of the spatial mineral distribution 'GRAD'. 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-intersections. This is a circular limit so that each estimate is made only from the assay composites occuring 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, this grid-intersection's point 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 'P', 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-intersections, 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 further 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 assumed search parameters. Alternatively, the assay composites could be used to estimate the mineral values of a previously mined out 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:

```
EV=A+(B*AV)
```

where:

'EV' is the estimated value,
'AV' is the actual value, and
'A' and 'B' are constants.

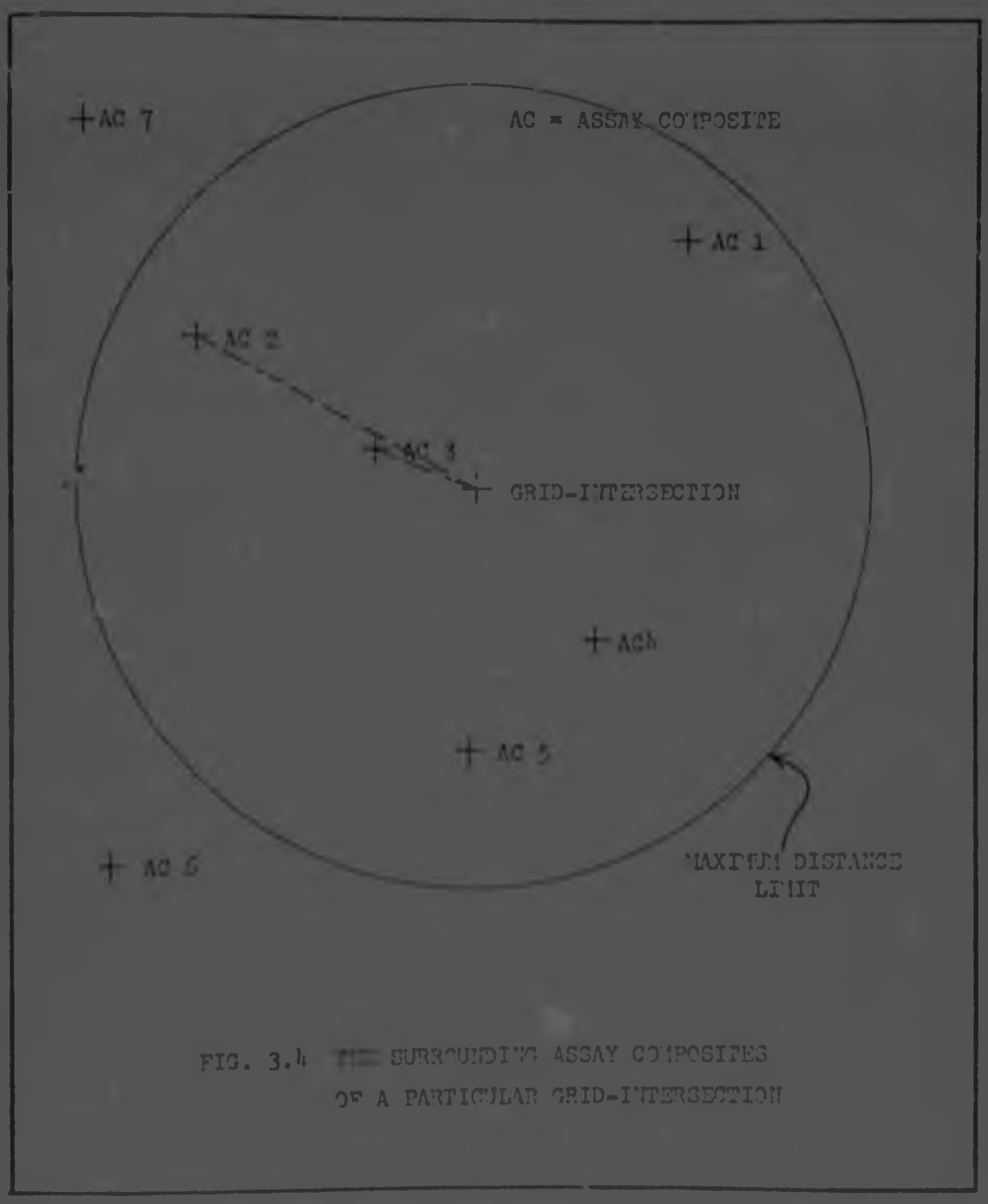


FIG. 3.4 SURROUNDING ASSAY COMPOSITES OF A PARTICULAR GRID-INTERSECTION

The constants--A and B--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=((Y*X2)-(X*XY))/((NS**2)-(X**2))
      B=((NS*XY)-(X*Y))/((NS*X2)-(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=((A+(B*AV(N)))-M)**2
   10 D=(EV(N)-M)**2
      R=SQRT(C/D)
```

where:

'NS' is the total number of samples,

'A' and 'B' are the constants from the regression line,

'AV(N))' are the actual values.

'(A (B*AV(N)))' are the estimated values given by the regression line,

'M' is the mean of the estimated values,

'EV(N)' are the estimated values, and

'R' is the correlation coefficient.

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=(R*SQRT(N-2))/SQRT(1-(R**2))
```

where:

'R' is the given correlation coefficient from the assumed sample,

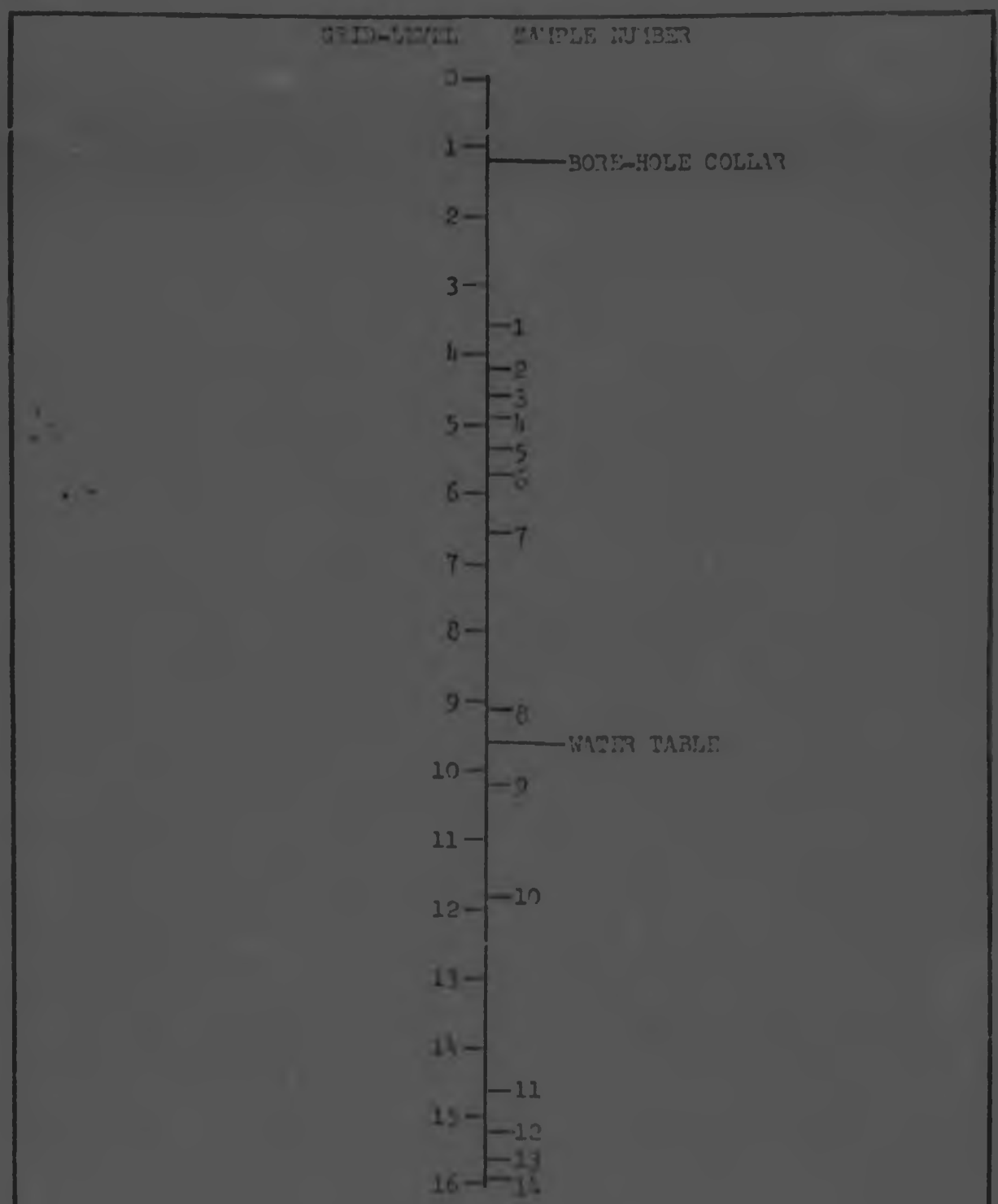


FIG. 3.3 VERTICAL CROSS-SECTION OF A BORE-HOLE

'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 "Theory and 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 3.4.1. 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 made with the 'TOPO' indicator array used as an example in Section 3.3. 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 by-passed 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 3•

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•4,

2) are given for the DEFINE FILE statement in Section 4•3 for the direct access data sets•

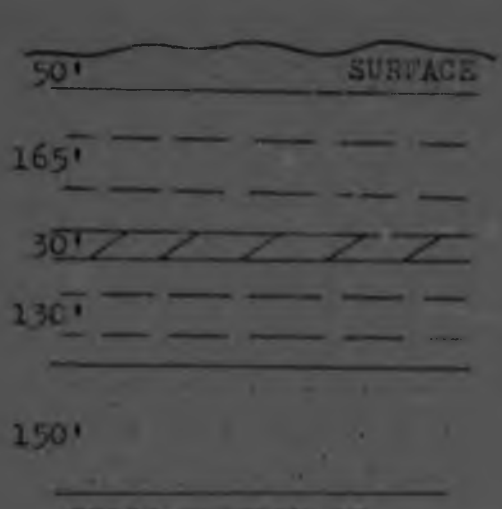
STEP 3:

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•

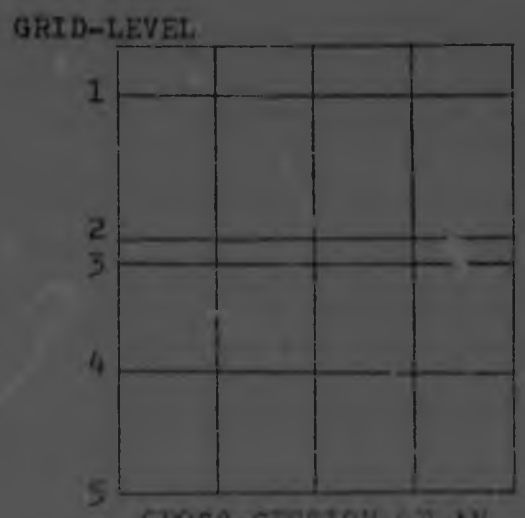
A) 'KINT' is a constant and equal to the unit interval distance between the vertical grid co-ordinates for a regularly spaced and rectangular grid system•

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



CROSS-SECTION OF
A LAYERED DEPOSIT



CROSS-SECTION OF AN
IRREGULARLY SPACED GRID SYSTEM

THE SCALE FACTOR 'KINT' FOR THE ABOVE DEPOSIT

KINT(1)=50

KINT(2)=165

KINT(3)=30

KINT(4)=130

KINT(5)=150

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

-rectangular grid system. It is illustrated in Fig. 4.2 for a non-rectangular grid.

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 092 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 conversly. The constant scale factor described in Step 3A is used to convert scales on regularly spaced axes. The array described in Step 3B 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 Fig. 4.2 is:

$$K = \frac{\tan(a) * \left(I * SFI + \frac{V}{\tan(a)}\right)}{SFK}$$

where:

'K' is the grid-level.

'I' is the horizontal grid co-ordinate.

'V' is the vertical distance in feet.

'SFI' is a scale factor equal to the interval distance

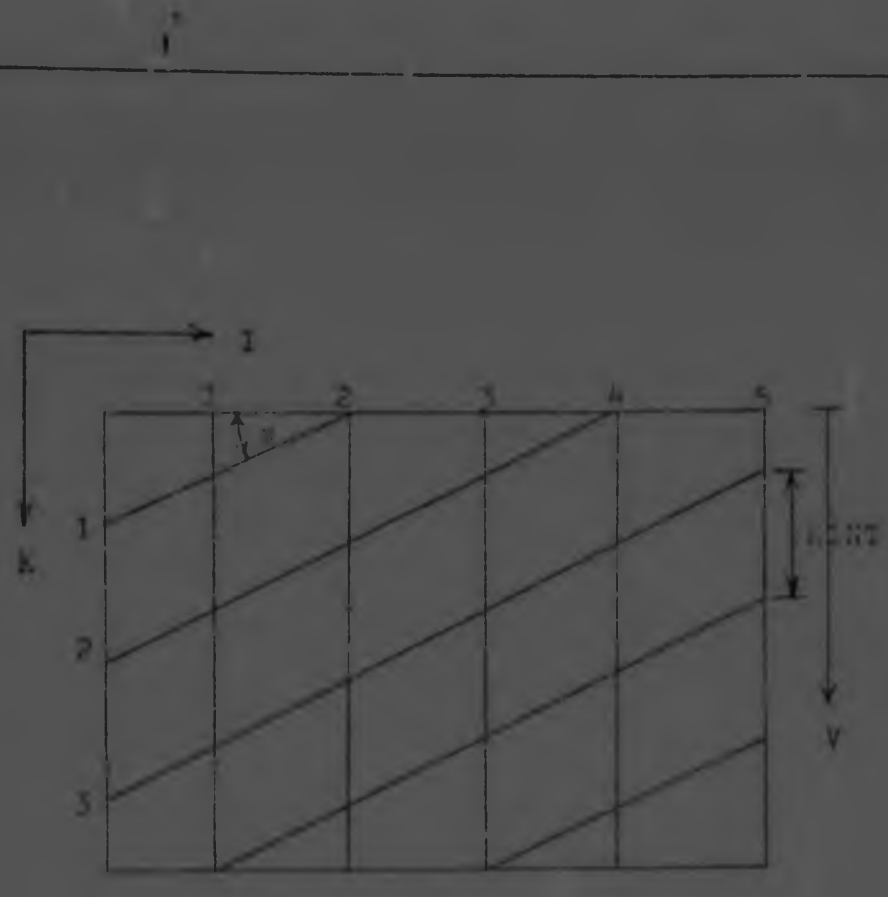


FIG. 6.2 CROSS-SECTION OF A NON-RECTANGULAR GRID

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 statments.

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 programmes.

LINES 016 - 022:

DO loop 16 composites the bore-hole samples 'SAMP' between grid-levels to give an average mineral grade 'SACO' 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 'SACO', if the assay composite should represent air space. Whenever 'SACO' 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(M,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 (M) 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
NO.

001         REAL SAMP(2,60,18),INT(60,18),CORI(60),CORJ(60),
           1CUMINT(17),INTA,INTB,SACO(19,2,60),
           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 = NBH * NK, jj = (number of recoverable minerals + 1)*4, kk = NBH.

```
001         DEFINE FILE 11(ii,jj,JI),12(kk,8,JJ)

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

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 programme.

---------------------------------------------------------

004           READ (5,3) KINT,NMIN,NBH,NSAMP,NK,NZ
005         3 FORMAT (F5.1,5I4)
006           DO 801 N=1,NBH
007           DO 801 L=1,NSAMP,3
008       801 READ (5,1) SAMP(1,N,L),SAMP(2,N,L),INT(N,L),
             1SAMP(1,N,L+1),SAMP(2,N,L+1),INT(N,L+1),
             2SAMP(1,N,L+2),SAMP(2,N,L+2),INT(N,L+2)
009         1 FORMAT (2F7.4,F6.1,2F7.4,F6.1,2F7.4,F6.1)
010           DO 803 N=1,NBH
011           READ (5,2) SURFAC(N),CORI(N),CORJ(N)
012           X=CORI(N)
013           Y=CORJ(N)
014       803 WRITE (12'N) X,Y
015         2 FORMAT (F2,2F7.1)
016           DO 152 L=1,NSAMP
017       152 CUMINT(L)=0
018           DO 16 K=1,NMIN
019           DO 16 N=1,NBH
020           CUMINT(1)=INT(N,1)
021           DO 4 L=2,NSAMP
022         4 CUMINT(L)=CUMINT(L-1)+INT(N,L)
023           DO 16 K=1,NK
024           SUMG=0
025           INTA=0
026           INTB=0
027           SAMPA=0
028           SAMPB=0
029           IF (SURFAC(N)-K) 131,131,130
030       130 SACO(K,N,N)=-10.0
031           GO TO 16
032       131 SUM=0
033           DO 51 LL=1,NSAMP
034        51 LC(LL)=0
035           I=0
```

```
036          L=1
037        5 IF (K*KINT-CUMINT(L)) 6,8,7
038        6 INTA=INT(N,L)-CUMINT(L)+K*KINT
039          IF (INTA-KINT) 14,9,9
040       14 SAMPA=SAMP(M,N,L)
041          GO TO 10
042        7 IF (L-NSAMP) 101,100,100
043      101 L=L+1
044          IF (INT(N,L)) 52,52,5
045       52 INTA=KINT-CUMINT(L)+K*KINT
046          IF (INTA-KINT) 53,112,112
047       53 SAMPA=0
048          GO TO 10
049      100 IF ((K*KINT-CUMINT(L))-KINT) 111,111,112
050      111 INTA=K*KINT-CUMINT(L)
051          SAMPA=0
052          SUM=INTA+INT(N,L)
053          SUMG=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 16
057        8 IF (KINT-INT(N,L)) 9,9,110
058        9 SACO(K,M,N)=SAMP(M,N,L)
059          GO TO 16
060      110 INTA=INT(N,L)
061          SAMPA=SAMP(M,N,L)
062       10 SUM=INTA
063      150 LL=0
064       11 IF (SUM-KINT) 12,151,13
065       12 IF (L-1) 13,13,102
066      102 SUM=SUM+INT(N,L-1)
067          LL=LL+1
068          LC(LL)=L-1
069          L=L-1
070          I=I+1
071          GO TO 11
072       13 SUM=SUM-INT(N,L)
073          I=I-1
074      151 INTB=KINT-SUM
075          SAMPB=SAMP(M,N,L)
076          IF (I) 120,120,140
077      140 DO 15 LL=1,I
078          IF (LC(LL)) 120,120,15
079       15 SUMG=SUMG+SAMP(M,N,LC(LL))*INT(N,LC(LL))
080      120 SUMT=SUMG+SAMPA*INTA+SAMPB*INTB
081          SACO(K,M,N)=SUMT/KINT
```

```
082      16 CONTINUE
083         DO 1112 K=1,NK
084         DO 1112 N=1,NBH
085         DO 1112 M=1,NZ
```

---COMMENT---

A variable 'ASAYn' is added for each recoverable mineral at lines 085 and 102. In other words, ASAY1(K,N,M)=0, 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.

```
086         ASAY1(K,N,M)=0.
087    1112 ASAY2(K,N,M)=0.
088         I=0
089         DO 1010 N=1,NBH
090         DO 1010 K=1,NK
091         I=I+1
```

---COMMENT---

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 Programme 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. 'CORI' and 'CORJ' are measured in feet but 'K' is measured in grid co-ordinates. This difference is accounted for by including scale factors in the expression's constants.

```
092         IF(1274.075+0.663*CORI(N)-1.3875*CORJ(N)
           1-37.625*K) 1006,1007,1007
093    1006 IF (K-10) 2001,2001,2002
094    1007 IF (K-10) 1008,1008,1009
095    1008 M=1
096         GO TO 2003
097    1009 M=2
098         GO TO 2003
099    2001 M=1
100         GO TO 2003
```

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 descritption 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

```
101    2002 M=4
102    2003 ASAY1(K,N,M)=SACO(K,1,N)
103 .       ASAY2(K,N,M)=SACO(K,2,N)
104         WRITE (11'I) ASAY1(K,N,M),ASAY2(K,N,M),M
105    1010 CONTINUE
106         STOP
107         END
```

4. 4 GLOSSARY

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:

NAME(i,j,k) - ( ) 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:

   a. 'D' is double precision, with eight storage locations;
   b. 'R' is real precision, with four storage locations;
   c. 'I' is integer precision, with four storage locations; and
   d. 'I*2' is integer precision, with two storage locations.

```
101    2002 M=4
102    2003 ASAY1(K,N,M)=SACO(K,1,N)
103         ASAY2(K,N,M)=SACO(K,2,N)
104         WRITE (11'I) ASAY1(K,N,1),ASAY2(K,N,M),M
105    1010 CONTINUE
106         STOP
107         END
```

4.4 GLOSSARY

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:

NAME(i,j,k) - ( ) 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:

   a. 'D' is double precision, with eight storage locations;
   b. 'R' is real precision, with four storage locations;
   c. 'I' is integer precision, with four storage locations; and
   d. 'I*2' is integer precision, with two storage locations.

4) A statement of the variable's significance appears next.

5) 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.

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

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

ASAY1(i,j,k), ASAY2(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.

CORI(i), CORJ(i) - (R) the rectangular I and J co-ordinate locations, respectively, of the bore-holes in feet;

i - bore-hole identification number.

CUMINT(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 cummulative sample length 'CUMINT'.

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

LC(i) - (P) 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.

NSAMP - (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) - (R) assay composites;

i - grid-level
j - mineral identification number
k - bore-hole identification number.

SAMP(i,j,k) - (R) 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 5.3. 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*2 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*2 variables. For example, assume that the given deposit is 3000 ft. by 5000 ft. by 1600 ft., and that the desired 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 'IS' and 'JS' scale factors must be increased to reduce the number of grid-blocks to the maximum allowable limit.

STEP 2:

The following search paramters 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) 'DEAR' - the minimum angular separation,

4) 'P' - 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•3 and 3•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 - 186:

The nested DO loops 201 and 45 estimate the point mineral densities 'GRAD1' and 'GRAD2' at each grid-intersection (I,J,K) from the assay composites 'ASAY1' and 'ASAY2'•

LINE 048:

The 'LOC' 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 R 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 by-passed, 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 gridintersection 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 'M'.

LINES 077 - 089:

DO 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 'B' and 'B1' from the grid-intersection (I,J,K) to the selected (NC(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 seperation but a large difference between their bearings. Therefore, the two bearings are required to test the angular seperation close to the opposite base line.

LINES 145 - 149:

DO loops 33 and 37 test the angular separation between the selected assay composites (NC(L))• If the angular separation is less than the minimum 'DTAR', 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 'WRITE' statment 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 'R' by the power 'P' to estimate the point mineral densities 'GRAD1' and 'GRAD2'•

5• 3 A LIST OF THE GRADE PREDICTION PROGRAMME

```
001          REAL COPI(60),COPJ(60),MAXD(4),ASAY1(19,60,4),
            1ASAY2(19,60,4),R(60),B(60),B1(60),ATAN,ABS,SQRT,
            2STORE(30)
002          INTEGER NCLUS(4),NC(60),T,SCORE(750)
003          INTEGER*2 TOPO(30,25,18)
-------------------------COMMENT-----------------------------
```

The parameters of the following DEFINE FILE statement are:

1) ii = NI•NJ•NK,

2) jj = (number of recoverable minerals)•4,

3) kk = NBH•NK,

4) 11 = (number of recoverable minerals + 1)*4, and

5) mm = NBH.

```
004          DEFINE FILE 10(ii,jj,L,IJ),11(kk,ll,L,JI),
            112(mm,8,L,JJ)
005          READ (5,1) NI,NJ,NK,NBH,NZ,IS,JS
006        1 FORMAT (7I4)
007          READ (5,1004) (NCLUS(M),MAXD(M),BEAR(M),P(M),
            1mind(m),m=1,4)
008     1004 FORMAT (I2,F6.1,F7.4,F3.1,F4.2)
```

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

The appropriate number of variables and expressions must be added to the Programme for each recoverable mineral. Parallel expressions are created for ASAYn(K,N,M), ASSAYn, GRADn and SASAYn, where 'n' identifies each recoverable mineral, at Lines 012, 018, 020, 045, 073, 169, 180, 184 and 185.

```
009          DO 1112 K=1,NK
010          DO 1112 N=1,NBH
011          DO 1112 M=1,NZ
012          ASAY1(K,N,M)=0
013     1112 ASAY2(K,N,M)=0
014          I=0
015          DO 1010 N=1,NBH
016          DO 1010 K=1,NK
017          I=I+1
018          READ (11'I) ASSAY1,ASSAY2,M
019          IF (ASSAY1) 1010,1010,1006
020     1006 ASAY1(K,N,M)=ASSAY1
021          ASAY2(K,N,M)=ASSAY2
022     1010 CONTINUE
023          DO 803 N=1,NBH
024          READ (12'N) X,Y
025          CORI(N)=X
026      803 CORJ(N)=Y
027          NN=NI*NJ
028          READ (5,1051) (SCORK(N),N=1,NN)
029     1051 FORMAT (30I2)
030          DO 1054 K=1,NK
031          N=0
032          DO 1054 J=1,NJ
033          DO 1054 I=1,NI
```

```
034          N=N+1
035          IF (SCORK(N)-K) 1055,1053,1053
036     1055 II=I
037          GO TO 1054
038     1053 II=0
039     1054 TOPO(I,J,K)=II
040          EQGRAD=0.
041          DO 901 K=1,NK
042          DO 101 J=1,NJ
043          DO 45 I=1,NI
044          GRAD1=0.
045          GRAD2=0.
046          DO 161 N=1,NBH
047      161 R(N)=0
048          LOC=NJ*NI*(K-1)+NI*(J-1)+1
049          FIND (10'LOC)
050          IF (TOPO(I,J,K)) 46,46,999
051      999 DO 17 N=1,NBH
052          IF (13096.3+2F5.2*I-1110.*J-150.5*K) 1011,1012,1012
```

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

Lines 052 - 062 are tailored to the user's mineralization zones.

```
053     1011 IF (K-10) 2015,2015,2016
054     1012 IF (K-10) 1013,1013,1014
055     101  M=1
056          IF (13090.3+2.652*CORI(N)-5.55*CORJ(N)-150.5
            1*K) 17,75,75
057     1014 M=2
058          IF (13096.3+2.652*CORI(N)-5.55*CORJ(N)-150.5
            1*K) 17,75,75
059     2015 M=3
060          IF (13096.3+2.652*CORI(N)-5.55*CORJ(N)-150.5
            1*K) 75,17,17
061     2016 M=4
062          IF (13096.3+2.652*CORI(N)-5.55*CORJ(N)-150.5
            1*K) 75,17,17
063       75 IF (ABS(CORI(N)-IS*I)-MIND(M)) 90,90,91
064       90 IF (ABS(CORJ(N)-JS*J)-MIND(M)) 1016,1016,92
065       91 IF (ABS(CORJ(N)-JS*J)-MIND(M)) 93,93,1015
066       93 R(N)=(ABS(CORI(N)-IS*I))
067          GO TO 17
068       92 R(N)=(ABS(CORJ(N)-JS*J))
069          GO TO 17
```

```
070    1015 R(N)=SQRT((ABS(CORI(N)-IS*I))**2+(ABS(CORJ(N)
           1-JS*J))**2)
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=NCLUS(M)
077         DO 24 L=1,LMAX
078         DO 1018 N=1,NBH
079         IF (R(N)) 1018,1018,19
080    1018 CONTINUE
081      19 SMEL=R(N)
082      20 DO 23 N=1,NBH
083         IF (R(N)) 23,23,21
084      21 IF (R(N)-SMEL) 22,22,23
085      22 NC(L)=N
086         SMEL=R(N)
087      23 CONTINUE
088     171 R(NC(L))=90000000.
089      24 CONTINUE
090         DO 26 L=1,LMAX
091         IF (ABS(CORI(NC(L))-IS*I)-MIND(M)) 180,180,181
092     180 R(NC(L))=ABS(CORJ(NC(L))-JS*J)
093         GO TO 184
094     181 IF (ABS(CORJ(NC(L))-JS*J)-MIND(M) 182,182,183
095     182 R(NC(L))=ABS(CORI(NC(L))-IS*I)
096         GO TO 184
097     183 R(NC(L))=SQRT((ABS(CORI(NC(L))-IS*I))**2+
           (ABS(CORJ(NC(L))-JS*J))**2)
098     184 IF (R(NC(L))-MAXD(M)) 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 (CORJ(NC(L))-JS*J) 1102,300,1101
104    1101 IF (CORI(NC(L))-IS*I) 106,303,103
105    1102 IF (CORI(NC(L))-IS*I) 105,304,104
106     103 B(NC(L))=ATAN ((ABS(CORJ(NC(L))-JS*J))/
           1(ABS(CORI(NC(L))-IS*I)))
107         GO TO 28
108     104 B(NC(L))=4.71238+ATAN ((ABS(CORI(NC(L))-IS*I))/
           1(ABS(CORJ(NC(L))-JS*J)))
109         GO TO 28
110     105 B(NC(L))=3.14159+ATAN ((ABS(CORJ(NC(L))-JS*J))/
           1(ABS(CORI(NC(L))-IS*I)))
```

```
111         GO TO 28
112     106 B(NC(L))=1.57080+ATAN ((ABS(CORI(NC(L))-IS*I))/
           1(ABS(CORJ(NC(L))-JS*J)))
113         GO TO 28
114     300 IF (CORI(NC(L))-IS*I) 301,308,302
115     301 B(NC(L))=3.14159
116         GO TO 28
117     302 B(NC(L))=0.0
118         GO TO 28
119     303 B(NC(L))=1.57080
120         GO TO 28
121     304 B(NC(L))=4.71238
122      28 IF (CORJ(NC(L))-JS*J) 201,310,202
123     201 IF (CORI(NC(L))-IS*I) 203,313,204
124     202 IF (CORI(NC(L))-IS*I) 206,314,205
125     203 B1(NC(L))=ATAN ((ABS(CORJ(NC(L))-JS*J))/
           1(ABS(CORI(NC(L))-IS*I)))
126         GO TO 29
127     204 B1(NC(L))=1.5708+ATAN ((ABS(CORI(NC(L))-IS*I))/
           1(ABS(CORJ(NC(L))-JS*J)))
128         GO TO 29
129     205 B1(NC(L))=3.14159+ATAN ((ABS(CORJ(NC(L))-JS*J))/
           1(ABS(CORI(NC(L))-IS*I)))
130         GO TO 29
131     206 B1(NC(L))=4.71238+ATAN ((ABS(CORI(NC(L))-IS*I))/
           1(ABS(CORJ(NC(L))-JS*J)))
132         GO TO 29
133     310 IF (CORI(NC(L))-IS*I) 311,308,312
134     311 B1(NC(L))=0.0
135         GO TO 29
136     312 B1(NC(L))=3.14159
137         GO TO 29
138     313 B1(NC(L))=1.5708
139         GO TO 29
140     314 B1(NC(L))=4.71238
141         GO TO 29
142     308 WRITE (6,309)
143      29 CONTINUE
144     309 FORMAT (' TILT')
145         DO 33 L=1,LMAX
146         DO 33 T=1,LMAX
147         IF (R(NC(L))) 33,33,1120
148    1120 IF (R(NC(T))) 33,33,121
149     121 IF (ABS(B(NC(T))-B(NC(L)))-BEAR(M)) 30,33,33
150      30 IF (R(NC(T))-R(NC(L))) 31,33,32
151      31 R(NC(L))=0
```

```
      GO TO 33
   32 R(NC(T))=0
   33 CONTINUE
      DO 37 L=1,LMAX
      DO 37 T=1,LMAX
      IF (R(NC(L))) 37,37,122
  122 IF (R(NC(T))) 37,37,123
  123 IF (ABS(B1(NC(T))-B1(NC(L)))-BEAR(M)) 34,37,37
   34 IF (R(NC(T))-R(NC(L))) 35,37,36
   35 R(NC(L))=0
      GO TO 37
   36 R(NC(T))=0
   37 CONTINUE
      DO 38 L=1,LMAX
      IF (R(NC(L))) 38,38,39
   38 CONTINUE
      GRAD1=0.
      GRAD2=0.
      WRITE (10'LOC) GRAD1,GRAD2,EQGRAD
      WRITE (6,998) I,J,K
  998 FORMAT (' MAY BE OUT OF RANGE',3I3)
      GO TO 45
   39 SASAY1=0
      SASAY2=0
      SUMW=0
      DO 44 L=1,LMAX
      IF (R(NC(L))) 44,44,43
   43 SASAY1=SASAY1+ASAY1(K,NC(L),M)/(R(NC(L))**P(M))
      SASAY2=SASAY2+ASAY2(K,NC(L),M)/(R(NC(L))**P(M))
      SUMW=SUMW+1./(R(NC(L))**P(M))
   44 CONTINUE
      GRAD1=SASAY1/SUMW
      GRAD2=SASAY2/SUMW
   46 WRITE (10'LOC) GRAD1,GRAD2
  901 CONTINUE
      STOP
      END
```

5. 4 GLOSSARY

This glossary defines the FORTRAN variables of the Grade Prediction Programme. 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

4.4.

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

ASAY1(i,j,k), ASAY2(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.

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) - (R) a search parameter which is the minimum distance between assay composite and grid-intersection;

i - mineralization zone identification number.

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

NC(i) - (I) 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) - (I) 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.

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

NZ - (I) the total number of mineralization zones.

P(i) - (R) a search parameter which is the inverse proportional weighting factor;

i - mineralization zone identification number.

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

i - bore-hole identification number.

TOPO(i,j,k) - (I*2) an indicator array used to simulate the deposit's surface;

i,j,k - the I, J and K grid co-ordinates, respectively.

CHAPTER 6 OPEN PIT PLANNING SYSTEM

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 value. 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 6 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

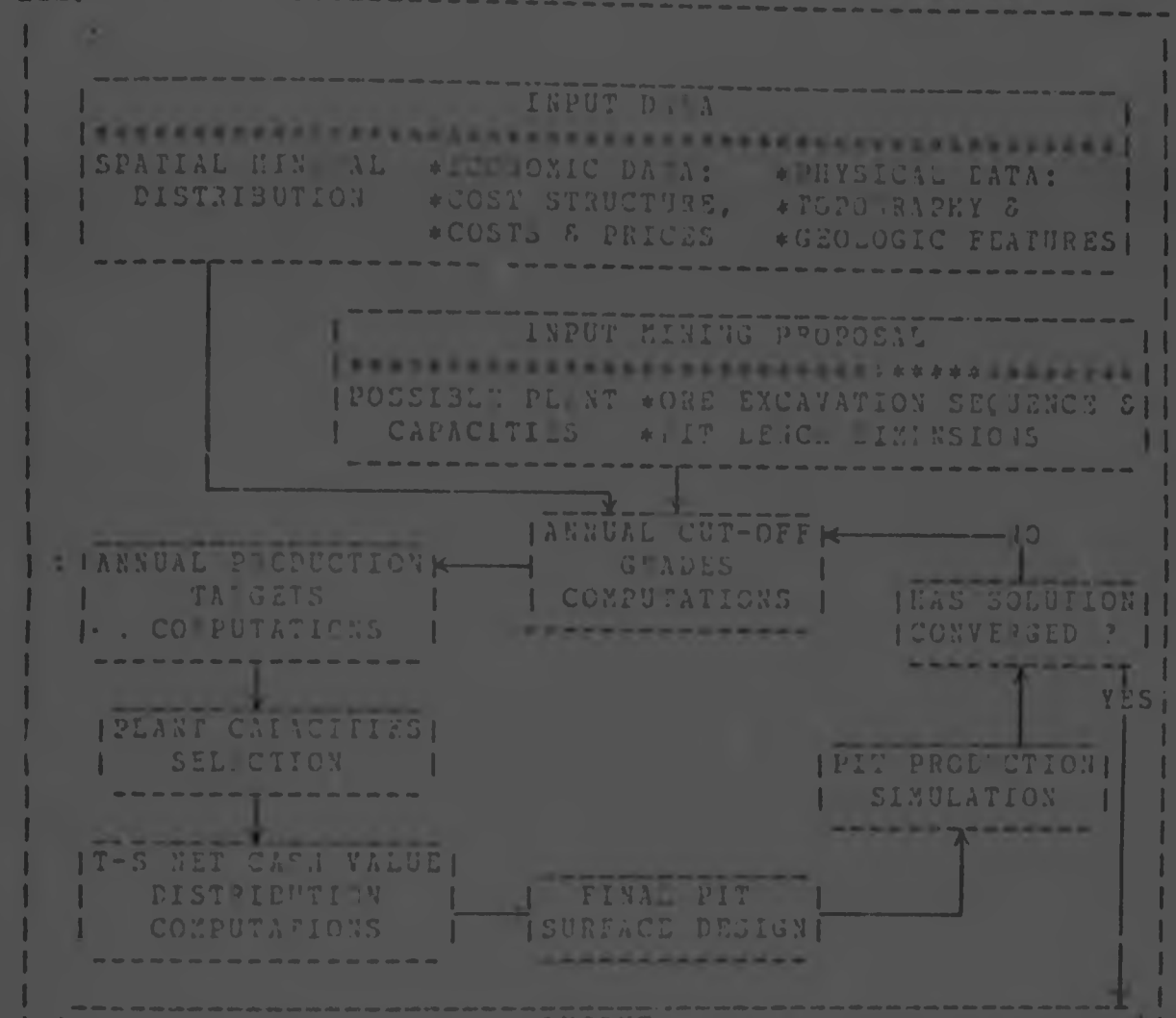


TABLE 6. 1

capacities, and gives 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

OVERALL RECOVERY EFFICIENCY	REFINERY CAPACITY (T)	CONCENTRATOR CAPACITY (M) 1.5	1.6	1.7
85%	20.0	37.75	32.00	35.15
	22.5	32.50	27.65	42.00
	25.0	28.50	27.25	30.25
90%	20.0	37.50	26.85	31.85
	22.5	38.50	41.25	27.00
	25.0	34.75	40.50	38.25
95%	20.0	30.75	40.00	25.50
	22.5	25.75	41.00	37.75
	25.0	28.25	25.65	35.00

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 include: 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 6.1. The three regions are delimited by their orientation with the weak their strength direction of the deposit's bedding planes.

Working Wall Angles:

The working wall-angles define the allowable minimum angle

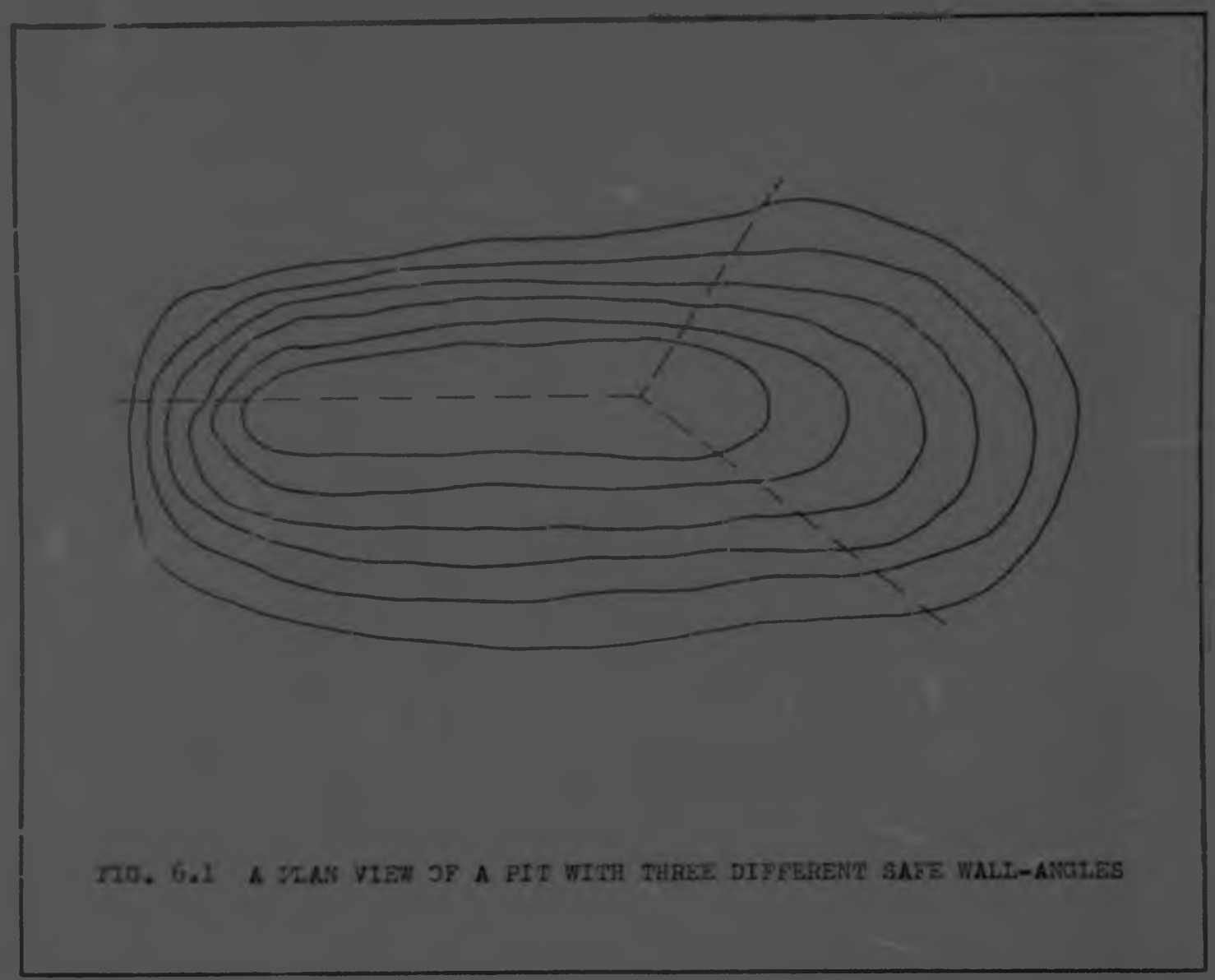


FIG. 6.1 A PLAN VIEW OF A PIT WITH THREE DIFFERENT SAFE WALL-ANGLES

to the vertical that the pit walls 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 3.2.2. 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 alter-

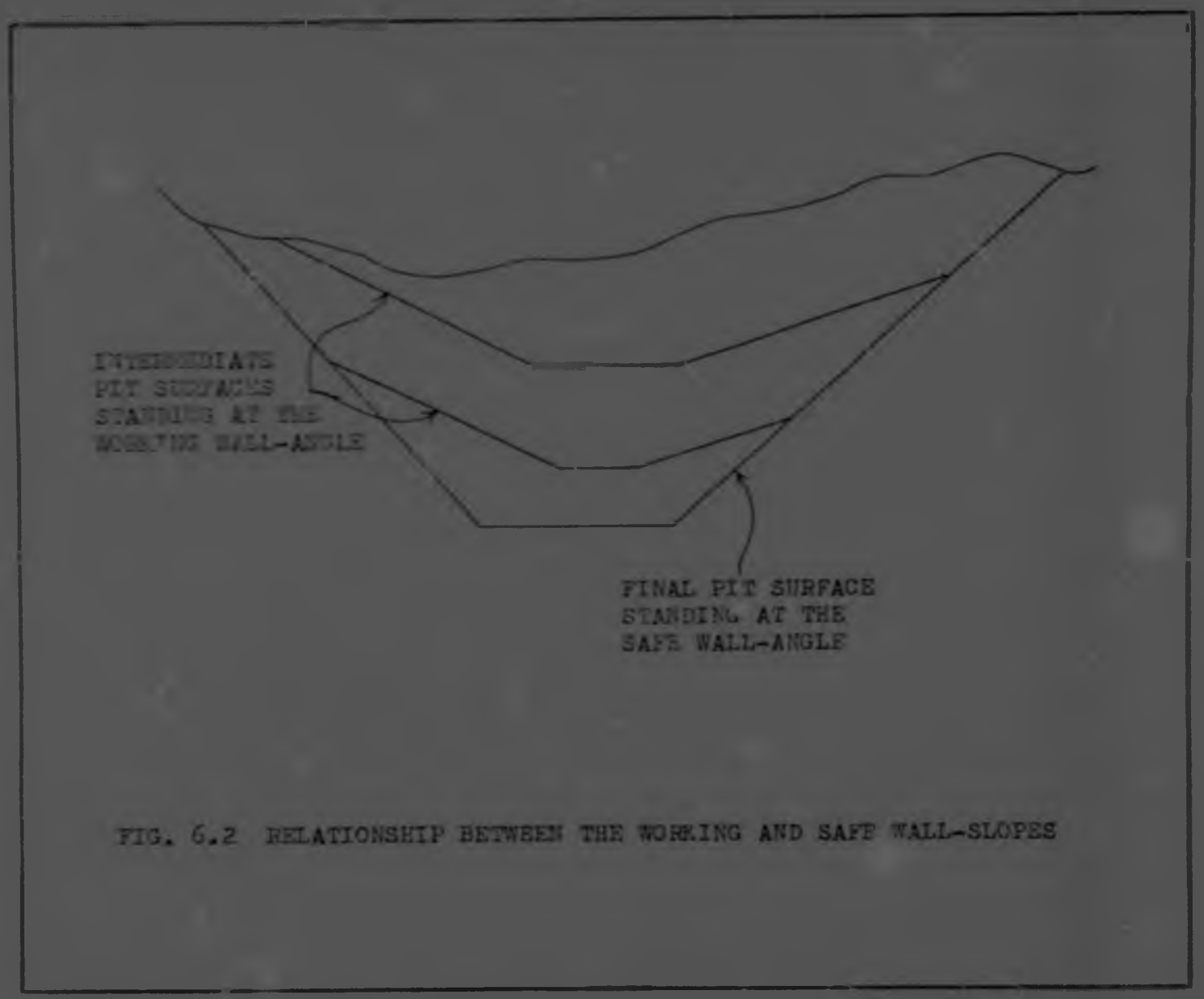


FIG. 6.2 RELATIONSHIP BETWEEN THE WORKING AND SAFE WALL-SLOPES

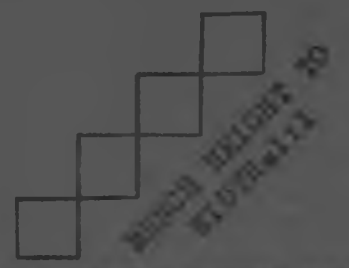


FIG. 6.3A SINGLE BENCH OFFSET

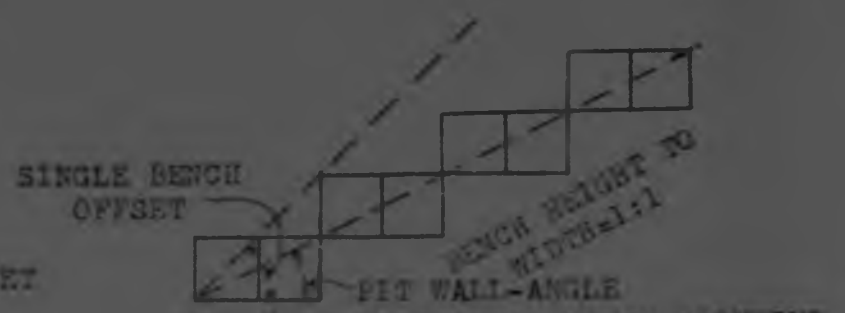


FIG. 6.3B HORIZONTAL BENCH GROUPING

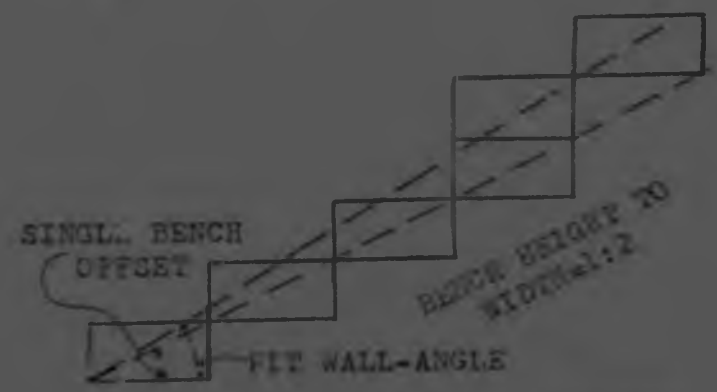


FIG. 6.3C VERTICAL BENCH GROUPING

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 6.3C. 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 Programme permits the following specifications of the 'INPUT MINING PROPOSAL' to be made independently:

1) the bench height and width as defin[illegible] the excavation equipments working space requirement[illegible],

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 [illegible] by the MP Programme. 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 MP 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 MP 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 MP 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. This

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 distribution 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 MP 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 standards are adjusted and totalled individually to give the total net cash value of each grid-block according to the peculiar conditons 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 excavating 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 MP 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 MP 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 MP Programme is written to recognize any of these

DEFINITION OF UNIT PRODUCTION COST TERMS

ANNUAL UNIT COSTS		STANDARD UNIT COSTS
DEFINITIONS FOR CUT-OFF GRADES & PRODUCTION TARGETS COMPUTATIONS	ALGEBRAIC EQUIVALENCE	DEFINITIONS FOR T-S NET CASH VALUE DISTRIBUTION COMPUTATIONS
E= EXCAVATION COST PER UNIT WEIGHT OF PIT MATERIAL	E=OB	OB= WASTE EXCAVATION COST PER UNIT WEIGHT OF WASTE
		OR= ORE EXCAVATION COST PER UNIT WEIGHT OF ORE
C= CONCENTRATION COST PER UNIT WEIGHT OF ORE	C=MILC+OR-OB	MILC= MILLING COST PER UNIT WEIGHT OF MILL INPUT
R= REFINING COST PER UNIT WEIGHT OF END PRODUCT	R=REFC	REFC= REFINING COST PER UNIT WEIGHT OF END PRODUCT

TABLE 6• 3

features so that it can modify the open pit mining solution for their occurences• The programming techniques described in Section 3•3 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 R encompassing the user's deposit. The region R is subdivided into subregions, called parallelipipeds, 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 R.

Let $P(C(x,y,z),t)$ be defined as the net cash value density function $C(x,,z)$ 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 $P(C(x,,z),t)$ 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, and

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 PV of the following multiple integral with the preceding restrictions:

$$PV = \int_T \left[\iiint_R P(C(x,y,z),t)\,dV \right] dt$$

The three innermost integrals are evaluated first to find the final pit surface yielding a maximum total profit. Then, the remaining integration is performed to find the exploitation solution that will maximize the mine's present value. Unfortunately, a simple relationship does not exist for the function $P(C(x,y,z),t)$. Additionally, the limits for the time 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 MP 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 dispersion of mineral quantities and net cash values by their spatial location and by the period in which they are mined. The time and space dimensions are used to account for the differences in the mineral quantities and net cash values available for exploitation at any moment 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. Each point density is defined by its three-dimensional location in the user's 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 $P(C(x,y,z,t))$. Secondly, the T-S mineral distribution is the deposit's tonnages per mineral grade interval per time

period. 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 6.3 and 6.5.

B) Annual unit production costs: These costs are the annual excavation, concentration and refining costs per 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 distibution 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 valid 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. It is with the above assumptions that this is achieved. In the end, the iteration converges as corrections to the assumptions become compatible with the final pit surface

design and exploitation solution. Table 6.1 illustrates the process graphically.

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

$$PV = \int_T [\iiint_R (\quad (x,y,z),t)\,dV]dt$$

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 densities of the function P(C(x,y,z),t) are computed in the routine T-5 NET CASH VALUE DISTRIBUTION COMPUTATIONS assuming the time dimension. Then, the operation

$$\iiint_R (C(x,y,z),t)\,dV$$

is executed to find the pit yielding the maximum total profit over the region R, i.e. the grid system, by the routine FINAL PIT SHELL DESIGN. Thereafter, the operation

$$\int_T (\qquad\qquad)dt$$

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

A) with the times computed by the routine PIT PRODUCTION SIMULATION,

B) 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:

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 Programme (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 totalling the mineral quantities of succes-

B) 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:

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 Programme (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 totalling the mineral quantities of succes-

sive groups of grid-levels from the surface down. The subscript 'N' denotes the grid-level groups. The following routine illustrates how this assumption is made:

```
      DO 11 N=1,MAXN
      K1=4*N
      K2=K1-3
      DO 10 K=K2,K1
      DO 10 J=1,NJ
      DO 10 I=1,NI
      IF (TOPO(I,J,K)) 10,10,5
    5 M=GRAD(I,J,K)*GS
      IF (M-1) 6,7,7
    6 M=1
    7 ORERES(M,N)=ORERES(M,N)+TONF
   10 CONTINUE
   11 CONTINUE
```

where (in order of appearance):

'N' identifies 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 directions, respectively.

'TOPO' 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'.

In DO loop ten, the 'TOPO' indicator array is used to branch past the remaining computations, if the grid-intersection (I,J,K) is above the deposit's surface. Otherwise, the point mineral densities 'GRAD' are multiplied by a scale factor 'GS' to increment the mineral grades and to make their values suitable for the integer subscript 'M' of 'ORDRES'. 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 'GRAD' is .036 and 'GS' equals 100. Then, the tonnage represented by this point density is added to grade increment M=3.

The second IF statement checks the value of 'M' to see if it is less than one. If so, statement number six reassigns 'M' a value of one to prevent 'ORDRES' from being out of subscript range. Consequently, 'GS' must be defined small enough so that all array members where M=1 can represent the pit material with a mineral content range of 0≤M<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 3.3. 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 variations. This simulation technique was explained in greater detail in Subsection 3.3.2. As pointed out in that Subsection, it has a comparatively 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-5) 1,1,2
    1 MM=1
      GO TO 3
    2 MM=2
    3 •
      •
      •
   10 ORERES(M,N)=ORERES(M,N)+TONF(MM)
```

The IF statement expression represents the boundary between the two unit weights. The 'TONF' array members contain the unit weight of the grid-blocks whose relative positions with grid-level five are given by their 'K' grid co-ordinate. 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 'ORERES' that there was only one recoverable mineral reported. A deposit containing several recoverable mineral presents a difficulty for computing 'ORERES'.

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 occuring 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, assume that:

1) there are three recoverable minerals 'SPAM1', 'SPAM2'

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 'GS', 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 K=1,NK
      DO 10 J=1,NJ
      DO 10 I=1,NI
      LOC=NJ*NI*(K-1)+NI*(J-1)+I
      IF (TOPO(I,J,K)) 10,10,2
    2 IF (YP(LOC)) 10,10,3
    3 PITINC=(YP(LOC)-1)/PERIOD+1
      IF (K-5) 4,4,5
    4 MM=1
      GO TO 6
    5 MM=2
    6 EQGRAD(I,J,K)=GRAD1(I,J,K)+((S2/S1)*(Y2/Y1)*
     1GRAD2(I,J,K))
      M=EQGRAD(I,J,K)*GI
      IF (M-1) 7,7,8
    7 M=1
    8 ORERES(M,PITINC)=ORERES(M,PITINC)+TONS(MM)
   10 CONTINUE
```

Definitions (in order of appearance):

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

'NI', 'NJ' and 'NK' 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.

'TOPO' 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.

'MM' 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.

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

'GI' is a scale factor.

'M' is the integer value of 'EQGRAD.'

'CRERES' is the T-S mineral distribution.

'TONF' 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 excavated is stored in data set 'Y2' by the routine PIT PRODUCTION SIMULATION. Statement number two establishes whether the grid-block named by the DO loop parameters (I,J,K) has been

mined• If not, the routine branches past the remaining computations• A negative or zero array value for 'YR' indicates that the given grid-block (I,J,K) is unmined• 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 'TONF' of the given grid-block (I,J,K) is added to the appropriate array member of 'OREPES' 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 in 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 'TONF', 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 data set with the increasing accuracy of the time factor as provided by the iterative solution•

6• 4 ECONOMIC MANAGEMENT OF THE OPEN PIT PROPOSAL

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)

6• 4• 1 TRADITIONAL METHOD OF ECONOMIC POLICY DECISIONS

The traditional method of calculating this economic policy (9; page 141, 874 and 897) 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 th ee 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:

ORE VALUE/TON - ORE PRODUCTION COST/TON

-WASTE STRIPPING COST/TON

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:

"BREAK-EVEN STRIPPING RATIO=

$$\frac{\text{RECOVERABLE VALUE/ORE TON - PRODUCTION COST/ORE TON}}{\text{STRIPPING COST/WASTE TON}}\text{•}$$

In this formula, 'PRODUCTION COST' is the total of all costs through to the refined metal, exclusive of stripping costs•" (9, page 146) 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 assumed for the pit• Any combination of these assumptions fixes the capacity requirements of the

production [illegible] limits, 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 [illegible] mineral distribution, the changing present value over the pit's life span and the time inflation of costs and prices. The economic environment provided by these changing circumstances demands that the cut-off grades vary accordingly, to maximize the pit's present value. Additionally, the traditional method makes no effort to combine or unite into one interdependent function the computations for the pit design, cut-off grade and plant capacities. As a result the final cut-off grade chosen to make the ore 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 is the outcome 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 interdependent functions of plant capacities, cut-off grades and production targets. They are included in the MP Programme in order to furnish the economic decisions that will maximize the ultimate present value. This computational interdependence is an advantage of the MP Programme because it is the unification of the open pit relationships that exist conditionally upon one another. Recall that those decisions provide the relationship between the function $P(\zeta(x,y,z),t)$ and the path T of the multiple integral developed in Section 6.2.

$$PV = \int_T \left[\iiint_R (P(\zeta(x,y,z),t)\, dV \right] dt$$

The MP Programme computes 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 cut-off

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 productuon 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"(4, page 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.

$$P=[(s-r)\cdot Qr]-[c\cdot Qc]-[m\cdot Qm]-[f\cdot T]$$

Where:

'p' is the profit.

's' is the selling price per unit weight of product.

'm' 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).

'r' 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 81').

'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 = the maximum possible present value of profits from the deposit and

2) W = the maximum possible present value of future profits after the next Qm of material has been mined.

Then, from the definition of present value:

$$PV=(P+W)/(1+d)^{T}$$

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) 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:

$$PV-W=P-(d*V*T)$$

Further, substituting:

1) V for (PV-W), the increase in present value from mining the new Qm of material and

2) the profit expression

yields the basic present value equation:

$$V=[(s-r)*Qr]-[c*Qc]-[m*Qm]-[(f+d*PV)*T]$$

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

Lane (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 same 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 MP 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 'M' is the maximum pit capacity; and

$$Vm=[(s-r)*Qr]-[c*Qc]-[[(f+d*PV)/M+m]*Qm]$$

Given s, the cut-off grade affects only Qr and Qc. Therefore the cut-off grade must be chosen to make

[(s-r)*Qr]-[c*Qc]

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

(s-r)*y*Gm=c

Gm=c/[(s-r)*y]." (4, page 816)

where:

'y' is the overall recovery efficiency of the equivalent mineral.

Similarly, if 'C' and 'R' are the maximum capacity per period for the concentrator and refinery, then:

T=Qc/C
Vc=[(s-r)*Qr]-[[(f+d*PV)/C+c]*Qc]-[m*Qm]
Gc=[(f+d*PV)/C+c]/[(s-r)*y]

T=Qr/R
Vr=[[-(f+d*PV)/R+(s-r)]*Qr]-[c*Qc]-[m*Qm]
Gr=c/[[-(f+d*PV)/R+(s-r)]*y]

The three limiting economic cut-off grades Gm, Gc, and Gr will be the ones which most increase Vm, Vc and Vr 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 'PV' 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." (4)

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: pit material, and refined product: ore." (4, page 817). 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:

Gmc is the cut-off where:

$$\frac{\text{ORE}}{\text{PIT MATERIAL}} = \frac{\text{CONCENTRATOR CAPACITY}}{\text{PIT CAPACITY}},$$

similarly:

$$G_{mr}: \quad \frac{\text{REFINED PRODUCT}}{\text{PIT MATERIAL}} = \frac{\text{REFINERY CAPACITY}}{\text{PIT CAPACITY}}$$

$$G_{rc}: \quad \frac{\text{REFINED PRODUCT}}{\text{ORE}} = \frac{\text{REFINERY CAPACITY}}{\text{CONCENTRATOR CAPACITY}}$$

"Sometimes a balancing position may not exist within the permissible range of grades. In such cases the corresponding balancing cutoff grade should be defined as 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, Qr and Qc vary; hence Vm and

Vc• At low cut-off grades Vm is smaller than Vc• 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 Fig• 6•4B and C• As these graphs show, when the balancing cut-off grade Grc 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•" (4, page 319)•

Three optimum cut-off grades GMC, GRC and GMR 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:

GMC=Gm if Gmr≤Gm
=Gc if Gmc≥Gr
=Gmc otherwise

For the mine and refinery:

GMR=Gm if Gmr≤Gm
=Gr if Gmr≥Gr
=Gmr otherwise

For the concentrator and refinery:

GRC=Gr if Grc≤Gr
=Gc if Grc≥Gc
=Grc otherwise•

The overall optimum cut-off grade 'G' is selected from GMC, GRC and GMR, and always is the middle value of the three• This can be demonstrated with the three V-curves illustrated

in Fig• 6•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 Vm, Vr and Vc• This is the curve1 polygon barred 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 GMC, GMR and GRC• In the case illustrated, it is GMR• 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 821)

The annual plant production targets for any year are computed from the cut-off grade 'G' of that year and the T-S mineral distribution 'ORER.S'•

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 T-S mineral distribution 'ORE-RES' 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• The 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 7•9• The data 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 7•9 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 Subsection are as follows in the subroutine COG1, COG2 and COG3 listed in Sections 8•4 and 8•5•

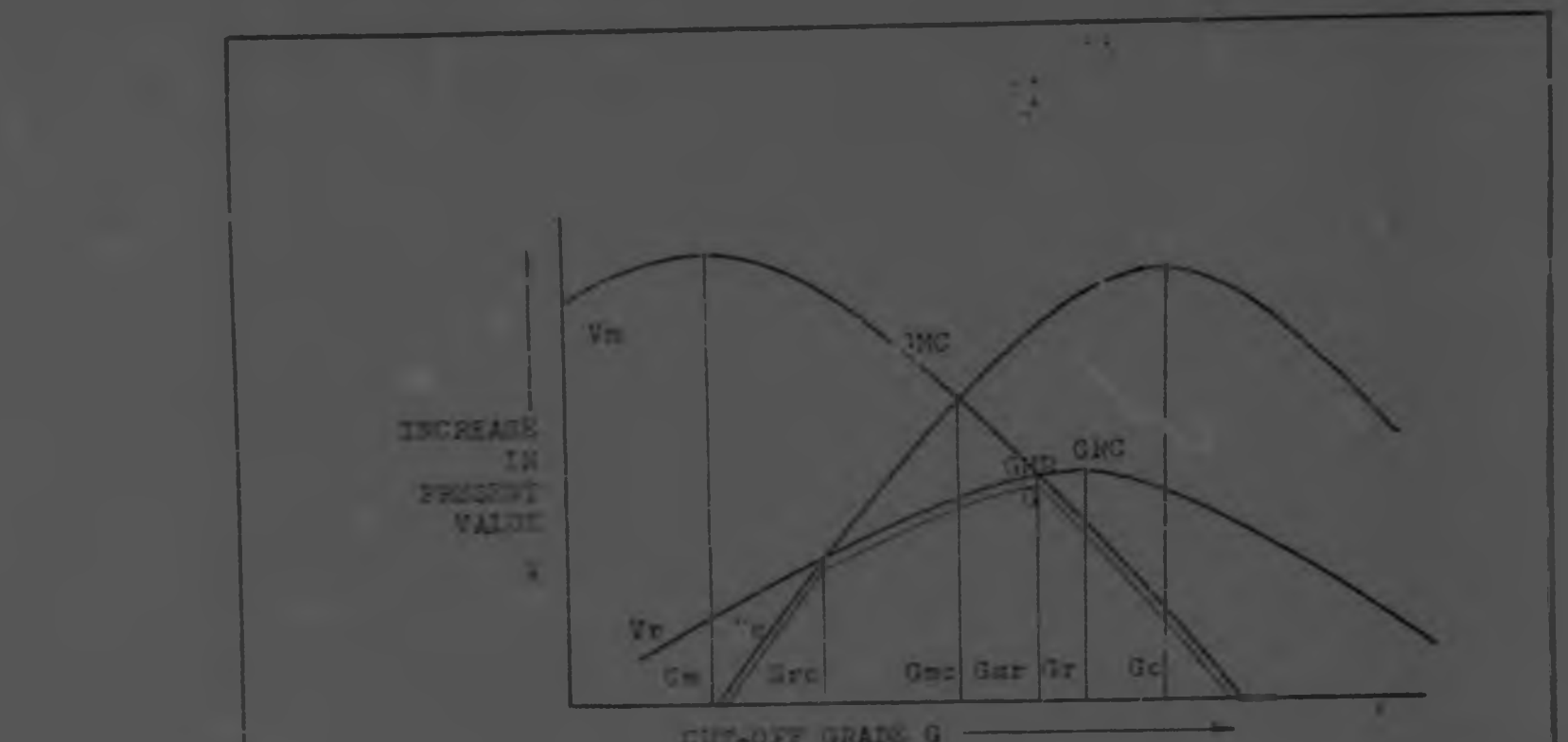

Vm
IMC
INCREASE IN PRESENT VALUE
GMC
Vr
Gm
Grc
Gmc
Gar
Tr
Gc
CUT-OFF GRADE G

FIG. 6.5

6• 5 TIME-SPACE DISTRIBUTION OF NET CASH VALUES

The time-space (T-S) distribution of net cash values is the concentration and dispersal 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 store 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 (as defined in Table 6•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.

```
LINE
 NO.

 01         DO 38 T=1,L
            .
            .
            .
 02         DO 34 K=1,NK
 03         DO 34 J=1,NJ
 04         DO 34 I=1,NI
 05         LOC=(J-1)*NI*(K-1)+NI*(J-1)+I
 06         IF (TOPO(LOC)) 34,34,14
 07     14  IF (YR(LOC)) 15,15,17
 08     15  M=1
 09         IF (T-M) 16,16,34
```

```
10      16 IF (EQGRAD(I,J,K)-AVEG) 19,23,23
11      17 IF (YR(LOC)-L) 18,18,261
12     261 M=1
13         GO TO 16
14      18 IF (T-YR(LOC)) 34,262,34
15     262 M=T
16         IF (EQGRAD(I,J,K)-CUTOFF(T)) 19,23,23
17      19 REV=0.0
           .
           .   (standard cost adjustments)
           .
18         COST=TONF*(X+(F(M)/MIN(M)))
19         GO TO 33
20      23 .
           .   (standard price adjustments)
           .
21         REV=TONF*(GRAD1(I,J,K)*S1(M)*YIELD1+
          1GRAD2(I,J,K)*S2(M)*YIELD2+...
          2GRADn(I,J,K)*Sn(M)*YIELDn)
           .
           .   (standard cost adjustments)
           .
22         COST=TONF*(Y+Z+W+(F(M)/MIN(M)))
23      33 VAL(I,J,K)=REV-COST
24      34 CONTINUE
           .
           .
           .
25      38 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.

'NI', 'NJ' and 'NK' are the maximum (I,J,K) grid-co-ordinates respectively.

'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).

'TOPO' is an indicator array of the deposit's surface.

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

'M' is a transfer variable used to expedite programming. It equals the production year.

'EQGRAD' 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).

'TONF' 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).

'GRAD1', 'GRAD2', . . . 'GRADn' are the point mineral grades at grid-intersection (I,J,K) for 'n' recoverable minerals, respectively.

'S1', 'S2', . . . 'Sn' are the selling prices per unit weight for the 'n' recoverable minerals, respectively, in year (M).

'YIELD1', 'YIELD2', . . . 'YIELDn' 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 point 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 38 names the production year for which the following computations are made.

02) DO loop 31 computes the point 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 PIT 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 38 cycles to the second year.

10) The annual cut-off grades 'CUTOFF' cannot be used if the the production year 'YR' is unknown. Therefore, an average 'AVEG' of the annual cut-off grades is used for the ore or waste decision in Line 10. The routine branches to statement 13, 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. The IF statement compares the mine's total life span 'L' with the production year 'YR'. The grid-block (I,J,K) is assumed to be mined in the first year by Line 12, if the production year 'YR' is greater than the mine life 'L'.

14) The IF statement in Line 14 compares the production

year 'YR' with the 'I' 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 'EQGRAD' 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 the ore 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 MP 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 PIT PRODUCTION SIMULATION in the first cycle of the iterative 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 unmined 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 spans 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 'YR' are unknown or exceed the life span 'L'. Therefore, the mean 'AVEG' of the annual cut-off grades 'CUTOFF' is employed to make the ore or waste decision. Additionally, these grid-blocks are assured 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 'M = 1' were substituted with 'M = 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 MP Programme 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: F(M), W, X, Y, and Z;

Selling Prices: S1(M), S2(M), . . . Sn(M); 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. The factors are chosen by the user to simulate the peculiar costing conditions that will be encountered by him. The following list suggests some of the factors which can be applied to adjust the standards:

TYPE 1:

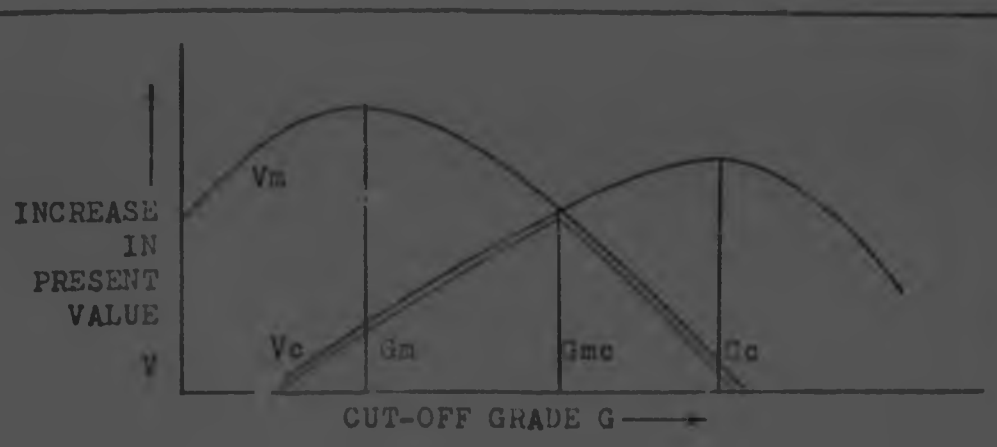


FIG. 6.4A

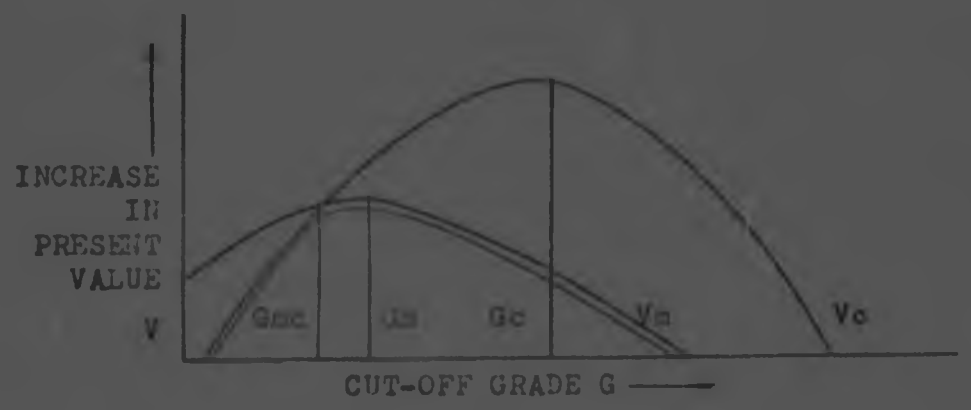


FIG. 6.4B

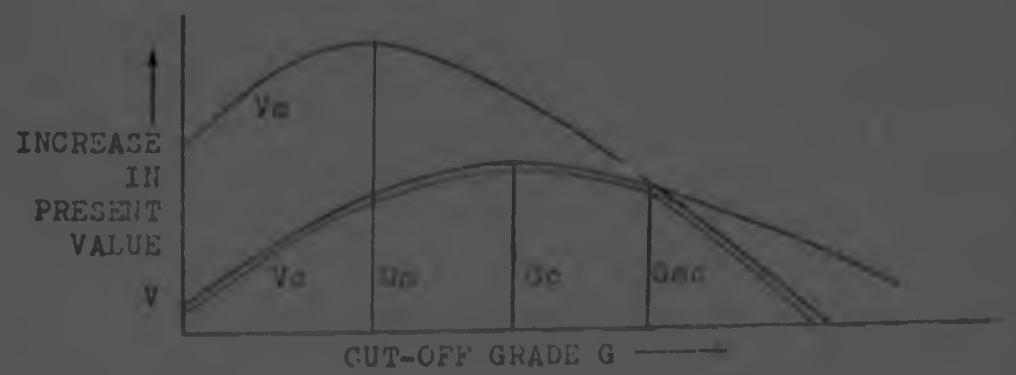


FIG. 6.4C

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.+INFRC)**M
```

where:

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

'M' 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 'M'. 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 3.1, 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 'OC' to get the additional cost of ore excavation at depth:

```
(K*OBCI)
```

where:

'K' is the vertical grid co-ordinate of the particular grid-block being evaluated.

'OBCI' 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 'K' grid-level. Similar factors can be devised for all the costing 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=OB*FCW(NN)
      Y=OR*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.

'OB' and 'OR' are the standard waste and ore excavation costs respectively.

'X' amd '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'.

```
      PLANE=0.22*I-J-(0.11*K)+10
      IF (ABS(PLANE-3.0) 30,30,31
```

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

```
      IF (FAULT(I,J,K)) 30,30,31
   30 NN=1
      GO TO 32
   31 NN=2
   32 X=OB(NN)
      Y=OR(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 few deposits 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 user'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:

```
      RAIL=10*I-J
      IF (RAIL) 10,10,11
   10 VAL(I,J,K)=10.0E-75
   11 CONTINUE
```

This routine causes 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 3:

Production penalties or rewards can be charged to the grid-blocks by identifying their minerals. For example, there 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 (GRADB(I,J,K)-0.005) 11,11,10
   10 W=REFC+CHARGE
      GO TO 12
   11 W=REFC
   12 CONTINUE
```

where:

'GRADB' 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 'GRADB' that interferes with production. The grid-block (I,J,K) is given a metallurgical cost penalty, 'CHARGE', if the proportional mineral content of 'GRADB' 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 coating 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 vary; consequently, the adjustments of the standards described in

this subsection must be tailored to the peculiar costing 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 'B', 'C', 'Y' and 'L' 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 'B', 'C' and 'L' 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 Lerchs-Grossman algorithm (5) outlined in Chapter 2. Recall from Section 6.2 that the objective is to find a pit volume 'V' of the maximum deposit 'P' that yields the maximum total profit from the three innermost integrals of the multiple integral:

$$PV = \int_T \left[\iint_R P(C(x,y,z),t)\,dx\,dy\,dz \right] dt$$

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

6.6.1 A DIRECTED GRAPH OF AN OPEN PIT MINE

The Lerchs-Grossman algorithm (5) uses the directed graph defined in Chapter 2. The following definitions relate the directed graph to the specific application in this study.

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 any 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 them 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. 6.6 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 difficulty 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 over-lapping 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. 6.7 shows how a cone can be used to simulate the arcs. 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 over-lapping cones that

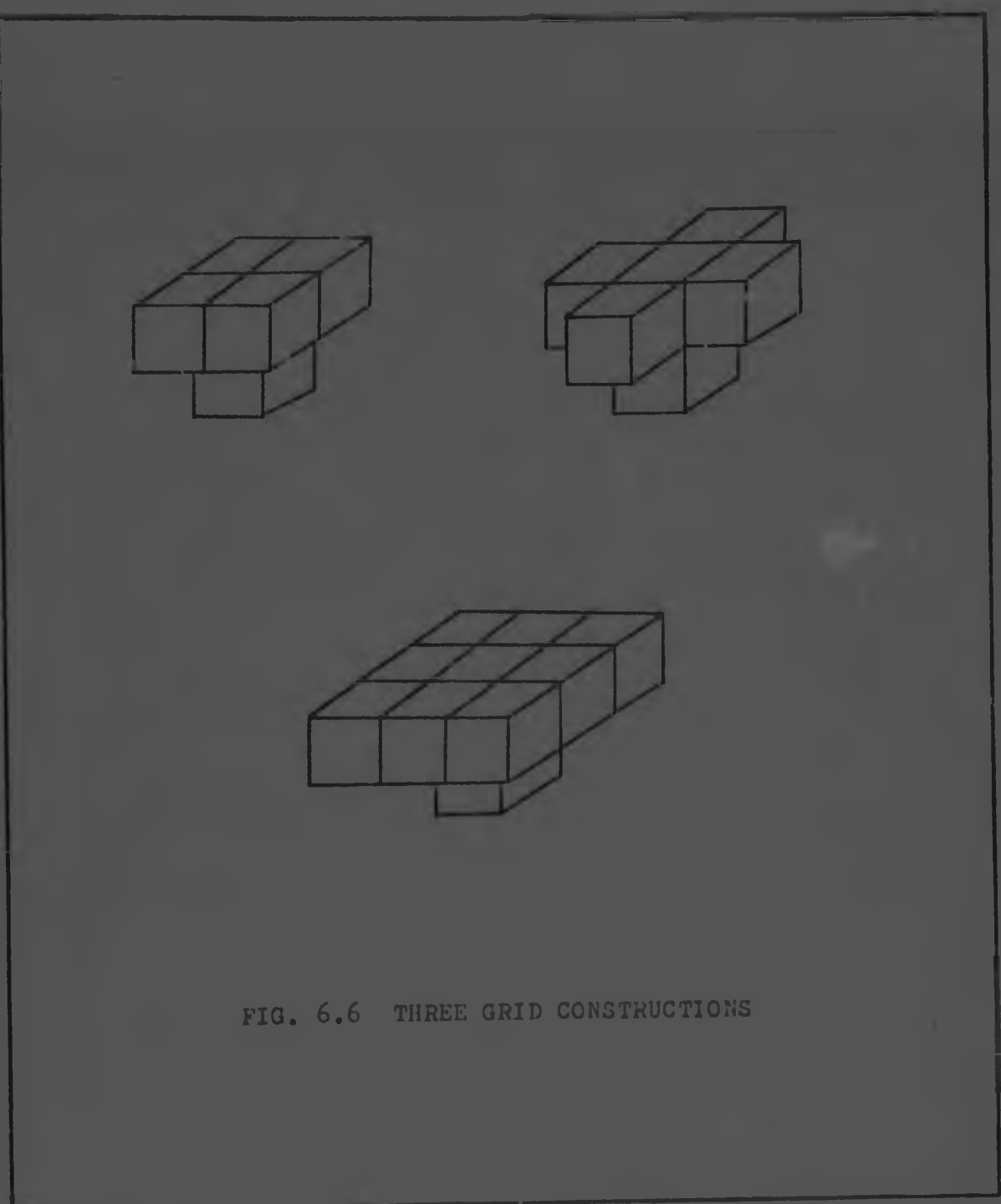


FIG. 6.6 THREE GRID CONSTRUCTIONS

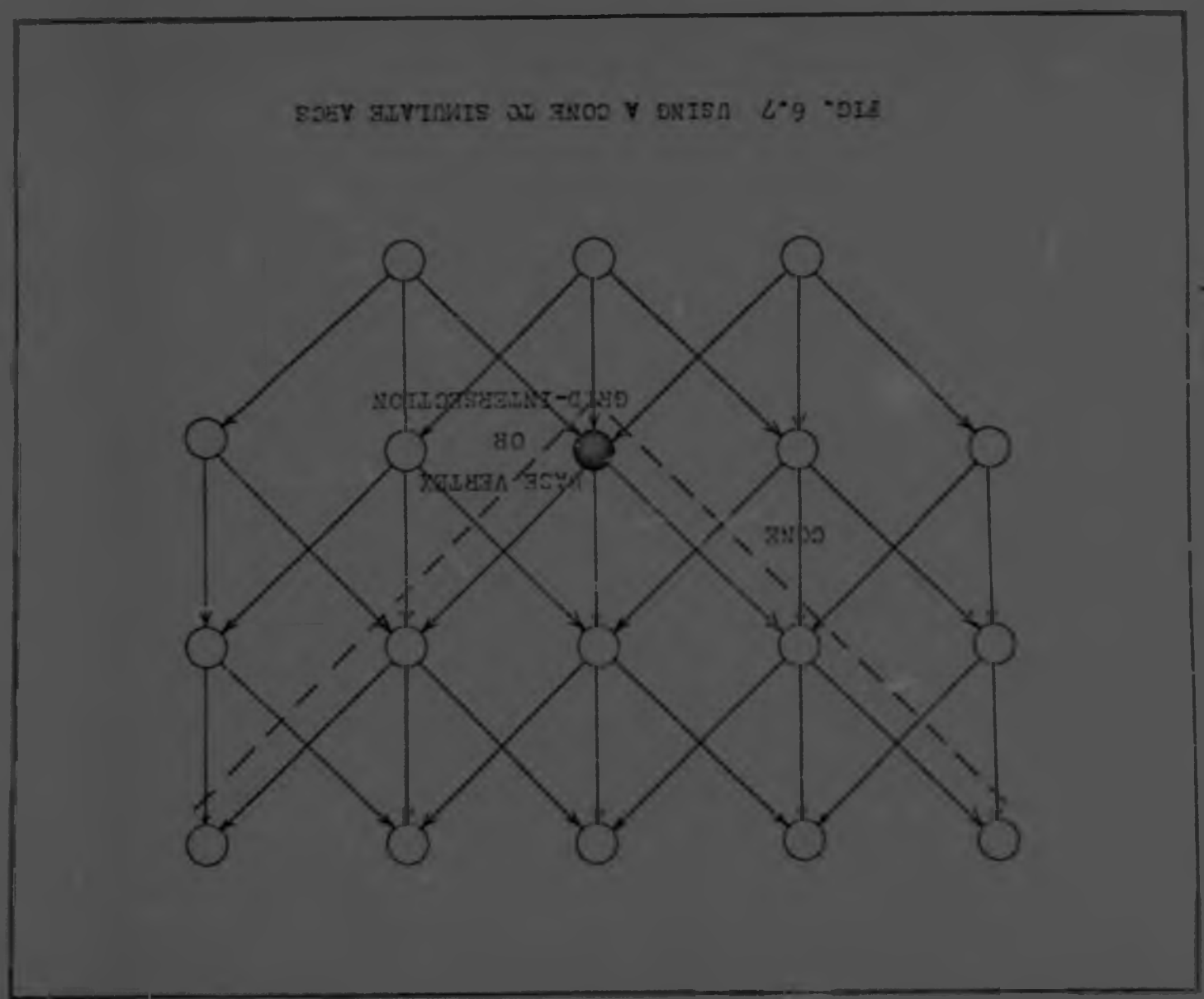


FIG. 6.7 USING A CONE TO SIMULATE ARCS

are a maximum closure of the directed graph. These cones are programmed by deriving a FORTRAN expression from the analytical equation of a cone, the grid co-ordinates of the vertices and the pit's safe wall- angles. This expression is:

```
CONE=SLOPE*((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 conditions. 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 constructing 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. 6.1. It is required to identify the vertices within the closure shown

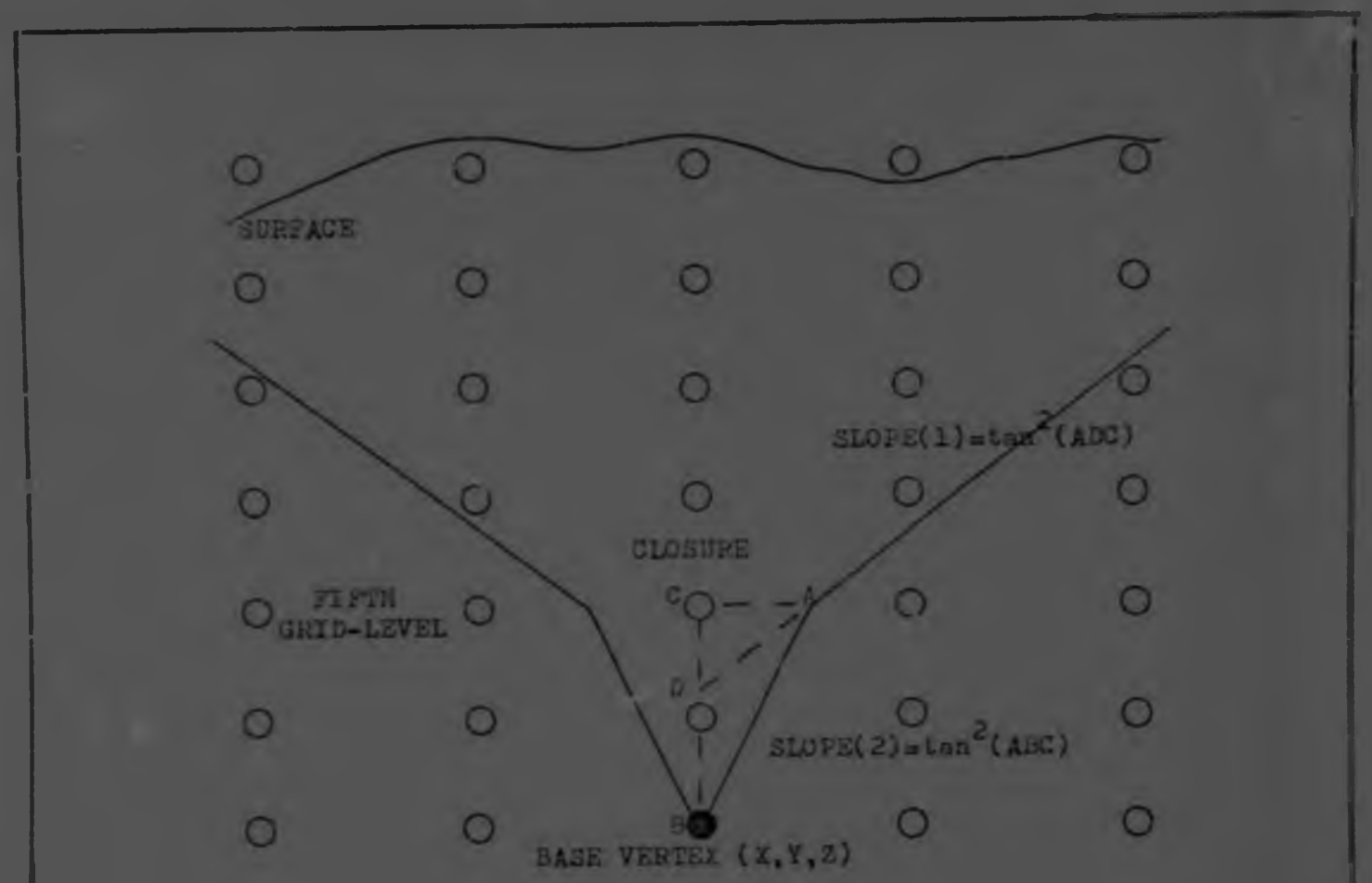


FIG. 6.8 A CROSS-SECTION OF A SEMI-CONE SET WITH TWO WALL-ANGLES

to expose the base vertex (X,Y,Z)• The vertices below the fifth grid-level are within the closure, if they satisfy the above 'CONE' expression using SLOPE = SLOPE(2)• The vertices above the water table are within the closure, if they satisfy the 'CONE' 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 trignometric 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,B,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 K, J, and I directions were 100 ft, 200 ft, and 100 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•3• It can be seen from the analogue that the grid-blocks have either an upward or downward force• The

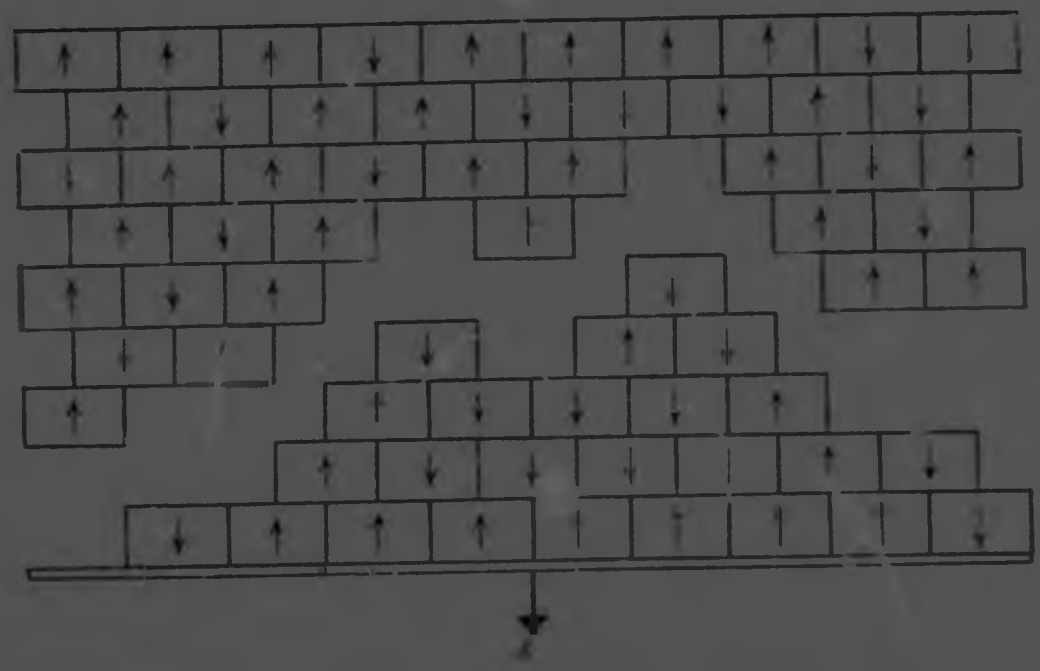


FIG. 6.9 MECHANICAL ANALOGUE OF THE LERCHS-GROSSMAN GRAPH ALGORITHM

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 Lerchs and Grossman.

STEP1:

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

STEP2:

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 totalled. 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.

An illustration of the algorithm can be seen in Fig. 6.10. Note that the directed graph is represented in arc form rather than semi-cone form. In Fig. 6.10B, the first positive vertex has a value of +1. This is the base of a test closure formed by including all vertices coming 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 closures shown in Fig. 6.10C and D are included in the maximum closure because they both have a total positive value. The test closure shown in Fig. 6.10E has three vertices (+7, -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. 6.11e has the highest value possible from the vertices with the constraints of the arcs.

6.7 PIT PRODUCTION SIMULATION

The routine PIT PRODUCTION SIMULATION performs the outermost integration of the multiple integral:

$$PV = \int_T \left[\iiint_R P(C(x,y,z),t)\,dx\,dy\,dz \right] dt \quad \text{(see Section 6.2)}.$$

In order to optimize the mine's present value, the path of this integration T is computed by the routines:

1) PLANT CAPACITIES SELECTION,

2) CUT-OFF GRADE COMPUTATIONS,

3) PRODUCTION TARGET COMPUTATIONS, and

the ore excavation sequence given by the user. Simulation of the pit production is essentially an exercise in developing a digital computer analogue to imitate the excavation sequence of the pit material

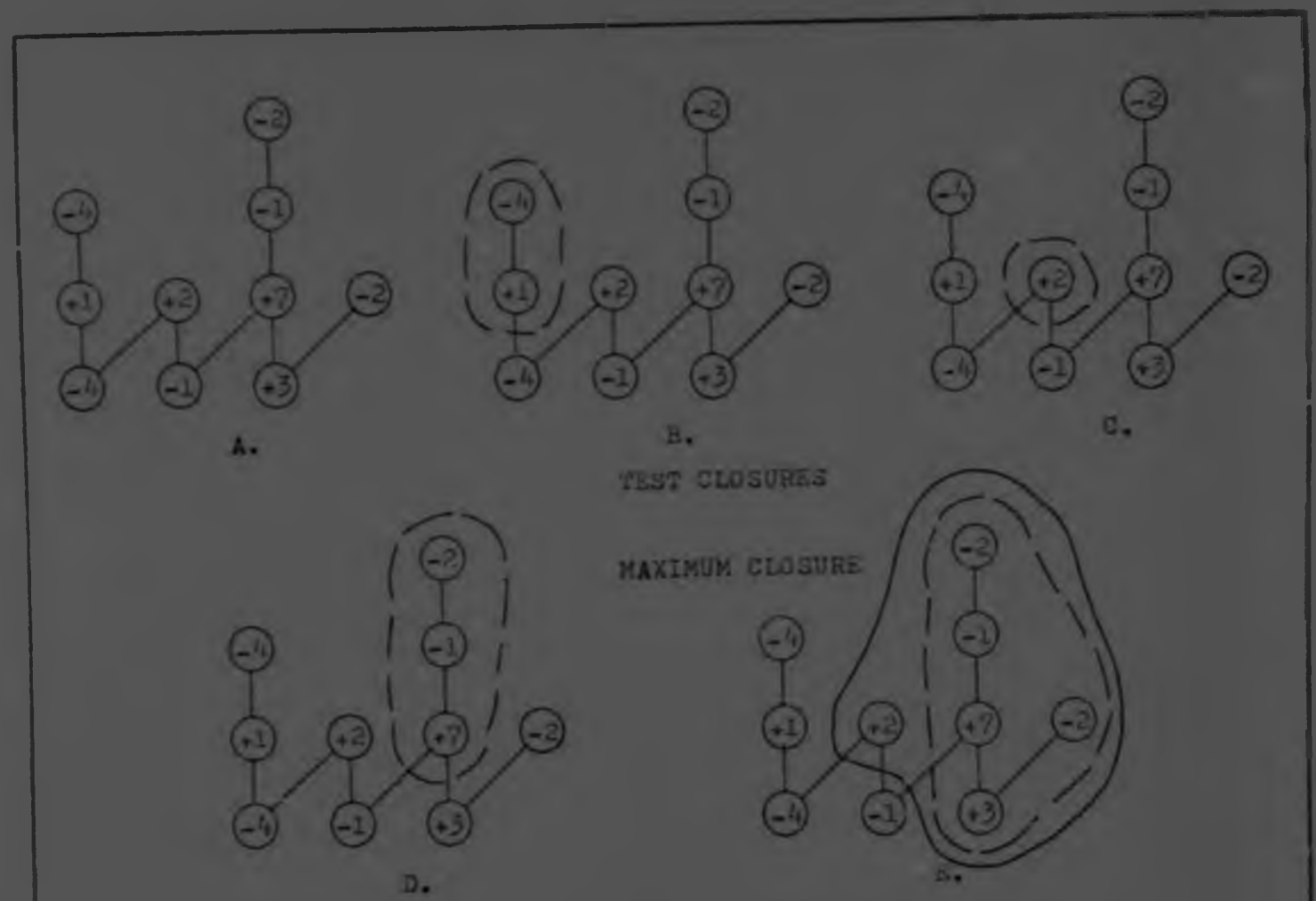


FIG. 6.10 AN ILLUSTRATION OF THE LERCHS-GROSSMAN GRAPH ALGORITHM

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 6.6 and continued in Table 6.7 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:

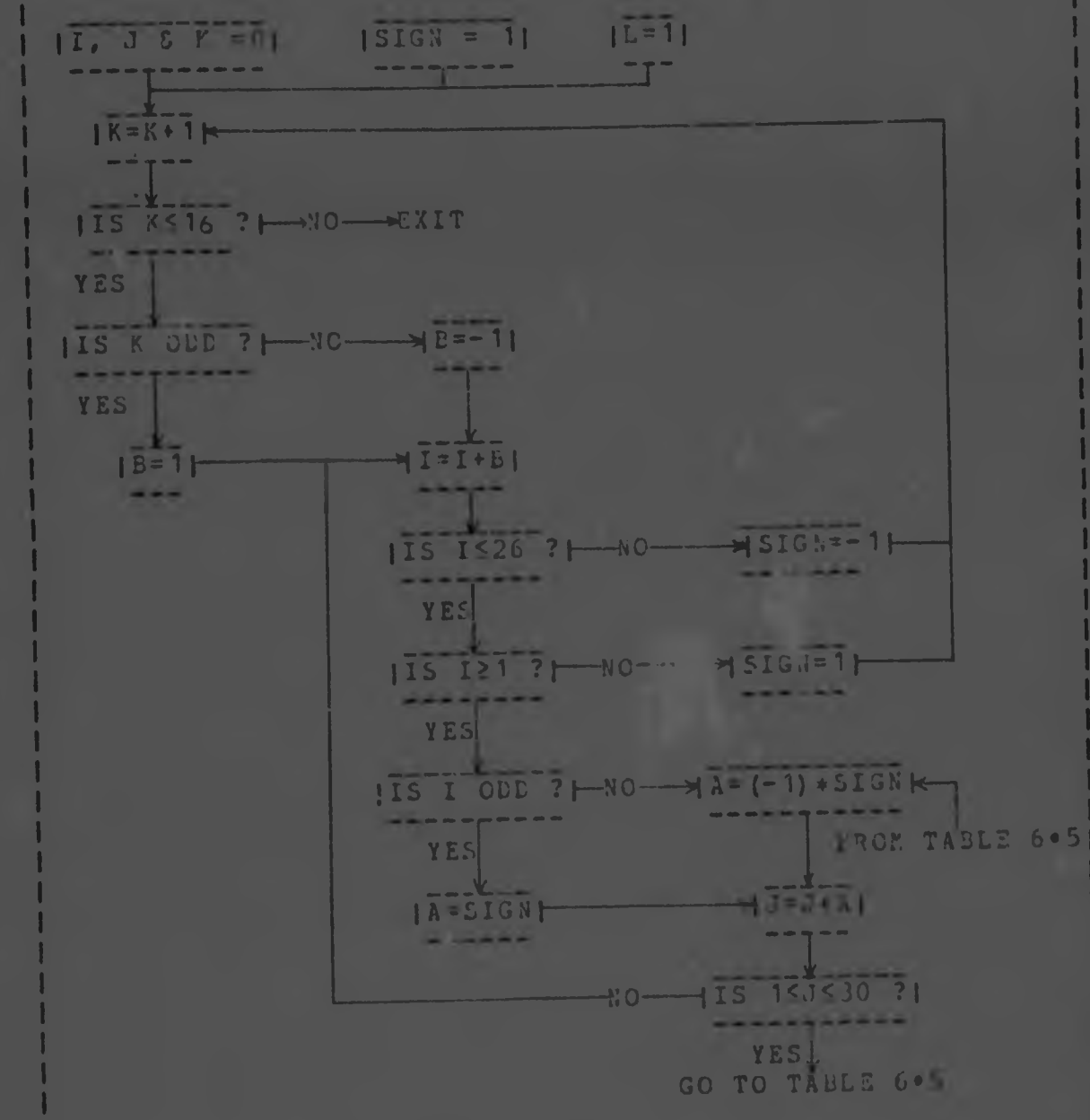


TABLE 6• 4

1) 'L' is the production year

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.

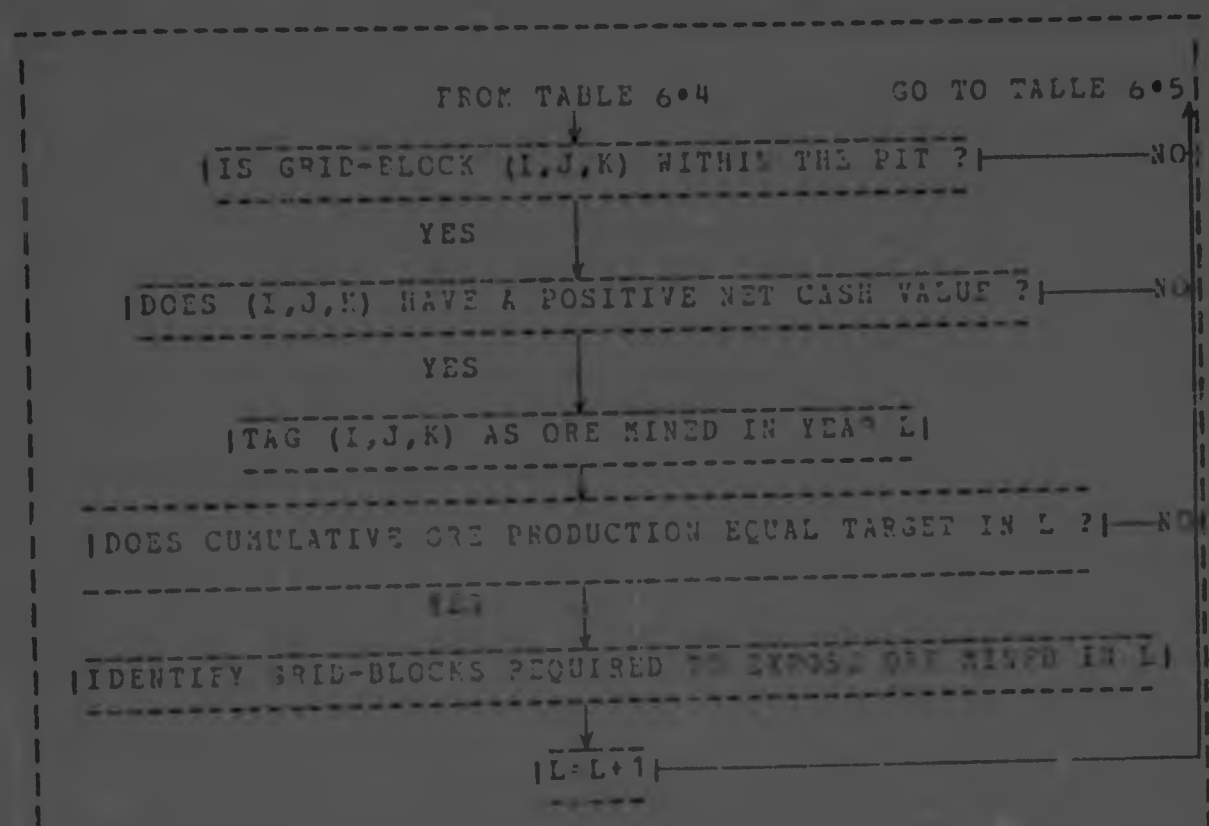


TABLE 6.5

The computations illustrated in table 6.4 define the grid co-ordinates of the next grid-block that should be found in the ore excavating sequence. Then, this grid-block is tested as illustrated in table 6.5. Firstly, it is determined whether the grid-block is within the final pit surface. Secondly, 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 by-passed 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 'L' 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 6.8 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 6.8) 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 6.6 define the grid co-ordinates of the next ore bench to be mined. Secondly, the latter part illustrates the following computations:

1) the ore grid-blocks on the particular ore bench (I,K) are identified and their J co-ordinate is stored in the array NC(N).

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.

Tables 6.7 and 6.8 are similar but not identical. In both Tables, the first 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.

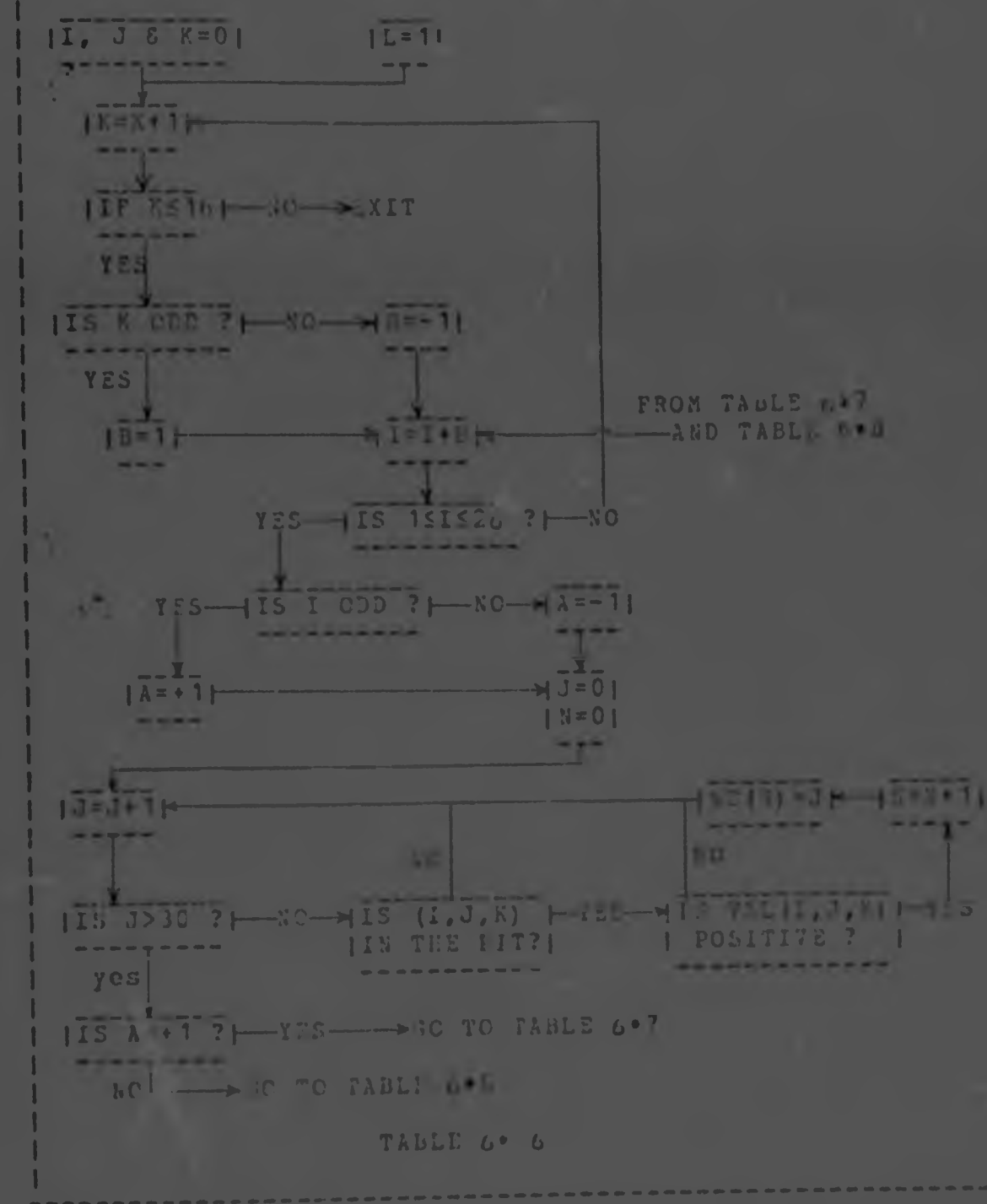


TABLE 6• 6

The two above examples illustrate how the hit production

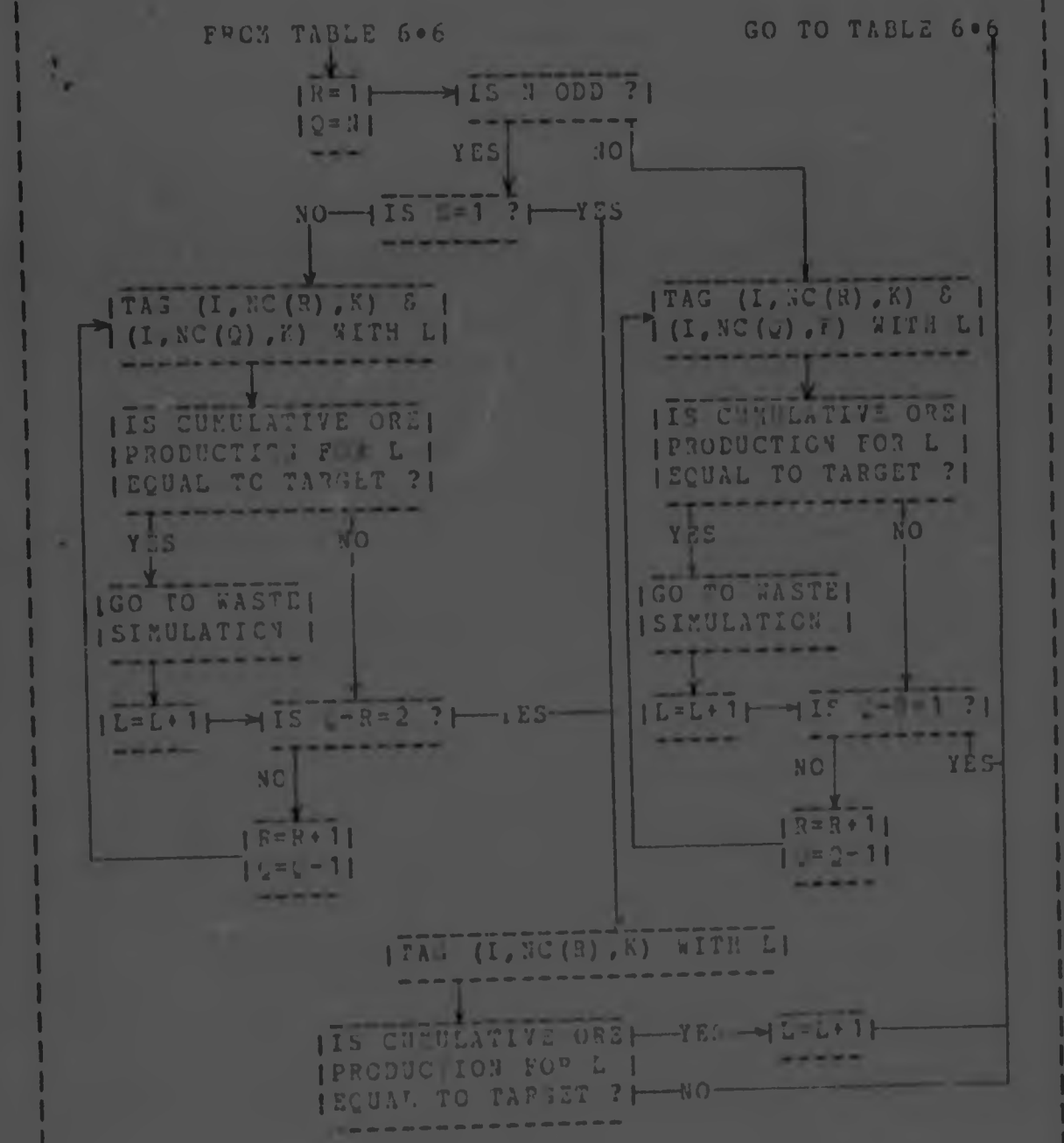


TABLE 6.7

simulation can be tailored to fit the user's ore excavation sequence. A variety of ore excavation sequences can be

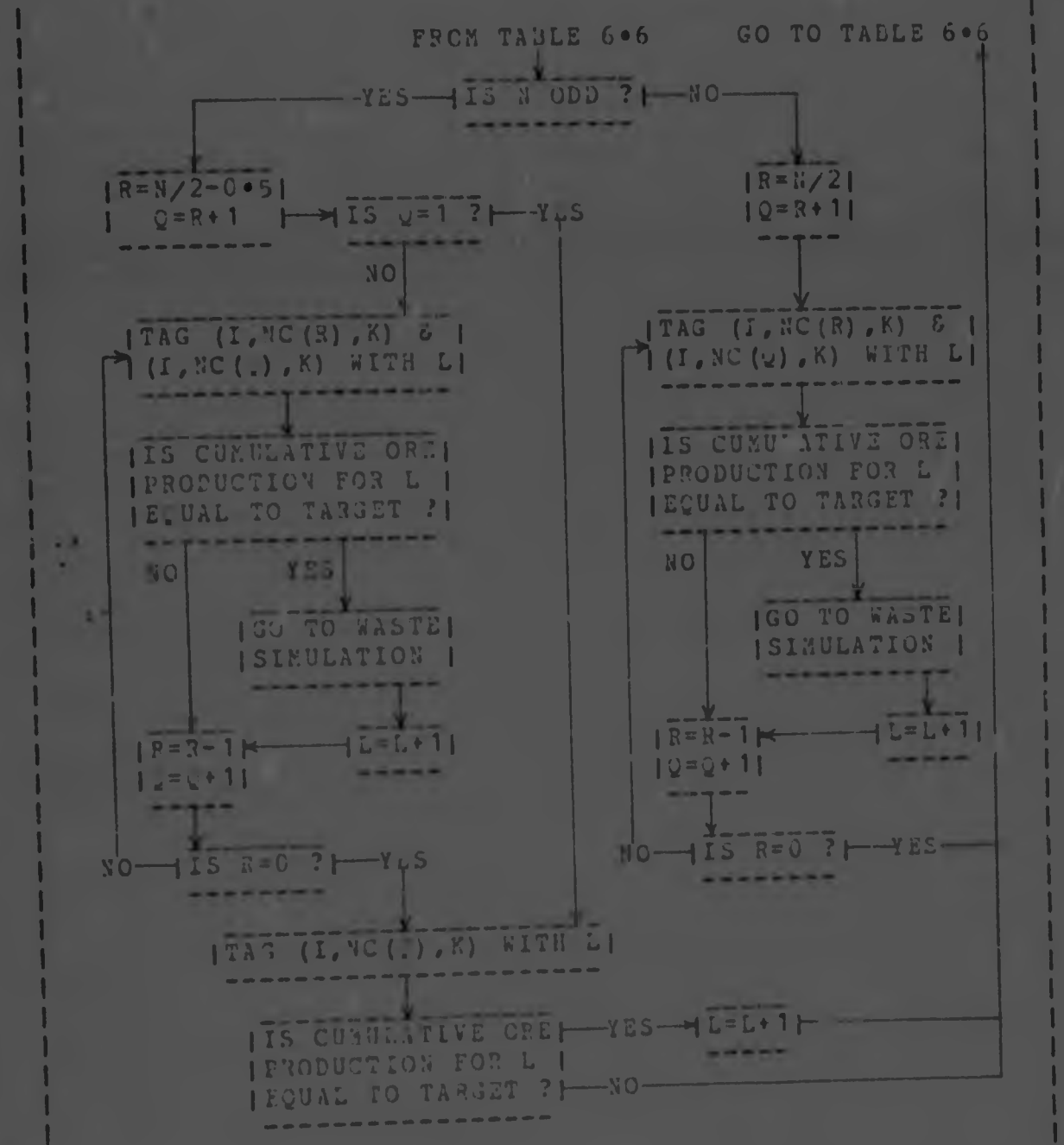


TABLE 6• 8

substituted into the MP Programme in a similar fashion.

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 3.3. 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:

```
CONE=WORKSL*((A(K-Z))**2)-((B(J-Y))**2)-((C(I-X))**2)
```

where:

'WO SL' 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 semicone 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-angle 'WORKSL' and the same constants (A,B,C) as described in Subsection 6.6.1.

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 3.3. The ore excavation sequence is programmed to recognize these features in order to take account of their occurrence. For example, a 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:

```
CONE=WORKSL*((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 semicone 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 'WORKSL' and the cone constants (A,B,C) as described in Subsection 6.6.1.

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 'YR'. 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 'WORKSL' 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 is 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 7.4.

The waste production simulation routine can be used for any surface deposit including those with variable working wall

-angles. 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 Programme. This example does not necessarily duplicate any existing mining conditions. It was devised to demonstrate the simulation techniques. The Programme'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 900.00 and 300.00 cash units per ton of end product. It is anticipated that these selling prices will inflate at a 1.1% 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 down to a depth of 250 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 500 ft. squares of the grid. The drilling progressed from the center of the deposit out to the limit of mineralization. Once the outer mineralization limit was found no further bore-holes were drilled.

The vertical cross-sections shown in Fig 7.2 were drawn from the exploration drilling. The primary mineral occurs within the outline shown by the dashed lines. The secondary mineral occurs below the past levels of the water table. Of all the geologic features reported from the bore holes, only one



CONTOURS AT 100FT. INTERVALS FROM THE SURFACE'S HIGHEST POINT DOWN

FIG. 7.1 PLAN MAP AND EXPLORATION DRILLING PATTERN OF THE EXAMPLE DEPOSIT

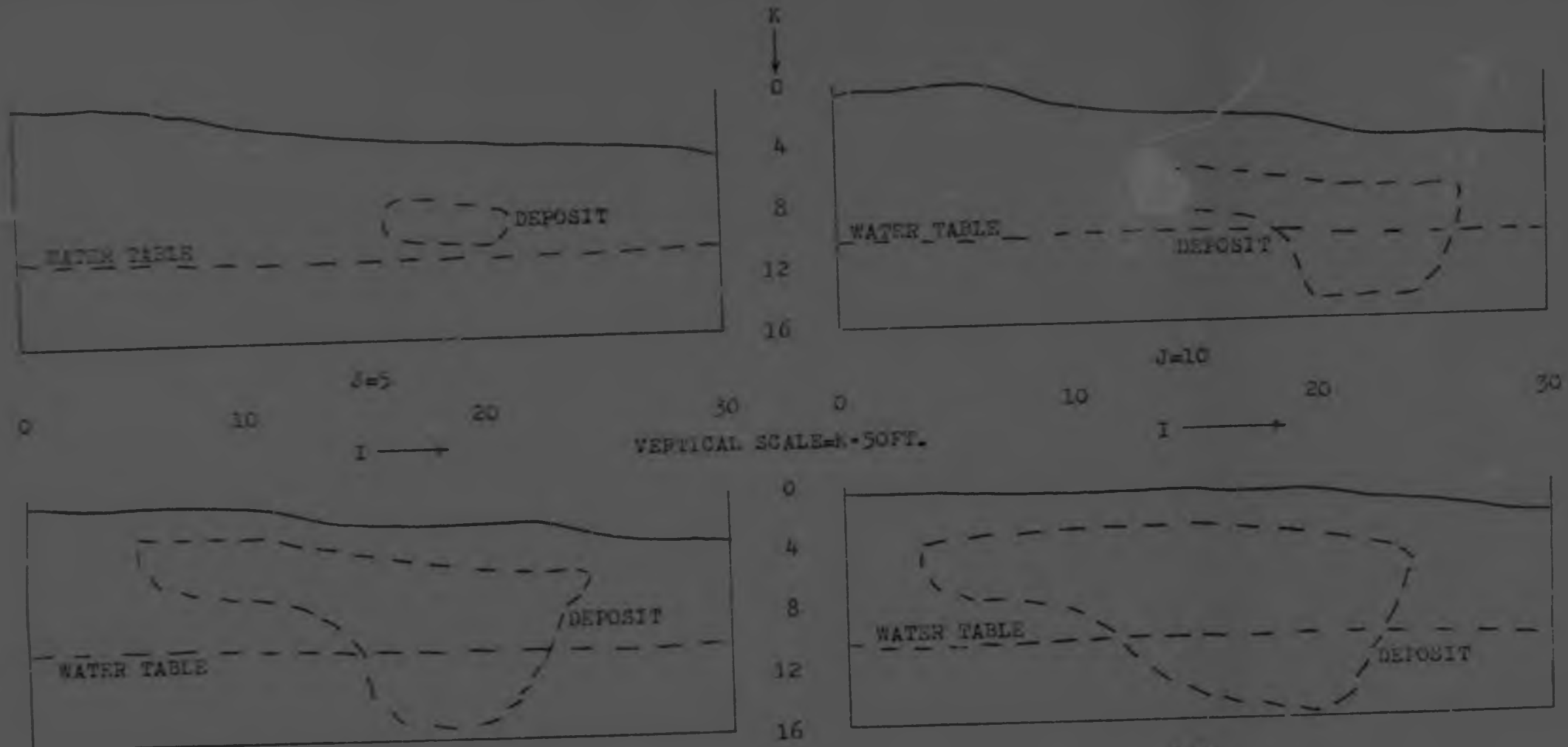


FIG. 7.2 CROSS-SECTIONS OF THE EXAMPLE DEPOSIT

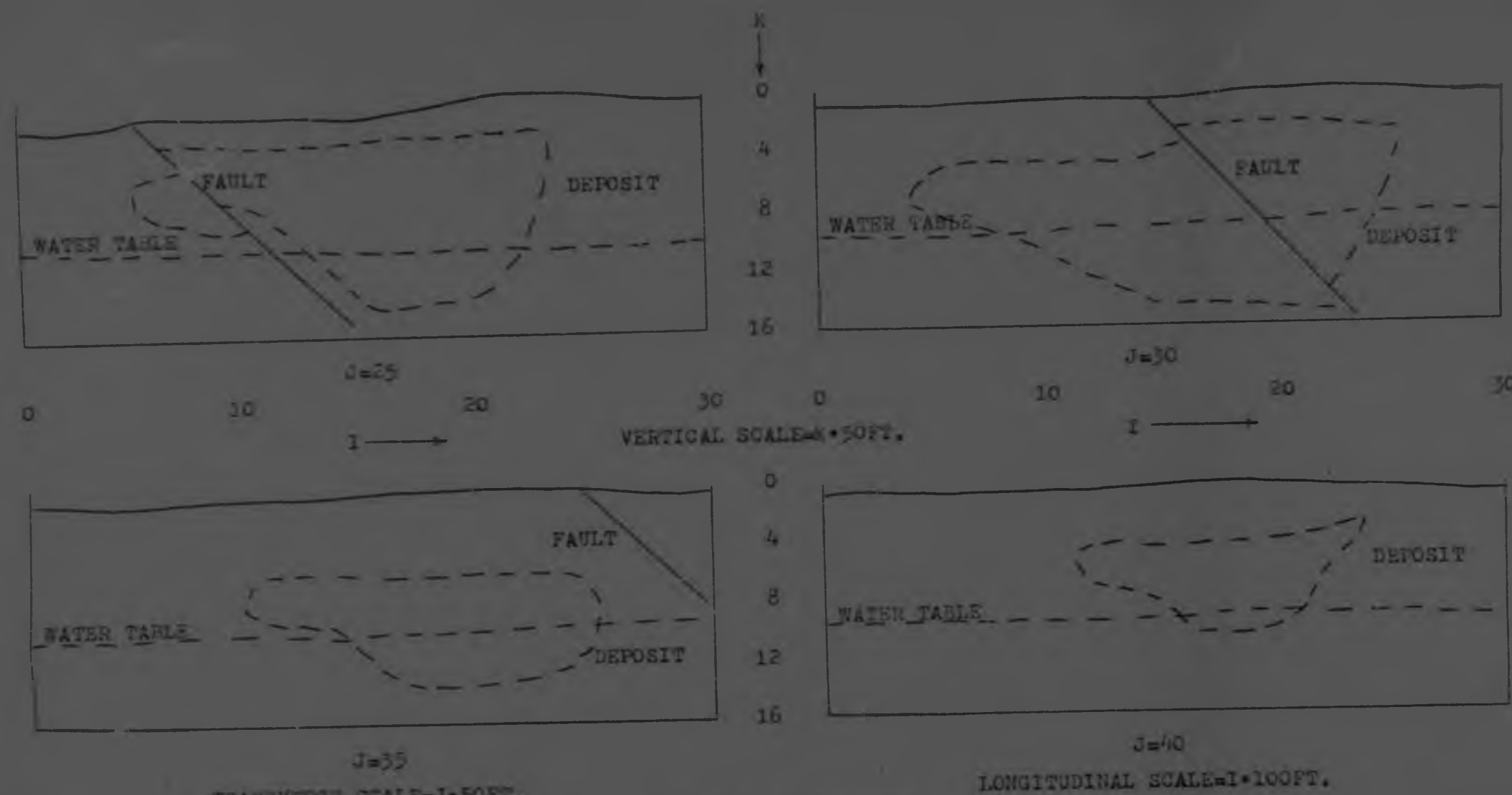


FIG. 7.2 (CONTINUED)

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 foot wall side of the fault are better than those on the hanging wall side. The minimum safe wall-angles as measured from vertical are 0.698 radians and 0.796 radians for the foot wall and hanging wall sides, respectively. Additionally, a metallurgical penalty will have to be imposed on the ore coming from a 300 ft. 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 concentration originating from this area require an additional refining expense. The standard refining cost is 49.50 cash units per ton of end product. Whereas, the adjusted refining cost is 50.00 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 3,520 tons and 3,700 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 minimum working space requirements. Above the water table the equipment requires a minimum working wall angle of 1.466 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•35¢ per ton above the water table to 0•39¢ per 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 strengths is accounted for with the following adjustments of the standards: The overburden excavation cost on any particular grid-level is equal to 0•0024¢ per ton above the water table or 0•0054¢ per ton below, multiplied by the grid-level number plus the respective standard excavation cost•

The ore 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 ore 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 ore excavation costs are 0•40¢ per ton plus 0•0030¢ per ton per grid-level above the water table and 0•50¢ per ton plus 0•0064¢ per ton per grid-level below•

The milling cost of the ore varies depending on whether it contains the secondary mineral• It is 0•82¢ per ton of mill input for ore without the secondary mineral and 0•93¢ 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•

THE CAPITAL COST OF 27 PLANT CAPACITY COMBINATIONS

OVERALL RECOVERY EFFICIENCY	REFINERY CAPACITY (T)	CONCENTRATOR CAPACITY (M) 1•5	1•6	1•7
85%	20•0	37•75	32•00	35•15
	22•5	32•50	25•65	42•00
	25•0	28•50	27•25	30•25
90%	20•0	37•50	26•85	31•85
	22•5	38•50	41•25	27•00
	25•0	34•75	40•50	38•25
95%	20•0	30•75	40•00	25•50
	22•5	25•75	41•00	37•75
	25•0	28•25	25•65	35•00

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 7• 1

The grid system used to simulate the deposit has rectangular axes which have regularly spaced grid co-ordinates• The distance between grid co-ordinates in the longitudinal direction is 100 ft• The transverse and vertical interval

distances are equal to the bench width and height which are 100 ft and 50 ft, respectively. Ore excavation comes 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 MP 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 'N' of the T-S mineral distribution's data set 'ORERES' 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:	S = 0.43¢ 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 MP 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 profits from its production targets discounted to the first production year less the capital costs illustrated in

THE NET PRESENT VALUE OF 27 PLANT CAPACITY COMBINATIONS
(FIRST ITERATION CYCLE)

OVERALL RECOVERY EFFICIENCY	REFINERY CAPACITY (T)	CONCENTRATOR CAPACITY (M) 1•5	1•6	1•7
	20•0	-	-23•55	-
85%	22•5	-		-
	25•0	-	-	-
	20•0	-	-	-
90%	22•5	-	-	-
	25•0	-	-	-
	20•0	-	-	-
95%	22•5	-	-	-
	25•0	-	-	-

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

Table 7•1• 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 deposit'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)

OVERALL RECOVERY EFFICIENCY	REFINERY CAPACITY (T)	CONCENTRATOR CAPACITY (M) 1•5	1•6	1•7
85%	20•0	-16•11	-8•41	-9•72
	22•5	-10•81	-2•01	-16•46
	25•0	-6•74	-3•21	-4•64
90%	20•0	-16•13	-3•50	-6•67
	22•5	-17•04	-17•86	-1•76
	25•0	-13•26	-16•80	-12•93
95%	20•0	-9•52	-16•83	-0•48
	22•5	-4•48	-17•82	-12•72
	25•0	-6•98	-2•17	-9•96

PIT CAPACITY=7 MILLION TONS PER YEAR

TABLE 7• 3

THE NET PRESENT VALUE OF 27 PLANT CAPACITY COMBINATIONS
(THIRD ITERATION CYCLE)

OVERALL RECOVERY EFFICIENCY	REFINERY CAPACITY (T)	CONCENTRATOR CAPACITY (T) 1•5	1•6	1•7
85%	20•0	-10•11	-2•56	-3•79
	22•5	-5•18	3•86	-10•54
	25•0	-1•20	2•16	1•05
90%	20•0	-10•13	2•35	-0•72
	22•5	-11•39	-11•99	4•21
	25•0	-7•58	-11•35	-7•16
95%	20•0	-3•60	-11•03	5•40
	22•5	1•19	-12•00	-6•82
	25•0	-1•13	3•33	-4•10

1.1 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 excepting 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•

THE NET PRESENT VALUE OF 27 PLANT CAPACITY COMBINATIONS

OVERALL RECOVERY EFFICIENCY	REFINERY CAPACITY (T)	CONCENTRATOR CAPACITY (M) (FOURTH ITERATION CYCLE) 1•5	1•6	1•7
85%	20•0	-10•20	-2•66	-3•84
	22•5	-5•26	3•78	0•58
	25•0	-1•29	2•06	•01
90%	20•0	-10•21	2•25	-0•77
	22•5	-11•43	-12•08	4•16
	25•0	-7•67	-11•44	-7•18
95%	20•0	-3•66	-11•12	5•34
	22•5	1•10	-12•10	-6•88
	25•0	-1•41	-13•24	-4•16

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/YEA
REFINERY CAPACITY (REF) = 20•0 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 85%

YR	COG (%)	PIT (M)	CON (M)	REF (T)	PROFIT (M)
1	•11	7•00	0•30	3•82	-0•52
2	•11	7•00	0•30	3•82	-0•50
3	•11	7•00	0•30	3•82	-0•52
4	•11	7•00	0•30	3•82	-0•55
5	•11	7•00	0•30	3•82	-0•57
6	•11	7•00	0•30	3•82	-0•59
7	•11	7•00	0•30	3•82	-0•62
8	•11	7•00	0•30	3•82	-0•64
9	•11	7•00	0•30	3•82	-0•67
10	•11	7•00	0•30	3•82	-0•69
11	•11	5•29	1•60	6•88	1•88
12	•11	3•25	1•60	6•01	2•08
13	•11	3•25	1•60	6•01	2•09
14	•11	3•25	1•60	6•01	2•09
-	-	-	-	-	-
63	•14	7•00	0•59	5•06	-1•65
64	•14	7•00	0•59	5•06	-1•71
65	•14	7•00	0•59	5•06	-1•78
66	•14	7•00	0•59	5•06	-1•84
67	•14	7•00	0•59	5•06	-1•91
68	•14	7•00	0•59	5•06	-1•97
69	•14	7•00	0•59	5•06	-2•04
70	•14	7•00	0•59	5•06	-2•11
71	•14	7•00	0•59	5•06	-2•18
72	•14	7•00	0•59	5•06	-2•26
73	•14	7•00	0•59	5•06	-2•33
74	•14	7•00	0•59	5•06	-2•41
75	•15	7•00	0•57	5•03	-2•49
76	•15	6•13	0•50	4•40	-2•44

PRESENT VALUE OF PROFITS LESS CAPITAL COST =
-17•05 MILLION CASH UNITS

TABLE 7• 6

CUT-OFF GRADES & TARGETS FOR OPTIMUM PLANT COMBINATION:
(SECOND ITERATION CYCLE)
PIT CAPACITY = 7•0 MILLION TONS/YEAR
CONCENTRATOR CAPACITY (CON) = 1•7 MILLION TONS/YEA
REFINERY CAPACITY (REF) = 20•0 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 95%

YR	COG (%)	PIT (M)	CON (M)	REF (T)	PROFIT (M)
1	•09	7•00	0•74	4•84	-0•01
2	•09	7•00	0•73	4•80	0•00
3	•09	7•00	0•65	4•47	-0•24
4	•09	7•00	0•65	4•47	-0•26
5	•09	4•26	1•70	6•28	1•72
6	•10	4•13	1•70	6•39	1•38
7	•10	4•13	1•70	6•39	1•89
8	•10	4•13	1•70	6•39	1•89
9	•10	4•06	1•70	6•38	1•92
10	•11	3•64	1•70	6•40	2•16
11	•11	3•64	1•70	6•40	2•16
12	•11	3•64	1•70	6•40	2•17
13	•11	3•64	1•70	6•40	2•17
14	•11	3•64	1•70	6•40	2•18
-	-	-	-	-	-
31	•12	1•77	1•70	7•98	5•40
32	•11	1•77	1•70	7•96	5•42
33	•11	1•86	1•70	6•95	4•16
34	•11	1•87	1•70	6•90	4•12
35	•11	1•87	1•70	6•90	4•15
36	•11	1•87	1•70	6•90	4•17
37	•11	1•87	1•70	6•76	4•00
38	•11	1•91	1•70	6•74	3•99
39	•11	1•92	1•70	6•74	4•01
40	•11	1•92	1•70	6•74	4•04
41	•11	1•92	1•70	6•7•	4•06
42	•11	1•92	1•70	5•36	2•28
43	•11	1•81	1•70	2•40	0•47
44	•11	0•83	0•78		

PRESENT VALUE OF PROFITS LESS CAPITAL COST =
-0•[illegible] MILLION CASH UNITS

TABLE 7• 7

CUT-OFF GRADES & TARGETS FOR OPTIMUM PLANT COMBINATION:
(THIRD ITERATION CYCLE)

PIT CAPACITY = 7•0 MILLION TONS/YEAR
CONCENTRATOR CAPACITY (CON) = 1•7 MILLION TONS/YEA
REFINERY CAPACITY (REF) = 20•0 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 95%

YR	COG (%)	PIT (M)	CON (M)	REF (T)	PROFIT (M)
1	•12	7•00	1•17	5•88	0•49
2	•12	7•00	1•17	5•88	0•41
3	•12	7•00	1•17	5•88	0•43
4	•12	7•00	1•41	6•65	0•96
05	•12	4•47	1•70	6•87	2•01
6	•12	4•47	1•70	6•87	2•03
7	•12	4•47	1•70	6•87	2•10
8	•11	4•41	1•70	6•81	2•14
9	•11	4•40	1•70	6•79	2•22
10	•13	4•48	1•70	6•61	2•06
11	•13	4•48	1•70	6•61	2•09
12	•13	4•48	1•70	6•61	2•14
13	•13	4•48	1•70	6•61	2•20
14	•13	4•48	1•70	6•61	2•24
-	-	-	-	-	-
30	•10	1•86	1•70	7•96	6•13
31	•09	1•75	1•70	7•27	5•42
32	•09	1•74	1•70	7•24	5•37
33	•09	1•74	1•70	7•24	5•43
34	•09	1•74	1•70	7•24	5•61
35	•09	1•74	1•70	7•24	5•70
36	•09	1•87	1•70	6•44	4•70
37	•09	1•83	1•70	6•31	4•60
38	•09	1•89	1•70	6•31	4•68
39	•09	1•85	1•70	6•21	4•63
40	•09	1•85	1•70	6•21	4•72
41	•08	1•85	1•70	6•15	4•63
42	•08	1•77	1•70	4•83	2•91
43	•08	0•37	0•36	1•02	-0•12

PRESENT VALUE OF PROFITS LESS CAPITAL COST =
5•40 MILLION CASH UNITS

TABLE 7• 8

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/YEA
REFINERY CAPACITY (REF) = 20•0 THOUSAND TONS/YEAR
OVERALL RECOVERY EFFICIENCY = 95%

YR	COG (%)	PIT (M)	CON (M)	REF (T)	PROFIT (M)
1	•12	7•00	1•17	5•88	0•49
2	•12	7•00	1•17	5•88	0•41
3	•12	7•00	1•17	5•88	0•43
4	•12	7•00	1•41	6•65	0•96
5	•12	4•47	1•70	6•87	2•01
6	•12	4•47	1•70	6•87	2•03
7	•12	4•47	1•70	6•87	2•10
8	•11	4•41	1•70	6•81	2•14
9	•11	4•40	1•70	6•79	2•22
10	•13	4•48	1•70	6•63	2•06
11	•13	4•48	1•70	6•61	2•09
12	•13	4•48	1•70	6•61	2•14
13	•13	4•48	1•70	6•61	2•20
14	•13	4•48	1•70	6•61	2•24
–	–	–	–	–	–
30	•10	1•86	1•70	7•96	6•13
31	•09	1•75	1•70	7•27	5•42
32	•09	1•74	1•70	7•24	5•37
33	•09	1•74	1•70	7•24	5•43
34	•09	1•74	1•70	7•24	5•61
35	•09	1•74	1•70	7•24	5•70
36	•09	1•87	1•70	6•44	4•70
37	•09	1•89	1•70	6•31	4•60
38	•08	1•85	1•70	6•21	4•57
39	•08	1•85	1•70	6•21	4•63
40	•08	1•85	1•70	6•21	4•71
41	•08	1•85	1•70	6•18	4•73
42	•08	1•77	1•70	4•83	2•01
43	•11	0•41	0•38	1•11	-0•35

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)

YR	PIT(E) CASH UNITS/ OUTPUT TON	CONCENTRATOR(C) CASH UNITS/ INPUT TON	REFINERY(R) CASH UNITS/ OUTPUT TON
1	0•41	1•02	50•363
2	0•43	1•06	50•446
3	0•43	1•06	50•463
4	0•43	1•07	50•480
5	0•42	1•07	50•497
6	0•43	1•07	50•515
7	0•43	1•07	50•533
8	0•43	1•07	50•551
9	0•42	1•07	50•563
10	0•43	1•07	50•575
11	0•43	1•08	51•483
12	0•43	1•08	51•458
13	0•43	1•08	51•738
14	0•44	1•09	52•254
15	0•44	1•08	51•593
16	0•44	1•08	51•942
-	-	-	-
28	0•45	1•11	53•110
29	0•45	1•11	53•232
30	0•45	1•11	53•357
31	0•45	1•12	53•483
32	0•46	1•15	55•075
33	0•47	1•17	55•502
34	0•46	1•13	54•151
35	0•46	1•14	54•627
36	0•46	1•13	54•186
37	0•46	1•14	54•377
38	0•46	1•14	54•518
39	0•46	1•14	54•985
40	0•46	1•14	54•720
41	0•46	1•15	54•856
42	0•48	1•18	55•815
43	0•48	1•18	56•009
44	0•48	1•17	55•414

TABLE 7•11

-evaluated• These pit surfaces for the four iteration cycles

AVERAGE UNIT PRODUCTION COSTS
(THIRD ITERATION CYCLE)

YR	PIT(E) CASH UNITS/ OUTPUT TON	CONCENTRATOR(C) CASH UNITS/ INPUT TON	REFINERY(R) CASH UNITS/ OUTPUT TON
1	0•41	1•01	50•350
2	0•37	0•99	51•511
3	0•37	1•01	52•284
4	0•38	1•02	52•643
5	0•39	1•04	53•325
6	0•39	1•06	54•125
7	0•40	1•07	55•056
8	0•41	1•09	55•962
9	0•41	1•10	57•140
10	0•42	1•12	57•753
11	0•42	1•14	58•897
12	0•43	1•15	59•431
13	0•44	1•17	60•247
14	0•45	1•19	61•255
15	0•45	1•21	62•510
16	0•46	1•23	63•357
-	-	-	-
27	0•55	1•45	74•204
28	0•56	1•47	75•858
29	0•57	1•52	76•893
30	0•58	1•52	78•086
31	0•59	1•54	79•125
32	0•65	1•64	79•880
33	0•70	1•72	81•720
34	0•69	1•72	82•818
35	0•72	1•77	83•578
36	0•71	1•77	84•598
37	0•71	1•80	85•867
38	0•76	1•96	87•218
39	0•77	1•88	89•132
40	0•78	1•91	90•696
41	0•80	1•94	91•750
42	0•82	1•97	93•126
43	0•48	1•18	56•009

TABLE 7•12

are illustrated in Fig 7•3• Each figure on the page is the

AVERAGE UNIT PRODUCTION COSTS
(FOURTH ITERATION CYCLE)

YR	PIT (E) CASH UNITS/ OUTPUT TON	CONCENTRATOR (C) CASH UNITS/ INPUT TON	REFINERY (R) CASH UNITS/ OUTPUT TON
1	0•41	1•01	50•350
2	0•37	0•99	51•511
3	0•37	1•01	51•284
4	0•38	1•02	52•643
5	0•39	1•04	53•325
6	0•39	1•06	54•125
7	0•40	1•07	55•056
8	0•41	1•09	55•962
9	0•41	1•10	57•140
10	0•42	1•12	57•753
11	0•42	1•14	58•897
12	0•43	1•15	59•431
13	0•44	1•17	60•247
14	0•45	1•19	61•255
15	0•45	1•21	65•510
16	0•46	1•23	63•357
-	-	-	-
27	0•55	1•45	74•204
28	0•56	1•47	75•858
29	0•57	1•52	76•893
30	0•58	1•52	78•086
31	0•59	1•54	79•125
32	0•65	1•64	79•880
33	0•70	1•72	81•720
34	0•69	1•72	82•818
35	0•72	1•77	83•578
36	0•71	1•77	84•598
37	0•71	1•80	85•867
38	0•76	1•86	87•218
39	0•77	1•88	89•132
40	0•78	1•91	90•696
41	0•80	1•94	91•750
42	0•82	1•97	93•126
43	0•82	1•52	94•839

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 intercepts at every other 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 these boundaries must be simulated as described in Example 3, Subsection 6.5.2. in which case, the pit design routine will insure that the pit closes 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 7.14 - 7.17. The annual overburden stripping costs listed in these tables are the expenses of exposing the ore. The annual revenue is the gross profits earned from the ore less its production costs. The pit production simulation uses the information in Tables 7.6 - 7.9 as targets. However, if the mining conditions are such that the targets cannot be met, they are then superseded, as explained in Subsection 6.7.2. 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 grade and concentrator input -- information already presented.

Finally, there are two last items of information used to monitor the iterative solution that are printed out with the production schedules. Firstly, there is the number of pit increments in the T-S mineral distribution. This turns out 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:

OPTIMUM PIT SURFACE

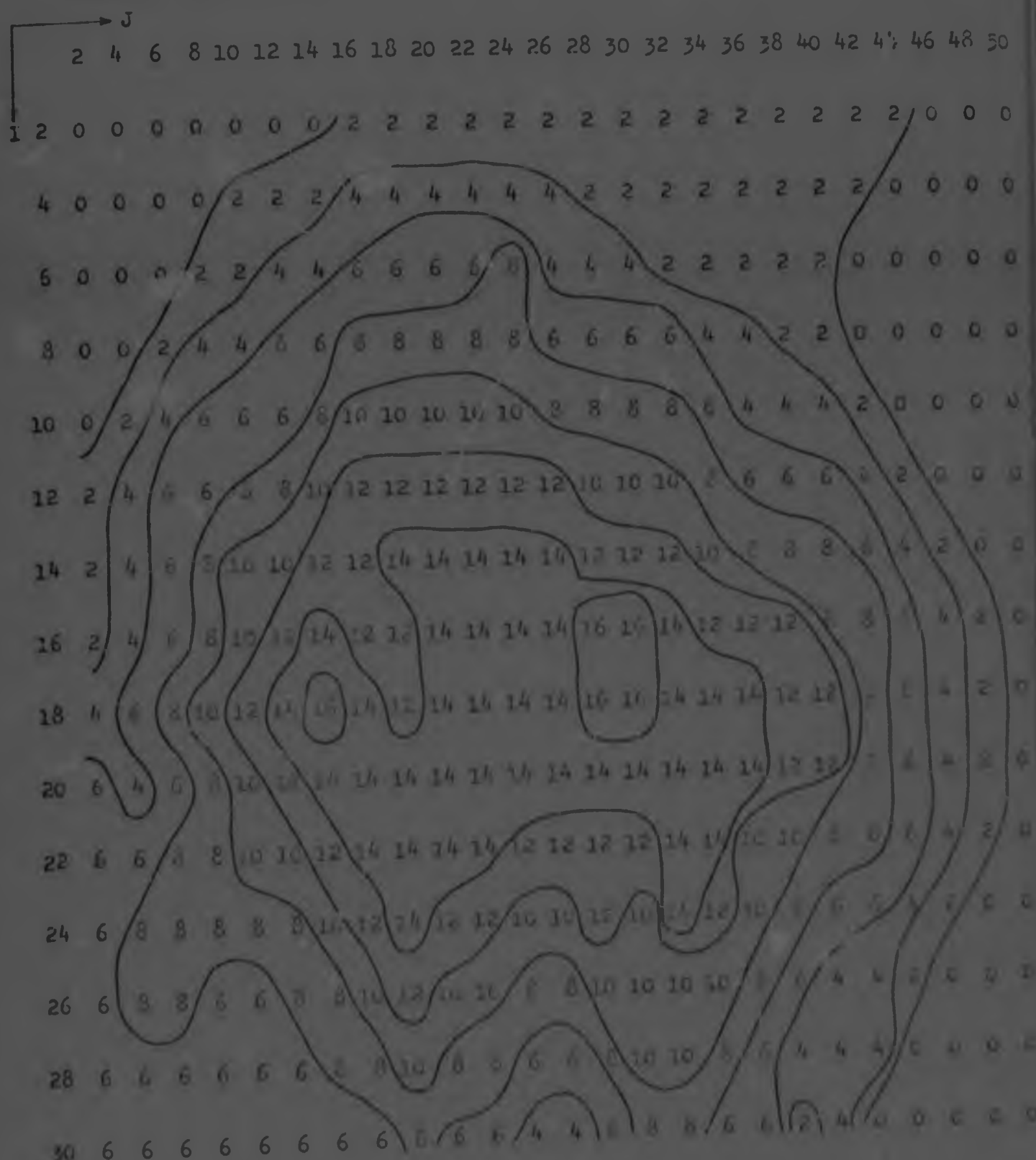


FIG. 7.3A THE FINAL SURFACE OF THE PIT (FIRST ITERATION CYCLE)

OPTIMUM PIT SURFACE

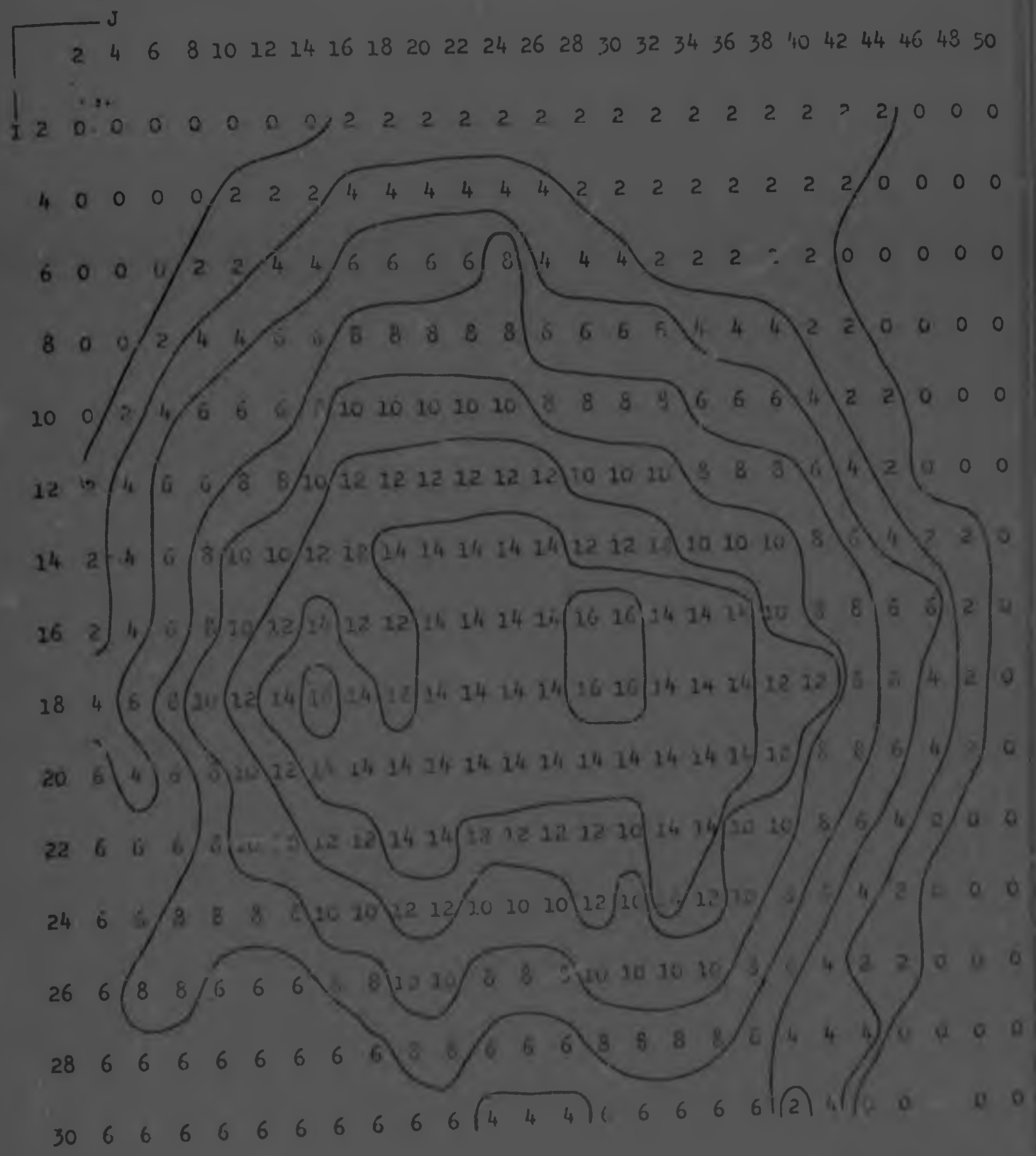


FIG. 7.3B THE FINAL SURFACE OF THE PIT (SECOND ITERATION CYCLE)

OPTIMUM PIT SURFACE

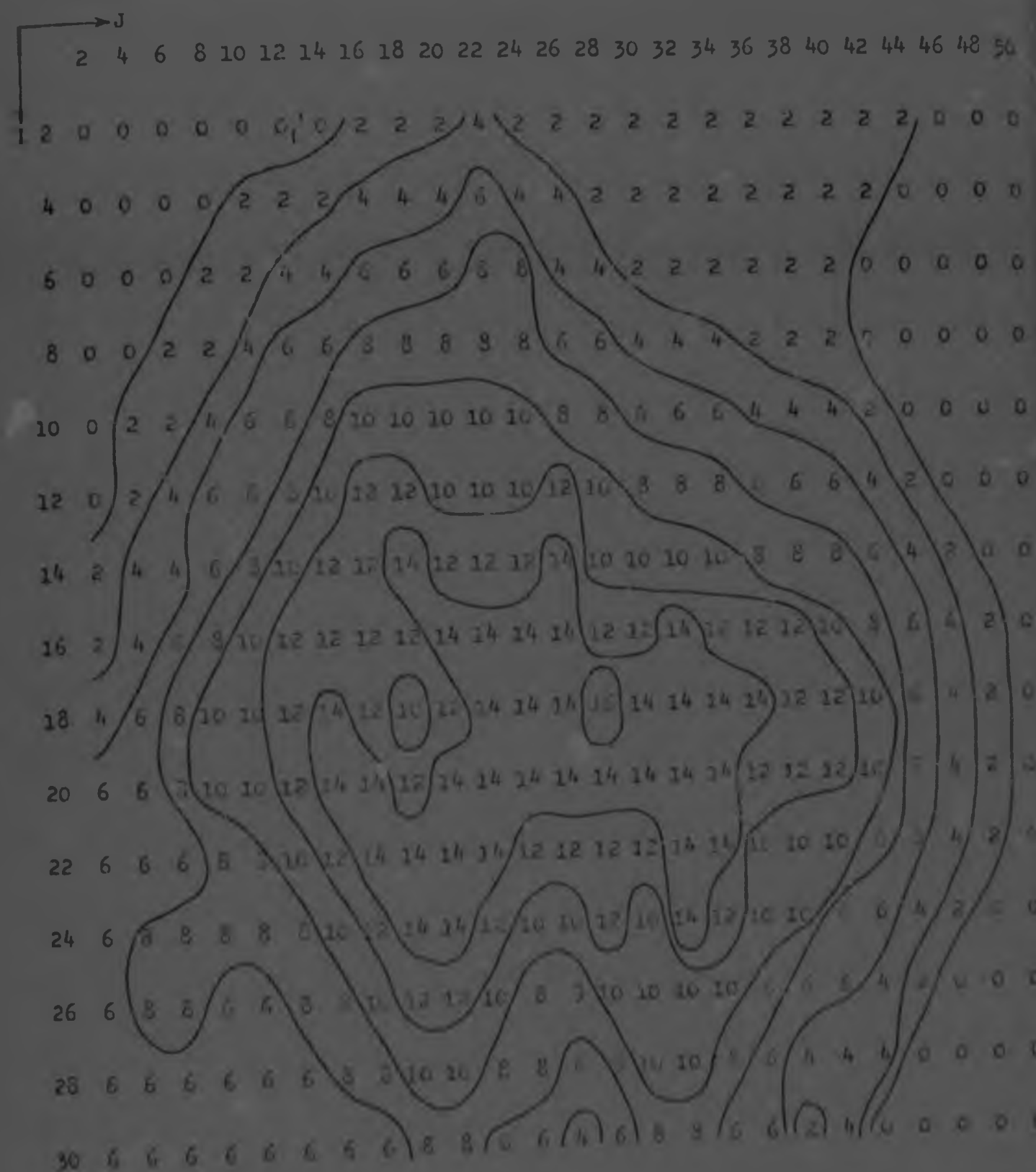


FIG. 7.3C THE FINAL SURFACE OF THE PIT (THIRD ITERATION CYCLE)

CPTIMUM PIT SURFACE

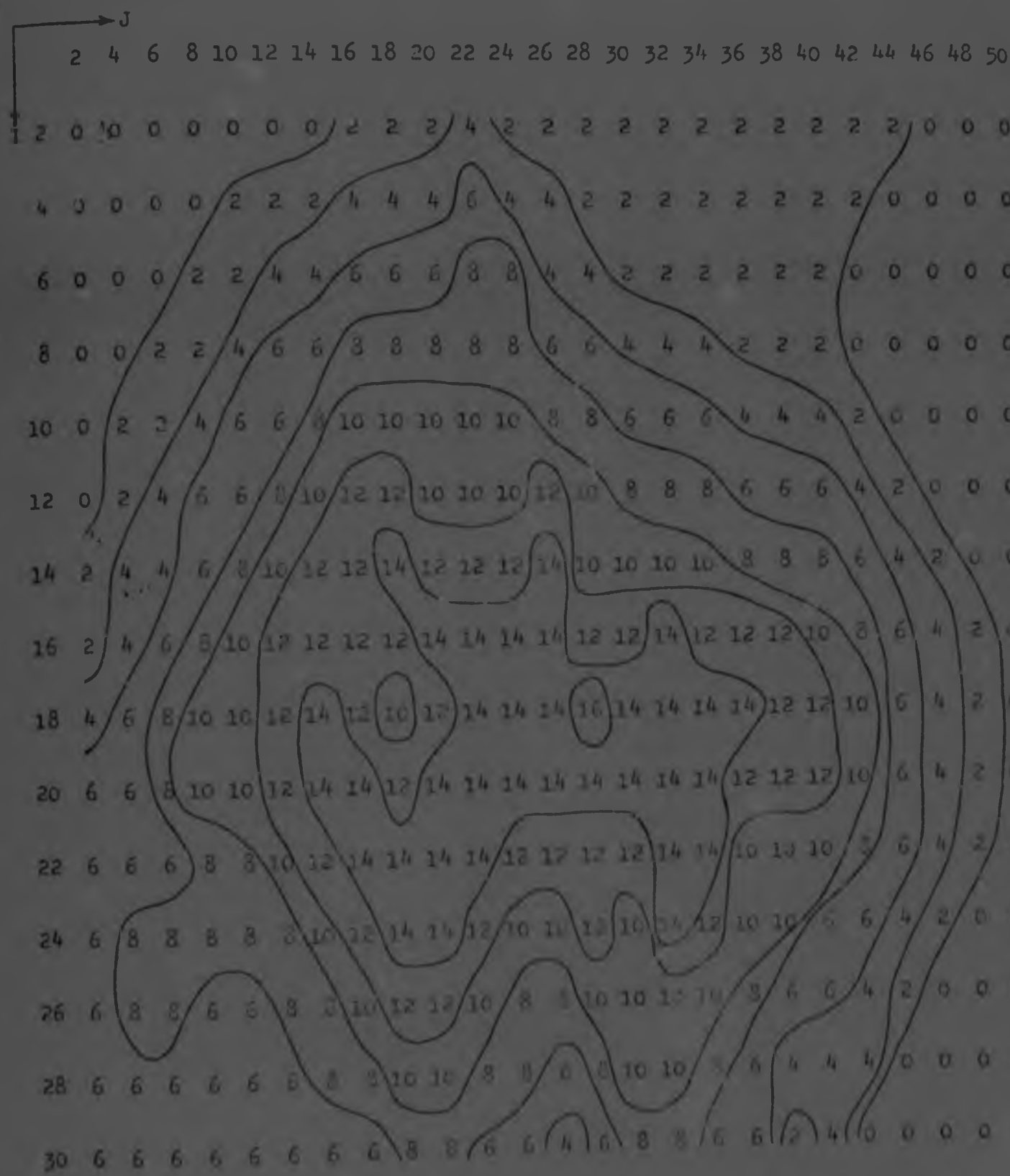


FIG. 7. 3D THE FINAL SURFACE OF THE PIT (FOURTH ITERATION CYCLE)

SIMULATED FIT PRODUCTION SCHEDULE(FIRST ITERATION CYCLE)

YR	OVERBURDEN M TONS	STRIPPING COSTS M CASH UNITS	ORE M TONS	REVENUE M CASH UNITS
1	6•34	2•74	0•28	0•07
2	0•99	0•43	0•28	0•10
3	0•00	0•00	0•28	0•14
4	1•41	0•61	0•28	0•14
5	3•10	1•34	0•28	0•07
6	0•00	0•00	0•28	0•07
7	0•28	0•12	0•28	0•06
8	0•84	0•36	0•28	0•03
9	6•61	2•87	0•28	0•07
10	6•05	2•63	0•28	0•07
11	4•08	1•80	1•55	1•60
12	1•59	0•75	1•55	1•71
13	1•69	0•75	1•55	1•95
14	2•53	1•03	1•55	1•70
15	2•11	0•85	1•55	1•50
16	1•69	0•72	1•55	1•81
-	-	-	-	-
35	0•28	0•01	1•55	3•59
36	0•28	0•13	1•55	5•65
37	0•00	0•00	1•55	3•33
38	0•00	0•00	1•55	2•24
39	0•14	0•03	1•55	2•00
40	0•00	0•00	1•55	2•75
41	0•00	0•00	1•58	2•58
42	0•28	0•07	•63	2•20
43	0•42	0•09	1•63	2•18
44	0•28	0•13	1•63	3•54
45	0•42	0•10	1•63	3•27
46	0•00	0•00	1•63	2•70
47	0•00	0•00	1•63	2•78
48	1•18	0•56	1•63	2•22
49	0•15	0•03	1•63	1•73
50	0•00	0•00	1•63	3•50
51	0•00	0•00	1•63	1•65
52	0•15	0•09	0•74	0•46

TOTAL REVENUE=109•91 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREMENTS=11

TABLE 7•14

SIMULATED PIT PRODUCTION SCHEDULE (SECOND ITERATION CYCLE)

YR	OVERBURDEN M TONS	STRIPPING COSTS M CASH UNITS	ORE M TONS	REVENUE M CASH UNITS
1	6•20	2•68	0•70	0•47
2	5•3	2•58	0•70	0•39
3	1•13	0•60	0•70	0•24
4	7•3	4•04	0•70	0•27
5	2•3	1•37	1•69	2•28
6	4•2	2•43	1•69	2•68
7	2•5	1•50	1•69	2•63
8	2•5	1•52	1•69	2•58
9	3•1	1•73	1•69	2•38
10	2•25	1•19	1•69	3•37
11	2•25	1•46	1•69	4•00
12	3•38	2•37	1•69	2•11
13	4•93	3•25	1•69	2•40
14	2•82	2•05	1•69	2•11
15	2•63	1•98	1•69	1•98
16	1•13	0•73	1•69	4•73
-	-	-	-	-
26	0•42	0•36	1•69	5•91
27	0•56	0•61	1•69	8•14
28	1•41	0•06	1•69	7•10
29	0•42	0•09	1•69	4•45
30	0•00	0•00	1•69	2•88
31	0•00	0•00	1•69	4•41
32	0•20	0•09	1•75	3•67
33	0•00	0•00	1•63	3•84
34	0•27	0•27	1•63	4•84
35	0•00	0•00	1•63	4•24
36	0•42	0•20	1•63	3•87
37	0•71	0•31	1•63	3•35
38	0•44	0•22	1•63	2•64
39	0•89	0•31	1•63	2•72
40	0•30	0•04	1•63	3•72
41	0•15	0•08	1•63	1•53
42	0•00	0•00	0•44	0•54

TOTAL REVENUE=148•05 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREMENTS=9

TABLE 7•15

1) annual topographic maps of the pit, for example the

SIMULATED PIT PRODUCTION SCHEDULE (THIRD ITERATION CYCLE)

YR	OVERBURDEN M TONS	STRIPPING COSTS M CASH UNITS	ORE M TONS	REVENUE M CASH UNITS
1	6•05	2•61	1•13	0•53
2	5•77	2•53	1•13	0•56
3	6•05	2•78	1•13	0•68
4	6•76	3•33	1•41	2•14
5	3•10	1•59	1•69	2•33
6	2•68	1•42	1•69	2•26
7	3•10	1•51	1•69	2•08
8	2•68	1•28	1•69	2•31
9	2•82	1•49	1•69	3•33
10	2•68	1•50	1•69	3•84
11	2•53	1•35	1•69	2•52
12	2•96	1•32	1•69	2•32
13	2•96	1•74	1•69	2•67
14	2•25	1•34	1•69	4•02
15	0•56	0•38	1•69	3•68
16	0•28	0•21	1•69	6•51
-	-	-	-	-
25	0•84	0•65	1•69	4•79
26	0•00	0•00	1•69	8•79
27	0•14	0•14	1•69	5•89
28	0•84	0•40	1•69	2•06
29	0•28	0•12	1•69	3•59
30	0•14	0•03	1•69	4•23
31	1•41	0•05	1•73	2•97
32	0•00	0•00	1•63	3•27
33	1•83	1•01	1•63	3•03
34	0•00	0•00	1•63	5•81
35	0•00	0•00	1•63	3•70
36	0•14	0•01	1•63	3•93
37	0•28	0•13	1•63	3•77
38	0•30	0•27	1•63	3•27
39	0•00	0•00	1•63	4•73
40	0•00	0•00	1•63	3•24
41	0•00	0•00	0•30	0•29

TOTAL REVENUE=14 •51 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREMENTS=9

TABLE 7•16

five-yearly topographic map illustrated in Fig 7•4•

SIMULATED PIT PRODUCTION SCHEDULE (FOURTH ITERATION CYCLE)

YR	OVERBURDEN M TONS	STRIPPING COSTS M CASH UNITS	ORE M TONS	REVENUE M CASH UNITS
1	6•05	2•61	1•13	0•59
2	5•77	2•53	1•13	0•56
3	6•05	2•78	1•13	0•68
4	6•76	3•33	1•41	2•14
5	3•10	1•59	1•69	2•93
6	2•68	1•42	1•69	2•26
7	3•10	1•51	1•69	2•08
8	2•68	1•28	1•69	2•31
9	2•82	1•49	1•69	3•33
10	2•68	1•50	1•69	3•84
11	2•53	1•35	1•69	2•52
12	2•36	1•32	1•69	2•32
13	2•96	1•74	1•69	2•67
14	2•25	1•34	1•69	4•02
15	0•56	0•38	1•69	3•68
16	0•28	0•21	1•69	6•51
-	-	-	-	-
25	0•84	0•66	1•69	4•70
26	0•00	0•00	1•69	8•79
27	0•14	0•14	1•69	5•89
28	0•84	0•40	1•69	2•96
29	0•28	0•12	1•69	1•59
30	0•14	0•03	1•69	4•23
31	0•14	0•01	1•73	2•37
32	0•00	0•00	1•63	1•27
33	1•83	1•01	1•63	3•03
34	0•00	0•00	1•63	5•81
35	0•00	0•00	1•63	3•70
36	0•13	0•01	1•63	3•93
37	0•28	0•13	1•63	3•76
38	0•30	0•27	1•63	3•26
39	0•00	0•00	1•63	4•79
40	0•00	0•00	1•63	3•24
41	0•00	0•00	0•30	0•29

TOTAL REVENUE=148•50 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREMENTS=0

TABLE 7•17

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 MP Programme.

Finally, it should be pointed out that the MP Programme can be used to study constrictions in the production. These constrictions are identified in the output illustrated in Table 7.9. In this example, note that the pit constricts 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 constrictions by specifying a more compatible combination of plant capacities and pit production simulation. 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 of the preceding Section is evaluated economically to measure its profitability. The revenue (RLV) and overburden stripping costs (OS) of Table 7.17 are analyzed in the cash flow of Table 7.18. The cash flow out is the equity capital (ECAP). The cash flow (CFI) in is the revenue less the overburden stripping costs, the tax and the interest (IST) on loan capital (LC). The interest rate on loan capital is 7.5%. The opportunity cost of investment capital is 5%. The profitability of the total cash flow (CF) is measured by computing its present value (PV) and discounted cash flow yield. The PV entries are the present value in the given year of the future profits. The cash flow tabulation has no entry for fixed or administrative costs because they have already been accounted for in the revenue and overburden stripping costs by the standard costing method used in the MP Programme. The taxes are computed according to the South African tax structure.

TOPOGRAPHIC MAP OF PIT IN YEAR 1

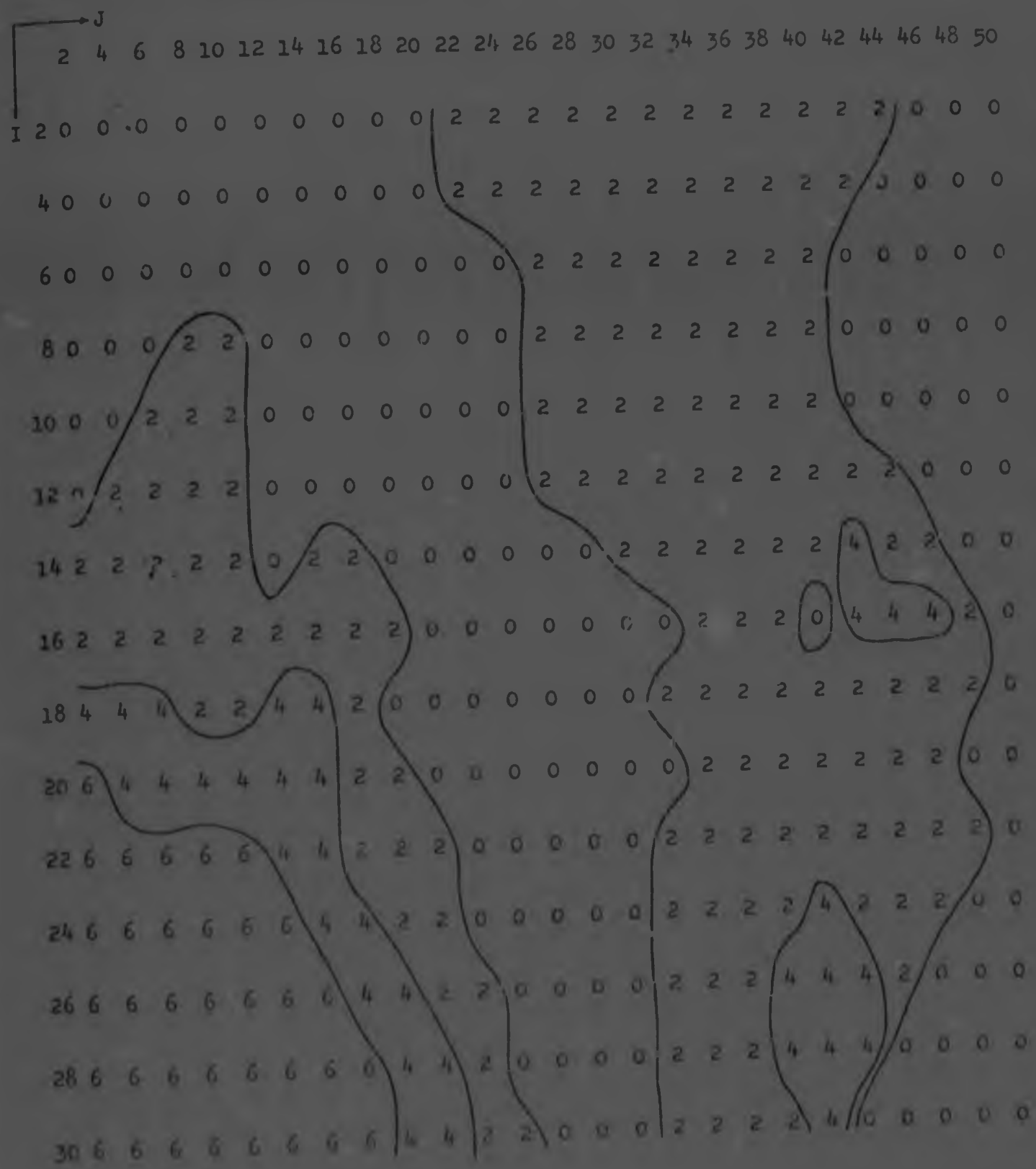


FIG. 7.4 FIVE YEARLY PLAN MAPS OF THE PIT

TOPOGRAPHIC MAP OF FIT IN YEAR 6

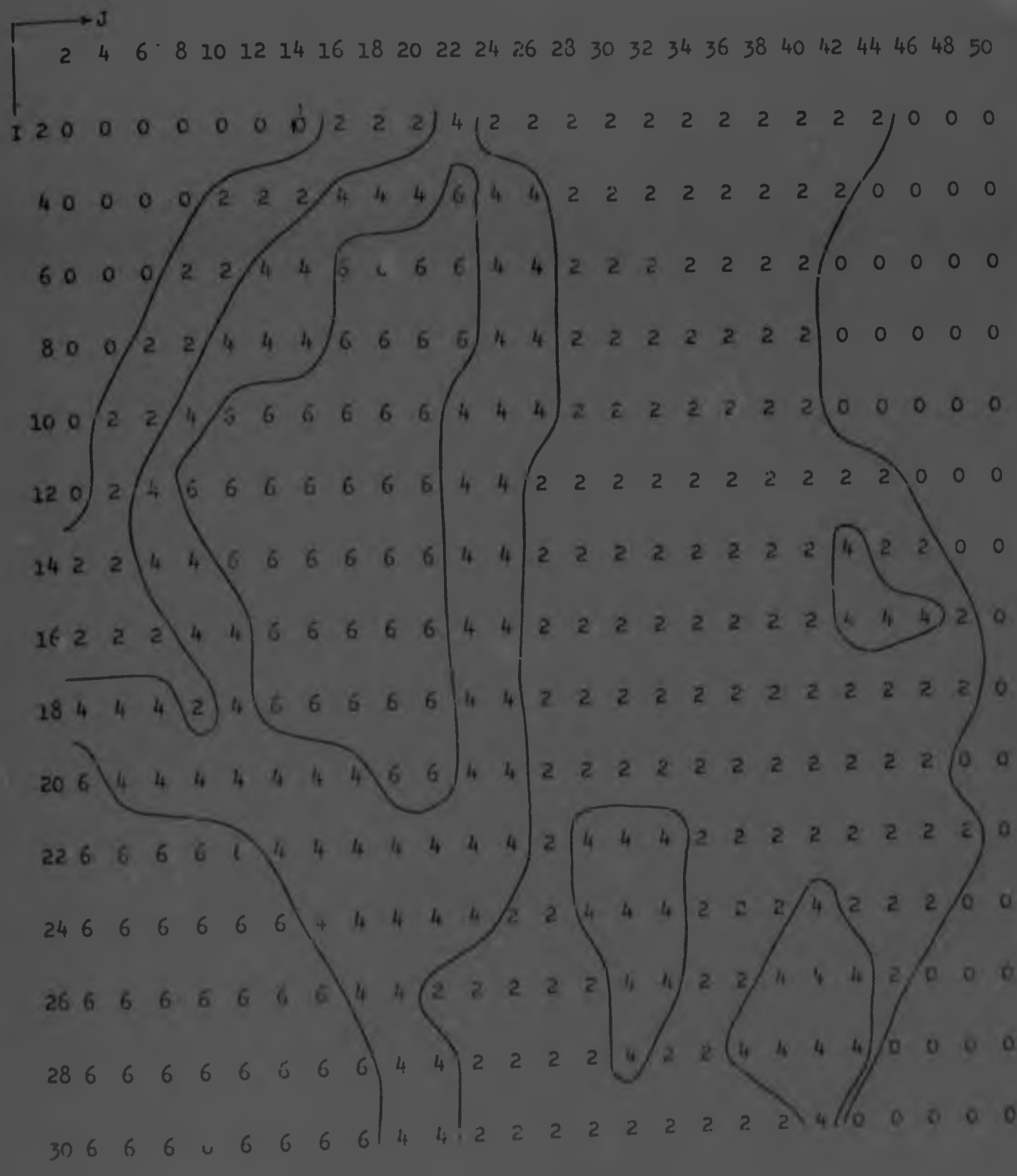


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 11

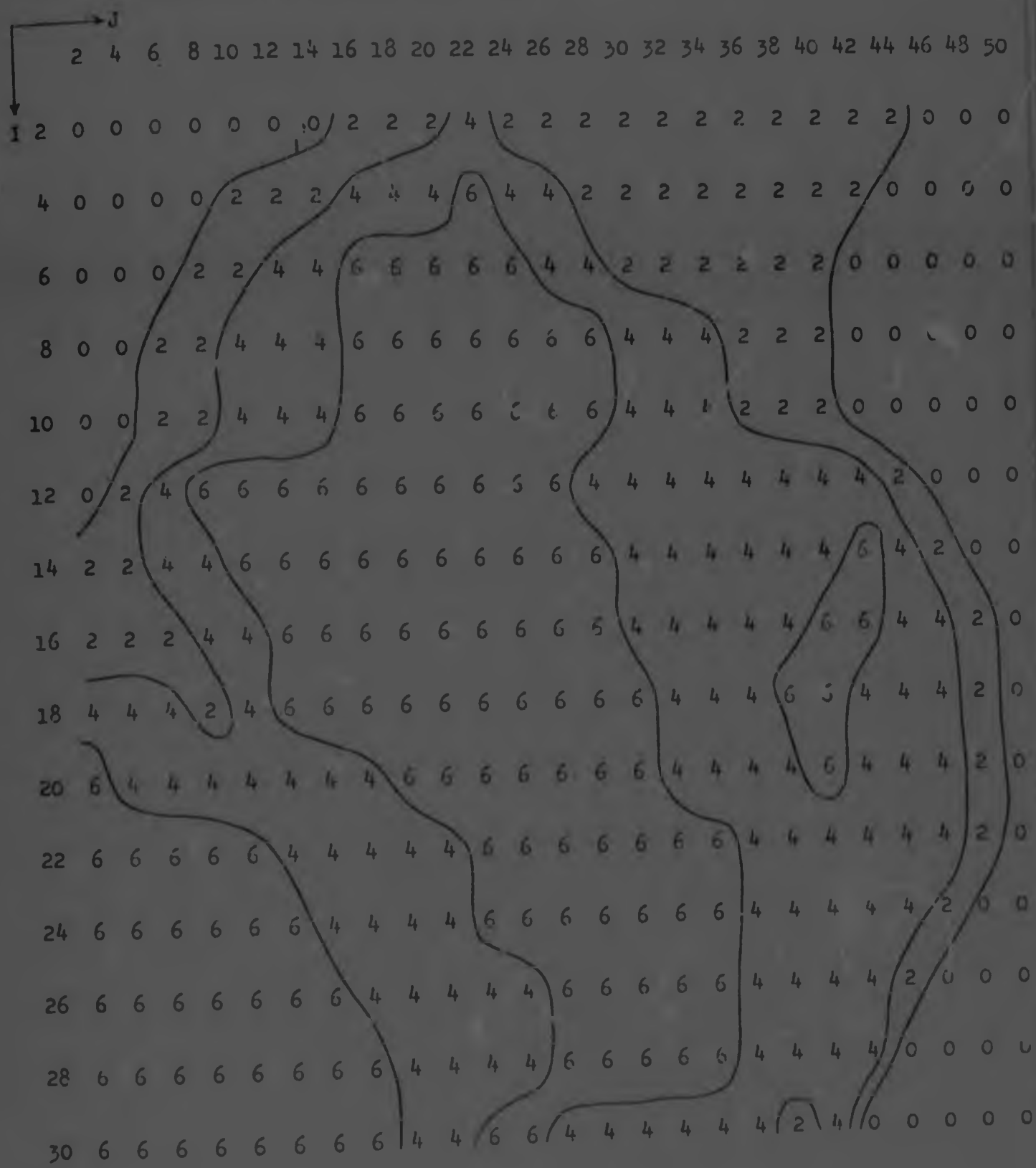


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 16

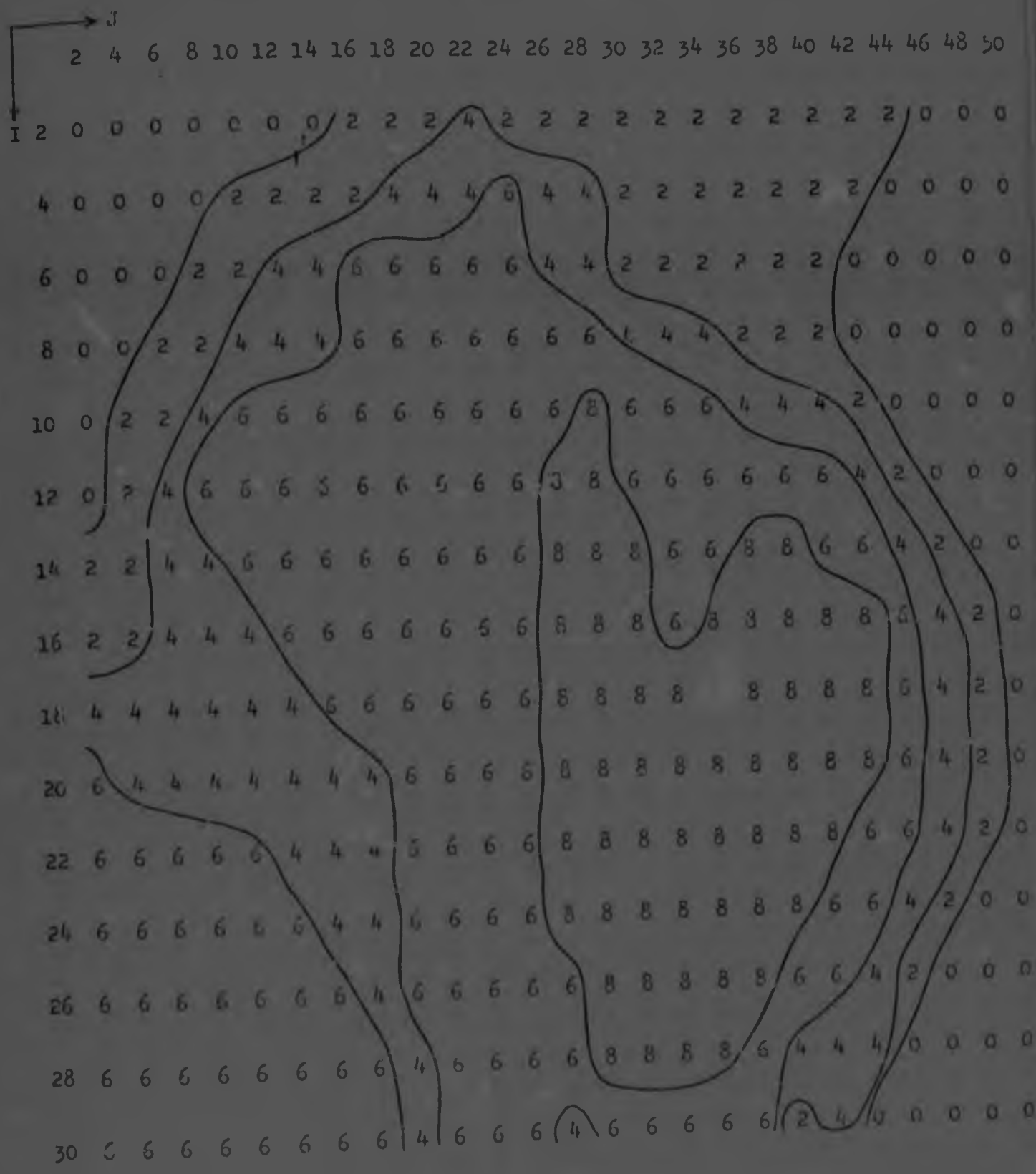


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 21

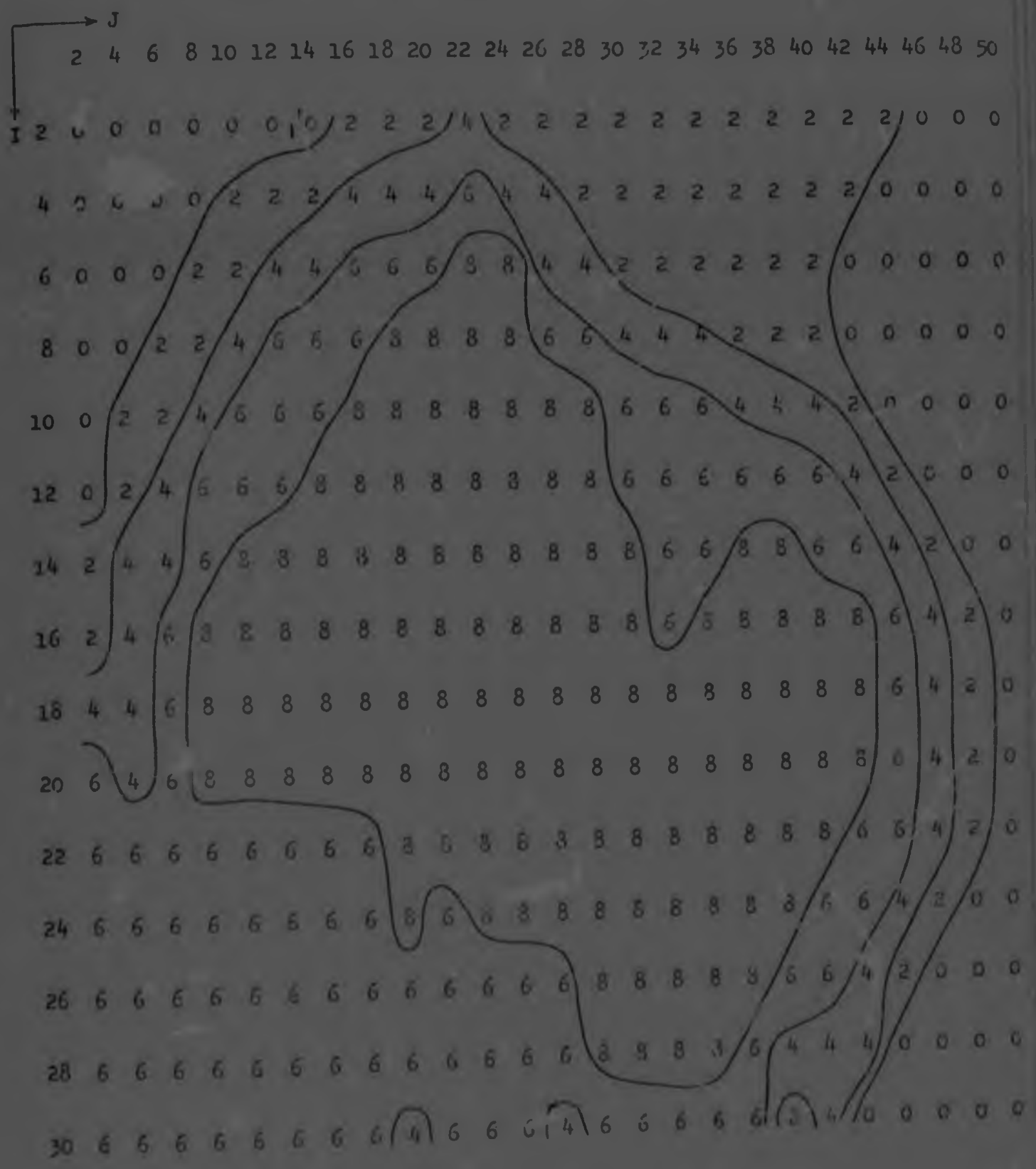


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 26

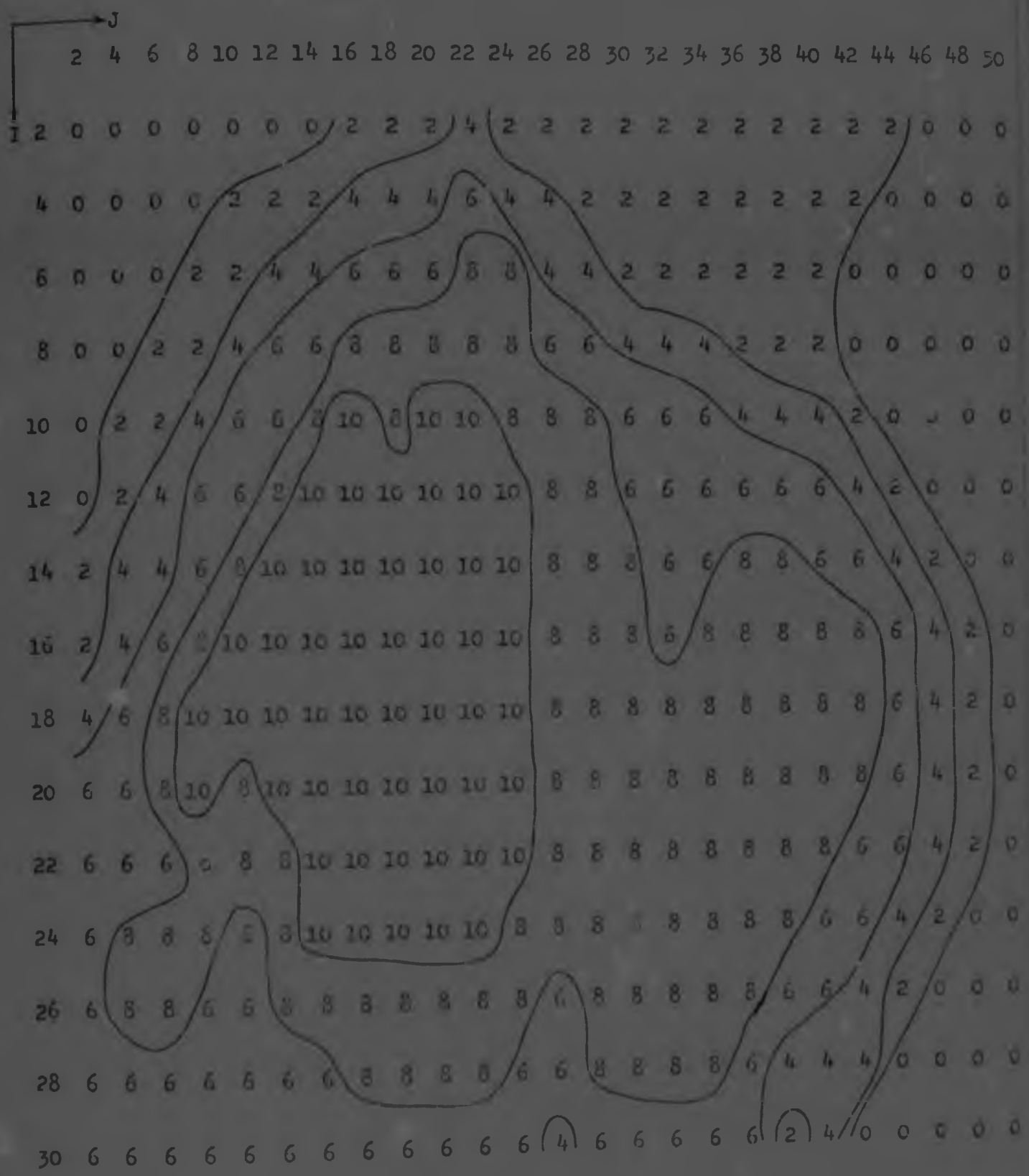


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 31

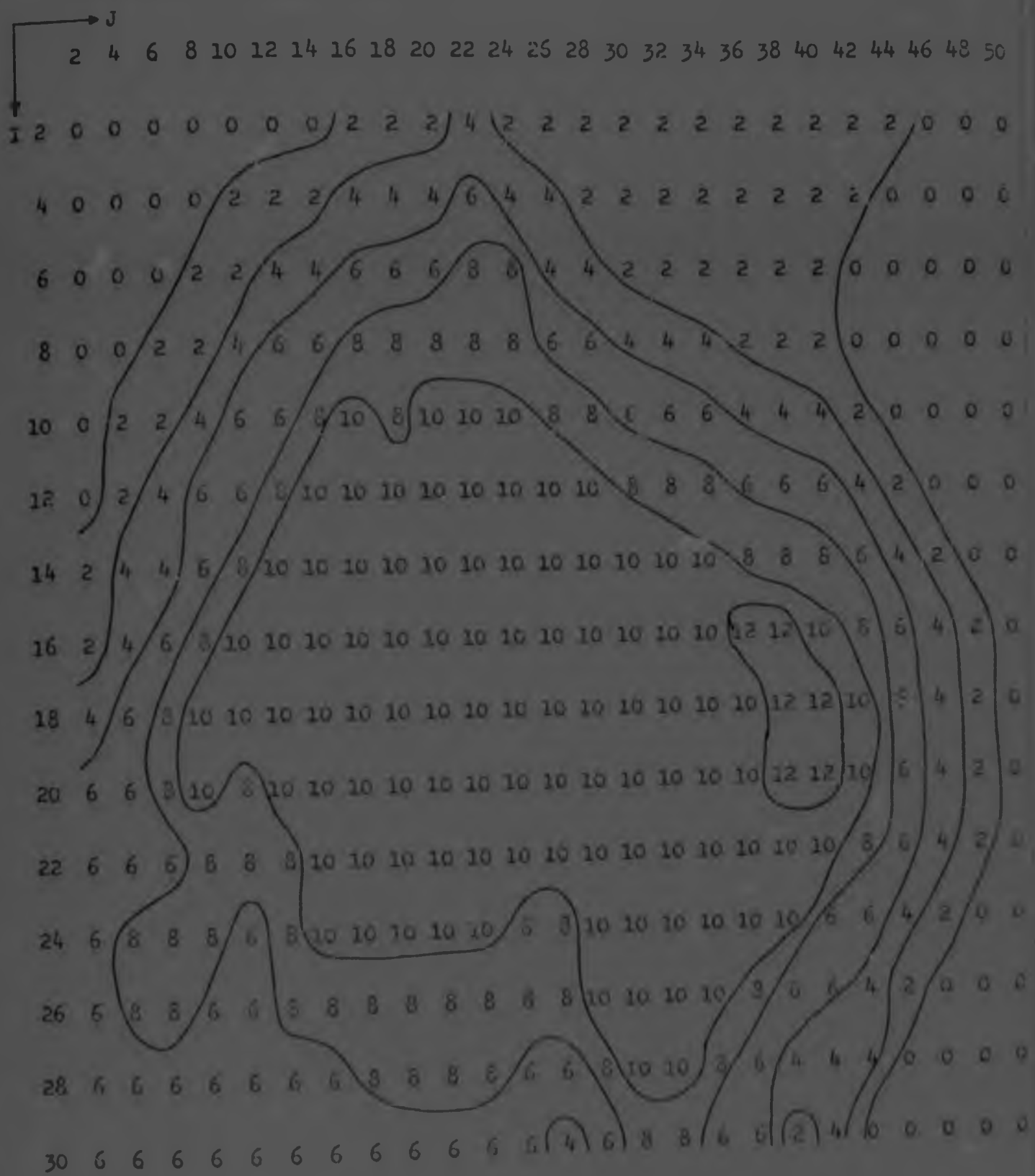


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 36

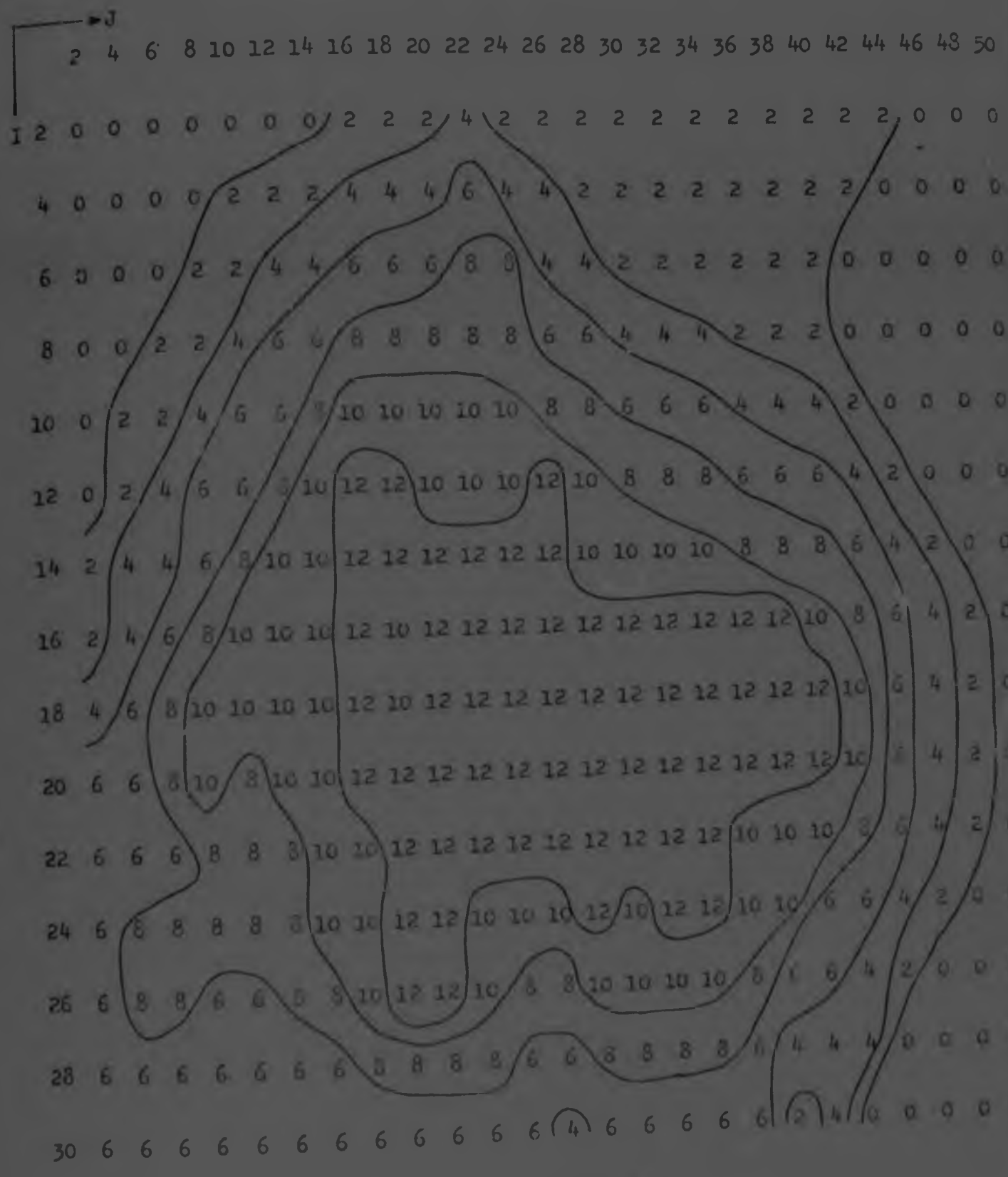


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 36

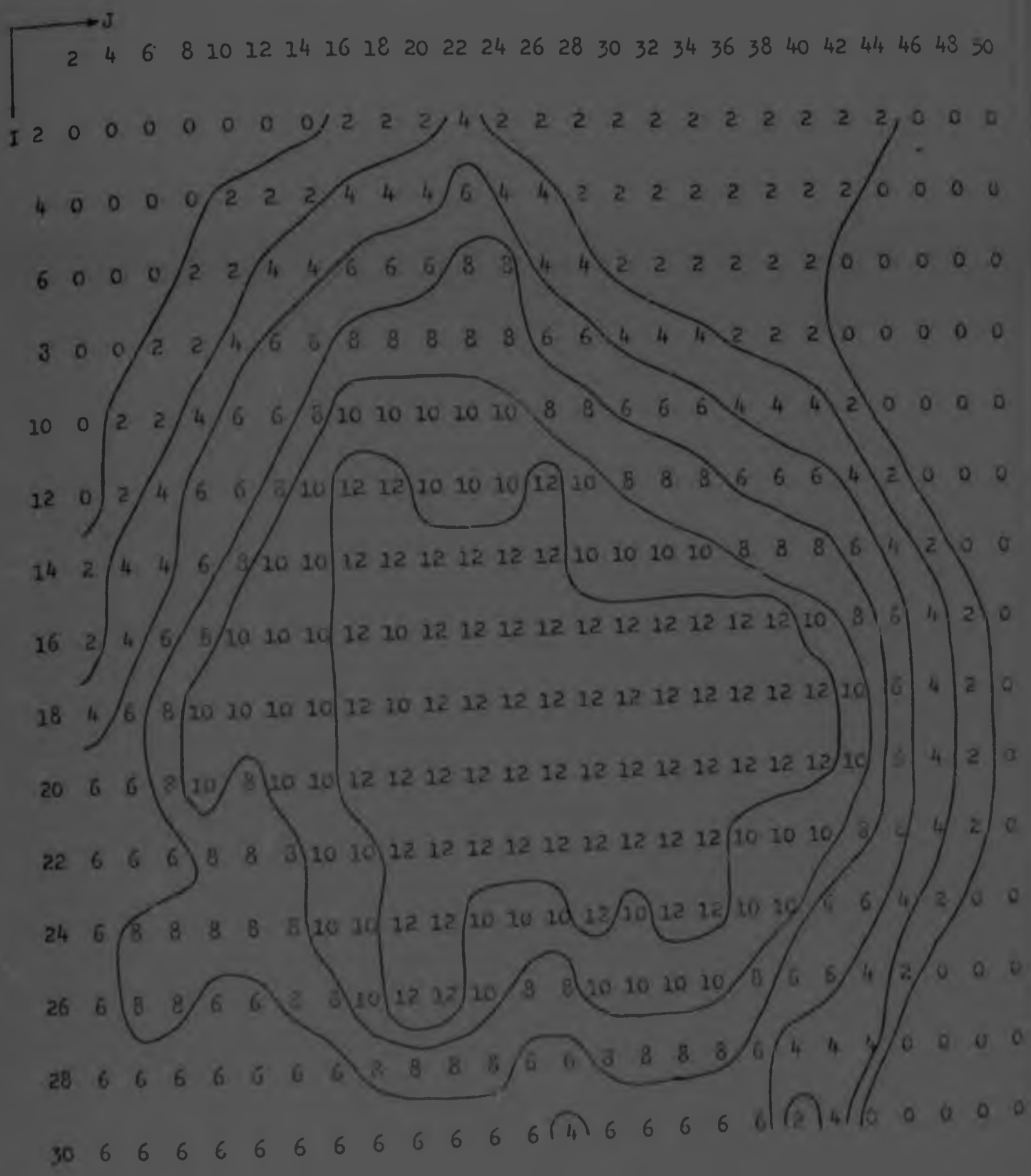


FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 41

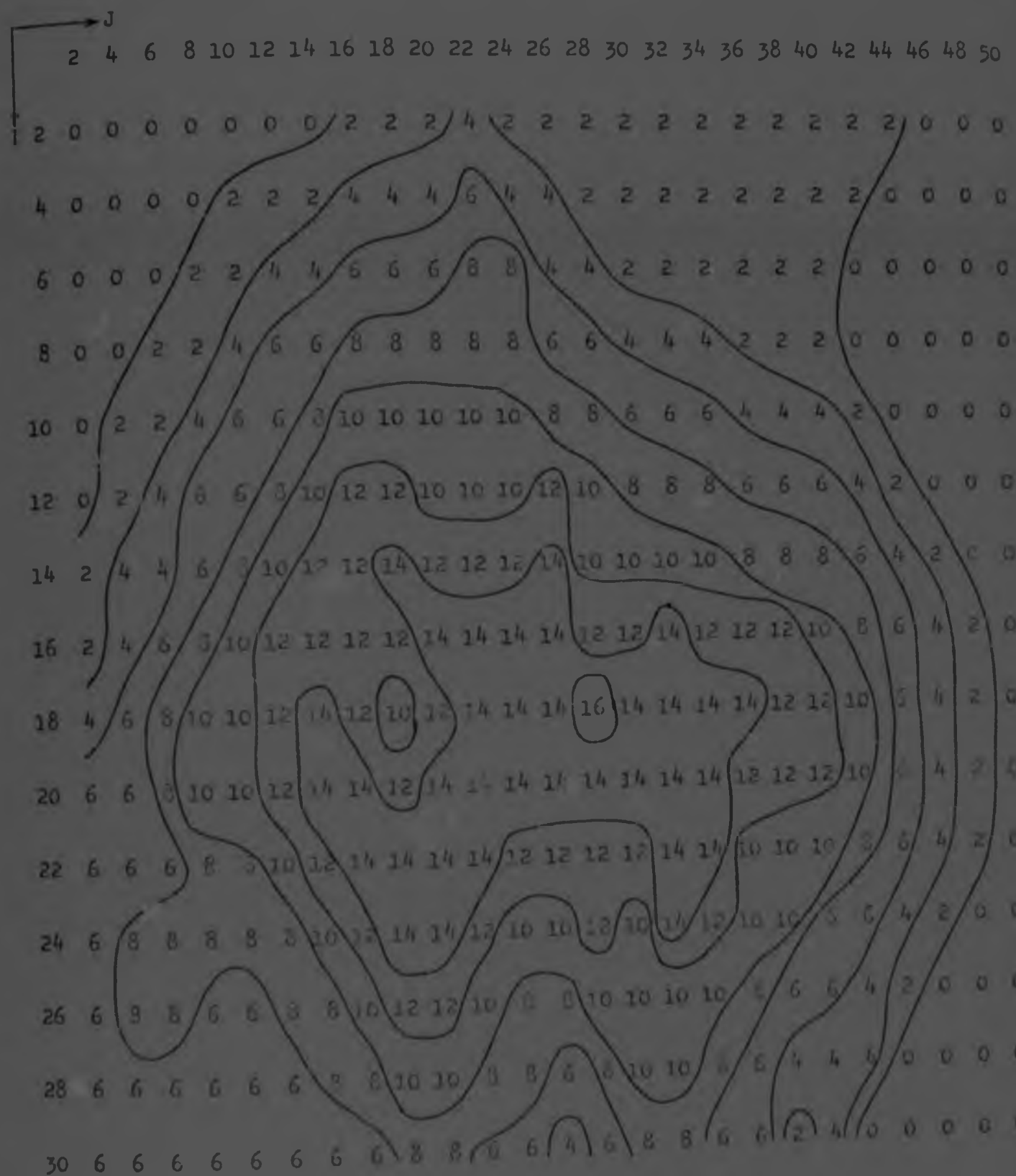


FIG. 7.4 (CONTINUED)

The capital requirements (CR) listed in the cash flow are the development expenditures for the combination of plant facilities selected 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 = 7m tons/yr•, concentrator = 1•7m tons/yr•, refinery = 20t tons/yr• and overall recovery efficiency = 95%•

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

The total present value (-5•39) 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 solution's worth• That solution resulting from the input mining proposal which yields the greatest profitability, is the obvious choice for exploiting the asset'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• 4 AN INVALID SOLUTION

An invalid solution of the MP Programme is illustrated in Table 7•19• This table was generated by the Programme for the example deposit with the following plant capacities:

1) Pit = 5•0m tons per year•

2) Concentrator = 3m tons per year, and

3) Refinery = 20t tons per year•

Recall from Subsection 6•7•2 that an invalid solution arises when the simulated waste plus ore production exceeds the

YR	REV	INT	TAX	OB	CFI	ECAF	CF	PV	LC	CR	LR
1	0•59	•0	0•0	2•61	-2•02	11•0	-13•02	-5•39	0•0	7•1	0•0
2	0•56	•0	0•0	2•53	-1•97	4•2	-6•22	7•19	0•0	5•6	0•0
3	0•68	•0	0•0	2•78	-2•10	0•0	-2•10	13•99	0•0	6•5	0•0
4	2•14	•56	0•0	3•33	-1•75	0•0	-1•75	17•21	7•5	6•5	0•0
5	2•93	•81	0•0	1•53	0•53	0•0	0•53	20•34	3•2	7•6	0•0
6	2•26	•81	0•0	1•42	0•03	0•0	0•03	21•43	0•0	7•5	0•0
7	2•08	•81	0•0	1•51	-0•24	0•0	-0•24	23•11	0•0	0•0	2•0
8	2•31	•66	0•0	1•28	0•37	0•0	0•37	25•20	0•0	0•0	2•0
9	3•33	•51	0•0	1•49	1•33	0•0	1•33	26•84	0•0	0•0	2•0
10	3•84	•36	0•0	1•50	1•98	0•0	1•98	27•65	0•0	0•0	2•0
11	2•52	•21	0•0	1•35	0•96	0•0	0•96	27•83	0•0	0•0	2•0
12	2•32	•06	0•0	1•32	0•94	0•0	0•94	29•15	0•0	0•0	0•7
13	2•67	•0	0•0	1•74	0•93	0•0	0•93	30•54	0•0	0•0	0•0
14	4•02	•0	1•2	1•34	1•43	0•0	1•43	32•05	0•0	0•0	0•0
15	3•68	•0	1•0	0•38	2•30	0•0	2•30	33•19	0•0	0•0	0•0
16	6•51	•0	1•9	0•21	4•41	0•0	4•41	33•54	0•0	0•0	0•0
-	-	-	-	-	-	-	-	-	-	-	-
25	4•79	•0	1•4	0•66	2•78	0•0	2•78	26•96	0•0	0•0	0•0
26	8•79	•0	2•6	0•0	6•18	0•0	6•18	26•33	0•0	0•0	0•0
27	5•89	•0	1•7	0•14	4•06	0•0	4•06	22•26	0•0	0•0	0•0
28	2•96	•0	0•8	0•40	1•79	0•0	1•79	19•93	0•0	0•0	0•0
29	3•59	•0	1•0	0•12	2•50	0•0	2•50	19•79	0•0	0•0	0•0
30	4•23	•0	1•2	0•03	3•03	0•0	3•03	18•87	0•0	0•0	0•0
31	2•37	•0	0•8	0•01	2•19	0•0	2•19	17•35	0•0	0•0	0•0
32	3•27	•0	0•9	0•0	2•40	0•0	2•40	16•55	0•0	0•0	0•0
33	3•03	•0	0•8	1•01	1•23	0•0	1•23	15•47	0•0	0•0	0•0
34	5•81	•0	1•7	0•0	4•14	0•0	4•14	15•43	0•0	0•0	0•0
35	3•70	•0	1•0	0•0	2•70	0•0	2•70	12•58	0•0	0•0	0•0
36	3•93	•0	1•1	0•01	2•84	0•0	2•84	10•89	0•0	0•0	0•0
37	3•76	•0	1•0	0•13	2•61	0•0	2•61	8•02	0•0	0•0	0•0
38	3•26	•0	0•9	0•27	2•13	0•0	2•13	7•02	0•0	0•0	0•0
39	4•79	•0	1•4	0•0	3•44	0•0	3•44	5•46	0•0	0•0	0•0
40	3•24	•0	0•9	0•0	2•38	0•0	2•38	2•45	0•0	0•0	0•0
41	0•29	•0	0•0	0•0	0•29	0•0	0•29	0•27	0•0	0•0	0•0

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.

SIMULATED PIT PRODUCTION SCHEDULE(FOURTH ITERATION CYCLE)

YR	OVERBURDEN M TONS	STRIPPING COSTS M CASH UNITS	ORE M TONS	REVENUE M CASH UNITS
1	2•53	1•25	1•27	0•68
2	2•06	1•40	1•27	0•51
3	6•62	3•41	1•27	1•06
4	5•43	2•69	1•27	1•66
5	2•25	1•12	1•41	2•20
6	4•64	2•45	2•25	3•30
7	1•55	0•78	2•25	2•62
8	1•55	0•79	2•25	3•19
9	1•13	0•60	2•25	5•07
10	5•49	2•65	2•11	3•05
11	6•62	3•48	1•97	1•84
12	5•43	2•35	1•97	3•09
13	5•35	2•90	1•97	2•56
14	0•28	0•15	1•97	4•53
15	1•69	1•02	1•97	7•26
16	0•84	0•53	2•53	11•35
17	0•56	0•20	2•96	6•90
18	1•27	0•77	2•96	8•01
19	0•14	0•09	2•96	5•97
20	4•22	2•85	2•96	7•85
21	0•14	0•03	2•96	12•24
22	1•13	0•57	2•96	8•79
23	1•97	1•22	2•96	6•50
24	0•99	0•43	3•04	5•99
25	0•0	0•0	2•96	6•61
26	0•0	0•0	2•96	8•93
27	0•28	0•18	2•96	7•61
28	0•56	0•27	2•96	5•43
29	2•12	1•07	2•96	7•88
30	4 85	3•26	1•18	1•00

TOTAL REVENUE=154•24 MILLION CASH UNITS
TOTAL NUMBER OF PIT INCREMENTS=6

TABLE 7•19

CHAPTER 8 THE MINE PLAN PROGRAMME

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 segment 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 take 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 involved, is substituted for 'disk pack label'. The position of the source decks 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 partioned data set. The control cards illustrated in Fig. 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.

	MP PROGRAMME		
	MODULE 1	MODULE 2	MODULE 3
CALLING PROGRAMMES	MAIN1	MAIN2	MAIN3
SUBROUTINES	COG1	COG2	---->
	COGSUB	---->	---->
	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 partioned data set. The modules are made by executing the control card list in Fig. 8.2 with the following substitutions:

1) 'n' and 'nn' are replaced with '1' to create the module MP1 which contains the first iteration cycle,

2) 'n' and 'nn' are replaced with '2' to create the module

```
(job card)
// EXEC FORTHCL,PARM.LKED='CAL
//SYSIN DD *
(source card deck for the calling programmes and subroutines, in turn)
/*
//LKED.SYSLMOD DD DSNAME=PROGLIB(mine),DISP=(NEW,KEEP),UNIT=SYSDA,      *
//                VOLUME=SER=disk pack label,SPACE=(CYL,(20,1,20))
//
```

The procedure control cards are used to load the first member of the partitioned data set only. The 'LKED.SYSLMOD' card must be replaced by the following card to compile the remaining data set members or to recompile any of these members:

```
//LKED.SYSLMOD DD DSNAME=PROGLIB(mine),DISP=OLD,VOLUME=SER=disk pack label
```

FIG. 8.1 COMPILATION AND LINK-EDIT JOB STEP

```
(job card)
// EXEC [illegible],PARM='[illegible]',REGION=150K
//SYSPRINT DD [illegible]=A
//SYSUT DD UNIT=SYSDA,VOLUME=SER=disk pack label,SPACE=(TRK,([illegible],10))
//SYSLIB DD DSNAME=SYS1.[illegible]TLIB,DISP=SHR
//SYSLMOD DD DSNAME=PROG([illegible]),DISP=([illegible]),UNIT=SYSDA,                X
//                VOLUME=SER=disk pack label,SPACE([illegible],(10,1,[illegible]))    X
//                DCB=(RECFM=U,BLKSIZE=3625)
//A DD DSNAME=PROG[illegible],DISP=OLD,VOLUME=SER=disk pack label,UNIT=SYSDA
//SYSLIN DD *,DCB=BLKSIZE=80
 ENTRY MAIN
 INCLUDE A(MAIN[illegible])
 INCLUDE A(CO[illegible])
 INCLUDE A(COGSUB)
 INCLUDE A(PD)
 INCLUDE A(SUBPD)
/*
//
```

The 'DISP' parameter must be changed to 'OLD', and the 'DCB' and 'SPACE' parameters deleted on the 'SYSLMOD' card for any executions of this job step after the first, as follows:

```
//SYSLMOD DD [illegible],DISP=OLD,UNIT=SYSDA,VOLUME=SER=disk pack label
```

FIG. 8.[illegible] CREATION OF MODULES JOB [illegible]

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 Fig. 8.3. 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 13,500 grid -intersections on the International Business Machines Corporation's IBM 360/50 computer.

8. 2 PREPARATION OF THE PROGRAMME FOR EXECUTION

The steps needed to implement the MP Programme are listed below. The Programme 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 BUFFER parameters and in subsection 8.1.2 for the DEFINE FILE statement.

STEP 2:

The grid system is already defined for the preceding Programme in the model. However, the maximum number of grid co-ordinates e.g. NJ and NK in the I, J and K co-ordinate

```
(job card)
//JOBLIB DD DSNAME=[illegible],DISP=([illegible]),UNIT=2311,VOLUME=[illegible]=disk pack label
//A EXEC PGM=MPn,TIME=30,REGION=550K
(sequential and direct access data sets--see the list below)
//GO.FT06F001 DD SYSOUT=A,DCB=(RECFM=FBA,LRECL=133,BLKSIZE=133,BUFNO=1)
//GO.FT05F001 DD *
(punched card input data)
/*
//
```

The sequential and direct access data sets required for the three modules (MP1 , MP2, and MP3) are:

1) For the first module (MP1):

```
//GO.FT1[illegible]01 DD DSNAME=GMODEL,DISP=OLD,UNIT=2311,VOLUME=[illegible]=disk pack label
//GO.FT11F001 DD DSNAME=UPDATE,DISP=([illegible]),UNIT=2311,                    X
//                VOLUME=[illegible]=disk pack label,[illegible]=([illegible],(3,1)),          X
//                DCB=(RECFM=VB,LRECL=[illegible],BLKSIZE=3204)
//GO.FT12F001 DD DSNAME=ORIGINAL,DISP=([illegible],K[illegible]),UNIT=2311                    X
//                VOLUME=[illegible]=disk pack label,[illegible]=([illegible],(3,1))           X
//                DCB=([illegible]=VB,LRECL=12,BLKSIZE=3[illegible])
//GO.FT13F001 DD [illegible]=EQGRAD,DISP=([illegible]),[illegible]=2311,                    X
//                VOLUME=[illegible]=disk pack label,[illegible]=([illegible],(1,1))           X
//                DCB=(RECFM=VB,LRECL=10,[illegible]=3204)
```

2) For the second module ([illegible]):

```
//GO.FT11F001 DD [illegible]=UPDATE,[illegible],[illegible]=2311,[illegible]=disk pack label
//[illegible]
//[illegible]
```

(continued--next page)

FIG. 8.3 [illegible]

```
//GO.FT14F001 DD DSNAME=UPDATE2,DISP=([illegible],[illegible]),UNIT=2311,          X
//               VOLUME=SER=[illegible] pack label,SPACE=(CYL,(2,1)),              X
//               DCB=(RECFM=VB,LRECL=12,BLKSIZE=3544)
//GO.FT15F001 DD DSNAME=CRIPES2,DISP=(NEW,KEEP),UNIT=2311                          X
//               VOLUME=SER=disk pack label,SPACE=([illegible],(2,1)),             X
//               DCB=(RECFM=VB,LRECL=1[illegible],BLKSIZE=3504)
//GO.FT16F001 DD DSNAME=PRICE[illegible],DISP=(NEW,KEEP),UNIT=2311                 X
//               VOLUME=SER=disk pack label,SPACE=([illegible],(1,1)),             X
//               DCB=(RECFM=VB,LRECL=32,BLKSIZE=3204)
```

3) For the [illegible] module (FT5):

```
//GO.FT13F001 DD DSNAME=[illegible],DISP=[illegible],UNIT=2311,VOLUME=SER=disk pack label
//GO.FT14F001 DD DSNAME=[illegible],DISP=[illegible],UNIT=2311,VOLUME=SER=disk pack label
//GO.FT15F001 DD DSNAME=[illegible],DISP=[illegible],UNIT=2311,VOLUME=SER=disk pack label
//GO.FT16F001 DD DSNAME=PRICE[illegible],DISP=[illegible],UNIT=2311,VOLUME=SER=disk pack label
```

FIG. 8.3 (CONTINUED)

rections, respectively, are required as part of the MP Programme's input data.

STEP 3:

The appropriate number of FORTRAN variables must be created for each recoverable mineral reported by the Grade Prediction Programme. This modification only is required for the calling programmes. The variables and their location in the programmes 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 'ORBES'. 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 ORBES 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 ORBES does not become excessively large. 'GI' and 'GS' are input data to the MP Programme.

STEP 5:

The initial assumption for the subscript 'M' of 'ORBES' is given by the user. (See Subsection 8.1.1 for a description of how this is done.) This assumption is located in the calling programme MAIN1. (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-S net cash value distribution,

B) design of the final pit surface,

C) the tonnage factors, and

D; pit production simulation•

Thi:e is a comment at those places in the Programme lists where the simulation of these features is re,uired•

STEP 7:

The variable 'D' in the cone expressions for the final pit surface desi,n and waste mining simulation routines must be dependant on the pit wall-angles, if thes• angles vary• The relationshi; between 'D' and the pit wall-angles is explained further in the comments in the lists of subroutines PD and SUBPD•

STEP 8:

The ore excavation se,uence programmed in subroutine PD must be tailored to the user's re,uirements• The desired number o, ore workin, 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 se,uence 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 SE,UENCE O, MAIN1

LINES 001 - 007

Specification statements•

LINES 008 - 054

Input statements•

LINES 059 - 080

DO loop 11 computes:

A) an equivalent mineral grade 'EQGRAD' to represent all the deposit's recoverable minerals, and

B) the initial assumption for the T-S mineral distribution.

LINES 083 - 093

The first estimate of the unit production costs is computed.

LINE 094

The following values are supplied by the nested subroutines COG1 and COGSUB for the selected plant combination:

A) the annual cut-off grades 'CUTOFF' and production targets 'TI', CON, CREF';

B) the overall recovery efficiency 'YIELD', and

C) an average 'AVEG' 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 'A, C, B'. At this point in the solution, it is not know when the grid-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 'AVEG'.

LINE 151

The following computations are executed by the nested subroutines PD and SUBFR:

A) the final pit surface is designed and

B) the pit production is simulated.

LINES 152 - 157

The annual revenue is totalled 'PNEW' and the number of pit increments computed 'MAXN'. 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 'MP1' 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, EQGRAD, IGRAD1, IGRAD2) are stored for rapid access rather than being recalculated in the other modules.

LINES 186 - 209

Yearly topographic maps are printed to illustrate the pit production simulation.

8. 3. 2 A LIST OF MAIN1

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

The example deposit has two recoverable minerals 'GRAD1' and 'GRAD2' whose present selling prices are 'S1' and 'S2', respectively. The following lines must be modified, if the user's deposit contains a different number:

MAIN1 : LINES 004, 005, 021-022, 036-038, 047, 071-072, 091-092, 124-126, 142 and 184;

MAIN2 : LINES 004, 005, 016-017, 040, 049-050, 090-092, 108 and 117;

MAIN3 : LINES 004, 005, 016-017, 046-047, 049, 052, 091-093, 109 and 118.

```
001      DOUBLE PRECISION CAP(3,3,3),CONCAP(3),REFCAP(3),
        1MINCAP,ORERES(99,20),P(99),MIN(99),Q6(20),
        2Q7(65,20),Q8(65,20),Q9(65,20),Q10(65,20),
```

```
        3Q11(65,20),Q12(3,3,3),Q14(99),POLD,PNEW,DABS,FIXC,
        4Q14A(99)
002      REAL MILC(2),INFRC,INFRP,MINC
003      INTEGER CUTOFF(99),ONE/1/,TWO/2/,GI
004      INTEGER*2 TOPO(13500),YR(13500),PITINC,SURFC(30),
        1Q33(13500),Q34(13500),Q35(13500),T,SCORK(750),
        2IGRAD1(30,25,18),IGRAD2(30,25,18),EQGRAD(30,25,18),
        3AVEG
005      DIMENSION OB(2),OR(2),OBCI(2),ORCI(2),REFC(2),
        1PITANG(2),WPKANG(2),SLOPE(2),WSLOPE(2),S1(99),
        2S2(99),TONF(2),REC(3),E(99),C(99),R(99),PROFIT(99),
        3CAPOB(99),WASTE(99) VAL(13500)
006      COMMON ORERES,CONCAP,REFCAP,MINCAP,F,Q6,Q7,Q8,Q9,
        1Q10,Q11,Q12,MIN,Q14,Q14A,CAP,PITANG,WPKANG,SLOPE,
        2WSLOPE,S1,D,C,R,E,TONF,PROFIT, CAPOB,WASTE,REC,
        3VAL,GS,NI,NJ,NK,NBLKS,MAXN,MAXG,CUTOFF,L,Q33,Q34,
        4Q35,SURFC,TOPO,YR
```

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

ii = NK*NJ*NK: jj = 4*(number of recoverable minerals)•

```
007      DEFINE FILE 10( ii  , jj ,L,IJ)
```

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

The scale factor 'GS' and grade increment 'GI' (Line 011) are given by the user to set up the subscript 'N' of 'ORERES' in lines 037, 076 and 177 as explained in Step 4, Section 8•2•

```
008      READ (5,15) NI,NJ,NK,GS
009   15 FORMAT (3I3,F7•1)
010      NBLKS=NI*NJ*NK
011      GI=5
012      READ (5,1051) (SCORK(N),N=1,750)
013 1051 FORMAT (30I2)
014      DO 210 LOC=1,NBLKS
015      TOPO(LOC)=0•0
016      VAL(LOC)=0•0
017  210 YR(LOC)=0
018      DO 260 K=1,NK
019      DO 260 J=1,NJ
020      DO 260 I=1,NI
021      IGRAD1(I,J,K)=0
022      IGRAD2(I,J,K)=0
```

```
  260 EQGRAD(I,J,K)=0
      DO 1054 K=1,NK
      N=0
      DO 1054 J=1,NJ
      DO 1054 I=1,NI
      N=N+ONE
      LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
      FIND (10'LOC)
      IF (SCORK(N)-K) 1055,1053,1053
 1055 II=I
      GO TO 1056
 1053 II=0
 1056 TOPO(LOC)=II
      READ (10'LOC) GRAD1,GRAD2
      IGRAD1(I,J,K)=GRAD1*GS
      IGRAD2(I,J,K)=GRAD2*GS
 1054 CONTINUE
      READ (5,60) (((CAP(I,J,K),I=1,3),J=1,3),K=1,3)
   60 FORMAT (3F13•1)
      READ (5,61) (OB(I),OR(I),OBCI(I),ORCI(I),MILC(I),
     1REFC(I),PITANG(I),WRKANG(I),I=1,2)
   61 FORMAT (5F6•3,F7•3,2F7•4)
      DO 63 I=1,2
      SLOPE(I)=(SIN(PITANG(I))/COS(PITANG(I)))**2•
   63 WSLOPE(I)=(SIN(WRKANG(I))/COS(WRKANG(I)))**2•
      READ (5,1) MAXN,S1(1),S2(1),D,INFRC,INFRP,TONF(1),
     1TONF(2)
    1 FORMAT (I2,2F3•2,F5•2,2F6•3,2F9•1)
      READ (5,2) (CONCAP(I),I=1,3),(REFCAP(J),J=1,3),
     1(REC(K),K=1,3)
    2 FORMAT (3F11•1/3F9•1/3F5•2)
      READ (5,3) MINCAP
    3 FORMAT (F11•1)
      READ (5,12) SINC,CONC,FIXC
   12 FORMAT (2F5•2,F10•1)
      DO 4 NN=1,20
      DO 4 M=1,99
    4 ORERES(M,NN)=0•0
      MAXS=0
```

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

Among other things, in loop 11 executes the initial assumption for 'ORERES', as explained in Subsection 6•3•1• and Step 5, Section 8•2•

```
059        DO 11 N=1,MAXN
060        K1=4*N
061        K2=K1-TWO
062     73 DO 11 K=K2,K1
063        DO 11 J=1,NJ
064        DO 11 I=1,NI
0065        LOC=NJ*NI*(K-ONE)+NI*(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 grid-level. The following four statements select the appropriate tonnage factor for the 'ORERES' computations. These statements are modified as explained in Subsection 6.3.1, 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, 086 & 144;
MAIN3: LINES 077, 087 & 145;
PD: LINES 118 & 143; AND
SUBPD: LINE 050.)

```
067     5 IF (K-10) 6,6,7
068     6 MM=ONE
069       GO TO 8
070     7 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.

```
071     8 X=(S2(I)/S1(I))**EGRAD2(I,J,K)
072       EQGTAD(I,J,K)=EQGTAD(I,J,K)*FTIX(X)
073       IF (EQGTAD(I,J,K)-GI) 9,9,10
074     9 M=ONE
075       GO TO 70
076    10 M=EQGTAD(I,J,K)/GI
077    11 IF (MAXG-M) 30,80,70
078    80 MAXG=M
```

```
079    70 ORDRES(M,N)=OPIPES(M,N)+TONF(MM)
080    11 CONTINUE
081       WRITE (6,22 ) MAXG
082   220 FORMAT (1X,' MAXG=',14)
083       DO 13 N=1,99
084       T=N
085       X=(1.+INFFC)**T
086       Y=(1.+INFRP)**T
087       M(N)=X*MINC
088       C(N)=Y*CONC
089       R(N)=X*REFC(1)
090       F(N)=Y*FIXC
091       S2(N)=Y*S2(1)
092       S1(N)=Y*S1(1)
093    13 CONTINUE
094       CALL COG (AVEG,YIELD)
095       T=ONE
096       LC=0
097       MC=0
098       SUME=0
099       SUMC=0
100       SUMR=0
```

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

DO loop 34 computes the T-S net cash value distribution. This routine, and its duplicates in SAINI and SAIN2 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,NI
104       LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
105       IF (TOPO(LOC)) 34,34,14
106    14 M=ONE
107    16 IF (EQUIPAD(I,J,K)-AVEG) 19,20,20
108    19 LC=LC+ONE
109       REV=0.0
110       IF (K-10) 20,20,21
111    20 N=ONE
112       GO TO 22
113    21 N=TWO
114    22 X=(OB(N)+(K*OBCI(N)))*((1.+INFRC)**N)
115       SUME=SUME+X
116       COST=(X+(C(M)/MIB(31)))*TONF(M)
```

```
      GO TO 33
   23 LC=LC+ONE
      MC=MC+ONE
      IF (K-10) 24,24,25
   24 N=ONE
      GO TO 26
   25 N=TWO
   26 I1=IGRAD1(I,J,K)
      I2=IGRAD2(I,J,K)
      REV=((FLOAT(I1)/GS*S1(M))+(FLOAT(I2)/GS*S2(M)))*
     1TONF(M)*YIELD
      X=(OB(M)+(F*DLC1(M)))*((1.+INFRC)**M)
      Y=(OR(M)+(F*DRC1(M)))*((1.+INFRC)**M)
      SUME=SUME+X
      IF (IGRAD2(I,J,K)) 27,27,28
   27 NN=ONE
      GO TO 29
   28 NN=TWO
   29 Z=(MILC(NN))*((1.+INFRC)**M)
      SUMC=SUMC+Y+MC+Z
      IF (ABS(((0.3*I)-(277.5*J)-(37.525*K)+3274.075)
     1/288)-3.0) 30,30,31
   30 NN=ONE
      GO TO 32
   31 NN=TWO
   32 W=(RLIC(NN))*((1.+INFRC)**M)
      SUMR=SUMR+W
      COST=(Y+Z+(W*YIELD*((FLOAT(I1+I2))/GS))+(F(M)/
     1MIN(M)))*TONF(M)
   33 VAL(LOC)=REV-COST
   34 CONTINUE
      IF (SUME) 36,36,35
   35 E(T)=SUME/LC
   36 IF (SUMC) 38,38,37
   37 C(T)=SUMC/MC
      R(T)=SUMR/MC
   38 CONTINUE
      CALL SD
      PNEW=0.0
      DO 39 M=1,L
   39 PNEW=PNEW+AMULT(M)*PNEW
      WRITE (1,221) PNEW
  222 FORMAT (1X,' TOTAL REVENUE=',F14.3)
      MAXN=(L-ONE)/5+ONE
      WRITE (1,221) MAXN,L
  221 FORMAT (1X,' MAXN=',I3,' L=',I4)
```

```
      DO 41 M=1,99
      DO 41 N=1,20
   41 ORERES(M,N)=0.0
      DO 48 K=1,NK
      DO 48 J=1,NJ
      DO 48 I=1,NI
      LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
      IF (TOPO(LOC)) 48,48,42
   42 IF (YR(LOC)) 48,48,51
   51 PITINC=(YR(LOC)-ONE)/5+ONE
  231 IF (K-10) 43,43,44
   43 MM=ONE
      GO TO 45
   44 MM=TWO
   45 IF (EQGRAD(I,J,K)-GI) 46,46,71
   46 M=ONE
      GO TO 47
   71 M=EQGRAD(I,J,K)/GI
   47 ORERES(M,PITINC)=ORERES(M,PITINC)+TONF(MM)
   48 CONTINUE
      WRITE (11) (TOPO(N),YR(N),N=1,NBLKS)
      END FILE 11
      WRITE (12) ((ORERES(M,N),M=1,MAXG),N=1,MAXN)
      END FILE 12
      WRITE (13) (((EQGRAD(I,J,K),IGRAD1(I,J,K),
     1IGRAD2(I,J,K),I=1,NI),J=1,NJ),K=1,NK)
      END FILE 13
   50 WRITE (6,201)
  201 FORMAT (' ANNUAL PIT OUTLINE')
      DO 211 N=1,NJ
  211 SURFC(N)=0
      DO 100 N=1,L
      WRITE (6,202) N
  202 FORMAT ('1',' TOPOGRAPHIC MAP OF MINE IN YEAR',I3)
      DO 100 I=1,NI
      DO 104 J=1,NJ
      M=0
      K=ONE
  151 IF (K-NK) 154,154,175
  154 LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
      IF (TOPO(LOC)) 152,152,155
  155 IF (YR(LOC)) 175,175,152
  152 IF (YR(LOC)-N) 152,152,175
  152 M=K
      K=K+ONE
      GO TO 151
```

```
205   175 SURFC(J)=S
206   104 CONTINUE
207         WRITE (6,203)  (SURFC(LEVEL),LEVEL=1,NJ)
208   100 CONTINUE
209   203 FORMAT (1H1,3014)
210    49 STOP
211       DEBUG TRACE,SUBTRACE,SUBCHK
212       END
```

8. 3. 3 A LIST OF MAIN2

The comments within the list describe how MAIN2 differs from MAIN1.

```
001       DOUBLE PRECISION CAP(3,3,3),CONCAP(3),REFCAP(3),
         1MINCAP,ORDER(99,20),P(99),XIN(99),L6(20),
         2Q7(65,20),Q8(65,20),Q9(65,20),Q10(65,20),
         3Q11(65,20),Z12(3,3,3),Z14(99),POLD,PNEW,PABS,FIXC,
         4Q14A(99)
002       REAL MILC(2),INFRC,INFRP,MINC
003       INTEGER CWEIGHT(99),ONE/1/,TWO/2/,GI
004       INTEGER*2 TOPO(11500),IP(11500),PERIOD,SURFC(10),
         1Q33(11500),Q34(11500),Q35(11500),Z,SCORE(700),
         2LOOPAR(30,25,10),REPAD2(99,25,10),LOOPAD(30,25,10),
         3AVES
005       DIMENSION DATE(2),OP(2),OBJI(2),OTCI(2),REFC(2),
         1PUTAGE(2),DAMAGE(2),SLOPE(2),WSLOPE(2),A1(99),
         2Z2(99),TOWP(2),Z3(11),E(99),C(99),M(99),ZPOINT(99),
         3CAPOD(99),WASTE(99),VAL(11500)
006       COMMON EPSILON,CONCAP,REFCAP,MINCAP,P,Q6,Q7,Q8,Q9,
         1Q10,Q11,Q12,Z13,Q14,Q14A,CAP,FIXAGE,PABS,SLOPE,
         2WSLOPE,GI,P,C,M,E,TOWP,ZPOINT, CAPOD,WASTE,VAL,
         3VAL,G5,Z1,Z3,Z5,MILC,MAKE,MAKD,CUTOFF,L,Q14,Q34,
         4Q35,ZSURC,TOPO,YT
```

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

The input data of line 007 are required from the results of the first module MP1:

A) the maximum equivalent mineral grade 'MAXG' reported in the results deposit 'MAXT' and

B) the mine's life span 'L' reported by the subroutine COG1.

```
      READ (5,62) MAXG,L
   62 FORMAT (2I3)
      READ (5,4) NI,NJ,NK,GS
      NBLKS=NI*NJ*NK
    4 FORMAT (3I3,F7.1)
      GI=5
      DO 260 K=1,NK
      DO 260 J=1,NJ
      DO 260 I=1,NI
      IGRAD1(I,J,K)=0
      IGRAD2(I,J,K)=0
  260 FQGRAD(I,J,K)=0
      DO 210 LOC=1,NBLKS
      VAL(LOC)=0.0
      YR(LOC)=0
  210 CONTINUE
      READ (5,60) (((CAP(I,J,K),I=1,3),J=1,3),K=1,3)
   60 FORMAT (3F12.1)
      READ (5,61) (OB(I),OB(I),OBCI(I),OBCI(I),MILC(I),
     1REFC(I),PITANG(I),WRKANG(I),I=1,2)
   61 FORMAT (5F6.3,F7.3,2F7.4)
      DO 63 I=1,2
      SLOPE(I)=(SIN(PITANG(I))/COS(PITANG(I)))**2.
   63 WSLOPE(I)=(SIN(WRKANG(I))/COS(WRKANG(I)))**2.
-------------------------COMMENT-------------------------
```

The total number of pit increments 'MAXN' reported by the preceding iteration cycle is required to modify the input of the following 'READ' statement.

```
-----------------------------------------------------------
      READ (5,1) MAXN,S1(1),S2(1),D,INFRC,INFRP,TONF(1),
     1TONF(2)
    1 FORMAT (I2,2F8.2,F5.2,2F6.3,2F9.1)
      READ (5,2) (CONCAP(I),I=1,3),(REFCAP(J),J=1,3),
     1REC(K),K=1,3)
    2 FORMAT (3F11.1/3F9.1/3F5.2)
      READ (5,3) MINCAP
    3 FORMAT (F11.1)
      READ (5,12) MINC,CONC,FIXC
   12 FORMAT (2F5.2,F10.1)
-------------------------COMMENT-------------------------
```

The two data sets 'YR' and 'CONCAP' are used to update the iterative solution. The data sets 'FQGRAD', 'IGRAD1', 'IGRAD2' and 'TOIO' are read from the disk pack where they were

stored rather than being recalculated.

```
038       READ (11) (TOPO(N),YR(N),N=1,NBLKS)
039       READ (12) ((ORDER(M,N),M=1,MAXG),N=1,MAXN)
040       READ (13) (((IGRAD(I,J,K),IGRAD1(I,J,K),
         1IGRAD2(I,J,K),I=1,NI),J=1,NJ),K=1,NK)
041       DO 13 N=1,99
042       T=N
043       X=(1.+INFRC)**T
044       Y=(1.+INFRP)**T
045       E(N)=Y*MINC
046       C(N)=X*CONC
047       R(N)=X*DEFC(1)
048       F(N)=Y*MINC
049       S2(N)=Y*S2(1)
050       S1(N)=Y*S1(1)
051    13 CONTINUE
052       CALL COG (AVEG,YIELD)
```

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

DO loop 38 has been added to this module to make the first estimate of the time dimension for the F-G net cash value distribution 'VAL' and the average unit production costs 'E', 'C' and 'D'.

```
053    92 DO 34 T=1,L
054       LC=0
055       MC=0
056       SUME=0
057       SUMC=0
058       SUMR=0
059       DO 34 K=1,NK
060       DO 34 J=1,NJ
061       DO 34 I=1,NI
062       LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
063       IF (TOPO(LOC)) 16,30,14
```

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

The time dimension 'T' and grid-block excavation time 'IT' were added to DO loop 34 with two statements in Lines 061, 062 and 063 = 071. Note that the time is assumed to be the first year for the grid-blocks whose pit production has not been calculated in the first module 'MPIT'. This assumption eventually becomes redundant as the solution converges.

```
   14 IF (YR(LOC)) 15,15,17
   15 M=ONE
      IF (T-M) 16,16,34
   16 IF (EQGRAD(I,J,K)-AVEG) 19,23,23
   17 IF (YR(LOC)-L) 18,18,261
  261 M=ONE
      GO TO 16
   18 IF (T-YR(LOC)) 34,262,34
  262 M=T
      IF (EQGRAD(I,J,K)-CUTOFF(T)) 19,23,23
   19 LC=LC+ONE
      REV=0.0
      IF (K-10) 20,20,21
   20 N=ONE
      GO TO 22
   21 N=TWO
   22 X=(OB(N)+(K*OBCD(N)))*((1.+INFRC)**M)
      SUME=SUME+X
      COST=(X+(F(M)/MIN(M)))*TONF(N)
      GO TO 33
   23 LC=LC+ONE
      MC=MC+ONE
      IF (K-10) 24,24,25
   24 N=ONE
      GO TO 26
   25 N=TWO
   26 I1=IGRAD1(I,J,K)
      I2=IGRAD2(I,J,K)
      REV=((FLOAT(I1)/GS*S1(M))+(FLOAT(I2)/GS*S2(M)))*
     1TONF(N)*YIELD
      X=(OB(N)+(K*OBCD(N)))*((1.+INFRC)**M)
      Y=(OB(N)+(K*OBCD(N)))*((1.+INFRC)**M)
      SUME=SUME+X
      IF (IGRAD2(I,J,K)) 27,27,28
   27 NN=ONE
      GO TO 29
   28 NN=TWO
   29 X=(BILC(NN))*((1.+INFRC)**M)
      SUMC=SUMC+Y+X+C
      IF (ABS(((66*3*I)-(277*S*J)-(37*625*K)+3274*675)
     1/288)-3.0) 30,30,31
     1-3.0) 30,30,31
   30 NN=ONE
      GO TO 32
   31 NN=TWO
```

```
   32 W=(REFC(NN))*((1**INFRC)**")
      SUMR=SUMR+W
      COST=(Y+Z+(W*YIELD*((FLOAT(I1+I2))/GS))+(F(M)/
     1MIN(M)))*TONF(N)
   33 VAL(LOC)=REV-COST
   34 CONTINUE
      IF (SUME) 36,36,35
   35 E(T)=SUME/LC
   36 IF (SUMC) 38,38,37
   37 C(T)=SUMC/MC
      R(T)=SUMR/MC
   38 CONTINUE
```

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

The re-evaluated unit production costs are stored on a disk pack by the following statement to replace previous estimates.

```
      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 ('1',' AVERAGE UNIT PRODUCTION COSTS')
      WRITE (6,241)
  241 FORMAT ('0','  E',1X,'  C',2X,'  R',3X,'REF')
      WRITE (6,242) (T,E(T),C(T),R(T),T=1,L)
  242 FORMAT (I3,2F5.2,F7.3)
      CALL PD
      PNEW=0.0
      DO 39 N=1,L
   39 PNEW=PROFIT(N)+PNEW
      WRITE ( 6,222) PNEW
      MAXN=(L-ONE)/5*ONE
  222 FORMAT (1X,' TOTAL REVENUE=',F14.1)
      WRITE (6,221) MAXN,L
  221 FORMAT (1X,' MAXN=',I4,'L=',I4)
      DO 41 M=1,99
      DO 41 N=1,20
   41 OREPES(M,N)=0.0
      DO 48 K=1,NK
      DO 48 J=1,NJ
      DO 48 I=1,NI
      LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
      IF (TOPO(LOC)) 48,48,42
   42 IF (YR(LOC)) 48,48,51
   51 DIFINE=(YR(LOC)-ONE)/5*ONE
```

```
144   231 IF (K-10) 43,43,44
145    43 MM=ONE
146       GO TO 45
147    44 MM=TWO
148    45 IF (EQGRAD(I,J,K)-GI) 46,46,71
149    46 M=ONE
150       GO TO 47
151    71 M=EQGRAD(I,J,K)/GI
152    47 ORERES(M,PITINC)=ORERES(M,PITINC)+TONT(MM)
153    48 CONTINUE
-------------------------COMMENT-------------------------

Additional external data sets are created by the following
four statements to store the updated variables 'YR' and
'ORERES' for the next iteration cycle. The 2-D net cash
value distribution 'VAL' is stored for informational pur-
poses only. The 'TOPO' data set is stored rather than being
recalculated for each module.

-----------------------------------------------------------
154       WRITE (14) (TOPO(N),YR(N),VAL(N),N=1,NBLKS)
155       END FILE 14
156       WRITE (15) ((ORERES(M,N),M=1,MAXG),N=1,MAXN)
157       END FILE 15
158    99 WRITE (6,201)
159   201 FORMAT (' ANNUAL PIT OUTLINE')
160       DO 211 N=1,NJ
161   211 SURFC(N)=0
162       DO 100 N=1,L
163       WRITE (6,202) N
164   202 FORMAT ('1',' TOPOGRAPHIC MAP OF MINE IN YEAR',I3)
165       DO 100 I=1,NI
166       DO 104 J=1,NJ
167       M=0
168       K=ONE
169   151 IF (K-NK) 154,154,175
170   154 LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
171       IF (TOPO(LOC)) 152,152,105
172   105 IF (YR(LOC)) 175,175,190
173   190 IF (YR(LOC)-M) 152,152,175
174   152 M=YR
175       K=K+ONE
176       GO TO 151
177   175 SURFC(J)=M
178   104 CONTINUE
179       WRITE (6,203) (SURFC(LEVEL),LEVEL=1,NJ)
```

```
100 CONTINUE
203 FORMAT ('0',30I3)
 49 STOP
    DEBUG SUBTRACE,SUBCHK
    END
```

8. 3. 4 A LIST OF MAIN 3

After the first two iteration cycles, only one module 'MP3' 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:

```
      DOUBLE PRECISION CAP(3,3,3),CONCAP(3),REFCAP(3),
     1MINCAP,OBERES(99,20),F(99),MIN(99),Q6(20),
     2Q7(65,20),Q8(65,20),Q9(65,20),Q10(65,20),
     3Q11(65,20),Q12(3,3,3),Q14(99),POLD,PNEW,DABS,FIXC,
     4Q14A(99)
      REAL MILC(2),INFRC,INFRP,MINC
      INTEGER CUTOFF(99),ONE/1/,TWO/2/,GI
      INTEGER*2 TOPO(13500),YR(13500),PITINC,SURFC(30),
     1Q33(13500),Q34(13500),Q35(13500),Z,SCORE(750),
     2IGRAD1(30,25,18),IGRAD2(30,25,18),EIGRAD(30,25,18),
     3AVEG
      DIMENSION DB(2),DR(2),ODCI(2),ORCI(2),REFC(2),
     1PITANG(2),WRKANG(2),SLOPE(2),WSLOPE(2),S1(99),
     2S2(99),TONE(2),REC(3),L(99),C(99),P(99),PROFIT(99),
     3CAPOB(99),WASTE(99),VAL(13500)
      COMMON OBERES,CONCAP,REFCAP,MINCAP,F,Q6,Q7,Q8,Q9,
     1Q10,Q11,Q12,MIN,Q14,Q14A,CAP,PITANG,WRKANG,SLOPE,
     2WSLOPE,S1,S,C,P,Z,TONE,PROFIT, CAPOB,WASTE,REC,
     3VAL,GS,NI,NJ,NK,NBLKS,MAXN,MAXG,CUTOFF,L,Q33,Q34,
     4Q35,SURFC,TOPO,YR
```

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

The mine's life span 'L' reported by the results of the preceding iteration cycle's subroutine COG2 is required to modify the input of the following statement.

```
      READ (5,62) MAXN,L
   62 FORMAT (2I3)
      READ (5,4) NI,NJ,NK,GS
      NBLKS=NI*NJ*NK
    4 FORMAT (3I3,F7.1)
      GI=5
```

```
      DO 260 K=1,NK
      DO 260 J=1,NJ
      DO 260 I=1,NI
      IGRAD1(I,J,K)=0
      IGRAD2(I,J,K)=0
  260 EQGRAD(I,J,K)=0
      DO 210 LOC=1,NBLKS
      VAL(LOC)=0.0
      YR(LOC)=0
  210 CONTINUE
      READ (5,60) (((CAP(I,J,K),I=1,3),J=1,3),K=1,3)
   60 FORMAT (3F12.1)
      READ (5,61) (OR(I),OR(I),OLCI(I),ORCI(I),AILC(I),
     1REFC(I),PITANG(I),WRKANG(I),I=1,2)
   61 FORMAT (5F6.3,F7.3,2F7.4)
      DO 63 I=1,2
      SLOPE(I)=(SIN(PITANG(I))/COS(PITANG(I)))**2.
   63 WSLOPE(I)=(SIN(WRKANG(I))/COS(WRKANG(I)))**2.
```

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

The total number of pit increments 'MAXN' reported by the preceding iteration cycle is required to modify the input of the following 'READ' statement.

```
      READ (5,1) MAXN,S1(1),S2(1),D,INFRC,INFRP,TONF(1),
     1TONF(2)
    1 FORMAT (I2,2F8.2,F5.2,2F6.3,2F9.1)
      READ (5,2) (CONCAP(I),I=1,3),(REFCAP(J),J=1,3),
     1REC(K),K=1,3)
    2 FORMAT (3F11.1/3F9.1/3F8.2)
      READ (5,3) MINCAP
    3 FORMAT (F11.1)
      READ (5,12) CINC,CONC,FIXC
   12 FORMAT (2F8.2,F10.1)
```

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

The unit production costs and selling prices estimated by DO loop 13 are replaced by overwriting them with the more accurate ones read in from the preceding iteration cycle (Line 057). However, the values read in from data set 1a have only been computed for the mine's life span 'L' of the preceding iteration cycle. Therefore, DO loop thirteen is required to estimate these costs for any years exceeding the previous cycle's mine life 'L', if the mine life of the current cycle exceeds that of the previous cycle. This

estimate becomes redundant as the solution converges.

```
------ ---------------------------------------------------------
038        DO 13 N=1,99
039        T=N
040        X=(1.+INFRC)**T
041        Y=(1.+INFRP)**T
042        E(N)=X*MINC
043        C(N)=Y*CONC
044        R(N)=X*REFC(1)
045        F(N)=X*FIXC
046        S2(N)=Y*S2(1)
047        S1(N)=Y*S1(1)
048     13 CONTINUE
-------------------------COMMENT-----------------------------
```

The following external data sets (13, 14, 15, & 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) (((IGRAD(I,J,K),IGRAD1(I,J,K),
          1IGRAD2(I,J,K),I=1,NI),J=1,NJ),K=1,NK)
050        READ (14) (TOPO(N),YR(N),VEL(N),N=1,NOLYR)
051        READ (15) ((OREPES(M,N),M=1,MAXG),N=1,MAXN)
052        READ (16) (E(T),C(T),R(T),S2(T),S1(T),F(T),T=1,L)
053        CALL COG (AVEG,YIELD)
054     92 DO 38 T=1,L
055        LC=0
056        MC=0
057        SUME=0
058        SUMC=0
059        SUMR=0
060        DO 34 K=1,NK
061        DO 34 J=1,NJ
062        DO 34 I=1,NI
063        LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
064        IF (TOPO(LOC)) 34,34,14
065     14 IF (YR(LOC)) 15,15,17
066     15 M=ONE
067        IF (T-M) 16,16,34
068     16 IF (IGRAD(I,J,K)-AVEG) 19,23,23
069     17 IF (YR(LOC)-L) 18,18,261
```

```
070   261 M=ONE
071       GO TO 16
072    18 IF (T-YR(LOC)) 34,262,34
073   262 M=T
074       IF (IGRAD(I,J,Y)-CUTOFF(T)) 19,23,23
075    19 LC=LC+ONE
076       REV=0.0
077       IF (K-10) 20,20,21
078    20 M=ONE
079       GO TO 22
080    21 N=TWO
081    22 X=(OB(N)+(K*OBCI(N)))*((1.+INFRC)**M)
082       SUME=SUME+X
083       COST=(X+(F(M)/MIN(M)))*TONF(N)
084       GO TO 33
085    23 LC=LC+ONE
086       MC=MC+ONE
087       IF (K-10) 24,24,25
088    24 N=ONE
089       GO TO 26
090    25 N=TWO
091    26 I1=IGRAD1(I,J,K)
092       I2=IGRAD2(I,J,K)
093       REV=((FLOAT(I1)/GS*S1(N))+(FLOAT(I2)/GS*S2(N)))*
      1TONT(N)*YIELD
094       X=(OB(N)+(K*OBCI(N)))*((1.+INFRC)**M)
095       Y=(OR(N)+(K*ORCI(N)))*((1.+INFRC)**M)
096       SUMX=SUMX+X
097       IF (IGRAD2(I,J,K)) 27,27,28
098    27 NN=ONE
099       GO TO 29
100    28 NN=TWO
101    29 Z=(MILC(MN))*((1.+INFRC)**M)
102       SUMC=SUMC+Y+X+Z
103       IF (ABS(((60.*J+I)-(277.5*J)-(37.625*K)+3274.075)
      1/288)-3.0) 30,30,31
      1-3.0) 30,30,31
104    30 NN=ONE
105       GO TO 32
106    31 NN=TWO
107    32 W=(*IRC(NN))*((1.+INFRC)**M)
108       SUMC=SUMC+W
109       COST=(Y+Z+(W*YIELD*((FLOAT(I1+I2))/GS))+(REV/
      1MIN(M)))*TONF(N)
110    33 VAL(LOC)=REV-COST
111    34 CONTINUE
```

```
112        IF (SUME) 36,36,35
113     35 E(T)=SUME/LC
114     36 IF (SUMC) 38,38,37
115      7 C(T)=SUMC/MC
116        R(T)=SUMR/MC
117     38 CONTINUE
```

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

The external data sets--'YR' only of 14, 15 and 16 (Lines 118 and 155 - 158)--used to update the solution are the same for all iteration cycles from the third on. Therefore, the obsolete data is lost when the current updating information is over-written on the same data sets. The 'VAL' data set is updated for informational purposes only. The 'TOPO' data set is stored rather than being recalculated for each iteration cycle.

```
118        WRITE (16) (E(T),C(T),R(T),S2(T),S1(T),F(T),T=1,L)
119        END FILE 16
120        WRITE (6,240)
121    240 FORMAT ('1',' AVERAGE UNIT PRODUCTION COSTS')
122        WRITE (6,241)
123    241 FORMAT ('0','  E',1X,'  C',2X,'  R',3X,'PEP')
124        WRITE (6,242) (T,E(T),C(T),R(T),T=1,L)
125    242 FORMAT (I3,2F5.2,F7.3)
126        CALL PD
127        PNEW=0.0
128        DO 39 N=1,L
129     39 PNEW=PROFIT(N)+PNEW
130        WRITE ( 6,222) PNEW
131        MAXN=(L-ONE)/N+ONE
132    222 FORMAT (1X,' TOTAL REVENUE=',F14.1)
133        WRITE (6,221) MAXN,L
134    221 FORMAT (1X,' MAXN=',I4,'L=',I4)
135        DO 41 N=1,99
136        DO 41 K=1,20
137     41 ORERES(K,N)=0.0
138        DO 48 K=1,NK
139        DO 48 J=1,NJ
140        DO 48 I=1,NI
141        LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
142        IF (TOPO(LOC)) 48,48,42
143     42 IF (YR(LOC)) 48,48,51
144     51 DIFFNCE(YR(LOC)-ONE)/N+ONE
145    231 IF (F-10) 43,43,44
```

```
146    43 MM=ONE
147       GO TO 45
148    44 MM=TWO
149    45 IF (EQGRAD(I,J,K)-GI) 46,46,71
150    46 M=ONE
151       GO TO 47
152    71 M=EQGRAD(I,J,K)/GI
153    47 ORERES(M,PITINC)=ORERES(M,PITINC)+TONF(MM)
154    48 CONTINUE
155       WRITE (14) (TOPO(N),YR(N),VAL(N),N=1,NBLKS)
156       END FILE 14
157       WRITE (15) ((ORERES(M,L),M=1,MAXG),N=1,MAXN)
158       END FILE 15
159    50 WRITE (6,201)
160   201 FORMAT (' ANNUAL PIT OUTLINE')
161       DO 211 N=1,NJ
162   211 SURFC(N)=0
163       DO 100 N=1,L
164       WRITE (6,202) N
165   202 FORMAT ('1',' TOPOGRAPHIC MAP OF MINE IN YEAR',I3)
166       DO 100 I=1,NI
167       DO 104 J=1,NJ
168       M=0
169       K=ONE
170   151 IF (K-NY) 154,154,175
171   154 LOC=NJ*NI*(K-ONE)+NI*(J-ONE)+I
172       IF (TOPO(LOC)) 152,152,105
173   105 IF (YR(LOC)) 175,175,110
174   110 IF (YR(LOC)-N) 152,152,175
175   152 M=K
176       K=K+ONE
177       GO TO 151
178   175 SURFC(J)=M
179   104 CONTINUE
180       WRITE (6,203) (SURFC(LEVEL),LEVEL=1,NJ)
181   100 CONTINUE
182   203 FORMAT ('0',30I3)
183    49 STOP
184       DEBUG SUBTRACE,SUBCHK
185       END
```

8. 4 SUBROUTINES COG1 AND COG 2

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 'OREBES'.

LINES 053 - 056

Subroutine 'COGSUB' is called to compute the following information for the combination of plant capacities (MINCAP, CONCAP, REFCAP, REC) named by (I,J,K):

A) the annual cut-off grades 'CUTOFF';

B) the annual production targets 'MIN', 'CON' and 'OREF'; and

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 maximum present value is selected later by replacing this subroutine with subroutine 'COG2' in subsequent iteration cycles.

LINES 057 - 061

These statements execute printer output used to monitor the

solution as described and illustrated in Section 7•2•

LINES 070 - 073

An average 'AVEG' of the annual cut-off grades 'CUTOFF' is computed•

8• 4• 2 A LIST OF COG1

```
001        SUBROUTINE COG (AVEG,YIELD)
002        DOUBLE PRECISION MAT(20),ORERES(99,20),
          WASTE(65,20),ORE(65,20),REF(20),DFLOAT,PROD(65,20),
          2R1(65,20),R2(65,20),R3(65,20),PVMAX,PV(3,3,3),
          3CAP(3,3,3),Q2(3),Q3(3),Q4,Q5(99),MIN(99),CON(99),
          4QREF(99),P(99)
003        INTEGER CUTOFF(99),ONE/1/,GUM1,NUM,GI
004        INTEGER*2 AVEG,Q33(13500),Q34(13500),Q35(13500),
          1Q37(30),Q38(13500),Q40(13500)
005        DIMENSION REC(3),Q16(2),Q17(2),Q18(2),Q19(2),
          1Q20(99),Q22(99),Q23(99),Q24(99),Q25(2),Q26(99),
          2Q27(99),Q28(99),Q29A(13500)
006        COMMON ORERES,Q2,Q3,Q4,Q5,MAT,ORE,PROD,R1,R2,R3,PV,
          1MIN,CON,QREF,CAP,Q16,Q17,Q18,Q19,Q20,Q21,Q22,Q23,
          2Q24,Q25,Q26,Q27,Q28,REC,Q29A,GS,GI,NS,SK,NBLKS,
          3MAXN,MAXG,CUTOFF,L,Q33,Q34,Q35,Q37,Q38,Q40
007        L=0
-----------------------COMMENT-----------------------

'GI' is given by the user and is equal to 'GI' in Line 011
of 'MAIN1'•

-----------------------------------------------------------
008        GI=5
009        DO 351 N=1,20
010        MAT(N)=0
011        REF(N)=0•0
012        DO 351 M=1,65
013        PROD(M,N)=0•0
014        WASTE(M,N)=0•0
015        ORE(M,N)=0•0
016        R1(M,N)=0•0
017        R2(M,N)=0•0
018        R3(M,N)=0•0
019    351 CONTINUE
020        DO 308 M=1,MAXN
021        DO 308 N=1,MAXG
```

```
022   308 MAT(N)=MAT(N)+ORERES(M,N)
023       DO 309 N=1,MAXN
024       WASTE(1,N)=ORERES(1,N)
025       IF (MAT(N)-WASTE(1,N)) 2,2,3
026     2 ORE(1,N)=ORERES(1,N)
027       GO TO 4
028     3 ORE(1,N)=MAT(N)-WASTE(1,N)
029     4 DO 309 M=2,MAXG
030       WASTE(M,N)=WASTE(M-1,N)+ORERES(M,N)
031   309 ORE(M,N)=MAT(N)-WASTE(M,N)
032       DO 310 N=1,MAXN
033       REF(N)=ORERES(1,N)*(GI/GS)
034       DO 310 M=2,MAXG
035   310 REF(N)=REF(N)+(DFLOAT(M*GI)/GS)*ORERES(M,N)
036       DO 311 N=1,MAXN
037       PROD(1,N)=REF(N)
038       DO 311 M=2,MAXG
039       IF (ORE(M,N)) 311,311,1
040     1 PROD(M,N)=PROD(M-1,N)-(DFLOAT(M*GI)/GS)*ORERES(M,N)
041   311 CONTINUE
042       DO 312 N=1,MAXN
043       DO 312 M=1,MAXG
044       R1(M,N)=ORE(M,N)/MAT(N)
045   312 R2(M,N)=PROD(M,N)/MAT(N)
046       DO 353 N=1,MAXN
047       DO 353 M=1,MAXG
048       IF (ORE(M,N)) 354,354,355
049   354 R3(M,N)=0
050       GO TO 353
051   355 R3(M,N)=PROD(M,N)/ORE(M,N)
052   353 CONTINUE
053       I=ONE
054       J=ONE
055       K=ONE
056   314 CALL COGSUB (I,J,K,P)
057       WRITE (6,6)
058     6 FORMAT ('1','DISCOUNTED PROFITS FROM THE PRODUCTION
          1TARGETS AT EACH PLANT COORDINATES')
059       WRITE (6,5) (((PV(II,JJ,KK),II=1,M),JJ=1,J),KK=1,I)
060     5 FORMAT (3F13.1)
061       YIELD=R1(K)
062       WRITE (6,7)
063     7 FORMAT ('1','COST-PER GRADES & TARGETS FOR THE PLANT
          1COORDINATES YIELDING THE GREATEST EXCESS OF PRESENT
          2VALUE OVER CAPITAL COST')
064       WRITE (6,8) PV(I,J,K),R2(I),R3(I),R1(K)
```

```
065      8 FORMAT ('0','PRESENT VALUE=',F13.1,'CONCENTRATOR CA
         1PACITY=',F11.1,'PRELIMINARY CAPACITY=',F9.1,'OVERALL R
         2ECOVERY EFFICIENCY=',F5.2)
066        WRITE (6,9)
067      9 FORMAT ('0',' YR  COG',4X,'MIN',8X,'CON',8X,'REF',
         1'PROFIT')
068        WRITE (6,10)  (N,CUTOFF(N),MIN(N),CON(N),REF(N),
         1P(N),N=1,L)
069     10 FORMAT (I3,I5,3F10.1,F13.1)
070        SUM=0
071        DO 165 N=1,L
072    165 SUM=SUM+CUTOFF(N)
073        AVEG=SUM/L
074        RETURN
075        DEBUG SUBTRACE,SUBCHK
076        END

0.4.3 A LIST OF COG2

001        SUBROUTINE COG (AVEG,YIELD)
002        DOUBLE PRECISION MAT(20),ORERES(99,20),
          WASTE(65,20),ORE(65,20),REF(20),DELORT,PROD(65,20),
          2R1(65,20),R2(65,20),R3(65,20),PVMAX,PV(3,3,3),
          3CAP(3,3,3),Q2(3),Q3(3),Q4,Q5(99),MIN(99),CON(99),
          4QREF(99),P(99)
003        INTEGER CUTOFF(99),ONE/1/,SUM1,SUM,GI
004        INTEGER*2 AVEG,Q33(13500),Q34(13500),Q35(13500),
          1Q37(30),Q38(13500),Q40(13500)
005        DIMENSION REC(3),Q16(2),Q17(2),Q18(2),Q19(2),
          1Q20(99),Q22(99),Q23(99),Q24(99),Q25(2),Q26(99),
          2Q27(99),Q29(99),Q30(13500)
006        COMMON P,Q2,Q3,Q4,Q5,MAT,ORE,PROD,R1,R2,R3,PV,
          1MIN,CON,Q22,Q23,Q16,Q17,Q18,Q19,Q20,Q21,Q24,Q25,
          2Q26,Q29,Q30,Q27,Q28,Q36,Q39,GI,Q31,Q32,Q33,Q34,Q35,
          3MAXN,MAX3,CUTOFF,L,Q38,Q38,Q40,Q37,Q37,Q40
007        L=0
008        GI=5
009        DO 351 N=1,20
010        MAT(N)=0
011        REF(N)=0.0
012        DO 351 M=1,65
013        PROD(M,N)=0.0
014        WASTE(M,N)=0.0
015        ORE(M,N)=0.0
016        R1(M,N)=0.0
017        R2(M,N)=0.0
```

```
018         R3(M,N)=0.0
019     351 CONTINUE
020         DO 309 N=1,MAXN
021         DO 309 M=1,MAXG
022     308 MAT(N)=MAT(N)+ORERES(M,N)
023         DO 300 N=1,MAXN
024         WASTE(1,N)=ORERES(1,N)
025         IF (MAT(N)-WASTE(1,N)) 2,2,3
026       2 ORE(1,N)=ORERES(1,N)
027         GO TO 4
028       3 ORE(1,N)=MAT(N)-WASTE(1,N)
029       4 DO 309 M=2,MAXG
030         WASTE(M,N)=WASTE(M-1,N)+ORERES(M,N)
031     309 ORE(M,N)=MAT(N)-WASTE(M,N)
032         DO 310 N=1,MAXN
033         REF(N)=ORERES(1,N)*(GI/GS)
034         DO 310 M=2,MAXG
035     310 REF(N)=REF(N)+(DFLOAT(M*GI)/GS)*ORERES(M,N)
036         DO 311 N=1,MAXN
037         PROD(1,N)=REF(N)
038         DO 311 M=2,MAXG
039         IF (ORE(M,N)) 311,311,1
040       1 PROD(M,N)=PROD(M-1,N)-(DFLOAT(M*GI)/GS)*ORERES(M,N)
041     311 CONTINUE
042         DO 312 N=1,MAXN
043         DO 312 M=1,MAXG
044         R1(M,N)=ORE(M,N)/MAT(N)
045     312 R2(M,N)=PROD(M,N)/REF(N)
046         DO 353 N=1,MAXN
047         DO 353 M=1,MAXG
048         IF (ORE(M,N)) 354,354,355
049     354 R3(M,N)=0
050         GO TO 353
051     355 R3(M,N)=PROD(M,N)/ORE(M,N)
052     353 CONTINUE
-------------------------COMMENT-------------------------

The annual cut-off grades 'CUTOFF' and production targets
'MIN' 'CON' and 'SALE' are calculated for all plant combina-
tions (I,J,K) rather than just the single trial combination
selected in subroutine 'CONST'.

---------------------------------------------------------

053         DO 314 I=1,3
054         DO 314 J=1,3
055         DO 314 K=1,3
```

```
056   314 CALL COGSUB (I,J,K,P)
057       WRITE (6,6)
058     6 FORMAT ('1','DISCOUNTED PROFITS FROM THE PRODUCTION
         1TARGETS OF EACH PLANT COMBINATION')
059       WRITE (6,5) (((PV(II,JJ,KK),II=1,3),JJ=1,3),KK=1,3)
060     5 FORMAT (3F13.1)
```

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

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.

```
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)) 161,162,162
066   161 PVMAX=PV(LL,M,N)-CAP(LL,M,N)
067       I=LL
068       J=M
069       K=N
070   162 CONTINUE
```

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

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.

```
071       CALL COGSUB (I,J,K,1)
072       YIELD=REC(K)
073       WRITE (6,7)
074     7 FORMAT ('1','CUT-OFF GRADES & TARGETS FOR THE PLANT
         1combination yielding the greatest excess of present
         2value over capital cost')
075       WRITE (6,3) PV(I,J,K),CUTOFF(I),CUTOFF(J),REC(K)
076     8 FORMAT ('0','PRESENT VALUE=',F11.1,'CONCENTRATOR CA
         1PACITY=',F11.1,'tonnes/ calendar',F9.1,'overall r
         2ecovery efficiency=',F5.2)
077       WRITE (6,9)
078     9 FORMAT ('0',' YR  COG',4X,'MIN',8X,'CON',8X,'REF',
         1'  profit')
079       WRITE (6,10) (N,CUTOFF(N),MIN(N),CON(N),CREF(N),
         1P(n),n=1,1)
```

```
080    10 FORMAT (I3,I5,3F10.1,F13.1)
081       SUM=0
082       DO 165 M=1,L
083   165 SUM=SUM+CUTOFF(M)
084       AVEG=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', 'GRM' and 'GRC'.

LINES 030 - 234

An implicit loop in which the optimum cut-off grades 'CUTOFF' and production targets 'MIN' 'CON' and 'ORES' 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 - 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 consisting entirely from the material within a single pit increment 'N' of 'ORDERS'. The second part performs the same computations for annual production straddling two pit increments.

LINES 044 - 046

The economic cut-off grades 'GM', 'GC' and 'GR' are computed for the year 'L'.

```
080    10 FORMAT (I3,I5,3F10.1,F13.1)
081       SUM=0
082       DO 165 M=1,L
083   165 SUM=SUM+CUTOFF(M)
084       AVEG=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', 'GRM' and 'GRC'.

LINES 030 - 234

An implicit loop in which the optimum cut-off grades 'CUTOFF' and production targets 'ATON' 'CON' and 'QNEF' 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 - 113 and Lines 114 - 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 'NPITS'. The second part performs the same computations for annual production straddling two pit increments.

LINES 044 - 046

The economic cut-off grades 'GM', 'GC' and 'GR' are computed for the year 'L'.

LINES 047 - 077

The optimum cut-off grade 'G' is chosen from the balancing cut-off grades (GMC, GRM, GRC) and the economic cut-off grades (GM, GC, GR) for the year 'L'.

LINES 078 - 111

The plant production targets 'QM', 'QC' and 'QR' are computed for the year 'L'.

LINES 112 - 113

The year 'L' and pit increment 'N' are controlled with these statements by comparing the cumulative quantity of material 'SUMQM' produced from the current pit increment 'N' to the total material 'MAT' contained in that pit increment.

LINES 114 - 117

The production targets (QM, QC, QR) are adjusted for the first part of production years 'L' that straddle two pit increments 'N'.

LINES 118 - 120

The plant capacities (MINCAP, CONCAP, REFCAP) 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'.

LINES 121 - 205

The cut-off grades 'G' and production targets 'QM', 'QC', and 'QR' are calculated for the second part of production years 'L' that straddle two pit increments 'N'. These computations are executed in a similar manner to those between Lines 047 and 111.

LINES 206 - 208

The year 'L' and pit increment 'N' are controlled with these statements.

LINES 209 - 229

The subscript 'M' is deleted from the cut-off grade 'G' and production targets 'QM', 'QC' and 'QR'. These data sets (CUTOFF, MIN, CON, QREF) are supplied to the other subroutine through the COMMON statement.

LINES 225 - 230

The annual profit 'P' and its present value 'SUMPV' 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 'SUMPV' 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),REFCAP(3),MINCAP,V(99),
         1SUMPV(20),R1(65,20),R2(65,20),R3(65,20),P(99),
         2QM(99,20),QC(99,20),QR(99,20),F(99),MAT(20),
         3ORE(65,20),PROD(65,20),MIN(99),CON(99),QMSF(99),
         4SUMPV(99),PV(3,3,3),S1,S2,S3,U1,U2,U3,ORE,PRODS
002       DOUBLE PRECISION DM2,DABS,DFLOAT,Q1(99,20),
         1Q15(3,3,3)
003       INTEGER A(99,20),OBJ/1/,CUTOFF(99),SI
004       INTEGER*2 GMC(20),GME(20),GRC(20),GS(99),GC(99),
         1GR(99),GSC1(99),GMC1(99),GRS1(99),GSC2(99),
         2GMC2(99),GRC2(99),IFIX,Q33(13500),Q34(13500),
         3Q35(13500),Q37(30),Q38(13500),Q40(13500),I
005       DIMENSION S(99),B(99),R(99),A(99),REC(3),Q15(2),
         1Q17(2),Q18(2),Q19(2),Q25(2),Q26(99),Q27(99),
         2Q28(99),Q29A(13500)
006       COMMON Q1,CONCAP,REFCAP,MINCAP,F,MAT,ORE,PROD,R1,
         1R2,R3,PV,MIN,CON,QREF,Q15,Q16,Q17,Q18,Q19,S,D,C,R,
         2E,Q25,Q26,Q27,Q28,REC,Q29A,GS,NI,NJ,NK,NDLKS,MAXN,
         3MAXG,CUTOFF,L,Q33,Q34,Q35,Q37,Q38,Q40
```

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

'GI' is given by the user and is equal to 'GI' in Line 011 of 'MAIN1'.

```
888 GI=5
    DO 160 L1=1,99
160 V(L1)=0.0
900 S1=CONCAP(I)/MINCAP
    S2=REFCAP(J)/MINCAP
    S3=REFCAP(J)/CONCAP(I)
    DO 6 N=1,MAXN
    SUMQM(N)=0
    GMC(N)=0
    DO 2 M=1,MAXG
    IF (DABS(R1(M,N)-S1)-.02) 1,1,2
  1 GMC(N)=M*GI
  2 CONTINUE
    GRM(N)=0
    DO 4 M=1,MAXG
    IF (DABS(R2(M,N)-S2)-.00010) 3,3,4
  3 GRM(N)=M*GI
  4 CONTINUE
    GRC(N)=0
    DO 6 M=1,MAXG
    IF (DABS(R3(M,N)-S3)-.00015) 5,5,6
  5 GRC(N)=M*GI
  6 CONTINUE
 52 DO 151 L1=1,99
    CUTOFF(L1)=0
    P(L1)=0.0
    DO 151 N=1,20
    QM(L1,N)=0.0
    QC(L1,N)=0.0
    QR(L1,N)=0.0
    G(L1,N)=0
151 CONTINUE
    L=0
    N=0
998 N=N+1
    SUMQM(N)=0
 10 L=L+ONE
    GM(L)=(C(L)/((R(L)-P(L))*REC(K)))*GS
    GC(L)=((C(L)+(P(L)+(D*V(L)))/CONCAP(I))/((R(L)-
   1R(L))*REC(K)))*GS
    GR(L)=(C(L)/(R(L)-R(L)-(P(L)+(D*V(L)))/REFCAP(J))*
   1REC(K))*GS
    IF (GM(L)-GMC(N)) 12,11,11
 11 GMC1(L)=GM(L)
    GO TO 70
```

```
050     12 IF (GC(L)-GMC(N)) 13,13,14
051     13 GMC1(L)=GC(L)
052        GO TO 70
053     14 GMC1(L)=GMC(N)
054     70 IF (GM(L)-GRM(N)) 16,15,15
055     15 GRM1(L)=GM(L)
056        GO TO 71
057     16 IF (GR(L)-GRM(N)) 17,17,18
058     17 GRM1(L)=GR(L)
059        GO TO 71
060     18 GRM1(L)=GRM(N)
061     71 IF (GR(L)-GRC(N)) 20,19,19
062     19 GRC1(L)=GR(L)
063        GO TO 72
064     20 IF (GC(L)-GRC(N)) 21,21,22
065     21 GRC1(L)=GC(L)
066        GO TO 72
067     22 GRC1(L)=GRC(N)
068     72 IF (GMC1(L)-GRM1(L)) 102,105,101
069    101 IF (GRC1(L)-GMC1(L)) 104,105,105
070    102 IF (GRC1(L)-GRC1(L)) 105,105,103
071    103 IF (GRM1(L)-GRC1(L)) 107,106,106
072    104 IF (GRM1(L)-GRC1(L)) 106,106,107
073    105 G(L,N)=GMC1(L)
074        GO TO 73
075    106 G(L,N)=GRC1(L)
076        GO TO 73
077    107 G(L,N)=GRM1(L)
078     73 T1=MAT(N)/MINCAP
079        IF (G(L,N)) 153,154,153
080    154 OREG=ORE(1,N)
081        PRODG=PROD(1,N)
082        GO TO 117
083    153 DO 118 M=1,MAXG
084        IF (G(L,N)-M*GI) 111,112,111
085    111 IF (G(L,N)-MAXG*GI) 118,112,116
086    118 CONTINUE
087    116 OREG=0
088        PRODG=0
089        GO TO 117
090    112 OREG=ORE(G(L,N)/GI,N)
091        PRODG=PROD(G(L,N)/GI,N)
092        GO TO 117
093    113 IF (G(L,N)-GI) 114,114,115
094    114 OREG=ORE(1,N)
095        PRODG=PROD(1,N)
```

```
096         GO TO 117
097     115 OREG=ORE(M,N)+(((DFLOAT(M*GI)-DFLOAT(G(L,N)))/GI)
           1*(ORE(M-1,N)-ORE(M,N)))
098         PROD=PROD(M,N)+(((DFLOAT(M*GI)-DFLOAT(G(L,N)))/GI)
           1*(PROD(M-1,N)-PROD(M,N)))
099     117 T2=OREG/CONCAP(I)
100         T3=PRODG/REFCAP(J)
101         IF (T1-T2) 501,502,502
102     501 IF (T2-T3) 506,505,505
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 QM(L,N)=MAT(N)/T
110         QC(L,N)=OREG/T
111         QR(L,N)=PRODG/T
112         SUMQM(N)=SUMQM(N)+QM(L,N)
113         IF (MAT(N)-SUMQM(N)) 23,996,10
114      23 QM2=QM(L,N)+(MAT(N)-SUMQM(N))
115         QC(L,N)=(QM2/QM(L,N))*QC(L,N)
116         QR(L,N)=(QM2/QM(L,N))*QM(L,N)
117         QM(L,N)=QM2
118         B2=MINCAP-QM(L,N)
119         C2=CONCAP(I)-QC(L,N)
120         P2=REFCAP(J)-QR(L,N)
121         U1=C2/B2
122         U2=P2/B2
123         U3=P2/C2
124         N=N+1
125         IF (MAXN-N) 43,130,130
126     130 GMC2(N)=0
127         DO 25 M=1,MAXG
128         IF (ABS(R1(M,N)-U1)-.02) 24,24,25
129      24 GMC2(N)=M*GI
130      25 CONTINUE
131         GRM2(N)=0
132         DO 27 M=1,MAXG
133         IF (ABS(R2(M,N)-U2)-.0001) 26,26,27
134      26 GRM2(N)=M*GI
135      27 CONTINUE
136         GRC2(N)=0
137         DO 28 M=1,MAXG
138         IF (ABS(R3(M,N)-U3)-.00010) 28,28,29
139      28 GRC2(N)=M*GI
```

```
      GO TO 217
  213 IF (G(L,N)-GI) 214,214,215
  214 ORE=ORE(1,N)
      PROD=PROD(1,N)
      GO TO 217
  215 ORE=ORE(M,N)+(((DFLOAT(M*GI)-DFLOAT(G(L,N)))/GI)
     1*(ORE(M-1,N)-ORE(M,N)))
      PROD=PROD(M,N)+(((DFLOAT(M*GI)-DFLOAT(G(L,N)))/GI)
     1*(PROD(M-1,N)-PROD(M,N)))
  217 T2=OREG/T2
      T3=PRODG/P
      IF (T1-T2) 601,602,602
  601 IF (T2-T3) 606,605,605
  602 IF (T1-T3) 606,604,604
  604 T=T1
      GO TO 79
  605 T=T2
      GO TO 79
  606 T=T3
   79 QM(L,N)=MAT(N)/T
      QC(L,N)=OREG/T
      QR(L,N)=PRODG/T
      SUMQM(N)=QM(L,N)
      IF (MAT(N)-SUMQM(N)) 23,996,10
  996 IF (MAXN-N) 43,43,998
   43 DO 45 M=1,L
      MIN(M)=0.0
      CON(M)=0.0
      QREF(M)=0.0
      SUMG=0
      NUM=0
      DO 49 N=1,MAXN
      IF (G(M,N)) 49,49,44
   44 NUM=NUM+ONE
      SUMG=SUMG+G(M,N)
      MIN(M)=MIN(M)+QM(M,N)
      CON(M)=CON(M)+QC(M,N)
      QREF(M)=QREF(M)+QM(M,N)
   49 CONTINUE
      CUTOFF(M)=SUMG/NUM
   45 CONTINUE
      DO 46 M=1,L
   46 P(M)=(S(M)-R(M))*QREF(M)-C(M)*CON(M)-E(M)*MIN(M)-
     1F(M)
      DO 47 M=1,L
      SUMPV(M)=0.
```

```
      DO 47 Y=M,L
47    SUMPV(M)=SUMPV(M)+P(Y)/((1.+D)**(Y-M+1))
      IF (DABS(SUMPV(1)-V(1))-1000000.0) 53,48,48
48    DO 51 M=1,L
51    V(M)=SUMPV(M)
      GO TO 52
53    IV(I,J,K)=V(1)
      RETURN
      DEBUG SUBTRACE
      END
```

8. 6 SUBROUTINE PD

8. 6. 1 COMPUTATIONAL SEQUENCE OF PD

LINES 001 - 006

Specification and input statements.

LINES 007 - 067

Design of the final pit surface.

LINE 013

The grid-blocks (X,Y,Z) with a positive net cash value 'VAL' are selected, in turn, to form the base of the semi-cone closures of the Lerchs Grossman algorithm (2).

LINES 020 - 057

The grid-blocks (I,J,K) required to expose the base grid -block (X,Y,Z) are located by storing their location in the array 'IV'. All grid-blocks (I,J,K) 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 'SCC' are totalled. If this total 'SUM' is positive, the semi-cone closure 'SCC' is included in the pit 'IP'; otherwise, it is excluded.

LINES 068 - 086

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 128 & 132

Subroutine SUBID is called to simulate the pit's waste production.

LINES 137 - 167

Schedules for the annual revenue, costs, ore production and waste production are computed and printed. Additionally the excavation times for ore 'IOR' and waste 'IWA' are combined into one array 'YR'.

8. 6. 2 A LIST OF PD

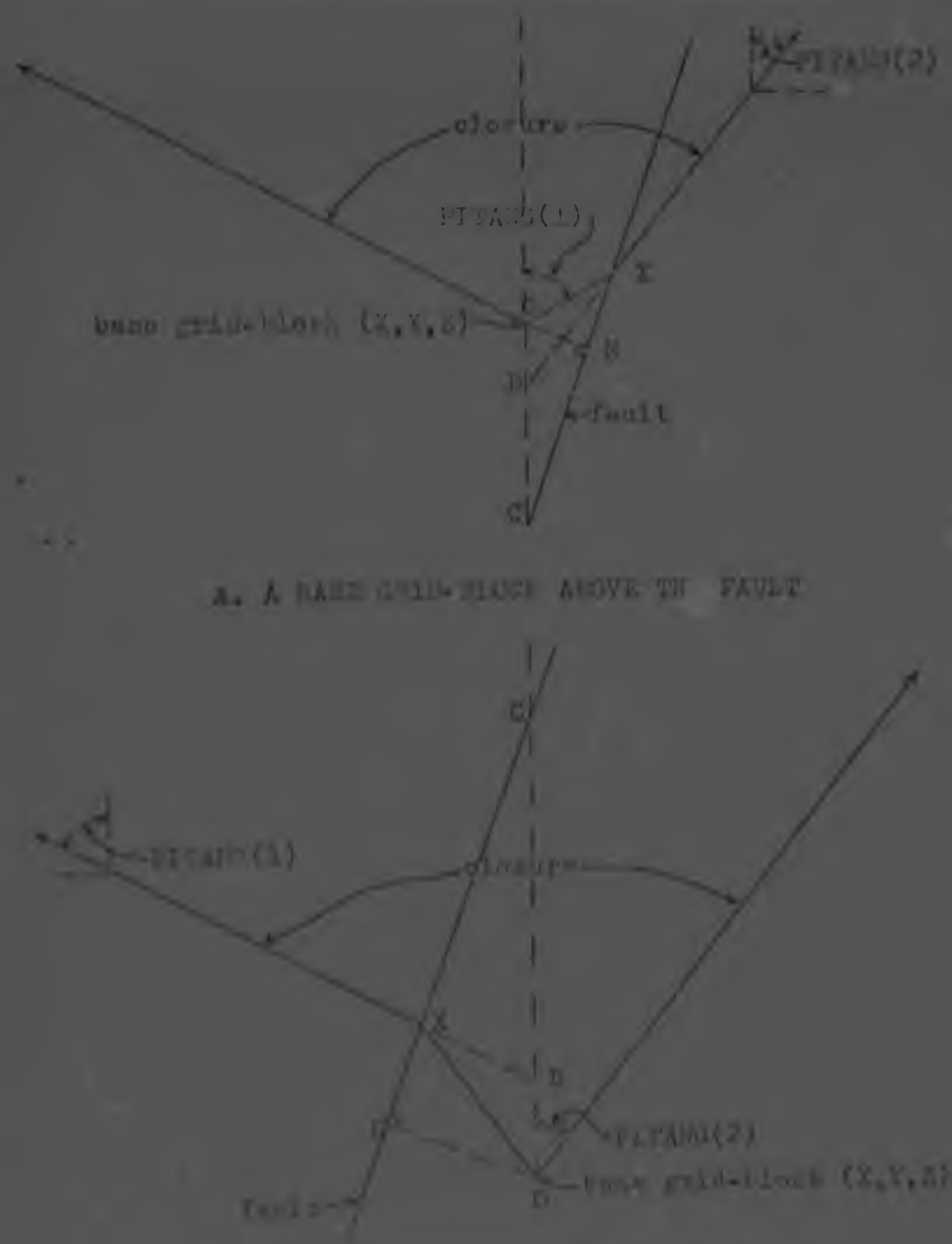
```
001        SUBROUTINE PD
002        DOUBLE PRECISION MAT(99),ORE(99),SUB,SOFE,BVAREE,
          1DRSE,DCLD,CASS,Q1(99,20),Q2(3),Q3(3),Q4,Q5(99),
          2Q6(20),Q7(65,20),Q8(65,20),Q9(65,20),Q10(65,20),
          3Q11(65,20),Q12(3,3,3),Q13(3,3,3),Q14A(99)
003        INTRUTE Q31(99),Q30,QSE/1/,TWO/2/
004        INTEGER*2 IP(13500),IWA(13500),ICL(13500),
          1IOR(13500),TOPO(13500),YR(13500),SURFC(30),X,Y,Z,
          2A,N
005        DIMENSION VAL(13500),PITANG(2),SLOPE(2),TONE(2),
          1WSLOPE(2),PROFIT(99),CAPOR(99),WASTE(99),Q17(2),
          2Q20(99),Q22(99),Q23(99),Q24(99),Q29(3)
006        COMMON Q1,Q2,Q3,Q4,Q5,Q6,Q7,Q8,Q9,Q10,Q11,Q12,MAT,
          1Q13,Q14A,Q15,PITANG,Q17,SLOPE,WSLOPE,Q20,Q21,Q22,
          2Q23,Q24,TONE,PROFIT,CAPOR,WASTE,Q29,VAL,Q3,Q1,Q3,
          3Q1,NBKLS,Q30,MAXO,Q31,Z,IWA,IP,IOR,SURFC,TOPO,YR
007        DO 4 N=1,NBLKS
008        YR(N)=0
009        IP(N)=0
010        IWA(N)=0
011      4 IOR(N)=0
012        DO 200 N=1,NBLKS
013    200 ICL(N)=0
014        IC=0
```

```
015        DO 20 Z=1,NK
016        DO 20 Y=1,NJ
017        DO 20 X=1,NI
018        LOC=NJ*NI*(Z-OSC)+NI*(Y-OSS)+X
019        IF (VAL(LOC)) 20,20,2
```

----------CONSENT----------

The constants (X,Y,Z) of the cone expression in Line 51 are given by the grid co-ordinates of the base grid-block (X,Y,Z). However, the constant 'D' must respond to variable safe wall-angles. For example, the given deposit (see Section 7.1) is divided into two zones by a fault (Line 020) either side of which has a different safe wall-angle 'PITANG'. The constant 'D' is adjusted to construct the overlapping cones required for the closures. The following statements (Lines 020 - 026 and 032 - 050), along with Fig. 8.4, demonstrate how this is accomplished for the example deposit. These statements must be tailored to the user's requirements.

```
020      2 FAULT=(66*3*X)-(277*5*Y)-(37*629*Z)+3274*075
021        IB=0
022        DC=ABS(Z-((66*3*X)-(277*5*Y)+3274*075)/37*625))
023        IF (DC) 210,210,211
024    211 SN=ABS(DC/DC)
025        C=(DX/DC)
026        ROC=ATAN(DEF(1)*-(C*CT)/C)
027    210 DO 240 F=1,NC
028        IF (F-Z) 5,8,208
029      8 DO 14 G=1,NJ
030        DO 14 I=1,NI
031       X=NJ*NI*(G-OSS)+NI*(I-OSS)+I
032        IF ((66*3*X)-(277*5*Y)-(37*625*Z)
          1+3274*075) 101,101,102
033    101 IB=TWO
034        IF (FAULT) 103,103,104
035    103 D=X
036        GO TO 104
037    104 ICC=3*16159-PITANG(1)
038        CD=DX/DC(ROC-ROC)
039        CD=(DX*SIN(PITANG(1)-PITANG(2)))/SIN(PITANG(2))
040        D=D+CD
041        GO TO 102
042    102 IB=ON
043        IF (FAULT) 104,105,102
```



A. A BASE GRID-BLOCK ABOVE TH FAULT

B. A BASE GRID-BLOCK BELOW THE FAULT

FIG. 8.4 CROSS-SECTIONS OF OVERLAPPING CONES USED AS CLOSURES

```
  105 D=Z
      GO TO 108
  106 XOC=PITANG(2)
    1 OX=ON/COS(ABS(NOC-XOC))
    5 OD=(OX*SIN(PITANG(1)-PITANG(2)))/SIN(3.14159-
     1PITANG(1))
      D=Z-OD
  107 IF (D) 14,14,108
  108 CONE=SLOPE(IS)*(K-D)*(K-D)-((2*J-2*Y)**2)-((I-X)
     1**2)
      IF (CONE) 14,12,12
   12 IF (IP(M)) 13,13,14
   13 IB=IB+ONE
      ICL(IB)=M
   14 CONTINUE
  240 CONTINUE
  233 IF (IB) 20,20,174
  174 SUM=0
      DO 15 M=1,IB
   15 SUM=SUM+VAL(ICL(M))
      IF (SUM) 18,18,16
   16 DO 17 M=1,IB
   17 IP(ICL(M))=ICL(M)
   18 DO 19 M=1,IB
   19 ICL(M)=M
   20 CONTINUE
      WRITE (6,99)
   99 FORMAT ('1',' OPTIMUM PIT SURFACE')
      DO 220 N=1,NJ
  220 SURFC(N)=0
      DO 97 X=1,NI
      DO 96 Y=1,NJ
      M=0
      Z=ONE
   91 IF (Z-NK) 87,87,92
   87 LOC=NJ*NI*(Z-ONE)+NI*(Y-ONE)+X
      IF (TOPO(LOC)) 88,88,89
   89 IF (IP(LOC)) 92,92,88
   88 M=Z
      Z=Z+ONE
      GO TO 91
   92 SURFC(Y)=M
   96 CONTINUE
   97 WRITE (6,98) (SURFC(N),N=1,NJ)
   98 FORMAT ('0',[illegible])
-----------------------CONTINE-----------------------
```

The following ore excavation sequence (Lines 088 - 114) illustrates a simulation for one working face that advances in opposite directions on alternate benches parallel to the I co-ordinate axes. 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       SORE=0
092       SWASTE=0
093    31 ISIGN=ONE
094       GO TO 33
095    32 ISIGN=-ONE
096    33 K=K+ONE
097       IF (K-NI) 34,34,160
098    34 N=K-((K/2)*2)+ONE
099       GO TO (35,36),N
100    35 B=-ONE
101       GO TO 37
102    36 B=ONE
103    37 J=J+B
104       IF (J-NJ) 38,39,32
105    38 IF (J-ONE) 31,39,39
106    39 N=J-((J/2)*2)+ONE
107       GO TO (40,41),N
108    40 A=(-ONE)*ISIGN
109       GO TO 42
110    41 A=ISIGN
111    42 I=I+A
112       IF (I-NI) 43,43,37
113    43 IF (I-ONE) 37,44,44
114    44 MC=750*(K-ONE)+30*(J-ONE)+I
115       IF (IOR(MC)) 130,130,42
116   130 IF (IOR(MC)) 42,42,110
117   110 IF (VAL(MC)) 42,42,45
118    45 IF (K-10) 46,46,47
119    46 N=ONE
120       GO TO 48
121    47 N=2
122    48 SORE=SORE+IORE(N)
123       IF (CRIT(N)-SORE) 50,50,49
124    49 IOR(MC)=L
125       GO TO 42
```

```
126    50 IF (SCRE-ORE(L)-(TONF(M)/2.)) 51,51,60
127    51 IOR(MC)=L
128       CALL SUBPD (SCRE,M,M)
129       SCRE=0
130       L=L+ONE
131       GO TO 42
132    60 CALL SUBPD (SCRE,M,M)
133       L=L+ONE
134       SCRE=TONF(M)
135       IOR(MC)=L
136       GO TO 42
137   160 DO 69 M=1,99
138       PROFIT(M)=0.0
139       CAPOB(M)=0.0
140       ORE(M)=0.0
141    69 WASTE(M)=0.0
142       DO 253 Z=1,NK
143       IF (Z-10) 71,71,72
144    71 II=ONE
145       GO TO 73
146    72 II=TWO
147    73 DO 76 Y=1,NJ
148       DO 86 X=1,NI
149       M=NJ*NI*(Z-ONE)+NI*(Y-ONE)+X
150       IF (TOPO(M)) 86,86,260
151   260 IF (IOR(M)) 84,84,180
152   180 IF (IOR(M)) 181,181,83
153   181 IF (IWA(M)) 182,182,84
154   182 IWA(M)=L
155    84 YR(M)=IWA(M)
156       CAPOB(IWA(M))=CAPOB(IWA(M))+VAL(M)
157       WASTE(IWA(M))=WASTE(IWA(M))+TONF(II)
158       GO TO 86
159    83 YR(M)=IOR(M)
160       PROFIT(IOR(M))=PROFIT(IOR(M))+VAL(M)
161       ORE(IOR(M))=ORE(IOR(M))+TONF(II)
162    86 CONTINUE
163   253 CONTINUE
164       WRITE (6,250)
165   250 FORMAT ('1',' SIMULATED PIT PRODUCTION SCHEDULE')
166       WRITE (6,251)
167   251 FORMAT ('0',' Y  OVERBURDEN STRIPPING COSTS ORE
      1 TONNAGE')
168       WRITE (6,252) (M,WASTE(M),CAPOB(M),ORE(M),
      1PROFIT(M),M=1,J)
169   252 FORMAT (I3,4F11.0)
```

```
170       RETURN
171       END
```

8. 7 SUBROUTINE SUBPD

8. 7. 1 COMPUTATIONAL SEQUENCE OF SUBPD

LINES 001 - 006

Specification and input statements.

LINES 007 - 066

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 'IKA'.

LINE 008

The appropriate minimum working wall-angle 'WSLOPE' is selected.

LINES 011 - 066

DO loop 56 increases the working wall-angle 'WSLOPE' in increments 'STEP' between its minimum 'WSLOPE' (Line 10) and its maximum '45.0' (Line 15) until the overburden 'EXASTE' within the waste closure approximates the waste production target 'WAT - ORE'. Then, the accuracy of the working wall-angle 'WSLOPE' is increased by splitting the increment 'STEP', changing its sign and repeating the computations within the scaling 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 limits without satisfying the waste production target. The maximum working wall-angle 'WSLOPE=35' 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 ore grid-blocks 'IORE' that form

```
   29 CONTINUE
      IF (GM(L)-GMC2(N)) 31,30,30
   30 GMC1(L)=GM(L)
      GO TO 75
   31 IF (GC(L)-GMC2(N)) 32,32,33
   32 GMC1(L)=GC(L)
      GO TO 75
   33 GMC1(L)=GMC2(N)
   75 IF (GM(L)-GRM2(N)) 35,34,34
   34 GRM1(L)=GM(L)
      GO TO 76
   35 IF (GR(L)-GRM2(N)) 36,36,37
   36 GRM1(L)=GR(L)
      GO TO 76
   37 GRM1(L)=GRM2(N)
   76 IF (GR(L)-GRC2(N)) 39,38,38
   38 GRC1(L)=GR(L)
      GO TO 77
   39 IF (GC(L)-GRC2(N)) 40,40,41
   40 GRC1(L)=GC(L)
      GO TO 77
   41 GRC1(L)=GRC2(N)
   77 IF (GMC1(L)-GRM1(L)) 202,205,201
  201 IF (GMC1(L)-GMC1(L)) 204,205,205
  202 IF (GRC1(L)-GMC1(L)) 205,205,203
  203 IF (GRM1(L)-GRC1(L)) 207,206,206
  204 IF (GRM1(L)-GRC1(L)) 206,206,207
  205 G(L,N)=GMC1(L)
      GO TO 78
  206 G(L,N)=GRC1(L)
      GO TO 78
  207 G(L,N)=GRM1(L)
   78 T1=MAT(N)/E2
      IF (G(L,N)) 155,152,155
  152 OREG=OPR(1,N)
      PRODG=PROD(1,N)
      GO TO 217
  155 DO 218 M=1,MAXG
      IF (G(L,N)-MAXG) 218,212,211
  211 IF (G(L,N)-MAXG+GI) 218,212,216
  218 CONTINUE
  216 OREG=0
      PRODG=0
      GO TO 217
  212 OREG=OPR(G(L,N)/GI,N)
      PRODG=PROD(G(L,N)/GI,N)
```

the base 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 ore grid-blocks are excavated as well, if they satisfy the semi-cone set.

8. 7. 2 A LIST OF SUBPD

```
001        SUBROUTINE SUBPD (SORE,M,K)
002        DOUBLE PRECISION SORE,SWASTE,Q1(99,20),Q2(3),Q3(3),
          1Q4,Q5(99),Q6(20),Q7(65,20),Q8(65,20),Q9(65,20),
          2Q10(65,20),Q11(65,20),Q12(3,3,3),MAT(99),ORE(99),
          3Q15(3,3,3),Q14A(99),DNEW,DOLD,DABS
003        INTEGER Q31(99),ONE/1/,TWO/2/
004        INTEGER*2 IOR(13500),IP(13500),IWA(13500),
          1TOPO(13500),X,Y,Z,Q37(30),Q40(13500)
005        DIMENSION WORKSL(2),VAL(13500),TOPT(2),Q16(2),
          1Q19(2),WSLOPE(2),Q20(99),Q22(99),Q23(99),Q24(99),
          2Q26(99),Q27(99),Q28(99),Q29(3)
006        COMMON Q1,Q2,Q3,Q4,Q5,Q6,Q7,Q8,Q9,Q10,Q11,Q12,MAT,
          1ORE,Q14A,Q15,Q16,WORKSL,Q18,WSLOPE,Q20,Q21,Q22,
          2Q23,Q24,TOPT,Q26,Q27,Q28,Q29,VAL,NS,NI,NJ,NK,
          3NBLKS,Q30,MAXS,Q31,L,IWA,IP,IOR,Q37,TOPO,Q40
077        STEP=5.0
008        WORKSL=WSLOPE(2)-STEP
009        IF (MAT(L)-ORE(L)) 121,59,121
010    121 DNEW=SORE-MAT(L)
011        DO 56 II=1,3
012     52 DOLD=DNEW
013        WORKSL=WORKSL+STEP
014        IF (WORKSL-WSLOPE(2)) 59,140,140
015    140 IF (WORKSL-35.0) 53,53,53
016     53 SWASTE=0
017        DO 171 Z=1,NK
018        DO 171 Y=1,NJ
019        DO 171 X=1,NI
020        N=NJ*NI*(Z-ONE)+NI*(Y-ONE)+X
021        IF (IWA(N)-Z) 171,173,171
022    173 IWA(N)=0
023    171 CONTINUE
024        DO 56 Z=1,NK
025        IF (K-Z) 232,54,54
026     54 DO 56 Y=1,NJ
027        DO 56 X=1,NI
```

```
      N=NJ*NI*(Z-ONE)+NI*(Y-ONE)+X
      IF (IOR(N)-L) 56,55,56
   55 DO 19 KK=1,NK
      IF (KK-Z) 2,2,56
    2 DO 22 J 1,NJ
      DO 9 I=1,NI
      N=NJ*NI*(KK-ONE)+NI*(J-ONE)+I
      IF (TOPO(N)) 9,9,10
   10 IF (IOR(N)) 1,1,9
    1 IF (IP(N)) 1,9,3
    3 IF (IWA(N)) 4,4,9
```

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

The constants (X,Y,D) of the cone expression in Line 47 are given by the grid co-ordinates of the base grid-blocks (X,Y,Z). 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 039) either side of which has a different minimum working wall-angle 'WRKANG'. The constant 'D' is adjusted to construct the overlapping cones required for the closure. The following statements (Lines 039 - 046), along with Fig. 8.5, demonstrate how this is accomplished for the example deposit. These statements must be tailored to the user's requirements.

```
    4 IF (Z-10) 14,14,15
   14 D=Z
      GO TO 16
   15 IF (KK-10) 17,14,14
   17 CO=Z-10.
      CS=SLPT(WORKSL)*CO
      OD=CO-CS*(COS(WRKANG(1))/SIN(WRKANG(1)))
      D=Z-OD
   16 IF (WORKSL*(KK-D)*(KK-D)-((Z*J-Z*Y)**2)-((I-X)
     1**2)) 9,5,5
    5 IF (KK-10) 6,6,7
    6 MM=ONE
      GO TO 8
    7 MM=NK
    8 IF (VAL(N)) 11,11,12
   11 IWA(N)=L
      SWASTE=SWASTE+TOMP(MM)
      GO TO 9
   12 IOR(N)=L
```

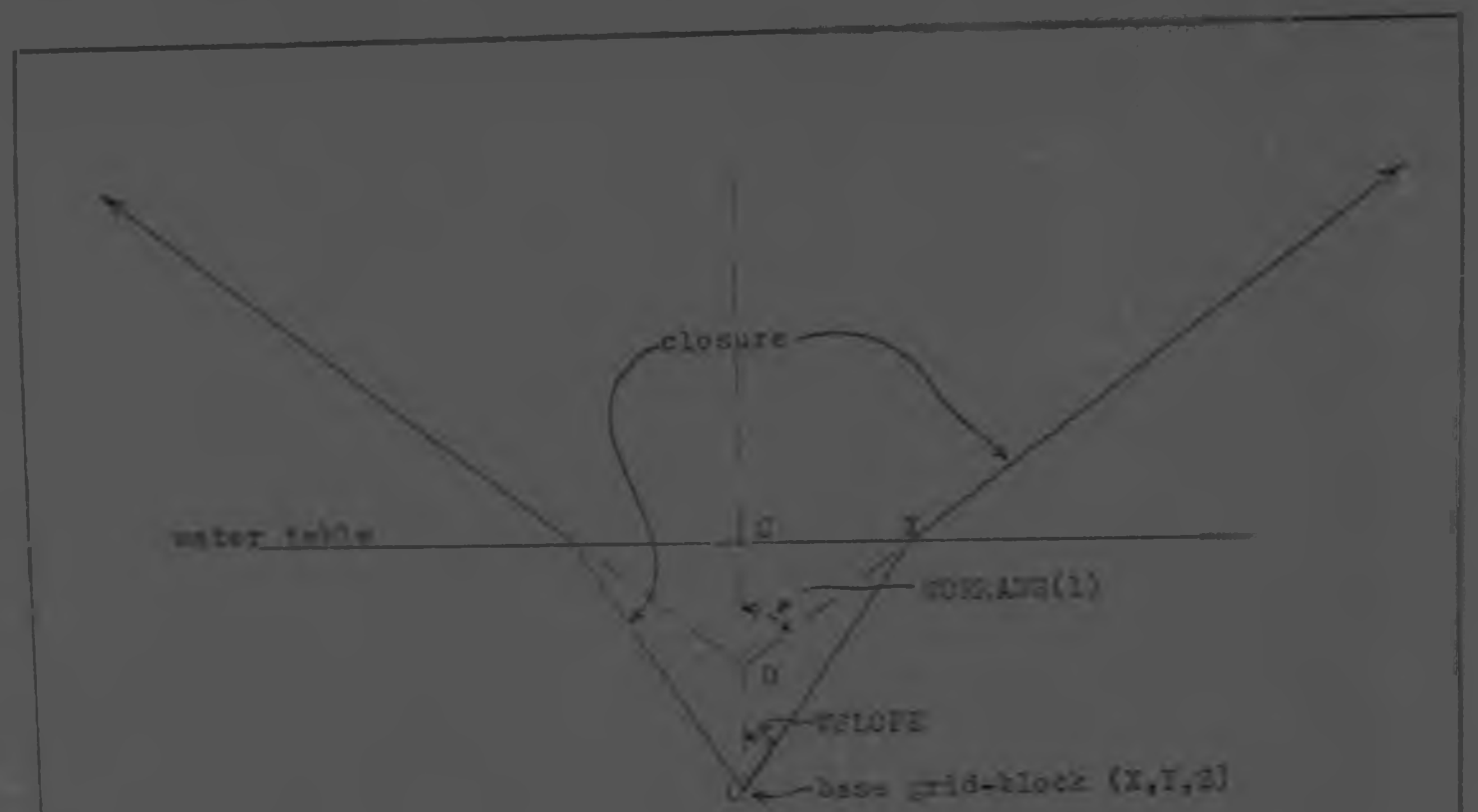


FIG. 8.5 A CROSS-SECTION OF OVERLAPPING CONES USED AS A CLOSURE

```
057        SORE=SORE+TONF(MM)
058      9 CONTINUE
059     22 CONTINUE
060     19 CONTINUE
061     56 CONTINUE
062    232 DNEW=SWASTE-MAT(L)+SORE
063        IF (ABS(DNEW)-TONF(M)) 59,57,57
064     57 IF (DOLD*DNEW) 58,52,52
065     58 STEP=-0.5*STEP
066     59 SWASTE=0
067     23 RETURN
068        END
```

8.8 GLOSSARY

The format of this glossary follows the same pattern as those of Chapters 4 and 5. Wherever a variable name appears twice in the glossary, it is the result of using the same name for two different applications in the XP programs. 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 'Cnn', where 'nn' is an identifying number, are only necessary to match the '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.

AVEG - (I*2) the average cut-off grade over the pit's life span.

C(i) - (R) the annual concentration costs per input ton. See subsection 4.1.2 for a rigorous definition;

i - year.

CAPI(i,k) - (R) the total capital cost of each possible plant combination discounted to the first year at the opportunity cost of investment capital 'r';

i - concentrator identification number
j - refinery identification number
k - overall recovery efficiency identification number.

CAPOB(i) - (R) the annual cost of stripping overburden;

i - year.

CON(i) - (I) the annual production target for the concentrator in input tons;

i - year.

CONCAP(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 'GS' to integer value;

i - year.

D - (R) the opportunity cost of investment capital (MAIN1-1, COG1, 2 and COGS 8).

D - (R) a constant used to define the semi-cone closures (PD & SUBPD).

E(i) - (R) the annual excavation costs per ton of pit output. See Subsection 6.1.2 for a rigorous definition;

i - year.

[illegible] - (I*2) the total equivalent mineral content at each grid-intersection;

I,J,K - I, J and K grid co-ordinates, respectively.

F(i) - (D) the fixed plus administrative costs per year;

i - year.

FIXC - (D) a variable used to transfer the fixed costs 'F' from the card reader to internal storage.

[illegible] - (I) the annual cut-off grades on each pit increment. They are scaled up to integer values with the grade scale factor 'GS';

i - year
j - pit increment.

GM(i), GC(i), GR(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 'GS';

i - year.

GMC(i), GRM(i), GRC(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 'GS';

i - pit increment.

GI - (I) the class interval for the increments in the T-S mineral distribution 'GRERES'.

GRAD1, GRAD2 . . . GRADN - (N) variables used to transfer the proportional mineral grades of the 'n' minerals reported in the user's deposit from external to internal storage.

GS - (S) a scale factor used for converting proportional mineral grades to integers so that they may be used as subscripts in arrays.

ICL(i) - (I*2) a pointer used to indicate the grid-blocks in a test closure of the Dercha-Grossman graph algorithm;

i - grid-block identification number.

IGRAD1(i,j,k), IGRAD2(i,j,k) . . . IGRADN(i,j,k) - (I * 2) the mineral grades of the 'n' minerals reported in the user's deposit at each grid-intersection. They are scaled up to integer values with the grade scale factor 'GS';

i,j,k - I,J and K grid co-ordinates, respectively.

INFRC, INFRP - (S) the interest rates for the time inflation of costs and selling prices, respectively.

IOP(i) - (I*2) the excavation year of the ore 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 (COM1 - [illegible] and [illegible]);

i - pit increment.

MAT(i) - (I) the annual pit production target, ore plus overburden, in tons (FD and SUBFD);

i - year.

MAXG - (I) the maximum equivalent mineral grade 'EQGRAD' reported in the user's deposit.

MAXN - (I) the total number of pit increments.

MILC(i) - (D) the milling cost per input ton for each recoverable mineral of the example deposit. See Section 7.1.

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 ton 'CP' from the card reader to internal storage.

MINCAP - (D) the pit capacity, ore plus overburden, per year measured in tons.

NBLKS - (I) the total number of grid-intersections or grid-blocks.

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

OB(i) - (R) 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.

OBCI(i) - (R) the increased cost of overburden stripping 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.

OR(i) - (R) 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.

ORCI(i) - (R) 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) - (R) the available ore tonnage, in pit increments, for the range of cut-off grades from zero to the maximum reported mineral grade 'GRAD' (CCGI - 2 and CCGSUB);

i - integer cut-off grade in cents
j - pit increment.

ORE(i) - (R) this variable has two different definitions in the PD subroutine. Firstly, it is used as the production target for the concentrator. Thus, it becomes the total simulated ore production in each year. Both data sets are measured in tons of concentrator input(CC);

i - year•

ORERES(i,j) the data set for the T-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 'REF';

i - year•

PITANG(i) - (D) the safe pit wall-angles 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 - (I*2) the pit increment•

PNEW - (D) the total revenue over the mine's life span•

PROD(i,j) - (D) 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) - the total present value of the annual profits 'P' estimated from the production targets 'MIN', 'CON' and 'REF' for each combination of the given plant capacities;

i - concentrator identification number
j - refinery identification number
k - overall recovery efficiency identification number•

PVMAX - (D) the present value of the combination of plant capacities yielding the maximum present value•

QC(i,j), QP(i,j), QR(i,j) - (D) the annual plant production targets for each pit increment in tons for the concentrator

pit and refinery, respectively;

i - year
j - pit increment.

QREF(i) - (D) the annual production target for the refinery;

i - year.

R(i) - (R) the annual cost of refining per ton of product;

i - year.

R1(i,j), R2(i,j), R3(i,j) - (D) the ratios, in each pit increment, of ore 'ORE' to pit material 'REM', refined product 'PROD' to pit material and refined product to ore, respectively, for the range of cut-off grades from zero to the maximum reported mineral grade 'MAXG';

i - integer cut-off grade increment
j - pit increment.

REC(i) - (D) the proportional overall recovery efficiency for each combination of plant capacities;

i - overall recovery efficiency identification number.

RMC(i) - (S) the refining cost per ton of product for the concentrates originating from either side of a zone surrounding the fault given for the example deposit. See Section 7.1.

i - pit zone identification number.

REFCAP(i) - (I) the plant capacity of the refinery in output tons per year;

i - refinery identification number.

S(i) - (D) the annual selling price per product ton of the primary material;

i - year.

S1(i), S2(i), . . . Sn(i) - (D) the annual selling price for product ton of the 'n' minerals recovered in the deposit

deposit (MAIN1 - 3, PD, SUBPD);

; i - year.

S1, S2, S3 - (D) the ratio of concentrator capacity 'CONCAP' to the pit capacity 'PITCAP', refinery capacity 'REFCAP' to pit capacity and refinery capacity to concentrator capacity, respectively, (COGSUB).

SLOPE(i) - (D) this variable is used to define the semi-cone closures of the final pit surface design routine. It is equal to the tangent squared of 'PITANG'.

i - pit zone identification number.

SORE - (D) cumulative total of the simulated ore production in any given year and measured in tons.

SUMPV(i) - (D) the present value of the future profits 'P' in each year;

i - year.

SUMQR(i) - (D) the cumulative total of the annual pit production targets 'QR' in each pit increment;

i - pit increment.

SWASTE - (D) the cumulative total of the simulated overburden production in any given year.

T1, T2, T3, T - (D) the time periods required to process a given amount of pit material at the optimum cut-off grade 'G' for the pit, concentrator and refinery, and the maximum of the three, respectively.

TCNF(i) - (D) the total number of the grid-blocks on each side of the tenth grid-level of the example deposit. See Section 7.1.

i - pit zone identification number.

TOPO(i) - (D) an indicator array for the given deposit's surface topography;

i - grid-intersection identification number.

V(i) - (D) the present value of the future profits 'P' in each year;

i - year.

VAL(i) - (D) the T-S net cash value distribution;

i - grid-block identification number.

WASTE(i,j) - (I) the tonnages of overburden in each pit increment for the range of cut-off grades from zero to the maximum reported mineral grade 'MAXG' (COG1 - 2);

i - integer cut-off grade increment
j - pit increment.

WASTE(i) - (R) the total simulated overburden production in tons for each year (COG1 - 3, PD, SUBPD);

i - year.

WORKSL - (R) this variable is used to define the semi-comp sets of the overall open production simulation. It is the current value of the working wall-angle.

WORKANG(i) - (R) the various pit wall-angle at which the excavation equipment can work on a sub-set of the tenth grid-level of the example deposit. See Section 7.1.

i - pit zone identification number.

WSLOPE(i) - (R) this variable is equal to the tangent squared of 'WORKANG';

i - pit zone identification number.

YR(i) - (I*2) the ore-rating year of each mined grid-block.

i - grid-block identification number.

YIELD - (R) the overall recovery efficiency of the combination 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 correctly illustrated with the following diagram:

MINE PLANNING--
|
|- +TIME FACTOR-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 proposal and any deposit by tailoring its computer programs. These computer programs then simulate:

1) the physical and geologic conditions of the user's 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 Lerchs and Grossmann (5), and

3) the economic planning decisions derived by Lane (6).

This thesis introduces simulation techniques for pit production, physical features and costing. Most important, it also introduces the time factor into planning and evaluation. The incorporation and synthesis of these innovations, as well as the preceding contributions, has this model as

improvement over previous open pit planning systems. It does, in fact, represent a significant development in the field.

This development assures that there is no loss of accuracy as a result of the model being a general planning tool. This is because its computer programmes are tailored to simulate the physio-economic 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 grades with the primary mineral only. The interpolation method used in the model to estimate mineral grades is more adaptable 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 variables. A sensitivity analysis also would reveal any pathological situations not encountered in the testing of the computer programmes.

B) A study of the model to establish whether its lengthy execution time can be cut down. For example, the Grade Prediction Programme could be made more efficient by eliminating some of the repetitious computations. The computations for the T-2 net cash value distribution can be improved by eliminating its outermost DO loop which cycles with time. The time factor could be obtained solely from the indicator array 'YR' 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 mineral situations. Then, rather than tailoring the programmes, the user would merely select those situations peculiar to his needs.

CHAPTER 10 REFERENCES

1) Hartman, B.J., Varga, C.G., A Three Dimensional Optimum Pit Program and a Basis for a Mining Engineering System, Proc. Symposium and Short Course on Computers and Operations Research in Mineral Industries, The Pennsylvania State University, University Park, Pa., V.2, April 1966.

2) International Business Machine Corp., IBM System/360 Operating System FORTRAN Language, IBM Corp., Poughkeepsie, N.Y., 1967.

3) International Business Machine Corp., IBM System/360 Operating System Job Control Language, IBM Corp., Poughkeepsie, N.Y., 1967.

4) Lane, K.F., Choosing the Optimum Cut-Off Grade, Quarterly of the Colorado School of Mines, Golden, Colo., V.59, no.4, p.811, Oct. 1964.

5) Lerchs, H., Grossman, I.F., Optimum Design of Open Pit Mines, Transactions of the Canadian Institute of Mining, V.118, p.17, 1965.

6) Ore, O., Graphs and their Uses, Random House, New York, N.Y., 1963.

7) Pana, S.T., [illegible], a Description of Computer Techniques Used in Mine Planning at the Utah Mine of Kennecott Copper Corporation, Proc. Symposium and Short Course on Computers and Operations Research in Mineral Industries, The Pennsylvania State University, University Park, Pa., V.2, April 1966.

8) Pana, S.T., The Simulation Approach to Open Pit Design, Transactions of the Short Course and Symposium on Computers and Computer Applications in Mining and Exploration, University of Arizona, Tucson, Ariz., March 1965.

9) Pfleider, E.P. (editor), Surface Mining, Seely W. Mudd Series of The American Institute of Mining, Metallurgical and Petroleum Engineers, New York, N.Y., 1968.

10) Spiegel, J.R., Theory and Problems of Statistics, Schaum Publishing Co., New York, N.Y., 1961.

11) Weaver, R.C., Relative Merits of Interpolation and Approximating Functions in the Grade Prediction Problem, Stanford University Publications in the Geological Sciences, Stanford, Calif., V.9, no.1, 1964.

Author Robinson R H
Name of thesis An Economic model of an open pit mine 1969

PUBLISHER:
University of the Witwatersrand, Johannesburg
©2013

LEGAL NOTICES:

Copyright Notice: All materials on the University of the Witwatersrand, Johannesburg Library website are protected by South African copyright law and may not be distributed, transmitted, displayed, or otherwise published in any format, without the prior written permission of the copyright owner.

Disclaimer and Terms of Use: Provided that you maintain all copyright and other notices contained therein, you may download material (one machine readable copy and one print copy per page) for your personal and/or educational non-commercial use only.

The University of the Witwatersrand, Johannesburg, is not responsible for any errors or omissions and excludes any and all liability for any errors in or omissions from the information on the Library website.