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 successive 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 'ORERES'. 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 'ORERES' 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 (K-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 'GRAD1', 'GRAD2'

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 (YR(LOC)) 10,10,3
    3 PITINC=(YR(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)+TONF(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.

'MK' 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.'

'ORERES' 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 was stored in data set 'YR' 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 'ORERES' in statement number eight.

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

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

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 facilities. Finally, that combination of cut-off grade and life span furnishing the highest present value is chosen to exploit the deposit. This final cut-off grade is not necessarily equal to the 'break-even cut-off grade.'

There are a number of deficiencies in this method of determining how to operate on a deposit's resources. Primarily, it ignores the T-S 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 mine's 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 these decisions provide the relationship between the function $P(C(x,y,z),t)$ and the path T of the multiple integral developed in Section 6.2.

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

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.

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

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 next 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 reserves 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:

$$T=Qm/M$$

if 'M' is the maximum pit capacity; and

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

"Given Qm, 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:

$$(s-r)*y*Gm=c$$

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

where:

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

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

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

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

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:

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

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

"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 Gmc is less than Gm, the mine is really the bottleneck in the operation and Gm is the optimum cut-off grade. On the other hand, when Gmc is greater than Gc, the concentrator is the bottleneck and Gc is the optimum cut-off grade." (4, page 319).

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

For the mine and concentrator:

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.

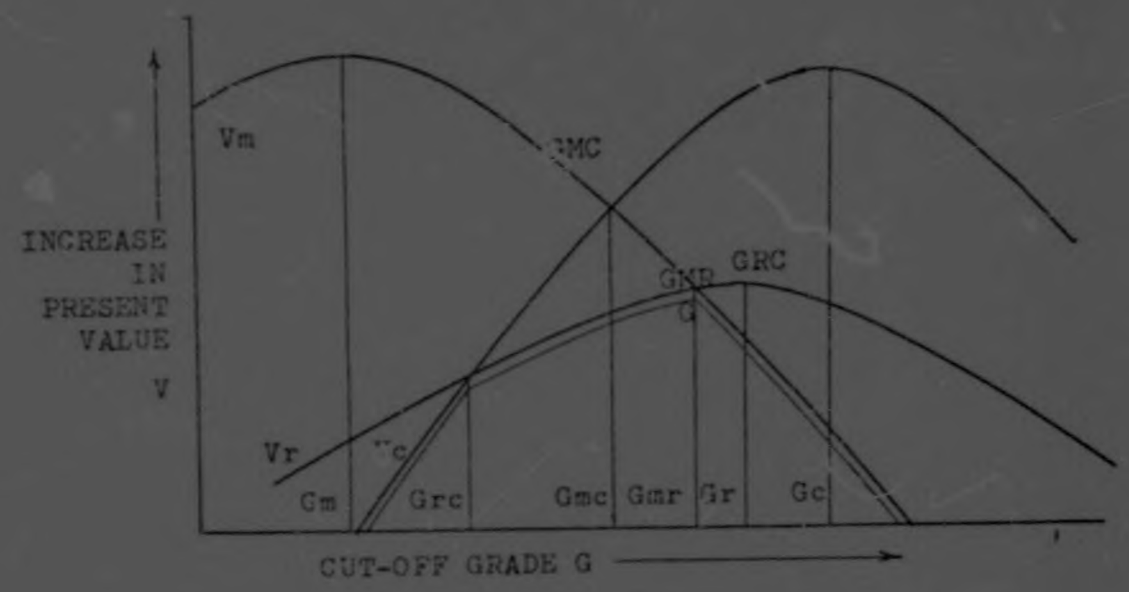
The overall optimum cut-off grade 'G' is selected from GMC, GRC and GMR, and always is the middle value of the three. This can be demonstrated with the three V-curves illustrated in Fig. 6.5. "The largest increase in present value that can be achieved at any cut-off grade, allowing for the capacity restrictions, is actually the least of Vm, Vr and Vc. This is the curved polygon marked 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 'ORERES'.

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 'ORERES' 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 can be followed in the subroutines COG1, COG2 and COGSUB listed in sections 8.4 and 8.5.



INCREASE IN PRESENT VALUE
V
Vm
GMC
GMR
GRC
G
Vr
Gm
Grc
Gmc
Gmr
Gr
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 be the net cash value gained from the material within the associated grid-block (I,J,K), if it is mined. The point net cash values are the total value of the respective grid-blocks to the mining venture. Consequently, they can be positive, zero or negative depending on their economic potential.

6. 5. 1 STANDARD COSTING METHOD

The point net cash values are computed with a standard costing method. This method is devised by appointing standard values for the various unit production costs (as defined in Table 6.2), the selling prices and recovery efficiencies. Then, those 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) it's 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•

```
      DO 38 T=1,L
      •
      •
      •
      DO 34 K=1,NK
      DO 34 J=1,NJ
      DO 34 I=1,NI
      LOC=NJ*NI*(K-1)+NI*(J-1)+I
      IF (TOPO(LOC)) 34,34,14
   14 IF (YR(LOC)) 15,15,17
   15 M=1
      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=1
      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 REV=0.0
      .
      .   (standard cost adjustments)
      .
      COST=TONF*(X+(F(M)/MIN(M)))
      GO TO 33
   23 .
      .   (standard price adjustments)
      .
      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)
      .
      COST=TONF*(Y+Z+W+(F(M)/MIN(M)))
   33 VAL(I,J,K)=REV-COST
   34 CONTINUE
      .
      .
      .
   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 3? 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 08, 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 19, if the grid-block (I,J,K) is waste or it branches to statement 23 if the grid-block is ore.

11) The routine branches to Line 11, if the production of grid-block (I,J,K) has been simulated. This IF statement compares the 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 'T' 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 assumed to be mined in the first year. These assumptions are crude, but this is inconsequential. The final pit surface becomes static as the solution converges, and as a result, the assumptions become redundant.

The preceding assumptions never become completely redundant in the last year of long term proposals. The reason is that the point net cash values outside the final pit surface are unrealistically inflated because of the assumptions. Therefore, the final pit surface oscillates at the point where the grid-blocks are affected. It could be possible to stop this oscillation, if '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 his deposit. The following list suggests some of the factors which can be applied to adjust the standards:

TYPE 1:

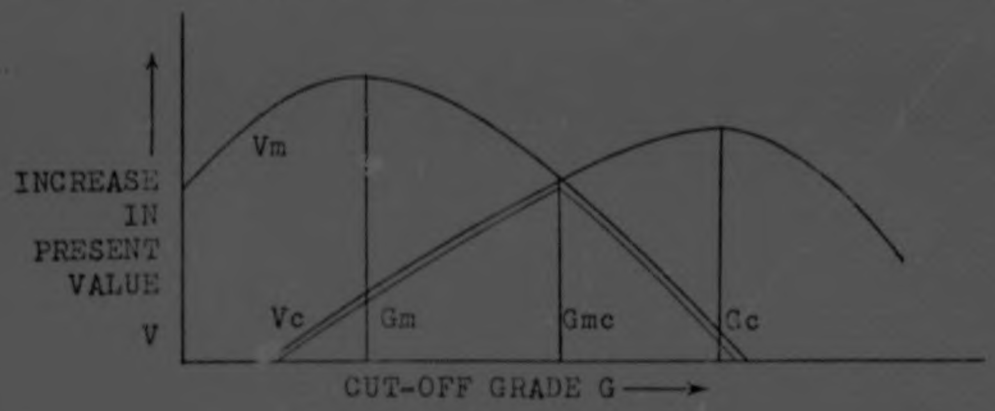


FIG. 6.4A

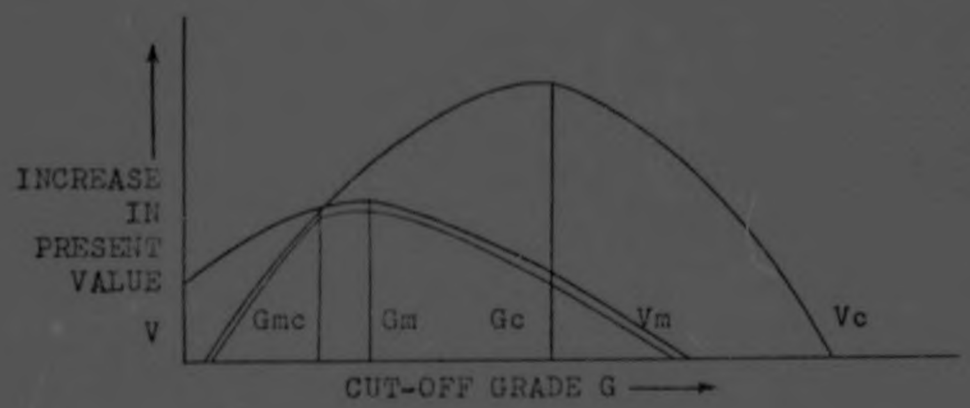


FIG. 6.4B

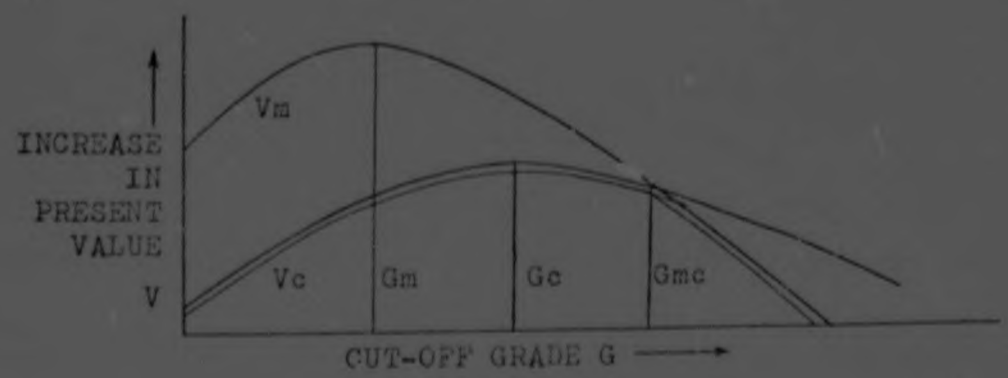


FIG. 6.4C

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 3.3, 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 'OB' 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' and 'Y' are the adjusted waste and ore excavation costs, respectively.

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

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

where:

'GRAD3' is the point density of the offending mineral at (I,J,K).

'CHARGE' is the metallurgical penalty factor.

'REFC' is the standard refining cost.

'W' is the adjusted refining cost.

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

The standard costing method is a flexible technique of computing the point net cash values. This flexibility allows the user to simulate the exact cost structure of his deposit. However, the cost structure of all deposits 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 'W', 'X', 'Y' and 'Z' 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 'E', 'C' and 'R' 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 user's deposit 'R' that yields the maximum total profit from the three innermost integrals of the multiple integral:

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

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

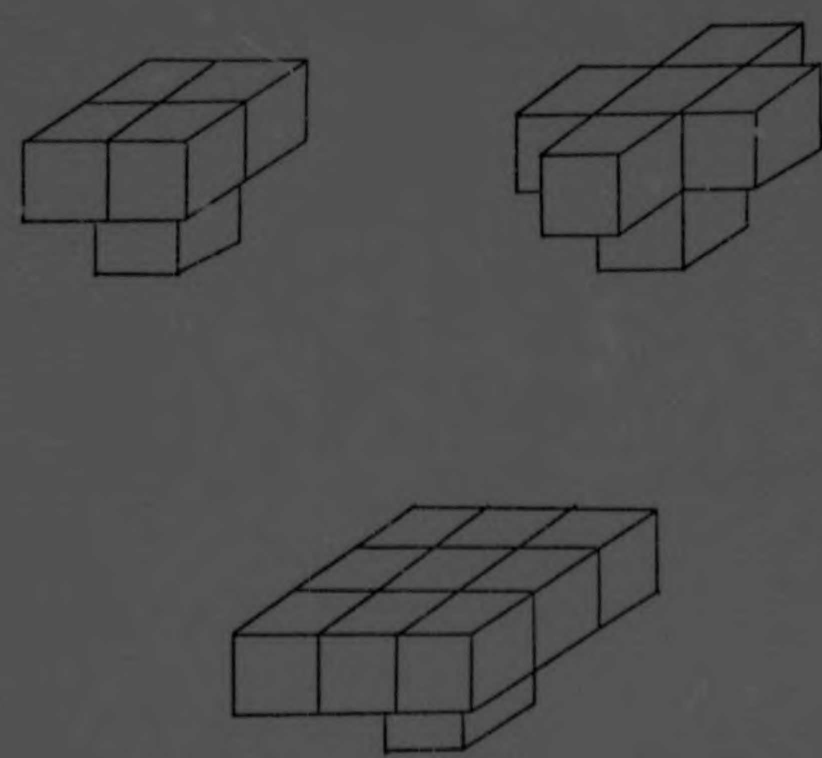


FIG. 6.6 THREE GRID CONSTRUCTIONS

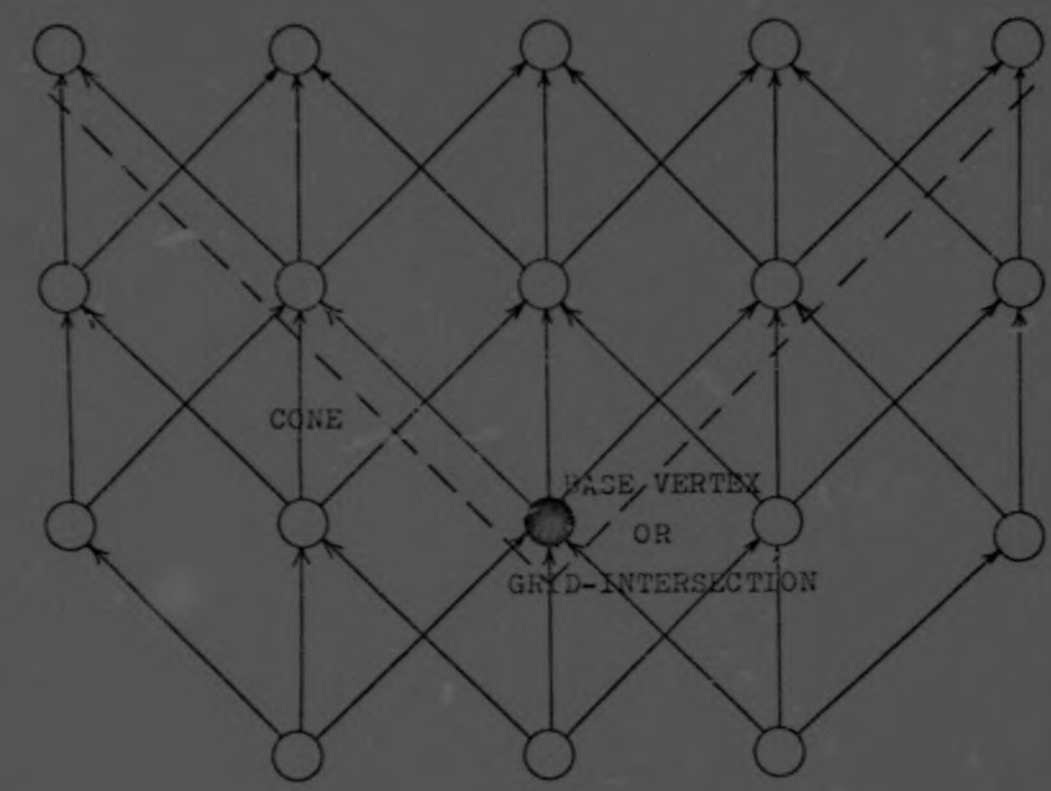


FIG. 6.7 USING A CONE TO SIMULATE ARCS

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. 6.8. It is required to identify the vertices within the closure shown

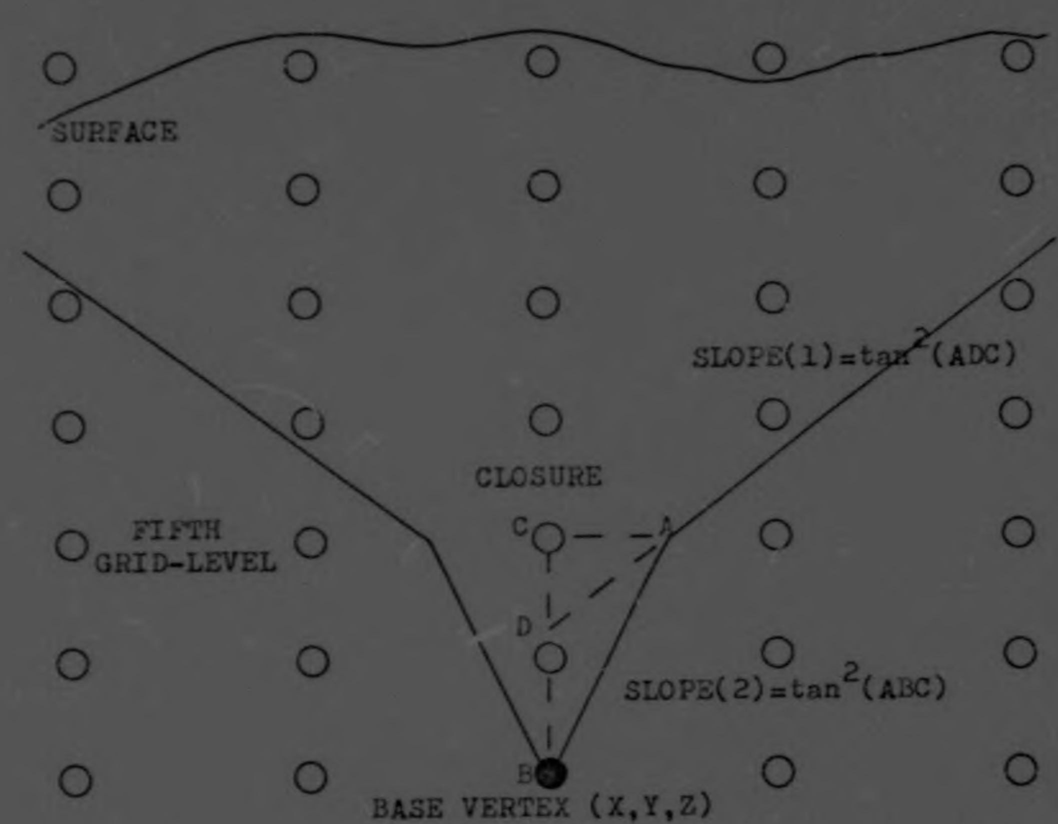


FIG. 6.8 A CROSS-SECTION OF A SEMI-CONE SET WITH TWO WALL-ANGLES

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. 6.9. It can be seen from the analogue that the grid-blocks have either an upward or downward force. The

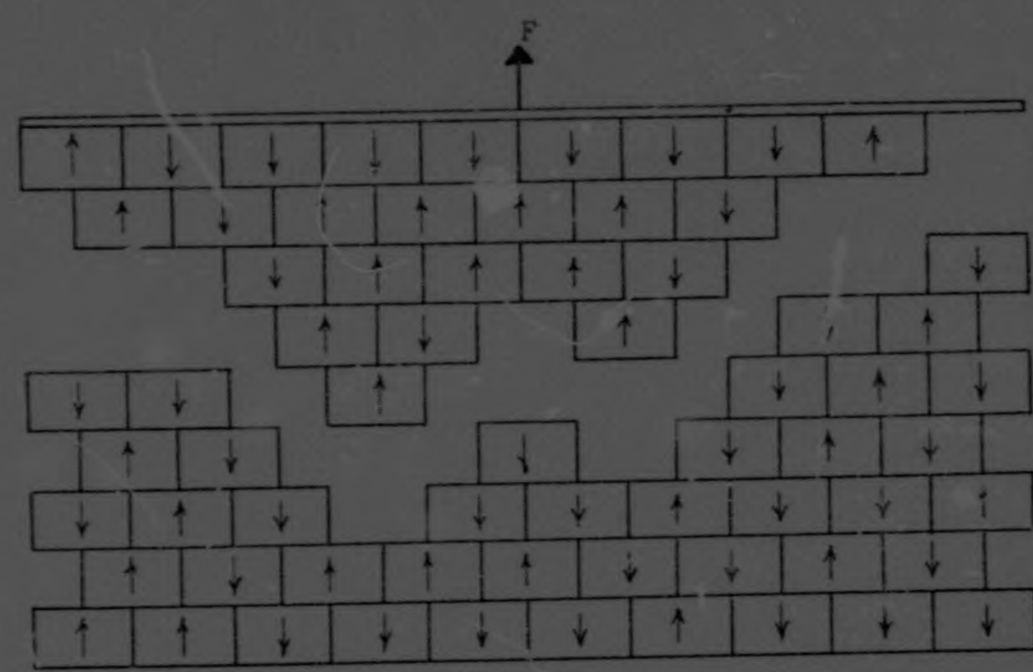


FIG. 6.9 MECHANICAL ANALOGUE OF THE LERCHS-GROSSMAN GRAPH ALGORITHM

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 bound 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:

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

In order to maximize 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 TARGETS COMPUTATIONS, and

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

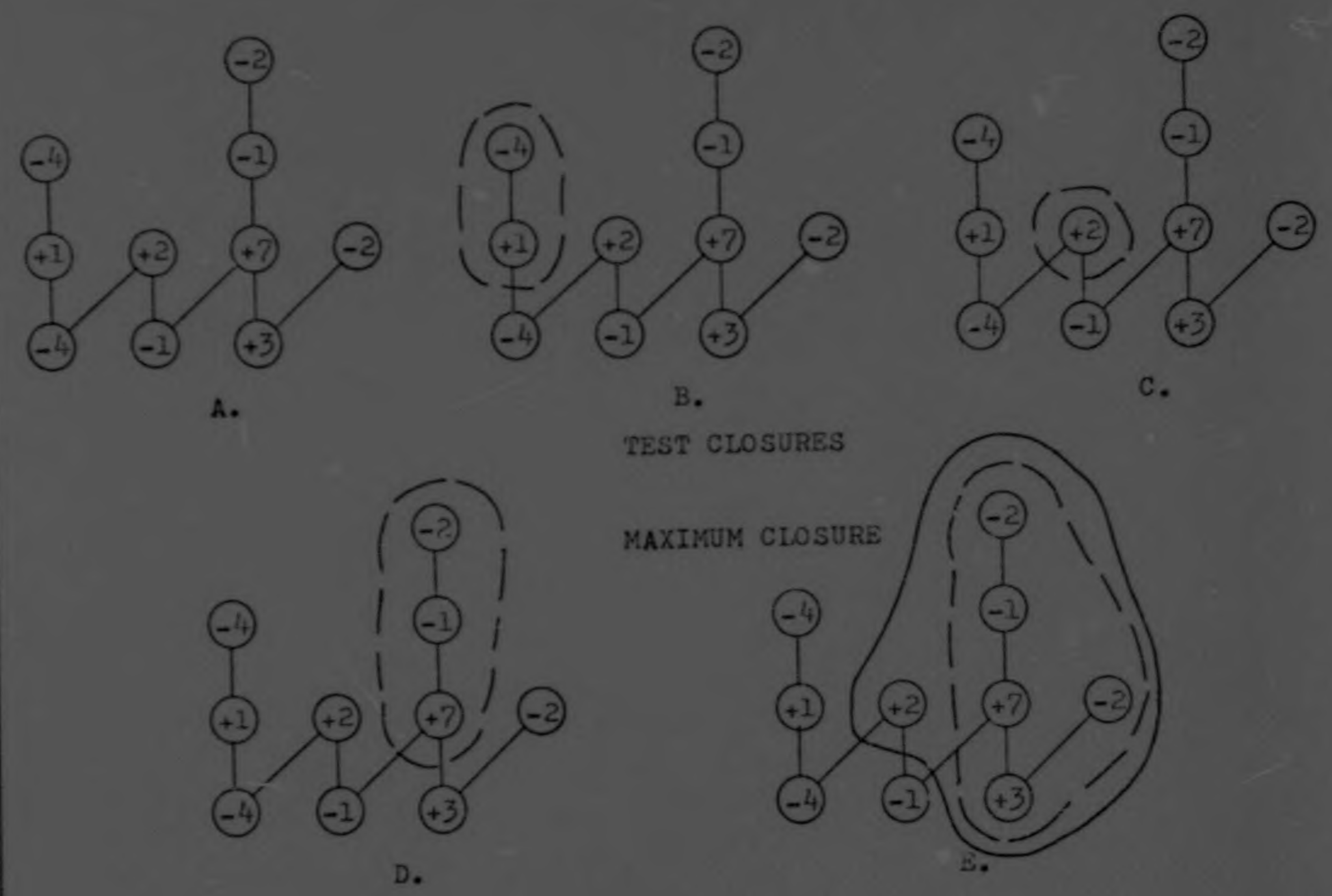


FIG. 6.10 AN ILLUSTRATION OF THE LERCHS-GROSSMAN GRAPH ALGORITHM

The simulation is programmed with a loop of computati ns 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 simula-tion, if the deposit's physical conditions make it impos-sible 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 produc-tion 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 produc-tion 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 6.4 and continued in Table 6.5 illustrates the rules used to programme 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:

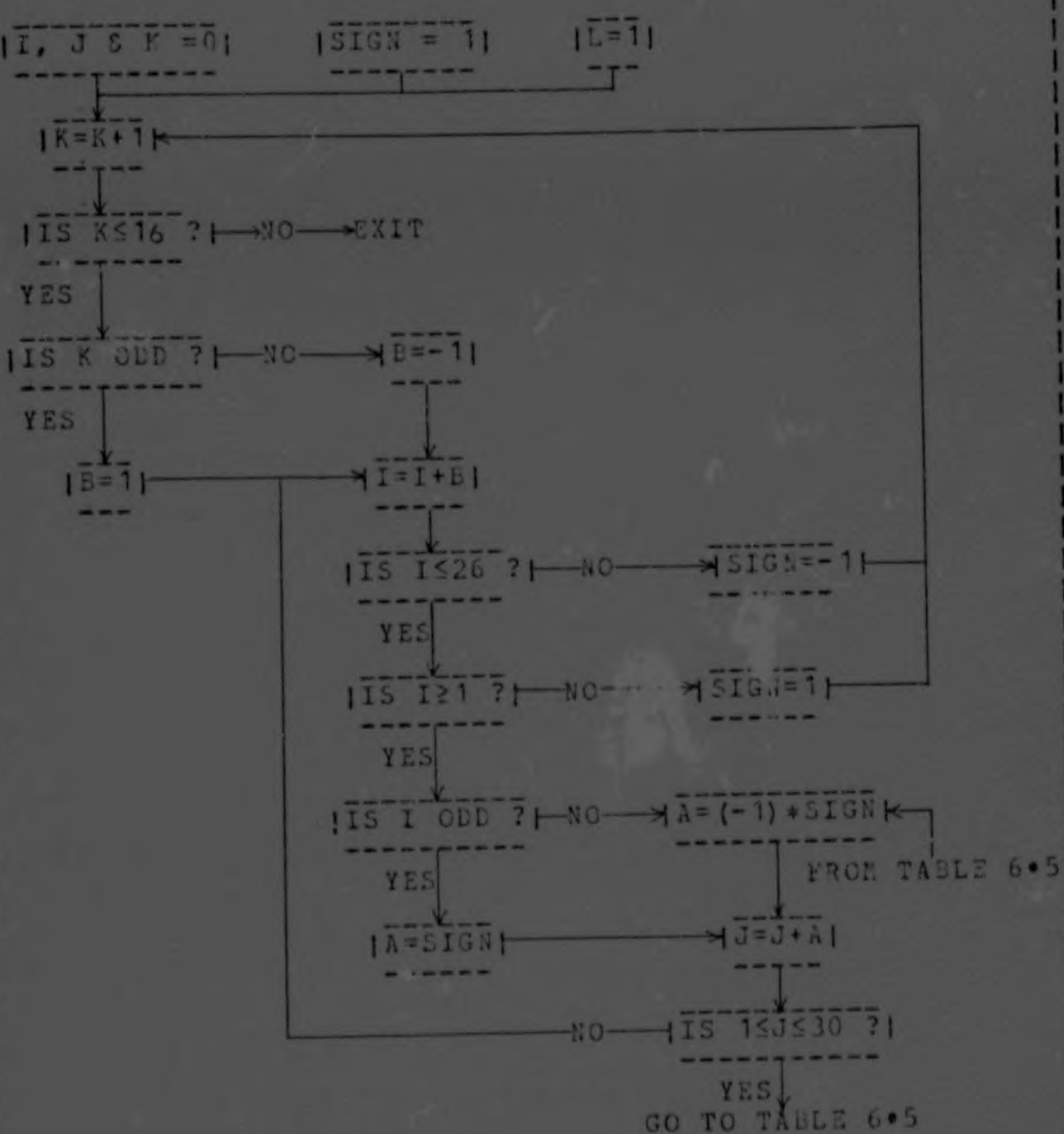


TABLE 6• 4

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.

```
FROM TABLE 6.4                                   GO TO TABLE 6.5
|IS GRID-BLOCK (I,J,K) WITHIN THE PIT ?|----------------NO
YES
|DOES (I,J,K) HAVE A POSITIVE NET CASH VALUE ?|---------NO
YES
|TAG (I,J,K) AS ORE MINED IN YEAR L|
|DOES CUMULATIVE ORE PRODUCTION EQUAL TARGET IN L ?|----NO
YES
|IDENTIFY GRID-BLOCKS REQUIRED TO EXPOSE ORE MINED IN L|
|L=L+1|
```

TABLE 6. 5

The computations illustrated in Table 6.4 define the grid co-ordinates of the next grid-block that should be mined in the ore excavation 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. Additionally, 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(M),

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.

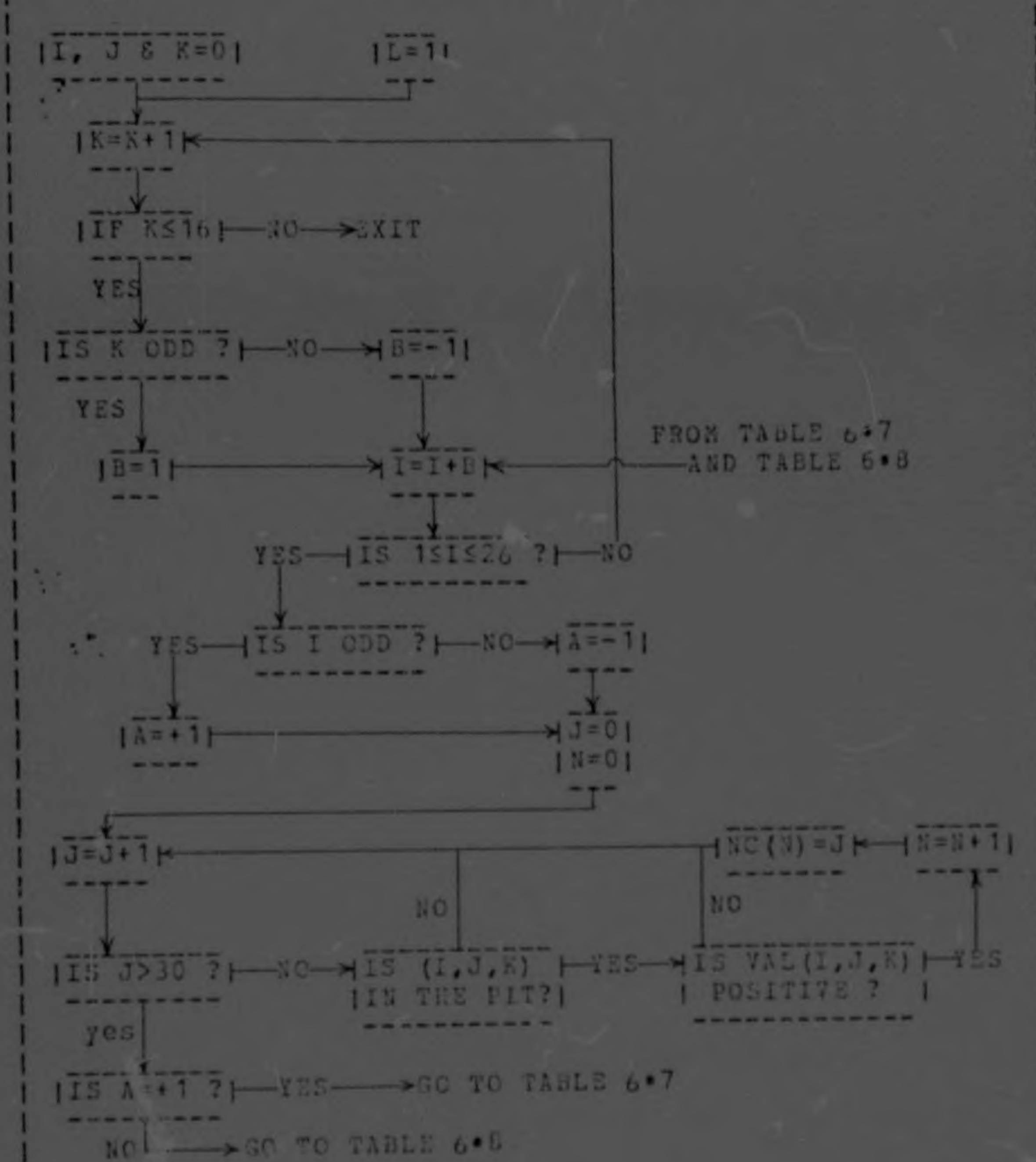


TABLE 6• 6

The two above examples illustrate how the pit production

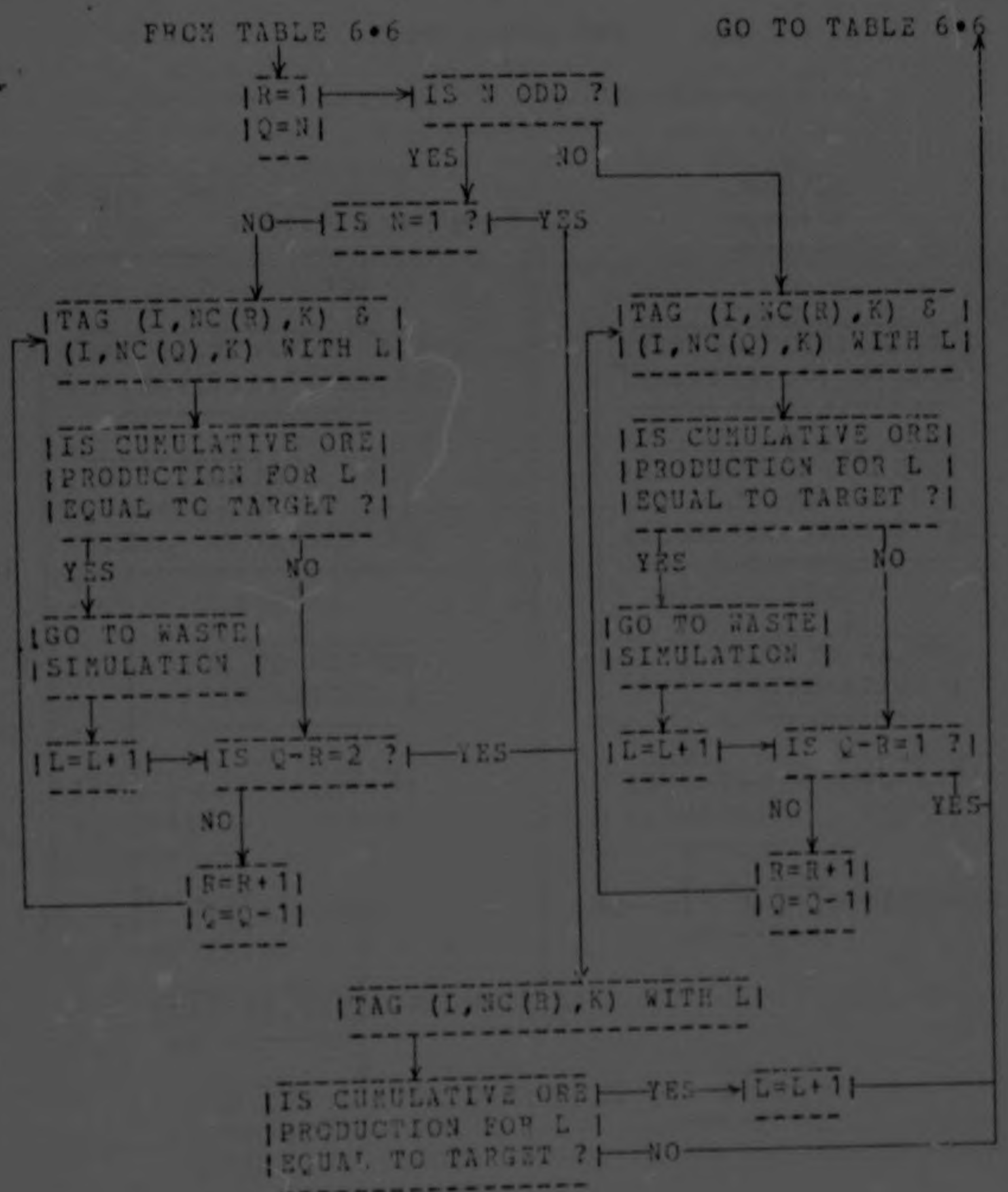


TABLE 6• 7

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

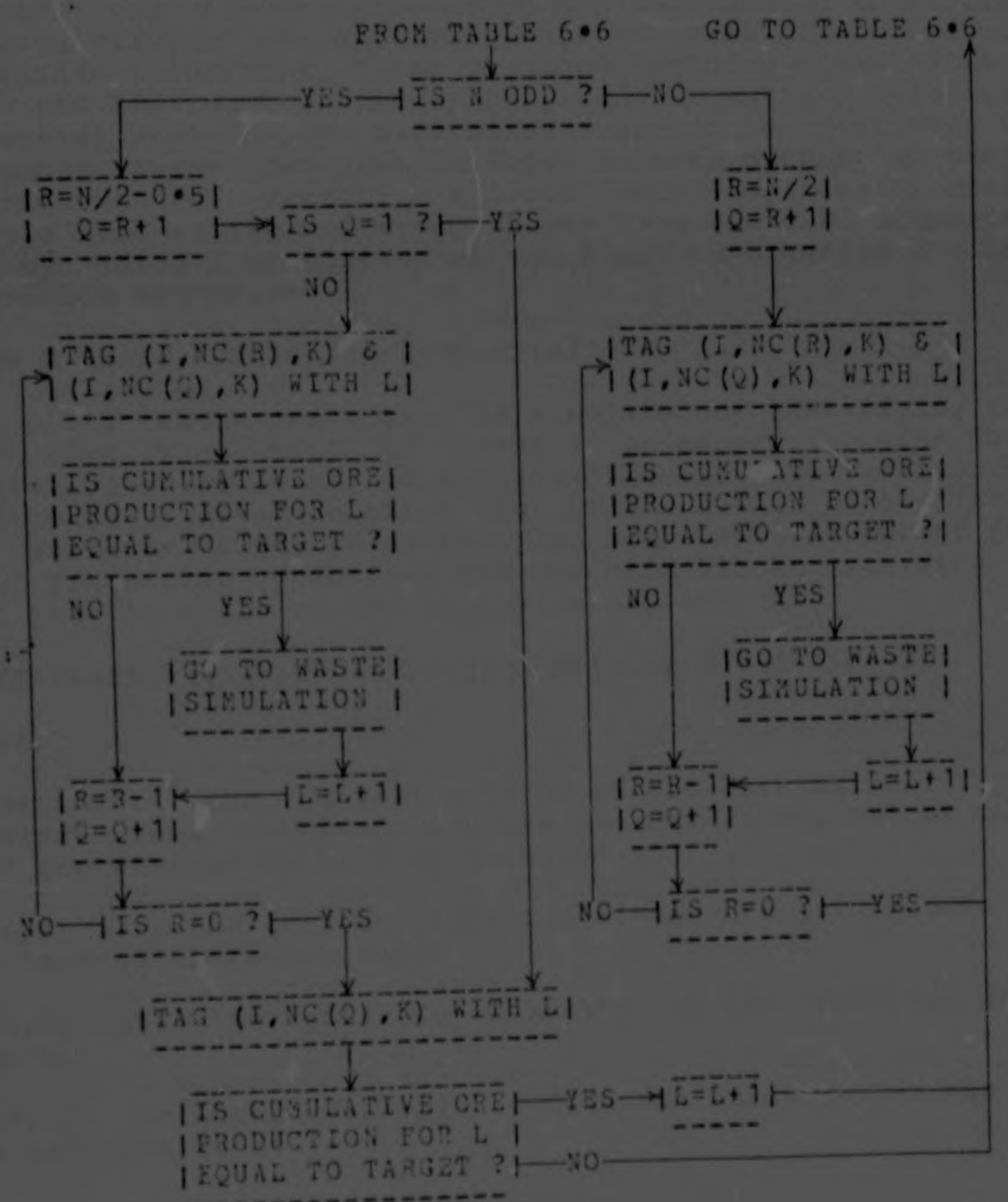


TABLE 6• 8

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. There 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:

'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.

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:

'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

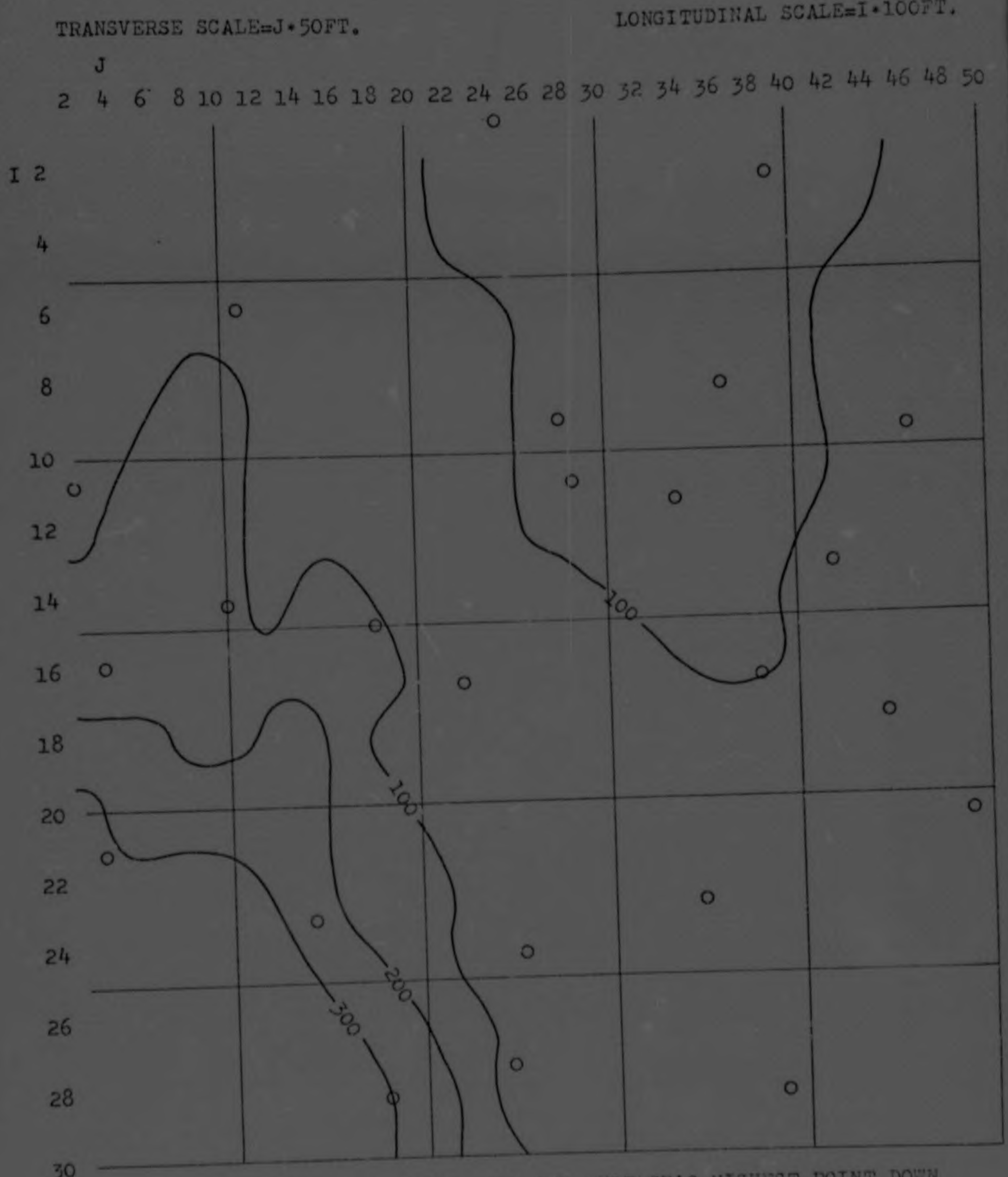
The surface deposit described broadly in Section 7. 1 is used as an example to illustrate an application of the MP Programme. This example does not necessarily duplicate any existing mining conditions. It was devised to demonstrate the simulation techniques. The Programme's output forms the open pit 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 800 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. When 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

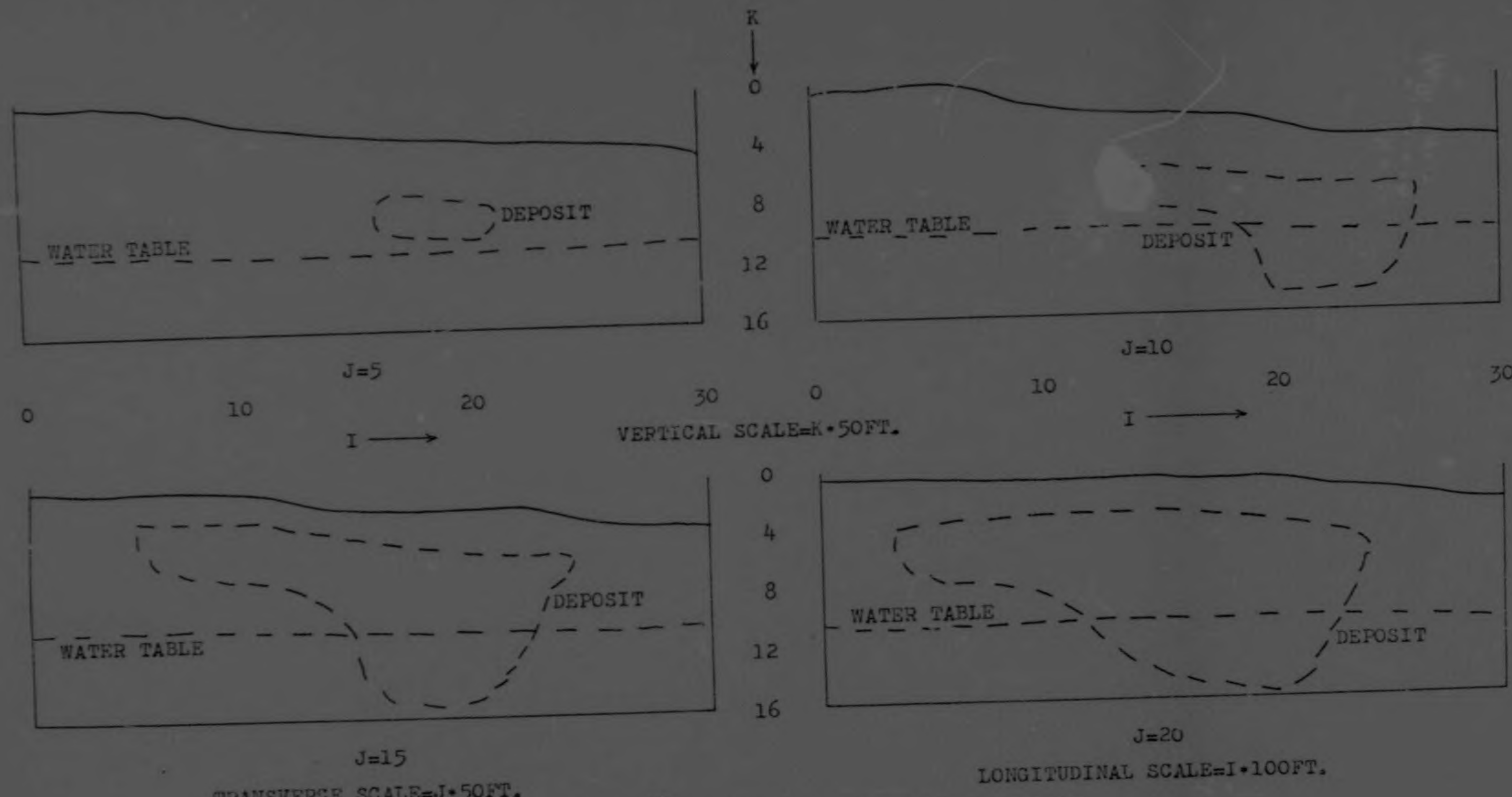


FIG. 7.2 CROSS-SECTIONS OF THE EXAMPLE DEPOSIT

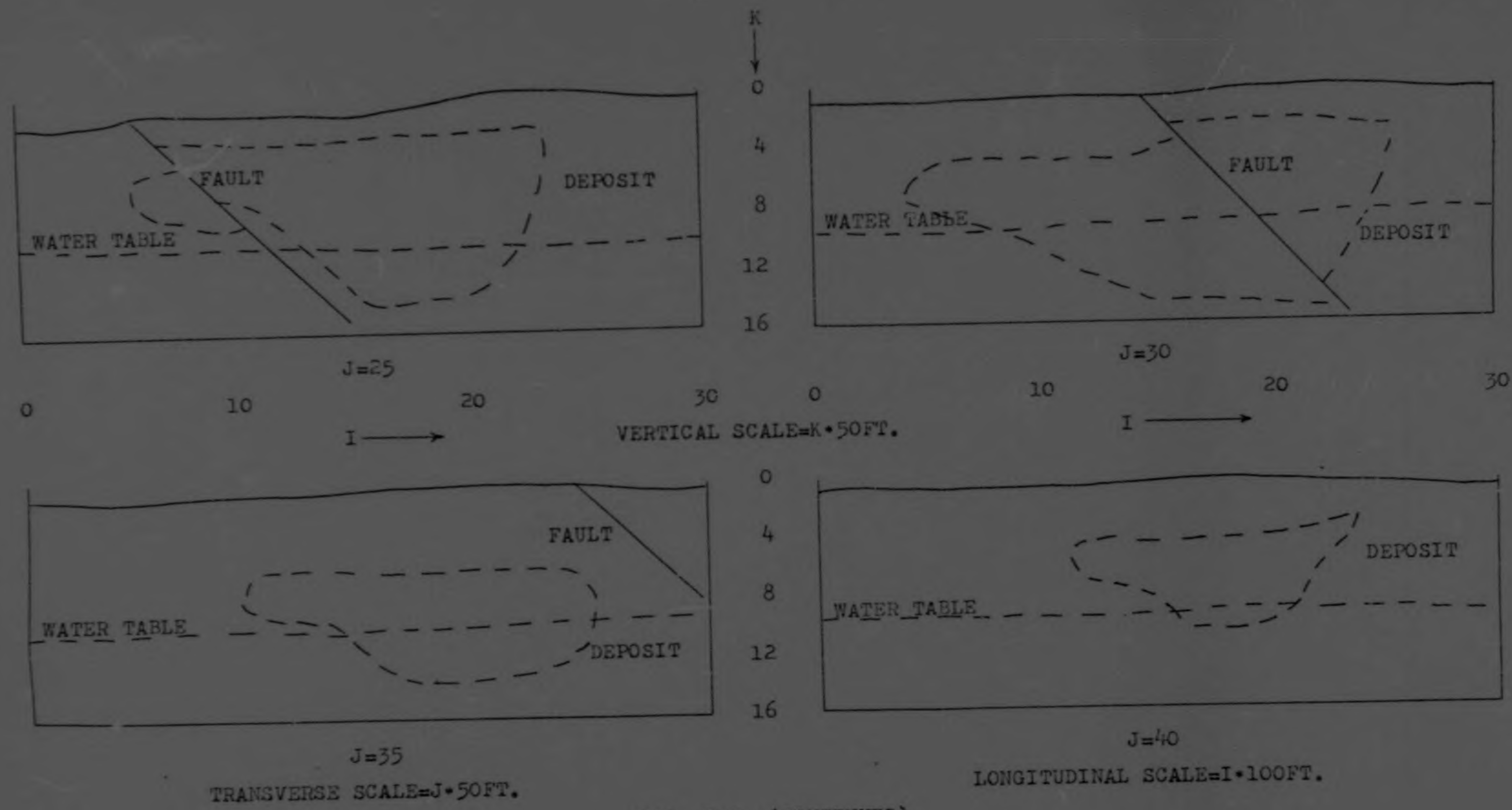


FIG. 7.2 (CONTINUED)

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•786 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 concentrates 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•100 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•49≠ per ton plus 0•003≠ per ton per grid-level above the water table and 0•50≠ per ton plus 0•004≠ 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
	20•0	37•75	32•00	35•15
85%	22•5	32•50	25•65	42•00
	25•0	28•50	27•[illegible]	30•25
	20•0	37•50	26•85	31•85
90%	22•5	38•50	41•25	27•00
	25•0	34•75	40•50	38•25
	20•0	30•75	40•00	25•50
95%	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:	E = 0.43¢ per output ton
Concentrator:	C = 0.80¢ per input ton
Refinery:	R = 50.00 cash units per output ton

Recall from Section 5.2 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 7.5. 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 user'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
	20•0	-16•19	-8•41	-9•72
85%	22•5	-10•81	-2•01	-16•46
	25•0	-6•74	-3•21	-4•64
	20•0	-16•13	-3•50	-6•67
90%	22•5	-17•04	-17•86	-1•76
	25•0	-13•26	-16•80	-12•98
	20•0	-9•52	-16•83	-0•48
95%	22•5	-4•48	-17•92	-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 (M) 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

PIT 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
	20•0	-10•20	-2•66	-3•84
85%	22•5	-5•26	3•76	-10•58
	25•0	-1•29	2•06	1•01
	20•0	-10•21	2•25	-0•77
90%	22•5	-11•43	-12•08	4•16
	25•0	-7•67	-11•44	-7•18
	20•0	-3•66	-11•12	5•34
95%	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•88
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•90	4•20
38	•11	1•91	1•70	6•76	4•00
39	•11	1•92	1•70	6•74	3•99
40	•11	1•92	1•70	6•74	4•01
41	•11	1•92	1•70	6•74	4•04
42	•11	1•92	1•70	6•74	4•06
43	•11	1•81	1•70	5•36	2•28
44	•11	0•83	0•78	2•40	0•47

PRESENT VALUE OF PROFITS LESS CAPITAL COST =
-0•48 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	1.89	1.70	6.31	4.60
38	.09	1.89	1.70	6.31	4.68
39	.08	1.85	1.70	6.21	4.63
40	.08	1.85	1.70	6.21	4.72
41	.08	1.85	1.70	6.15	4.69
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•79	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•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•91
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•790
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•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•48	1•18	56•009

TABLE 7•12

are illustrated in Fig 7•3• Each figure on the maps 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	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	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 comes 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 superceded, 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 grades 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-5 mineral distribution. This forms part 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

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	4	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	6	8	4	4	4	2	2	2	2	2	0	0	0	0	0
8	0	0	2	4	4	6	6	8	8	8	8	8	6	6	6	6	4	4	2	2	0	0	0	0	0
10	0	2	4	6	6	6	8	10	10	10	10	10	8	8	8	8	6	4	4	4	2	0	0	0	0
12	2	4	6	6	8	8	10	12	12	12	12	12	12	10	10	10	8	6	6	6	4	2	0	0	0
14	2	4	6	8	10	10	12	12	14	14	14	14	14	12	12	12	10	8	8	8	6	4	2	0	0
16	2	4	6	8	10	12	14	12	12	14	14	14	14	16	16	14	12	12	12	8	8	6	4	2	0
18	4	6	8	10	12	14	16	14	12	14	14	14	14	16	16	14	14	14	12	12	8	6	4	2	0
20	6	4	6	8	10	12	14	14	14	14	14	14	14	14	14	14	14	14	12	12	8	6	4	2	0
22	6	6	8	8	10	10	12	14	14	14	14	12	12	12	12	14	14	10	10	8	8	6	4	2	0
24	6	8	8	8	8	8	10	12	14	12	12	10	10	12	10	14	12	10	8	6	6	4	2	0	0
26	6	8	8	6	6	8	8	10	12	10	10	8	8	10	10	10	10	8	6	4	4	2	0	0	0
28	6	6	6	6	6	6	8	8	10	8	8	6	6	8	10	10	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	8	6	6	4	4	6	8	8	6	6	2	4	0	0	0	0	0

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

OPTIMUM PIT SURFACE

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	4	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	6	8	4	4	4	2	2	2	2	2	0	0	0	0	0
8	0	0	2	4	4	6	6	8	8	8	8	8	6	6	6	6	4	4	4	2	2	0	0	0	0
10	0	2	4	6	6	6	8	10	10	10	10	10	8	8	8	8	6	6	6	4	2	2	0	0	0
12	2	4	6	6	8	8	10	12	12	12	12	12	12	10	10	10	8	8	8	6	4	2	0	0	0
14	2	4	6	8	10	10	12	12	14	14	14	14	14	12	12	12	10	10	10	8	6	4	2	2	0
16	2	4	6	8	10	12	14	12	12	14	14	14	14	16	16	14	14	14	10	8	8	6	6	2	0
18	4	6	8	10	12	14	16	14	12	14	14	14	14	16	16	14	14	14	12	12	8	6	4	2	0
20	6	4	6	8	10	12	14	14	14	14	14	14	14	14	14	14	14	14	12	8	8	6	4	2	0
22	6	6	6	8	10	10	12	12	14	14	12	12	12	12	10	14	14	10	10	8	6	4	2	0	0
24	6	6	8	8	8	8	10	10	12	12	10	10	10	12	10	14	12	10	8	6	4	2	0	0	0
26	6	8	8	6	6	6	8	8	10	10	8	8	8	10	10	10	10	8	6	4	2	2	0	0	0
28	6	6	6	6	6	6	6	6	8	8	6	6	6	8	8	8	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	6	6	4	4	4	6	6	6	6	6	2	4	0	0		0	0

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

OPTIMUM PIT SURFACE

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	8	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	8	10	10	10	10	10	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	8	10	12	12	10	10	10	12	10	8	8	8	6	6	6	4	2	0	0	0
14	2	4	4	6	8	10	12	12	14	12	12	12	14	10	10	10	10	8	8	8	6	4	2	0	0
16	2	4	6	8	10	12	12	12	12	14	14	14	14	12	12	14	12	12	12	10	8	6	4	2	0
18	4	6	8	10	10	12	14	12	10	12	14	14	14	16	14	14	14	14	12	12	10	6	4	2	0
20	6	6	8	10	10	12	14	14	12	14	14	14	14	14	14	14	14	12	12	12	10	8	4	2	0
22	6	6	6	8	8	10	12	14	14	14	14	12	12	12	12	14	14	10	10	10	8	8	4	2	0
24	6	8	8	8	8	8	10	12	14	14	12	10	10	12	10	14	12	10	10	6	6	4	2	0	0
26	6	8	8	6	6	8	8	10	12	12	10	8	8	10	10	10	10	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	8	8	10	10	8	8	6	8	10	10	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	8	8	6	6	4	6	8	8	6	6	2	4	0	0	0	0	0

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

OPTIMUM PIT SURFACE

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	8	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	8	10	10	10	10	10	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	8	10	12	12	10	10	10	12	10	8	8	8	6	6	6	4	2	0	0	0
14	2	4	4	6	8	10	12	12	14	12	12	12	14	10	10	10	10	8	8	8	6	4	2	0	0
16	2	4	6	8	10	12	12	12	12	14	14	14	14	12	12	14	12	12	12	10	8	6	4	2	0
18	4	6	8	10	10	12	14	12	10	12	14	14	14	16	14	14	14	14	12	12	10	6	4	2	0
20	6	6	8	10	10	12	14	14	12	14	14	14	14	14	14	14	14	12	12	12	10	6	4	2	0
22	6	6	6	8	8	10	12	14	14	14	14	12	12	12	12	14	14	10	10	10	8	6	4	2	0
24	6	8	8	8	8	8	10	12	14	14	12	10	10	12	10	14	12	10	10	6	6	4	2	0	0
26	6	8	8	6	6	8	8	10	12	12	10	8	8	10	10	10	10	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	8	8	10	10	8	8	6	8	10	10	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	8	8	6	6	4	6	8	8	6	6	2	4	0	0	0	0	0

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

SIMULATED PIT 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•0[illegible]	1•55	3•59
36	0•28	0•13	1•55	5•65
37	0•00	0•00	1•55	3•99
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	[illegible]•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•79
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•35	2•58	0•70	0•39
3	1•13	0•60	0•70	0•24
4	7•32	4•04	0•70	0•27
5	2•39	1•37	1•69	2•28
6	4•22	2•43	1•69	2•68
7	2•53	1•50	1•69	2•63
8	2•53	1•52	1•69	2•58
9	3•10	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	3•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•28	0•09	1•75	3•67
33	0•00	0•00	1•63	3•84
34	0•28	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•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•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•96
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•79
40	0•00	0•00	1•63	3•24
41	0•00	0•00	0•30	0•29

TOTAL REVENUE=148•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•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•96
29	0•28	0•12	1•69	3•59
30	0•14	0•03	1•69	4•23
31	0•14	0•01	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•30	0•00	1•63	3•70
36	0•14	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=9

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 (REV) and overburden stripping costs (OB) 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 (INT) on loan capital (LC). The interest rate on loan capital is 7.5%. The opportunity cost of investment capital is 8%. 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

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	0	0	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	0	0	0	0	0
8	0	0	0	2	2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	0	0	0	0	0
10	0	0	2	2	2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	0	0	0	0	0
12	0	2	2	2	2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	2	2	0	0	0
14	2	2	?	2	2	0	2	2	0	0	0	0	0	0	2	2	2	2	2	2	4	2	2	0	0
16	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	2	2	2	0	4	4	4	2	0
18	4	4	4	2	2	4	4	2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	2	2	0
20	6	4	4	4	4	4	4	2	2	0	0	0	0	0	0	0	2	2	2	2	2	2	2	0	0
22	6	6	6	6	6	4	4	2	2	2	0	0	0	0	0	2	2	2	2	2	2	2	2	2	0
24	6	6	6	6	6	6	4	4	2	2	0	0	0	0	0	2	2	2	2	4	2	2	2	0	0
26	6	6	6	6	6	6	6	4	4	2	2	0	0	0	0	2	2	2	4	4	4	2	0	0	0
28	6	6	6	6	6	6	6	6	4	4	2	0	0	0	0	2	2	2	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	4	4	2	2	0	0	0	2	2	2	2	4	0	0	0	0	0

FIG. 7.4 FIVE YEARLY PLAN MAPS OF THE PIT

TOPOGRAPHIC MAP OF PIT IN YEAR 6

J

	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
I 2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	6	4	4	2	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	4	4	6	6	6	6	4	4	2	2	2	2	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	6	6	6	6	4	4	4	2	2	2	2	2	2	2	0	0	0	0	0
12	0	2	4	6	6	6	6	6	6	6	4	4	2	2	2	2	2	2	2	2	2	2	0	0	0
14	2	2	4	4	6	6	6	6	6	6	4	4	2	2	2	2	2	2	2	2	4	2	2	0	0
16	2	2	2	4	4	6	6	6	6	6	4	4	2	2	2	2	2	2	2	2	4	4	4	2	0
18	4	4	4	2	4	6	6	6	6	6	4	4	2	2	2	2	2	2	2	2	2	2	2	2	0
20	6	4	4	4	4	4	4	4	6	6	4	4	2	2	2	2	2	2	2	2	2	2	2	0	0
22	6	6	6	6	6	4	4	4	4	4	4	4	2	4	4	4	2	2	2	2	2	2	2	2	0
24	6	6	6	6	6	6	4	4	4	4	4	2	2	4	4	4	2	2	2	4	2	2	2	0	0
26	6	6	6	6	6	6	6	4	4	2	2	2	2	2	4	4	2	2	4	4	4	2	0	0	0
28	6	6	6	6	6	6	6	6	4	4	2	2	2	2	4	2	2	4	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	4	4	2	2	2	2	2	2	2	2	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 11

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	6	6	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	4	4	6	6	6	6	6	6	6	4	4	4	2	2	2	0	0	[illegible]	0	0
10	0	0	2	2	4	4	4	6	6	6	6	6	6	6	4	4	4	2	2	2	0	0	0	0	0
12	0	2	4	6	6	6	6	6	6	6	6	6	6	4	4	4	4	4	4	4	4	2	0	0	0
14	2	2	4	4	6	6	6	6	6	6	6	6	6	6	4	4	4	4	4	4	6	4	2	0	0
16	2	2	2	4	4	6	6	6	6	6	6	6	6	6	4	4	4	4	4	6	6	4	4	2	0
18	4	4	4	2	4	6	6	6	6	6	6	6	6	6	6	4	4	4	6	6	4	4	4	2	0
20	6	4	4	4	4	4	4	4	6	6	6	6	6	6	6	4	4	4	4	6	4	4	4	2	0
22	6	6	6	6	6	4	4	4	4	4	6	6	6	6	6	6	6	4	4	4	4	4	4	2	0
24	6	6	6	6	6	6	4	4	4	4	6	6	6	6	6	6	6	4	4	4	4	4	2	0	0
26	6	6	6	6	6	6	6	4	4	4	4	4	6	6	6	6	6	4	4	4	4	2	0	0	0
28	6	6	6	6	6	6	6	6	4	4	4	4	6	6	6	6	6	4	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	4	4	6	6	4	4	4	4	4	4	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 16

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	C	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	6	6	4	4	2	2	2	?	2	2	0	0	0	0	0
8	0	0	2	2	4	4	4	6	6	6	6	6	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	6	6	6	6	6	6	6	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	6	6	6	6	6	6	6	3	8	6	6	6	6	6	6	4	2	0	0	0
14	2	2	4	4	6	6	6	6	6	6	6	6	8	8	8	6	6	8	8	6	6	4	2	0	0
16	2	2	4	4	4	6	6	6	6	6	6	6	8	8	8	6	8	8	8	8	8	6	4	2	0
18	4	4	4	4	4	4	6	6	6	6	6	6	8	8	8	8	[illegible]	8	8	8	8	6	4	2	0
20	6	4	4	4	4	4	4	4	6	6	6	6	8	8	8	8	8	8	8	8	8	6	4	2	0
22	6	6	6	6	6	4	4	4	6	6	6	6	8	8	8	8	8	8	8	8	6	6	4	2	0
24	6	6	6	6	6	6	4	4	6	6	6	6	8	8	8	8	8	8	8	6	6	4	2	0	0
26	6	6	6	6	6	6	6	4	6	6	6	6	6	8	8	8	8	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	6	6	4	6	6	6	6	8	8	8	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	4	6	6	6	4	6	6	6	6	6	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 21

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	6	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	6	8	8	8	8	8	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	6	8	8	8	8	8	8	8	8	6	6	6	6	6	6	4	2	0	0	0
14	2	4	4	6	8	8	8	8	8	8	8	8	8	8	8	6	6	8	8	6	6	4	2	0	0
16	2	4	6	8	8	8	8	8	8	8	8	8	8	8	8	6	8	8	8	8	8	6	4	2	0
18	4	4	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	6	4	2	0
20	6	4	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	6	4	2	0
22	6	6	6	6	6	6	6	6	8	8	8	8	8	8	8	8	8	8	8	8	6	6	4	2	0
24	6	6	6	6	6	6	6	6	8	6	8	8	8	8	8	8	8	8	8	6	6	4	2	0	0
26	6	6	6	6	6	6	6	6	6	6	6	6	6	8	8	8	8	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	6	6	6	6	6	6	6	8	8	8	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	4	6	6	6	4	6	6	6	6	6	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 26

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	8	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	8	10	8	10	10	8	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	8	10	10	10	10	10	10	8	8	6	6	6	6	6	6	4	2	0	0	0
14	2	4	4	6	8	10	10	10	10	10	10	10	8	8	8	6	6	8	8	6	6	4	2	0	0
16	2	4	6	8	10	10	10	10	10	10	10	10	8	8	8	6	8	8	8	8	8	6	4	2	0
18	4	6	8	10	10	10	10	10	10	10	10	10	8	8	8	8	8	8	8	8	8	6	4	2	0
20	6	6	8	10	8	10	10	10	10	10	10	10	8	8	8	8	8	8	8	8	8	6	4	2	0
22	6	6	6	8	8	8	10	10	10	10	10	10	8	8	8	8	8	8	8	8	6	6	4	2	0
24	6	8	8	8	6	8	10	10	10	10	10	8	8	8	8	8	8	8	8	6	6	4	2	0	0
26	6	8	8	6	6	8	8	8	8	8	8	8	6	8	8	8	8	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	6	8	8	8	8	6	6	8	8	8	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	6	6	6	6	4	6	6	6	6	6	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 31

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	25	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	8	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	8	10	8	10	10	10	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	8	10	10	10	10	10	10	10	10	8	8	8	6	6	6	4	2	0	0	0
14	2	4	4	6	8	10	10	10	10	10	10	10	10	10	10	10	10	8	8	8	6	4	2	0	0
16	2	4	6	8	10	10	10	10	10	10	10	10	10	10	10	10	10	12	12	10	8	6	4	2	0
18	4	6	8	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	12	12	10	8	4	2	0
20	6	6	8	10	8	10	10	10	10	10	10	10	10	10	10	10	10	10	12	12	10	6	4	2	0
22	6	6	6	8	8	8	10	10	10	10	10	10	10	10	10	10	10	10	10	10	8	6	4	2	0
24	6	8	8	8	6	8	10	10	10	10	10	8	8	10	10	10	10	10	10	6	6	4	2	0	0
26	6	8	8	6	6	8	8	8	8	8	8	8	8	10	10	10	10	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	6	8	8	8	8	6	6	8	10	10	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	6	6	6	6	4	6	8	8	6	6	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 36

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	8	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	8	10	10	10	10	10	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	8	10	12	12	10	10	10	12	10	8	8	8	6	6	6	4	2	0	0	0
14	2	4	4	6	8	10	10	12	12	12	12	12	12	10	10	10	10	8	8	8	6	4	2	0	0
16	2	4	6	8	10	10	10	12	10	12	12	12	12	12	12	12	12	12	12	10	8	6	4	2	0
18	4	6	8	10	10	10	10	12	10	12	12	12	12	12	12	12	12	12	12	12	10	6	4	2	0
20	6	6	8	10	8	10	10	12	12	12	12	12	12	12	12	12	12	12	12	12	10	6	4	2	0
22	6	6	6	8	8	8	10	10	12	12	12	12	12	12	12	12	12	10	10	10	8	6	4	2	0
24	6	8	8	8	8	8	10	10	12	12	10	10	10	12	10	12	12	10	10	6	6	4	2	0	0
26	6	8	8	6	6	8	8	10	12	12	10	8	8	10	10	10	10	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	6	8	8	8	8	6	6	8	8	8	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	6	6	6	6	4	6	6	6	6	6	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)

TOPOGRAPHIC MAP OF PIT IN YEAR 36

I \ J	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48	50
2	0	0	0	0	0	0	0	2	2	2	4	2	2	2	2	2	2	2	2	2	2	2	0	0	0
4	0	0	0	0	2	2	2	4	4	4	6	4	4	2	2	2	2	2	2	2	2	0	0	0	0
6	0	0	0	2	2	4	4	6	6	6	8	8	4	4	2	2	2	2	2	2	0	0	0	0	0
8	0	0	2	2	4	6	6	8	8	8	8	8	6	6	4	4	4	2	2	2	0	0	0	0	0
10	0	2	2	4	6	6	8	10	10	10	10	10	8	8	6	6	6	4	4	4	2	0	0	0	0
12	0	2	4	6	6	8	10	12	12	10	10	10	12	10	8	8	8	6	6	6	4	2	0	0	0
14	2	4	4	6	8	10	10	12	12	12	12	12	12	10	10	10	10	8	8	8	6	4	2	0	0
16	2	4	6	8	10	10	10	12	10	12	12	12	12	12	12	12	12	12	12	10	8	6	4	2	0
18	4	6	8	10	10	10	10	12	10	12	12	12	12	12	12	12	12	12	12	12	10	6	4	2	0
20	6	6	8	10	8	10	10	12	12	12	12	12	12	12	12	12	12	12	12	12	10	6	4	2	0
22	6	6	6	8	8	8	10	10	12	12	12	12	12	12	12	12	12	10	10	10	8	6	4	2	0
24	6	8	8	8	8	8	10	10	12	12	10	10	10	12	10	12	12	10	10	6	6	4	2	0	0
26	6	8	8	6	6	8	8	10	12	12	10	8	8	10	10	10	10	8	6	6	4	2	0	0	0
28	6	6	6	6	6	6	6	8	8	8	8	6	6	8	8	8	8	6	4	4	4	0	0	0	0
30	6	6	6	6	6	6	6	6	6	6	6	6	4	6	6	6	6	6	2	4	0	0	0	0	0

FIG. 7.4 (CONTINUED)