



Productivity improvement model for planning two pass dragline operations
by Glen Arthur Sattoriva

A thesis submitted in partial fulfillment of the requirements for the degree of MASTER OF SCIENCE
in Industrial Engineering
Montana State University
© Copyright by Glen Arthur Sattoriva (1978)

Abstract:

The basic premise of the thesis topic is to yield a mathematical model, structured as a non-linear programming problem, that will allow the user to analyze the economic tradeoffs between the pit geometry and dragline dimensions for two pass dragline operations. This model is coded in Fortran IV and is linked with a computer coded direct search technique. Since the model is a resident of the computer mine planning system at Montana State University, its use is interactive with all results displayed via computer drawn casting diagrams.

STATEMENT OF PERMISSION TO COPY

In presenting this thesis in partial fulfillment of the requirements for an advanced degree at Montana State University, I agree that the Library shall make it freely available for inspection. I further agree that permission for extensive copying of this thesis for scholarly purposes may be granted by my major professor, or, in his absence, by the Director of Libraries. It is understood that any copying or publication of this thesis for financial gain shall not be allowed without my written permission.

Signature

Glen A. Sattorwa

Date

June 15, 1978

PRODUCTIVITY IMPROVEMENT MODEL FOR PLANNING
TWO PASS DRAGLINE OPERATIONS

by

GLEN ARTHUR SATTORIVA

A thesis submitted in partial fulfillment
of the requirements for the degree

of

MASTER OF SCIENCE

in

Industrial Engineering

Approved:

David J. Gibson
Chairperson, Graduate Committee

John A. Ritchey
Head, Major Department

Henry L. Parsons
Graduate Dean

MONTANA STATE UNIVERSITY
Bozeman, Montana

June, 1978

ACKNOWLEDGMENTS

I would like to express my appreciation to David F. Gibson for his guidance and support. Also, I would like to thank two members of the SEAM research team, Thomas Lehman and Edward Mooney, for the time they spent reviewing my thesis and for their constructive criticism.

I would like to express my gratitude to Jean Julian for the care and timeliness spent typing the thesis.

Lastly, I would like to dedicate this work to my wife, Karen, and daughter, Jessica, for the sacrifices they made so that this work was made possible.

TABLE OF CONTENTS

	<u>Page</u>
VITA	ii
ACKNOWLEDGMENTS	iii
TABLE OF CONTENTS	iv
LIST OF TABLES	v
LIST OF FIGURES	vi
ABSTRACT	viii
INTRODUCTION	1
Energy Crisis	1
Federal Funded Research	3
Western Surface Mining	5
Computerized Mine Planning System	9
Pit Design Methodologies	15
Statement of Thesis Topic	19
TWO PASS DRAGLINE MODEL	20
Technical Presentation	20
Parameter Descriptions	23
Technique Modeled	24
Design Constraints	34
EXAMPLE APPLICATION	39
Minimum Time Objective	43
Minimum Cost per Ton of Coal Objective	50
SUMMARY	57
Conclusions	57
Recommendations	59
APPENDIX	61
REFERENCES CITED	151
BIBLIOGRAPHY	154

LIST OF TABLES

<u>Table</u>	<u>Page</u>
I. Tonnages of Strippable Coal	2
II. Parameter Specifications	40
III. Initial Plan	41
IV. Summary of the Results for Example I of the Minimum Time Objective	45
V. Results of Optimization for Example II of Minimum Time Objective	47
VI. Results of Example III for the Minimum Time Objective	50
VII. Final Decision Vector	52
VIII. Results of Example II	54
IX. Results for Example III	54
X. Summary of Examples	56
XI. Contents of Dragline Decision Subset	110

LIST OF FIGURES

<u>Figure</u>	<u>Page</u>
1. Prominent Stripping Methods	7
2. Typical Western Strip Mine	8
3. MSU's Computer Mine Planning System	10
4. Design Levels of Production Analysis Module	12
5. Rumfeldt's Casting Diagram	17
6. Coal Fender	26
7. Sequence of Overburden Removal	27
8. Maximum Rehandle Area	35
9. Fill Bench Area	36
10. Plan View of Dragline Locations	38
11. Example I Minimum Time	44
12. Example II Minimum Time	46
13. Example III Minimum Time	49
14. Example I Minimum Cost	51
15. Example II Minimum Cost	53
16. Example III Minimum Cost	55
17. Model Location	58
18. Maximum Cut Length	65
19. Maximum Perpendicular Distance to Edge of the Cut	67
20. Dragline Distance from the Cut Face	68

<u>Figure</u>	<u>Page</u>
21. Final Spoil Configuration	71
22. Fill Bench	73
23. Key Cut Spoil Area Without Rehandle	75
24. Key Cut Spoil With Rehandle	77
25. Key Cut Spoil Rests on Previous Spoil Bank	80
26. Maximum Key Cut Spoil Area	82
27. Main Cut and Key Cut Widths	84
28. Typical Main Cut Spoil Situation	85
29. Maximum Rehandle Situation for First Pass Overburden	88
30. Possible Rehandle Configuration for Shallow First Pass	89
31. Main Cut Rehandle Overhangs Key Cut Spoil	92
32. Spoil Configuration for Main Cut Rehandle Area	97
33. Spoil Peak Configurations	99
34. First Pass Area Less Rehandle	101
35. Second Pass Key Cut Spoil	102
36. Main Cut Spoil Configuration	105
37. Final Spoil Configuration	107
38. Average Swing Angle	109
39. Centroids of Cut Areas on First Pass	115
40. Centroids of Cut Areas on Second Pass	116
41. Rehandle Configuration	117

ABSTRACT

The basic premise of the thesis topic is to yield a mathematical model, structured as a non-linear programming problem, that will allow the user to analyze the economic tradeoffs between the pit geometry and dragline dimensions for two pass dragline operations. This model is coded in Fortran IV and is linked with a computer coded direct search technique. Since the model is a resident of the computer mine planning system at Montana State University, its use is interactive with all results displayed via computer drawn casting diagrams.

INTRODUCTION

Energy Crisis

In recent years America has come to realize it is no longer self-sufficient in terms of supplying its own energy needs. In fact, our economy is highly dependent upon foreign oil imports to meet its energy demands. Statistics compiled by D. Sheridan reveal to us this dependency is increasing. For instance, in 1973-74 foreign oil filled approximately 15 percent of our total energy demand. In 1976 this figure rose to 20 percent. A distressing note is that this trend is expected to get worse--up to 24 percent or more by 1985 (16).

In recognition of this problem, the Carter administration has evolved a national energy plan. It has three essential features: (i) to use energy more wisely, i.e., conservation, (ii) increase coal production and develop nuclear energy, and (iii) increase exploration for oil and natural gas plus initiate and increase support of research for new energy sources. The second feature enables us to meet our short term and intermediate goals--that is, to decrease our dependency on oil imports. When one considers coal vs. nuclear energy, coal is more promising because of the quantity of reserves and because of the constraints we have imposed upon ourselves with respect to the time it takes to bring a nuclear facility on line. Therefore, the Carter Plan has called for an increase in coal production of tremendous proportions. This increase implies mining over 1 billion tons/year--almost doubling

present consumption and bringing the percentage of coal usage with respect to total energy consumption from 18 percent to 29 percent (14).

Coal constitutes 88 percent of the United States' known and hypothetical energy reserves. Seventy-two percent of these coal reserves lie in the Western Great Plains. Table I shows the estimated tonnage of strippable coal in five western states (7).

Table I. Tonnages of Strippable Coal

State	Million Tons
Montana	42,562
Wyoming	23,845
North Dakota	16,003
Texas	3,272
New Mexico	2,258

In addition to its abundance, Western Coal has several other advantages over coal found elsewhere: (i) it lies relatively close to and parallel to the surface, (ii) it is found in thick seams, (iii) it is low in sulfur and is attractive to utilities that must comply with the Clean Air Act. Therefore, much of the increased activity in the coal industry has occurred in the Western Great Plains.

Since oil and natural gas surpassed coal and wood as primary fuel sources in the mid-century, coal production has been demand constrained. However, the reverse will soon take precedence--not because of limited reserves but because of bounds on productivity and the

necessary strip mine reclamation laws. Through laws passed by Congress, the nation has committed itself to the goals of safe mines, clean air and water, and reclaimed lands along with meeting the production goals established in Carter's Energy Plan. In addressing such a commitment a rational and objective approach to strip mining must be taken.

Federal Funded Research

Surface mining in the West is relatively new. Therefore, strip mine planning and reclamation are not well established. A large number of problems concerning both topics remain to be studied and resolved. As a result, groups within the federal, state, and private sectors have increased their support of research in these areas. Specifically, the United States Forest Service and the Environmental Protection Agency are sponsoring a large scale research project termed SEAM (Surface Environment and Mining). Generally, SEAM is to provide and apply technology in order to maintain or restore the quality of the environment on forest and range lands subject to mining. Such an approach means designing the most productive mine with respect to environmental constraints and physical limitations of the system.

SEAM funded a research project at Montana State University. The research project featured the development of a computerized mine planning system (7). The development of such a system allows for

comprehensive planning of strip mine operations relative to environmental considerations and physical limitations inherent within mining operations. The system focuses on maximizing and monitoring dragline productivity, since the dragline operation is the single most costly activity during mining.

This thesis was developed in conjunction with the SEAM research at MSU. The thesis topic, entitled Productivity Improvement Model for Planning Two Pass Dragline Operations, is aimed at increasing dragline productivity through the use of computer modeling and non-linear optimization. The computerized mathematical model is used to design pit geometry vs. dragline dimensions; namely, boom length and bucket size, relative to two pass strip mining operations.

In the following presentation, a discussion about surface mine design and mining operations describes the interrelated mine planning activities. Next, a comparison is given relating MSU's computer planning system to the contemporary mine planning procedure.

The pit design phase of the planning process is stressed in the section entitled Pit Design Methodologies. This section establishes relevancy for the thesis topic with respect to MSU's computerized mine planning system and for application in the field of mine planning. A concise statement of the thesis topic will be given following this section.

Western Surface Mining

Planning a western surface coal mine is a complex and lengthy activity and may follow various procedures. Therefore, the planning sequence and design factors considered here may not be complete for every mine design, nor completely necessary. However, the thorough pattern established suggests the comprehensive approach that is necessary when undertaking such an ordeal (11).

To begin compiling the necessary parameters and data for the mine plan, the engineer/planner would analyze geographic influences on the mine site. For instance, extreme weather conditions would have an impact on the equipment selections, total working days per year, etc. Following the geographic analysis, various geological inputs should be evaluated. These inputs usually encompass: (i) the topography of the mine sites, (ii) the depth and type of overburden material, and (iii) the depth, quality and pitch of the seam. Other geological data such as subsurface water, chemical properties of the soil, etc., are also evaluated at this time. The primary sources for this data are topographic maps and drill-hole data. From this analysis indices such as the stripping ratio (cubic feet of overburden/tons of coal) for the site can be obtained. This gives the engineer/designer a handle on forecasting production rates throughout the mine life. With the geographic and geologic analysis complete, the engineer/designer can develop the initial pit layout. This layout should take into

consideration the location of existing roads and railroads, groundwater flow, coal quality, and above all meeting required annual production rates.

Once the preliminary design phase is complete, the method of overburden removal should be established. In the Western Great Plains, truck and shovel and dragline operations dominate the scene. Considering specifically a dragline operation, the engineer/designer would now evaluate stripping methods and design the optimal pit geometry. If the mining company doesn't own a dragline, this activity may involve sizing a dragline to the particular site to meet some production goal.

Stripping methods usually fall into one of two categories: (1) single coal seam methods, and (2) multiple coal seam methods. Figure 1 illustrates cross sectional views of four prominent stripping methods and their associated pit geometry. These methods are the most prominent throughout the western great plains. Figure 1d shows the two-pass strip mining method. This method of excavation as previously mentioned constitutes the subject of this thesis. With minimal extensions the two pass method becomes the primary method for extracting multiple seams of coal.

When the engineer/designer evaluates stripping methods by conventional means, he primarily investigates the feasibility of mining a tract of land with a given dragline--that is, does the dragline reach accommodate that required to spoil the overburden with minimum

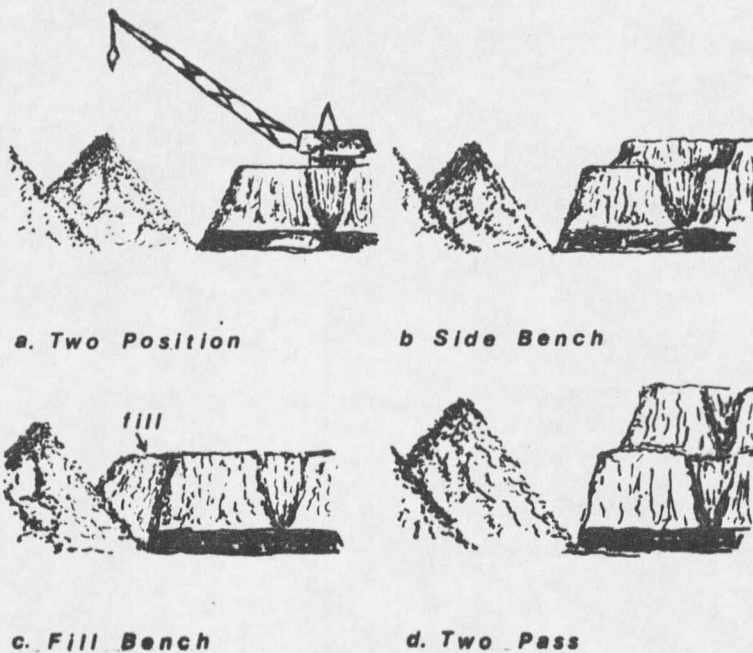


Figure 1. Prominent Stripping Methods

rehandle? This can be done by drawing casting diagrams (i.e., Figure 1) to analyze the geometric relationships between dragline reach and pit geometry. After evaluating numerous stripping techniques for feasibility, the engineer/ designer chooses the method which maximizes/ minimizes some measure of goodness (i.e., price per ton of exposed coal, etc.). Considerations such as reclamation practices, equipment constraints, geologic factors, and individual economic preferences are also important when selecting a particular stripping method. Upon the selection of the most appropriate mining method, the engineer/designer would locate the haul roads and tipples so as to average the haul

distance throughout the mine life. This level of design would complete the analysis--yielding the mine plan.

Now, with the mine plan established, the actual mining operations can commence. Figure 2 illustrates such operations at a "typical" western strip mine (11). First, the topsoil is removed by scrapers

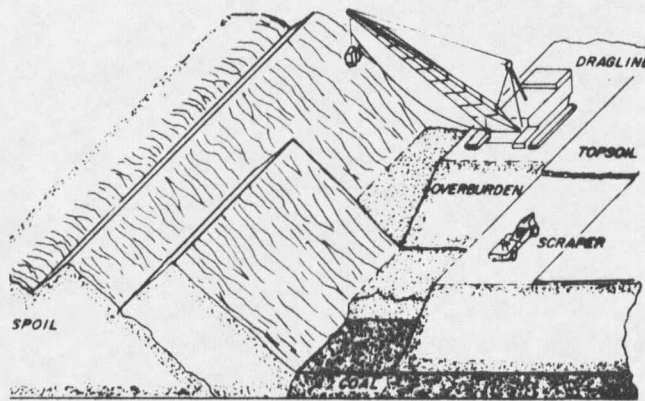


Figure 2. Typical Western Strip Mine

with a twenty cubic yard capacity. The topsoil is either stockpiled for later replacement or directly spread on regraded spoils. Overburden is drilled vertically on a grid spacing with a crawler-mounted rotary drill. The drill holes are loaded with a mixture of ammonium nitrate and fuel oil (ANFO), then blasted. This is done to loosen consolidated material and thus reduce overburden removal costs. Sixty

feet of overburden is removed by a walking dragline with a 225 foot boom and a thirty cubic yard bucket.

Prior to loading the coal, it is drilled and blasted. Then an electric shovel with an eleven cubic yard bucket loads eight ton haul trucks. The trucks haul the coal to a tippie for later loading onto unit trains.

Large bulldozers grade the spoil piles to approximately the original contour. Scrapers redistribute the topsoil which is then seeded and fertilized. Mulch is then generally crimped into the soil. Although the most prominent land use is grazing, other reclamation efforts may be taken on proposed future land use.

Computerized Mine Planning System

Figure 3 illustrates the essential features of the computerized mine planning system developed at Montana State University. This system, developed on a HP 21 MX mini-computer, will allow the mining engineer to follow the same planning methodology as outlined in the previous section. Gibson gave the following summary on system configuration and operation (8).

Notice the basic inputs to the system are of three main types. First, core or drill hole data taken at the property provides a description of the physical and chemical conditions under which mining takes place. Information extracted from the drill hole data might

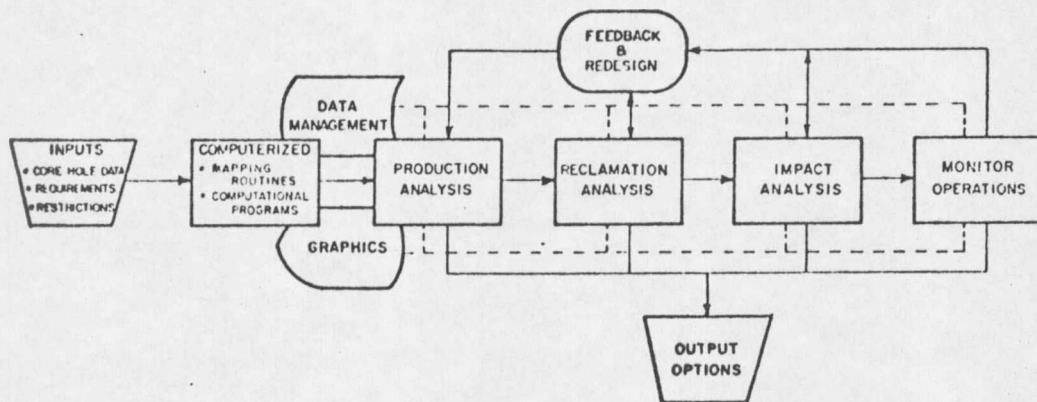


Figure 3. MSU's Computer Mine Planning System

include such things as the depth of overburden, the depth of coal, and various qualities and characteristics of each. A second type of information input to the system relates to production and capital requirements of the operation. Examples of this type of data might be required tons per year of coal, minimum required rate of return, and projected selling price of the coal. The third type of input is restrictions with respect to environmental protection, that is, those specified by state and federal laws or by particular company objectives. Such restrictions may place constraints on things such as the type of stripping technique used, the method of spoiling overburden, and/or the reclamation methods employed.

Once the basic input is provided to the system, a large array of computational and mapping routines is available to the user. Displays of geology, stratigraphy, and various other characteristics of the overburden and coal can be made in the form of three dimensional views, contour plots, bar charts, and various other graphical illustrations. Listing and computational summaries of the physical and chemical properties of the area can also be obtained. If desired, the user can "zoom in" on various parts of the area under consideration and obtain information in whatever detail and form he desires. The primary purpose of this part of the system is to allow the user to be able to recall, manipulate, and display information to aid him in designing the preliminary production and reclamation operations of the mine. In addition, the various computational and mapping routines available greatly facilitate the process of providing the various regulatory agencies required information about the mine site and planned operations.

After the user has analyzed the appropriate maps, he can proceed to utilize the production analysis module. This module provides an interactive, computer aided design tool for planning dragline stripping operations and auxiliary activities. The user can design the dragline operation for a particular pit layout within a region or simply generate designs relative to the average characteristics of the mine site.

The composition of the production analysis module is shown in Figure 4. The module consists of three design levels--each level corresponding to a particular step in a macro-to-micro design analysis hierarchy. Level 2 concerns itself with pit design and dragline sizing. This segment of the module allows the user to evaluate a number of stripping techniques (i.e., the two pass method) relative to an array of design objectives..

<u>Design Level</u>	<u>Analysis, Design, Evaluation</u>
1	Total Mine Plan -Transportation System -Equipment Balance -Interactions -Loading, Hauling -Production Analysis
2	Pit Design -Dragline Selection/Evaluation -Box Cut Location -Optimum Overburden Removal -Optimum Placement of Spoils -Optimum Move Sequence -Optimum Pit Geometry -Dragline Productivity
3	Detailed Evaluation -Swing by Swing Simulation -Overburden Removal Sequence -Dragline Performance - Energy Requirements - Forces - Operating Statistics

Figure 4. Design Levels of Production Analysis Module

Once the dragline operation has been designed, the user can employ either Level 1 or 3. The input required at each of these levels is the output from Level 2. At Level 1 all the auxiliary operations in support of overburden removal are analyzed. Based on production rate of the dragline, other operations are specified accordingly. Included are topsoil removal, drilling and blasting, coal loading and hauling, coal preparation, reclamation, and general support operations. Also available at this level are a series of cash flow analysis reports.

Level 3 is available to perform a detailed evaluation of the overburden removal operation. For a given location in the mine, stripping techniques and specified dragline operational sequence, this level can be employed to perform a swing by swing simulation. Such detail may not be required for production planning per se, but is required in order to predict environmental responses. For example, the techniques employed at this level will be able to trace a given material type from its location in the overburden to its location in the spoil pile. Various dragline performance statistics will also be available at this level.

Having specified the recovery methods and equipment, the user may analyze reclamation methods in a multilevel fashion similar to that employed during production analysis. Using outputs from the production analysis module as well as data regarding environmental limitations, the feasibility of a desired strategy will be evaluated. Next,

programs to estimate reclamation costs and suggested equipment mix can be executed. Feedback shown in Figure 3 to the production analysis module includes: (i) cost information and (ii) feasibility. Hence, the planner may iterate between production and reclamation design activities until he has balanced increased recovery costs with the (hopefully) lower reclamation costs. Or, he may find that the "least cost" recovery plan unduly restricts reclamation options and will cycle back to the production analysis to respecify the mining activities until the desired reclamation goals can be achieved.

Another major component of the computerized planning system is the impact analysis module. After a production plan has been specified, this module is utilized to predict various environmental responses. Examples of such responses would include wind and water erosion, characteristics of ground and surface water, subsidence, wildlife and vegetative recovery, etc. If some of these responses are unacceptable, the user can cycle back through the production analysis module (as illustrated by the feedback loop) and redesign his chosen method of operation. By identifying the causes of the adverse responses, additional constraints would be constructed for the strip-ping method being analyzed in the Production Module. Naturally, the redesign would result in a less productive plan--thus, revealing the trade-offs between production and environmental protection. Again, the plan's environmental impact would be assessed. The process would

continue until the proper balance between productivity and environmental quality was achieved. This procedure is perhaps the most important feature of the mine planning system. By documenting the process of cycling through the production and impact analysis modules, one can objectively evaluate the cost trade-offs between environmental protection and productivity. Once a satisfactory production plan has been achieved, the mine plan can be documented via a whole array of output options.

The final block of the flow diagram represents another useful aspect of the computer planning system. That is, the system is valuable not only for planning operations but also can be employed to monitor activity once mining has commenced. For instance, the features of the mine may not be as originally predicted, the economics of the market may change, equipment and methods may be improved, etc. Thus, the original plan may have to be modified or updated. The system can also be utilized to accomplish this task. Therefore, the computerized planning system is useful as a management tool in the entire spectrum of mining from preliminary design to monitoring existing operations.

Pit Design Methodologies

Since the overburden removal activity yields the greatest cost component of a strip mine, the design of efficient dragline operations is of major concern. To deal with this design problem, the mine

planner must effectively determine his pit geometry and dragline size so as to maximize or minimize some measure of productivity. Thus, the design problem becomes one of choosing the pit design methodology that yields the best results.

Design methodologies for planning pit geometry vs. dragline dimensions fall into three potential classifications: (1) mathematical modeling, (2) computer simulation, and (3) mathematical optimization. As will be pointed out, the first two classifications are highly restrictive in terms of obtaining the most productive plan. Neither group considers the level of productivity resulting from specific dragline size and pit geometry combinations.

Relatively speaking, mathematical modeling proved to be a significant breakthrough in effectively designing pit geometry with respect to dragline operations. Rumfeldt recognized that a functional relationship exists between certain pit design parameters; namely, the pit width, depth of overburden, and dragline reach (15). He developed an equation which expresses reach as a function of these design parameters. He also carried this analysis one step further by relating the dragline reach and bucket size to machine mass. This step gives an indication of the size of dragline necessary to accomplish the overburden removal operation.

Rumfeldt is also credited with illustrating this functional relationship via hand drawn casting diagrams, as shown in Figure 5.

Notice when values are specified for the spoil angle, θ , the highwall angle, ϕ , the overburden depth, d , the coal seam thickness, t , and pit width, w , the maximum effective reach can be obtained.

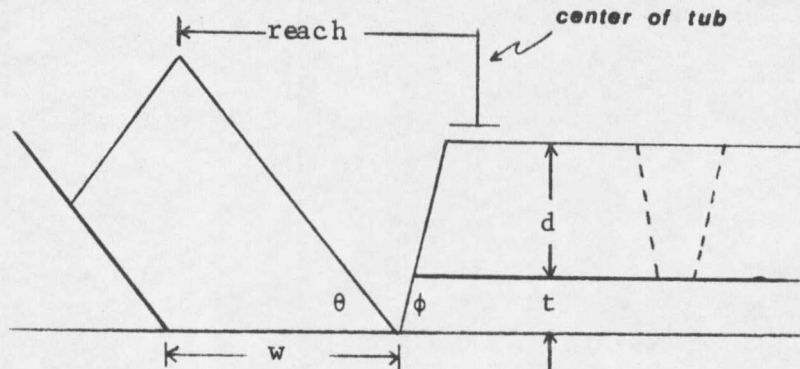


Figure 5. Rumfeldt's Casting Diagram

Rumfeldt's conception of the design problem, facing mining engineers, paved the way for more complex mathematical models. Tomica and Barron extended the technique for two pass systems and computerized their model (12). Their big achievement was the determination of feasible and infeasible dragline systems based on machine weight, required bucket size and maximum reach. As an addition, they combined their mathematical model with an economic package to estimate overall mine costs. This deterministic approach was so flexible, Finch and Dale utilized the concept to determine the maximum effective spoil radius for geometry associated with multiple seams of coal (5).

Simulation has also proven to be an effective tool for planning dragline operations. Dunlap and Jacobs reported the first computer aided simulation in 1955 (3). They designed a simulation model to depict dragline productivity for various sets of design parameters. More recently, computer simulation of dragline operations has become quite sophisticated. McDonnell-Douglas designed a simulation package that represents the overburden removal operation on a swing-by-swing basis (13). Interestingly enough, the required input parameters for such a simulation model are those computed, or specified, in the computation of the variables in Rumfeldt's equation. Perhaps their biggest achievement was adapting computer aided graphics to display intermediate results of the simulation.

Although satisfactory, these methods by no means guarantee the optimum or near optimum pit design with respect to dragline reach. Rumfeldt's method was the most widely applied. However, it fell short in that it forced the user to assume maximum reach yields maximum dragline productivity. This misconception was pointed out by Stefanko in 1973 (16). Three years later Mathematica, Inc., developed a model to compute dragline boom length, again applying Rumfeldt's equation (10). However, this time the model did consider swing time and dragline walk time in the efficiency of the overburden removal operation. The swing time was expressed as a function of average swing angle, pit width, and dragline reach. By linking pit geometry, dragline reach

and swing time, the model became a necessary milestone in predicting dragline efficiency. In fact, this model exposed the possibility of optimizing the tradeoffs between pit geometry parameters and dragline reach relative to some measure of system productivity.

Statement of Thesis Topic

As stated previously, the thesis topic was developed in conjunction with the SEAM program at MSU. The thesis topic features the development of a computer model that provides an answer to evaluating the tradeoffs between pit dimensions and dragline size as related to two pass dragline stripping operations. Upon development, the model was interfaced with the Production Analysis Module of MSU's computer mine planning system.

The model is coded in Fortran and contains relationships similar to those proposed by Rumfeldt. However, the design parameters now denoted decision variables (i.e., the pit width, depth of first pass, and boom and bucket size) are allowed to vary during the analysis. Thus, the coded model can be used simply to determine design feasibility or as a means to optimize the pit dimensions vs. dragline dimensions. Either application of the model allows for interactive computer aided design with all results displayed graphically via CRT.

Currently, the model is operational at Level 2 of the production analysis module of MSU's computerized planning system. It provides

the necessary balance when analyzing single coal seam dragline stripping techniques. With some modification the coded model could be adapted for use on any computer system with a Fortran compiler.

TWO PASS DRAGLINE MODEL

Technical Presentation

The two-pass dragline model was structured as a non-linear programming problem. The basic premise of the model can be expressed as:

$$\text{Minimize } z = F(x_1, x_2, \dots, x_i, \dots, x_n)$$

Subject to:

$$G_k(x_1, x_2, \dots, x_i, \dots, x_n) \geq 0 \quad k = 1, 2, \dots, m$$

The x_i belong to a vector, say $\bar{X}$, containing variables termed a decision vector. The decision variables are comparable to the critical parameters in Rumfeldt's analysis. Here the x_i represent the pit width, depth of cut, dragline boom length, and dragline bucket size. The decision vector can easily be subdivided into two parts: (1) the pit design variables, and (2) the dragline design variables. These variables are altered at each stage of the optimization. An improvement in the selection of x_i 's is reflected in the magnitude of the objective function.

The objective function $F(\bar{X})$ can be thought of as some measure of goodness which predicts dragline productivity for values of the

decision variables. The model allows the user to choose between two productivity measures. They are as follows:

- i. to minimize the time to remove the overburden covering a region,
- ii. to minimize the cost per ton of coal uncovered within a region.

Many times the mine planner must consider various economic factors when planning pit geometry and dragline dimensions. Specifying such factors as a maximum annual production rate, or a specific number of annual operating hours would surely constrain the design from being the most productive in terms of the objective function. Therefore, the model allows the user to make these types of economic considerations by selecting from the following design criteria:

- i. designing the most productive system with a specified maximum annual operating hours,
- ii. designing the most productive system to meet a specified annual production rate with varying operating hours,
- iii. designing the most productive system to meet a specified production rate with fixed operating hours,
- iv. designing the most productive system for a dragline with a maximum purchase price, and
- v. designing the most productive system for a specific dragline.

The design criteria selected have a significant effect on the outcome of the optimization. Thus, they become an integral part of the design process when analyzing the tradeoffs between designs in terms of system productivity. This preceding statement will become evident in the section containing actual examples of model application.

The functions $G_k(\bar{X})$ are the constraint equations. These equations are representative of the real world limitations upon the dragline operation. During the optimization they insure that the decision vector contains feasible values.

If the user doesn't elect to optimize the design, the model can be used simply to determine design feasibility. The value of $F(\bar{X})$ for the inputs would be a measure of design productivity. Negative values for the $G_k(\bar{X})$ would indicate constraint violations; thus, an infeasible design. Utilizing the model in this manner would be similar to the deterministic approaches taken by Rumfeldt, and Tomica and Barron (1,14).

If the optimization is elected, a computerized direct search is done over the x_i to minimize $F(\bar{X})$. The direct search algorithm employed is the Flexible-Tolerance algorithm as described by Himmelblau (9). To initiate the search procedure, the algorithm starts searching a near feasible region with respect to an initial plan at the same time monitoring the value of $F(\bar{X})$. When a specified feasibility tolerance is met, the search terminates. The structure of

the program allows the most recent solution to become a starting point for the next search--if elected. This allows for rapid convergence to a better solution. It should be noted the direct search technique does not guarantee the necessary and sufficient conditions for a global optimum to be satisfied. Therefore, it is unwise to make such a claim. In order to obtain the best solution, the user should alter the initial plan several times, compare the results of the optimization and select the best solution.

Parameter Descriptions

In order to utilize the model, the user must specify values for a number of parameters. These parameters are associated with both drag-line specifications and site characteristics. Appendix I defines the necessary parameters. Those marked with one asterisk are derived within the model; thus, they are not user specified. The parameters marked with two asterisks indicate the parameter may be in the decision vector.¹

Since the model is a subset of the SEAM computerized mine planning system, all parameters are initialized in subroutine ADM. This subroutine resides in the Production Analysis Module of MSU's .

¹The contents of the decision vector may be altered depending upon the design criteria selected. This situation is discussed on page 30.

computerized mine planning system. If the model was isolated from the SEAM system, an input routine must be supplied to initialize the parameters.

The parameters defined in Appendix I are subject to a standard notational convention. Relative to the subscripts of the parameters, the following subscripts are encountered in the model:

- i. ij - denotes the i^{th} pass and the j^{th} position, where
 $i = 1, 2; j = 1, 2, 3$
- ii. s - denotes the overburden in its natural state
- iii. tot - denotes a total
- iv. max - denotes a maximum value
- v. min - denotes a minimum value
- vi. t - denotes a temporary value
- vii. r - denotes rehandle
- viii. req - denotes a requirement

Technique Modeled

Since any mathematical model is an abstract representation of a real world situation, a number of simplifying assumptions must be made. These assumptions limit the complexity of the model to a workable level; yet, they do not or should not restrict the model's usefulness to any great extent. The underlying assumptions inherent within the two pass model can be summarized as follows:

- i. the coal seam is of constant thickness and is parallel to the surface
- ii. the overburden is of constant depth
- iii. the angle of coal after removal of the overburden is the same as the highwall
- iv. the spoil peak approximates a straight line
- v. the dragline operation avoids rehandle if possible
- vi. any rehandle accumulated may be composed of first pass key, first pass main cut volume, and second pass key cut volume

The dragline model incorporates the most common two pass stripping technique employed at western surface mines today (1,10). This stripping technique involves two dragline positions per lift; that is, a key cut and main cut position on each pass. However, if rehandle is accumulated it is possible to have a rehandle position on the second pass. The model allows for analysis of single or tandem dragline operations, and spoil piles with or without a coal fender. Also, all possible rehandle situations are considered to insure design feasibility. Thus, the model can be viewed as being composed of many sub-models, yielding a continuum of alternate pit designs.

The spoil pile configuration for spoiling with a coal fender is shown in Figure 6 by the shaded region. Notice the distance cc is the base of the coal fender and can be calculated as follows:

$$cc = t \cdot (\cot + \cot)$$

[1]

The dashed line in Figure 6 indicates the situation with no coal fender. This circumstance can be viewed as a special case of the spoil pile with a coal fender--that is, when cc equals zero. Since the case without a coal fender is much more common throughout industry, the remaining figures in this thesis will depict this case. However, the accompanying mathematical development does consider both situations.

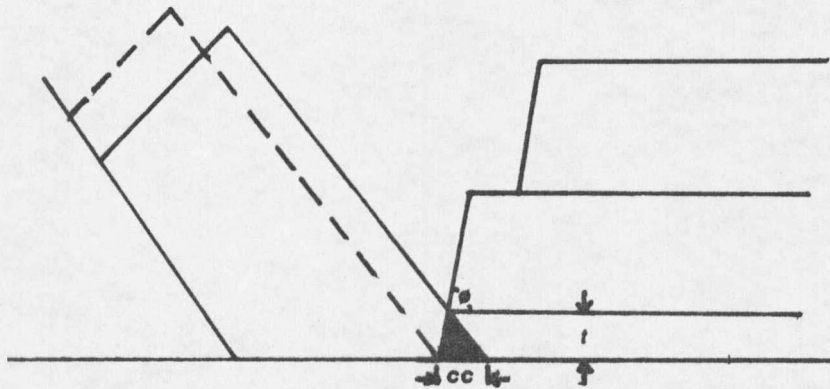


Figure 6. Coal Fender

The spoil pile configuration is determined by the sequence of overburden removal. The sequence of removal is illustrated in the casting diagram of Figure 7. The K indicates key cut material, the M indicates main cut material, and the R indicates rehandle material. Note the subscripts depict which pass the particular area was spoiled

from. Also, the dashed line outlines the fill bench, or rehandle area, to be spoiled.

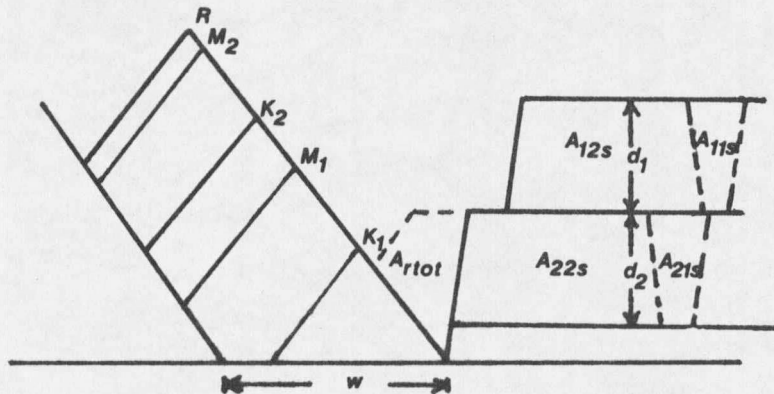


Figure 7. Sequence of Overburden Removal

To accomplish the task of obtaining the aforementioned spoil configuration, the model analyzes the stripping operation one step at a time. Each stage of the analysis will be discussed separately in the remaining paragraph.

Step I. Cut Lengths. First, the cut length is found for both lifts. This cut length is the maximum cut length possible after allowing for the assumption of a straight spoil peak. The calculations for the cut length are given in Appendix II.

Step II. Fill Bench Requirements. The model then commences by determining whether or not a fill bench is necessary to enable the

second pass dragline to reach the spoil peak. If a tandem dragline operation is not in use then the first pass and second pass dragline will be the same. Appendix III derives the equations necessary to make this determination.

Step III. First Pass Key Cut Spoil Configuration. After the fill bench requirements are determined, the key cut spoil configuration is determined for the first pass. The model positions the dragline immediately facing the cut area and allows the dragline to only remove an area based on the bucket width at the bottom of the cut. This area is called the minimum key cut area and is depicted in Figure 7 by A_{11s} . It may be necessary to accumulate some rehandle if the dragline can not reach the required spoil peak. The rehandle composed of first pass key cut material is denoted A_{r11} . Appendix IV contains the calculations for obtaining the key spoil configuration on the first pass.

Step IV. First Pass Main Cut Spoil Configuration. Next, the first pass main cut spoil configuration is dealt with. The main cut area is shown in Figure 7 and is denoted A_{12s} . The spoil peak location is dependent on whether or not there is a fill bench and whether or not the dragline can reach the required spoil peak. Tests are made within the model to satisfy these requirements. If the dragline can not reach the required spoil peak without rehandle or there exist fill

bench requirements, some rehandle is planned to satisfy the demand.

This rehandle is denoted A_{r12} . The calculations for determining the main cut spoil configuration on the first pass are given in Appendix V. This completes the removal of the first pass overburden.

Step V. Second Pass Key Cut Spoil Configuration. The removal of the second pass key cut begins the second pass stripping operation. Again, the dragline is positioned directly facing the cut and the amount of key cut material removed is dependent on the bucket width at the bottom of the cut. This area is denoted A_{21s} in Figure 7. The second pass key cut material may result in more rehandle if the second pass dragline can not reach significantly past the spoil peak created on the first pass. This rehandle is denoted A_{r21} in the model. Appendix VI contains the calculations necessary to determine the second pass key cut spoil configuration. Similarly, the second pass main cut spoil configuration is determined. The second main cut is shown in Figure 7 as A_{22s} . Again, the dragline position from the old highwall is determined by the location of the spoil peak required to spoil the area A_{22s} without rehandle. These calculations are given in Appendix VII.

Step VI. Rehandle Position. If rehandle exists then the spoil peak for depositing this volume must be determined. This area is shown in Figure 7 as A_{rtot} . The area A_{rtot} may be composed of A_{r11} ,

A_{r12} , and A_{r21} . The calculations for total spoil configuration including rehandle are in Appendix VIII.

Step VII. Dragline Distance from Cut Face. After the dragline locations from the old highwall are determined for each cut position, the model computes the distance the dragline remains from the cut face. This distance is highly dependent on the cut length and is calculated for each dragline position in Appendix II.

Performance Objectives and Design Criteria

Again, the two design objectives are

- i. to minimize the time to remove overburden,
- ii. and to minimize the cost per ton of coal exposed.

This section explains the mathematical derivation of these two performance measures. The effects of the design criteria on the decision vector is also discussed.

Both performance measures are a function of time to remove the overburden at each cut. This time is composed of dig time and walk time, and can be estimated from a formula expressing dig time as function of the average swing angle times the volume of soil removed plus the walk time. The exact mathematical expression for the time to excavate the overburden at the j^{th} position on the i^{th} pass is:

$$T_{ij} = \frac{l_i \cdot A_{ijs}}{B_i \cdot F} (40.2 + 2.88 \alpha_{ij}) + V_i (|x_{ij} - x_{\text{previous}}|) + (|y_{ij} - y_{\text{previous}}|) + CC2 \quad [2]$$

The walk time is expressed in terms of the rectilinear distance moved. The values relating average swing angle to dig time were obtained from Cohen (2). Appendix IX gives the calculations for determining the average swing angle.

When minimizing the total time to remove the overburden, the total time can be easily derived from the following equation:

$$T = \sum_{i=j}^2 \sum_{j=i}^2 \text{ or } 3 \frac{L_P W_P}{l_i w} T_{ij} + 2 \frac{W_P}{W} [CC2 + V(L_P + W)] \quad [3]$$

Note the last term expresses the deadhead time. Also the summation limits on T_{ij} vary since there are only two dragline positions on the first pass and possibly three positions on the second pass.

The second performance measure, minimizing the cost per ton of coal exposed, can be expressed as a function of the total time to excavate times the dragline hourly operating and ownership costs divided by the tons of coal uncovered. Restricting the cost function to these cost components minimizes the complexity of the problem. The hourly operating and ownership cost for the i^{th} dragline can be estimated from the following regression equations, respectively.

$$OP_i = 61.96 + 9.74 (MUFD_i) \quad [4]$$

$$OW_i = (17.695 + 7.04 (MUFD_i)) \cdot \left(\frac{7467}{OPHS} \right) \quad [5]$$

These equations were developed by Flour of Utah from 1975 cost figures (6). The MUFD is known as the maximum usefulness factor originally investigated by Rumfeldt (15). Flour of Utah developed the following relationship to compute its value.

$$MUFD_i = \frac{(b_i) \cos \beta \cdot B_i}{1000} \quad \text{for } i = 1, 2 \quad [6]$$

Finally, the cost per ton of coal can be estimated by:

$$C_t = \sum_{i=1}^2 (OP_i + OW_i) \sum_{j=1}^{2 \text{ or } 3} T_{ij} / [(L_P)(W_P)(t)(\rho)] \quad [7]$$

Notice two relationships have been defined as a function of both the pit design and the dragline variables. Therefore, any change in magnitude of the decision variables can be observed by the incremental change in the productivity measures.

Recall the two performance measures are functions of the decision vector which is composed of pit design variables and dragline design variables. The user has the option prior to using the model of explicitly excluding either decision subset from the decision vector by setting certain bits in the code word IVAR. The bit settings are described at the beginning of the program listing in Appendix XII.

This action would simply make the subset that was excluded fixed parameters during analysis.

The decision vector also may be implicitly altered by the selection of certain design criteria. However, in this case only one or both of the dragline design variables may be excluded from the decisions set. Again the excluded variable(s) is held constant during the analysis. Notice the possibility of implicitly excluding one or both dragline design variables from the decision set reinforces the statement that the design criteria have a significant effect on model results. To cite an example of this situation, suppose the user has elected to minimize time and explicitly included both the pit design and dragline design variables in the decision set. However, assume that he also selected to design the operation for a particular production rate within a fixed maximum annual operating hours. The bucket size is determined from the production rate and the boom length is set to its maximum, based on an equation from Stefanko.¹ Thus, in this case neither dragline decision variable has entered the true state of a decision variable. Other such combinations do exist and are discussed in full in Appendix X.

¹The equation is discussed more fully on page 32.

Design Constraints

The design constraints are simply physical limitations, inherent within the dragline operation. These restrictions usually result from (i) certain dragline boom length and bucket combinations, and (ii) working space requirements. With contemporary pit design techniques such as Rumfeldt's, many of these limitations are implicitly considered by the designer. However, within the context of the model, the capability exists to explicitly express many of the limitations in the form of mathematical equations. These equations, considered a constraint set, provide feedback concerning design feasibility and constrain the computer optimization algorithm.

The various boom length and bucket combinations that do exist are constrained in two ways. If a dragline purchase price has been specified, the dragline required to successfully operate with a particular boom and bucket combination should not exceed this price. In 1975 Flour of Utah developed an equation which expresses dragline purchase price as a function of the boom and bucket size (6). This equation was introduced in the model as a constraint and can be written as follows:

$$(\text{Dragline Price}) - (987.71 + .5106 (B_i)(b_i)) \geq 0 \quad [8]$$

Another situation that requires constraining is the boom length vs. the bucket size. A plot of dragline dumping radius versus bucket size

was obtained from Winczewski (18). The original data for the plot was extracted from a table developed by Stefanko (17). An equation was fit to the data expressing dumping radius or dragline reach as a function of the bucket size. The regression equation had an $R^2 = .79$. Therefore, the fit was assumed satisfactory and the following equation was introduced as a constraint.

$$351.67 - 1633.33/B_i - \text{Reach}_i \geq 0 \quad [9]$$

$$\text{where } \text{Reach}_i = (b_i) \cos \beta + c_i \quad [10]$$

Spatial limitations during a stripping operation are of major concern. Since these limitations are highly independent in nature, they will be discussed on an individual basis. First, let us consider the maximum area available for rehandle. This area is shown in Figure 8 and is depicted by the shaded region. The amount of rehandle

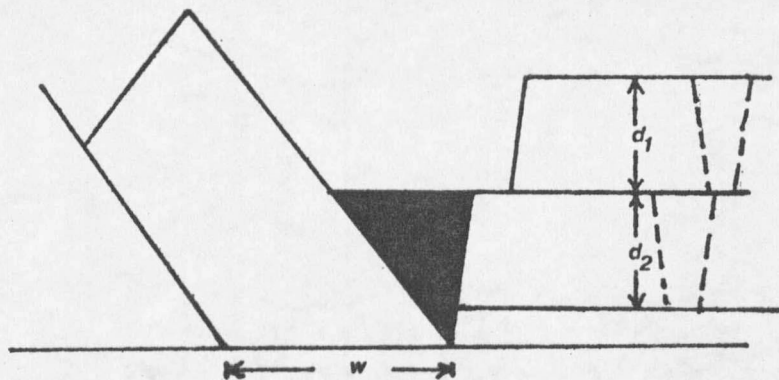


Figure 8. Maximum Rehandle Area

produced for a particular decision set cannot exceed this area. This area can be expressed mathematically as follows:

$$A_{rmax} = 1/2 \cdot (\cot\theta + \cot\phi) \cdot (d_2)^2 \quad [11]$$

Now a constraint can be written to restrict the total rehandle accumulated during the optimization below the maximum value. This constraint can be written as follows:

$$A_{rmax} - A_{rtot} \geq 0 \quad [12]$$

The amount of rehandle accumulated for a particular design must also satisfy the fill bench requirements. Again, the fill bench enables the dragline to reach the spoil peak on the second pass. This area is depicted in Figure 9 by the shaded region. The area required by the

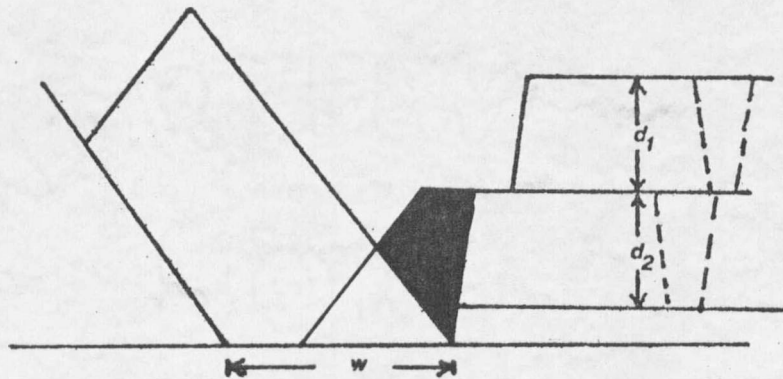


Figure 9. Fill Bench Area

fill bench is derived in Appendix II. Thus, we can constrain the rehandle accumulated for a particular design to satisfy the fill bench

requirement by the following equation:

$$A_{rtot} - A_{rreq} \geq 0 \quad [14]$$

The next constraint is relative to the assumption of spoiling only the minimum key cut area. This constraint restricts the model from spoiling less than the minimum area. This situation may result if A_{llmax} is less than A_{llmin} .¹

$$A_{llmin} - (BW_i + d_i \cdot \cot \phi) \cdot d_i \geq 0 \quad [15]$$

Another spatial consideration is the dragline location from both the cut face and the old highwall. The dragline must not hang over either edge during the stripping operation. Figure 10 is an artist's conception of the plan of the dragline location. The crosses mark the dragline positions on each pass of the operation. Had these positions been too close to the cut face or old highwall, the operation would be infeasible. The following constraints restrain the optimization to the desired feasible region in terms of dragline position.

$$x_{ij} - c_i \geq 0 \text{ for } i = 1, 2 \quad [16]$$

$$j = 1, 2, \text{ or } 3 \quad [17]$$

$$y_{ij} - c_i \geq 0$$

¹See Appendix IV for computation of A_{llmax} .

Notice the constraint set given here can easily be altered to conform to any mining situation. Also, new constraints easily may be added in order to account for particularly unique situations.

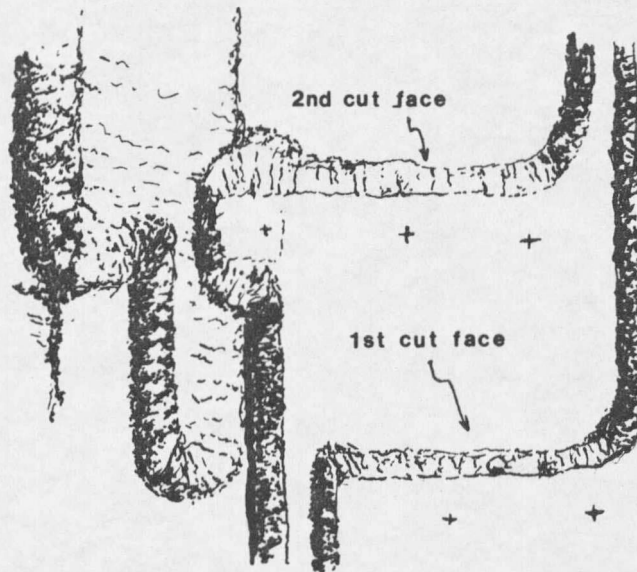


Figure 10. Plan View of Dragline Locations

A constraint directly related to neither the boom length an bucket size nor spatial limitation is the productivity constrain. This constraint is employed when tandem draglines are used. It insures the draglines are working with a certain time of one another. The purpose is to keep dragline idle time at a minimum; the constraint can be expressed as follows:

$$\left(\sum_{j=1}^2 T_{1j} - \sum_{j=1}^{2 \text{ or } 3} T_{2j} \right) - \text{DELTUB} \geq 0$$

EXAMPLE APPLICATION

This chapter illustrates the application of the two pass model by employing the optimization to design the "best" pit geometry for a specific mine site. To fully understand how the model responds to particular design situations, the optimization is employed for both the minimum time objective and the minimum cost per ton of coal objective. The design criteria are also varied for each objective function to indicate their respective effects on the results of the optimization. Computer drawn casting diagrams will be used to aid in the interpretation of model results.

Prior to running the model, the necessary design parameters must be specified by the user. These parameters describe the particular site characteristics and certain dragline features. Table II gives the values for the design parameters relative to the following examples. In order to more fully understand the model's response to the various design options, the above values will remain the same for all examples.

After specifying values for the design parameters, an initial plan is required of the user. This initial plan consists of specifying values for the decision variables. Recall the decision variables are the pit width, first pass depth, and the dragline boom length and bucket size. In the following examples both decision subsets are contained in the decision vector; thus, no variables are

Table II. Parameter Specifications

Parameter	Description	Value
W_p	the width of the total mine plan	2000 ft.
L_p	the length of the total mine plan	2000 ft.
f	the soil swell factor	.25
d	the depth of the overburden	150 ft.
t	the coal seam thickness	25 ft.
θ	the spoil angle of response	39°
ϕ	the highwall angle	72°
F	the bucket fill factor	.80
f	the boom angle for both the first and second pass draglines	35°
r	the height of the bottom of the boom for both the first and second pass draglines	15 ft.
c	the cab radius for both the first and second pass draglines	25 ft.
CC2	the fixed time to move a dragline	.5 hrs.
V	the dragline walking speed	.0013 $\frac{\text{hrs.}}{\text{ft.}}$
ρ	the coal density	.04 $\frac{\text{tons}}{\text{ft.}^3}$
castd	the dragline casting distance	15 ft.
Deltub	the time difference for the first and second pass excavation times	24 hr

explicitly excluded from the decision set. This will exhibit the most general application of the model in terms of decision variables. The values specified for the initial plan will be used as a starting point for the search algorithms in all the examples. Table III

contains the values for the initial decision set that is used in the examples. It should be pointed out that this initial plan is infeasible, because it violates the productivity constraint.¹ Using such a plan as a starting point for the search will give an indication of the ability of the optimization to strive for feasibility as well as optimality.

Table III. Initial Plan

Variable	Description	Value
w	the pit width	140 ft.
d ₁	the first pass depth	70 ft.
b ₁	the boom length of the first pass dragline	275 ft.
b ₂	the boom length of the second pass dragline	300 ft.
B ₁	the bucket size of the first pass dragline	63 yd. ³
B ₂	the bucket size of the second pass dragline	63 yd. ³

At this point the user may select the design criteria. Three examples, corresponding to different design options, are given for each objective function.² The first example for each objective

¹Refer to the section entitled Design Constraints for an explanation of the productivity constraint.

²Appendix X contains an explanation of the design options and their respective effects on the decision vector.

function derives the most productive plan for the site in terms of annual tons of coal uncovered; thus the annual production rate is derived in the model. This optimization problem is solved with the maximum annual operating hours set equal to 7000 hours. The next example requires the model to yield a specified annual production rate of 4 million tons of coal per year. For this situation the annual operating hours are derived in the model. The final example for each objective function employs the organization for a specified annual production rate of 4 million tons of coal and a specified annual operating hours of 7000 hours. Since these examples are run with the same initial plan, generalizations can be made relative to the economic tradeoffs associated with the various design criteria. Keep in mind the decision vector may be altered depending on the design criteria.¹

The other design options available will not be illustrated in the following examples, primarily because many of them are simply subsets of the design options selected for analysis. For instance, all the examples consider a tandem dragline operation, since the model views a single dragline system as a tandem operation with both draglines the same. Also, the examples consider a spoil configuration without a coal fender as opposed to the configuration with a coal fender. Again one can be viewed as a subset of the other; simply shifting the spoil

¹The alteration of the decision vector is discussed in Appendix X.

peak closer to the dragline and decreasing rehandle when a coal fender is selected. Further, the purchase price option has been neglected since this option basically adds a constraint which restricts the boom and bucket combination.

Since the two pass model is contained in MSU's Computerized Mine Planning System, the user is queried for the foregoing initial inputs on a question-answer basis. Therefore, if the model were isolated from the system, an alternate source of input must be devised.

Minimum Time Objective

Example I. The results of the optimization for the first example are shown in Figure 11 on the following page. The model results of the initial plan are not shown, however, a scaled casting diagram similar to Figure 11 would illustrate the relationships.

Model results for the initial plan yielded a time to remove the overburden of 8547 hours. Notice from Figure 11 the results of the optimization produced a time of 6466 hours. This time compares favorably to that of the initial plan. Thus, the optimization not only made the plan feasible in terms of the constraint set, but provided a decrease in the time to remove the overburden. Table IV summarizes the plan yielded by the optimization.

This plan is considered the most productive for the mine site in terms of annual coal production and with respect to the design objective, since all the decision variables are explicitly included in the

***** SITE CHARACTERISTICS *****

LENGTH OF MINE PLAN IS 2000. FT.
 WIDTH OF MINE PLAN IS 2000. FT.
 DEPTH OF OVERBURDEN IS 150. FT.
 COAL SEAM THICKNESS IS 25. FT.
 SPOIL ANGLE OF REPOSE IS 39. DEG.
 HIGHWALL ANGLE IS 72. DEG.
 SOIL SWELL FACTOR IS .25

***** DRAGLINE CHARACTERISTICS *****

1ST BUCKET SIZE IS 78. CU. YDS.
 2ND BUCKET SIZE IS 79. CU. YDS.
 1ST BOOM LENGTH IS 373. FT.
 2ND BOOM LENGTH IS 374. FT.
 BOOM ANGLE IS 35. DEG.
 TUB RADIUS IS 25. FT.
 ANNUAL OPERATING HOURS IS 7000.
 FIXED TIME TO MOVE DRAGLINE IS .5 HRS.
 DRAGLINE WALKING SPEED IS .0013 HR./FT.

***** COMPUTED PIT GEOMETRY *****

1ST WIDTH OF KEY CUT IS 72. FT.
 2ND WIDTH OF KEY CUT IS 58. FT.
 1ST WIDTH OF MAIN CUT IS 81. FT.
 2ND WIDTH OF MAIN CUT IS 95. FT.
 WIDTH OF PREVIOUS PIT IS 153. FT.
 1ST LENGTH OF CUT IS 101. FT.
 2ND LENGTH OF CUT IS 101. FT.
 WIDTH OF SAFETY BENCH IS 3. FT.
 DEPTH OF FIRST PASS IS 85. FT.

***** DRAGLINE LOCATIONS *****

1ST PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 157. FT.
 1ST PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 52. FT.
 FROM OLD HIGHWALL EDGE IS 128. FT.
 2ND PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 63. FT.
 FROM OLD HIGHWALL EDGE IS 132. FT.
 2ND PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 52. FT.
 FROM OLD HIGHWALL EDGE IS 51. FT.
 2ND PASS REHANDLE LOCATION IS -
 FROM CUT FACE IS 52. FT.
 FROM OLD HIGHWALL EDGE IS 27. FT.

***** DESIGN PRODUCTIVITY *****

TIME TO REMOVE OVERBURDEN IS 6466. HRS.
 COST PER TON OF COAL IS \$.786
 COAL UNCOVERED PER YEAR IS 8637. K TONS

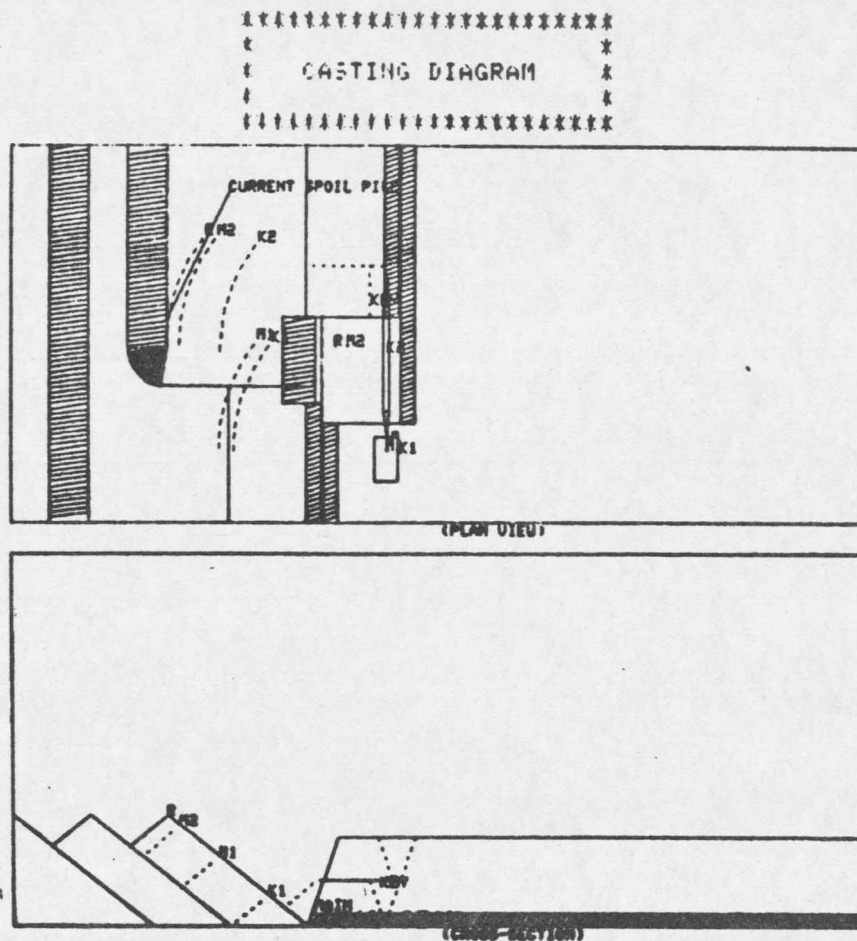


Figure 11. Example I Minimum Time

decision set. However, since only one initial plan was input, it is possible the optimization produced a local optimum. Thus, generally the optimization would be run a number of times with different initial

Table IV. Summary of the Results for Example I of the Minimum Time Objective

Variable	Description	Value
w	the pit width	153 ft.
d ₁	the first pass depth	85 ft.
b ₁	the boom length of the first pass dragline	373 ft.
b ₂	the boom length of the second pass dragline	374 ft.
B ₁	the bucket size of the first pass dragline	78 yd. ³
B ₂	the bucket size of the second pass dragline	79 yd. ³

inputs to gain confidence in the results. Since the decision vector contains the maximum number of variables and no design criteria are selected, these results will be considered the most productive plan with respect to minimum time objective, yielding an annual production rate of 8.637 million tons.

Example II. As stated previously, the second example requires an annual production rate of 4 million tons of coal with the annual operating hours derived in the model. The results of the optimization are shown in Figure 12. Notice the optimization decreased the time

***** SITE CHARACTERISTICS *****

LENGTH OF MINE PLAN IS 2000. FT.
 WIDTH OF MINE PLAN IS 2000. FT.
 DEPTH OF OVERBURDEN IS 150. FT.
 COAL SEAM THICKNESS IS 25. FT.
 SPOIL ANGLE OF REPOSE IS 39. DEG.
 HIGHWALL ANGLE IS 72. DEG.
 SOIL SWELL FACTOR IS .25

***** DRAGLINE CHARACTERISTICS *****

1ST BUCKET SIZE IS 68. CU. YDS.
 2ND BUCKET SIZE IS 76. CU. YDS.
 1ST BOOM LENGTH IS 370. FT.
 2ND BOOM LENGTH IS 373. FT.
 BOOM ANGLE IS 35. DEG.
 TUB RADIUS IS 25. FT.
 ANNUAL OPERATING HOURS IS 3437.
 FIXED TIME TO MOVE DRAGLINE IS .5 HRS.
 DRAGLINE WALKING SPEED IS .0013 HR./FT.

***** COMPUTED PIT GEOMETRY *****

1ST WIDTH OF KEY CUT IS 71. FT.
 2ND WIDTH OF KEY CUT IS 59. FT.
 1ST WIDTH OF MAIN CUT IS 80. FT.
 2ND WIDTH OF MAIN CUT IS 93. FT.
 WIDTH OF PREVIOUS PIT IS 151. FT.
 1ST LENGTH OF CUT IS 100. FT.
 2ND LENGTH OF CUT IS 101. FT.
 WIDTH OF SAFETY BENCH IS 3. FT.
 DEPTH OF FIRST PASS IS 85. FT.
 WIDTH OF FILL BENCH IS 11. FT.

***** DRAGLINE LOCATIONS *****

1ST PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 154. FT.
 1ST PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 127. FT.
 2ND PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 63. FT.
 FROM OLD HIGHWALL EDGE IS 129. FT.
 2ND PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 52. FT.
 2ND PASS REHANDLE LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 27. FT.

***** DESIGN PRODUCTIVITY *****

TIME TO REMOVE OVERBURDEN IS 5827. HRS.
 COST PER TON OF COAL IS \$ 1.089
 COAL UNCOVERED PER YEAR IS 4000. K TONS

|||||*****
 | CHETING DIAGRAM |
 | *****
 |||||*****

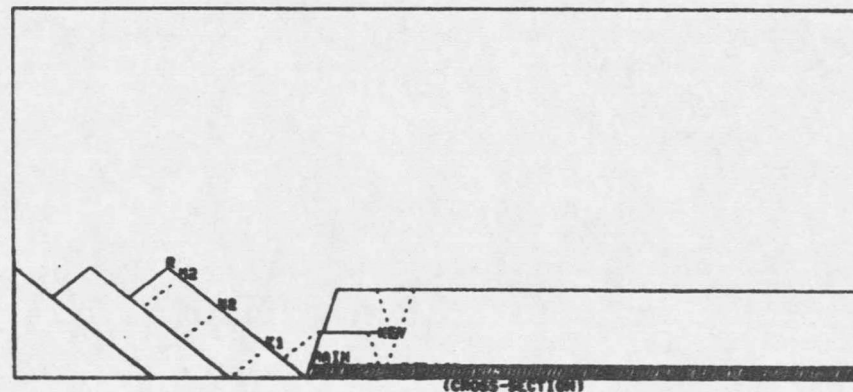
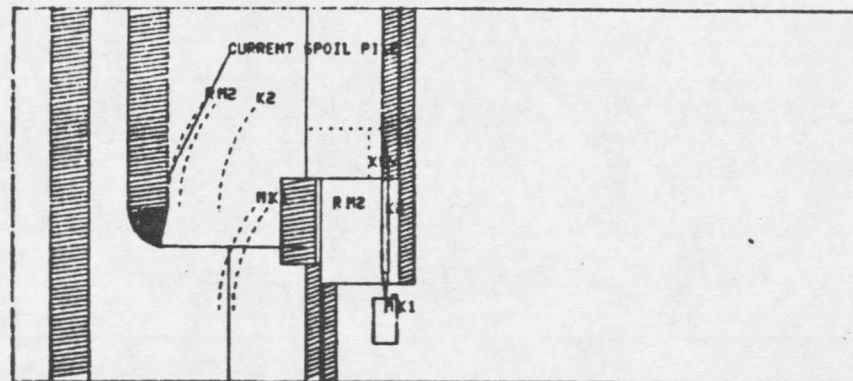


Figure 12. Example II Minimum Time

from 8547 hours for the initial plan to 6827 hours, and once again produced a feasible design. The solution is summarized in Table V.

Table V. Results of Optimization for Example II of Minimum Time Objective

Variable	Description	Value
w	the pit width	151 ft.
d ₁	the first pass depth	85 ft.
b ₁	the dragline boom length of first pass	370 ft.
b ₂	the dragline boom length of second pass	373 ft.
B ₁	the dragline bucket size on the first pass	68 yd. ³
B ₂	the dragline bucket size on the second pass	76 yd. ³

When comparing the minimum time from the first example to the results of this example, it is not unusual to find the improved time to be less than in the first example. This generalization seems to be logical, since the model is forced to yield a less productive plan in this example by decreasing the annual production rate to 4 million tons.

Example III. This example requires an annual production rate of 4 million tons of coal, with 7000 annual operating hours. Notice these requirements place even greater restrictions on the

optimization. Thus, constraining the optimization to generally yield a less productive plan than the ones generated in the first and second examples.

This set of design criteria slightly alter the initial plan by resizing the dragline buckets to meet the specified production rate with the annual operating hours.¹ The first pass dragline bucket is changed from 63 cubic yards to 26 cubic yards and the second pass dragline's bucket is changed from 63 cubic yards to 47 cubic yards. All other initial inputs remained the same. The altered initial plan is again infeasible and requires a time of 12050 hours to remove the overburden.

The results of the optimization are shown in Figure 13 on the following page. Notice to gain design feasibility requires an increase in the excavation time to 14016 hours. These results are summarized in Table VI.

¹Appendix X for details when solving for the bucket size.

***** SITE CHARACTERISTICS *****

LENGTH OF MINE PLAN IS 2000. FT.
WIDTH OF MINE PLAN IS 2000. FT.
DEPTH OF OVERBURDEN IS 150. FT.
COAL SEAM THICKNESS IS 25. FT.
SPOIL ANGLE OF REPOSE IS 39. DEG.
HIGHWALL ANGLE IS 72. DEG.
SOIL SWELL FACTOR IS .25

***** DRAGLINE CHARACTERISTICS *****

1ST BUCKET SIZE IS 35. CU. YDS.
2ND BUCKET SIZE IS 35. CU. YDS.
1ST BOOM LENGTH IS 342. FT.
2ND BOOM LENGTH IS 342. FT.
BOOM ANGLE IS 35. DEG.
TUB RADIUS IS 25. FT.
ANNUAL OPERATING HOURS IS 7000.
FIXED TIME TO MOVE DRAGLINE IS .5 HRS.
DRAGLINE WALKING SPEED IS .0013 HR./FT.

***** COMPUTED PIT GEOMETRY *****

1ST WIDTH OF KEY CUT IS 75. FT.
2ND WIDTH OF KEY CUT IS 55. FT.
1ST WIDTH OF MAIN CUT IS 69. FT.
2ND WIDTH OF MAIN CUT IS 89. FT.
WIDTH OF PREVIOUS PIT IS 144. FT.
1ST LENGTH OF CUT IS 97. FT.
2ND LENGTH OF CUT IS 97. FT.
WIDTH OF SAFETY BENCH IS 5. FT.
DEPTH OF FIRST PASS IS 91. FT.
WIDTH OF FILL BENCH IS 104. FT.

***** DRAGLINE LOCATIONS *****

1ST PASS KEY CUT LOCATION IS -
FROM CUT FACE IS 50. FT.
FROM OLD HIGHWALL EDGE IS 149. FT.
1ST PASS MAIN CUT LOCATION IS -
FROM CUT FACE IS 45. FT.
FROM OLD HIGHWALL EDGE IS 156. FT.
2ND PASS KEY CUT LOCATION IS -
FROM CUT FACE IS 61. FT.
FROM OLD HIGHWALL EDGE IS 124. FT.
2ND PASS MAIN CUT LOCATION IS -
FROM CUT FACE IS 45. FT.
FROM OLD HIGHWALL EDGE IS 53. FT.
2ND PASS REHANDLE LOCATION IS -
FROM CUT FACE IS 45. FT.
FROM OLD HIGHWALL EDGE IS 5. FT.

***** DESIGN PRODUCTIVITY *****

TIME TO REMOVE OVERBURDEN IS 14016. HRS.
COST PER TON OF COAL IS \$.827
COAL UNCOVERED PER YEAR IS 4000. K TONS

* CASTING DIAGRAM *

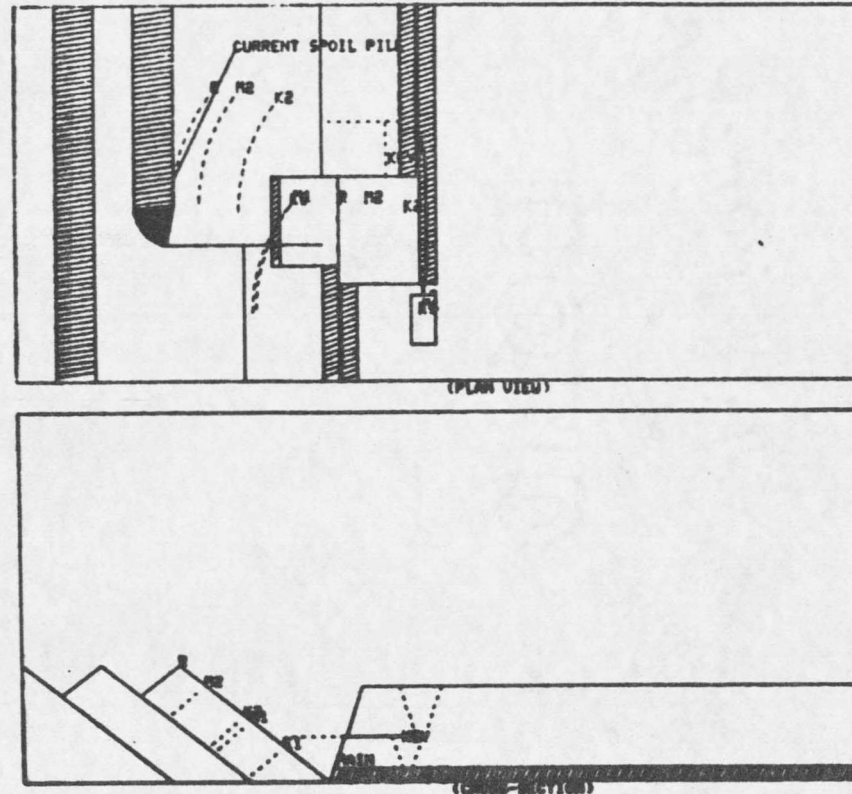


Figure 13. Example III Minimum Time

Table VI. Results of Example III for the Minimum Time Objective

Variable	Description	Value
w	the pit width	144
d ₁	the first pass depth	91
b ₁	the dragline boom length on the first pass	341
b ₂	the dragline boom length on the second pass	341
B ₁	the dragline bucket size on the first pass	35
B ₂	the dragline bucket size on the second pass	35

Minimum Cost per Ton of Coal Objective

Example I. The most productive plan in terms of the minimum cost objective is shown in Figure 14. For this design objective the initial plan yielded a cost per ton of coal of \$.712. Naturally, the productivity constraint is once again violated which makes the initial plan infeasible.

From Figure 14 we see the improved plan has a cost per ton of coal of \$.643 with an annual yield of 8475 tons. This plan is summarized in Table VII.

Example II. This example requires an annual production rate of 4 million tons with the annual operating hours derived in the model.

***** SITE CHARACTERISTICS *****

LENGTH OF MINE PLAN IS 2000. FT.
 WIDTH OF MINE PLAN IS 2000. FT.
 DEPTH OF OVERBURDEN IS 150. FT.
 COAL SEAM THICKNESS IS 25. FT.
 SPOIL ANGLE OF REPOSE IS 39. DEG.
 HIGHWALL ANGLE IS 72. DEG.
 SOIL SWELL FACTOR IS .25

***** DRAGLINE CHARACTERISTICS *****

1ST BUCKET SIZE IS 56. CU. YDS.
 2ND BUCKET SIZE IS 95. CU. YDS.
 1ST BOOM LENGTH IS 273. FT.
 2ND BOOM LENGTH IS 318. FT.
 BOOM ANGLE IS 35. DEG.
 TUB RADIUS IS 25. FT.
 ANNUAL OPERATING HOURS IS 7000.
 FIXED TIME TO MOVE DRAGLINE IS .5 HRS.
 DRAGLINE WALKING SPEED IS .0013 HR./FT.

***** COMPUTED PIT GEOMETRY *****

1ST WIDTH OF KEY CUT IS 70. FT.
 2ND WIDTH OF KEY CUT IS 60. FT.
 1ST WIDTH OF MAIN CUT IS 62. FT.
 2ND WIDTH OF MAIN CUT IS 72. FT.
 WIDTH OF PREVIOUS PIT IS 132. FT.
 1ST LENGTH OF CUT IS 88. FT.
 2ND LENGTH OF CUT IS 94. FT.
 WIDTH OF SAFETY BENCH IS 2. FT.
 DEPTH OF FIRST PASS IS 82. FT.
 WIDTH OF FILL BENCH IS 52. FT.

***** DRAGLINE LOCATIONS *****

1ST PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 45. FT.
 FROM OLD HIGHWALL EDGE IS 134. FT.
 1ST PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 40. FT.
 FROM OLD HIGHWALL EDGE IS 132. FT.
 2ND PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 109. FT.
 2ND PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 40. FT.
 FROM OLD HIGHWALL EDGE IS 33. FT.
 2ND PASS REHANDLE LOCATION IS -
 FROM CUT FACE IS 40. FT.
 FROM OLD HIGHWALL EDGE IS -14. FT.

***** DESIGN PRODUCTIVITY *****

TIME TO REMOVE OVERBURDEN IS 6581. HRS.
 COST PER TON OF COAL IS \$.643
 COAL UNCOVERED PER YEAR IS 8475. K TONS

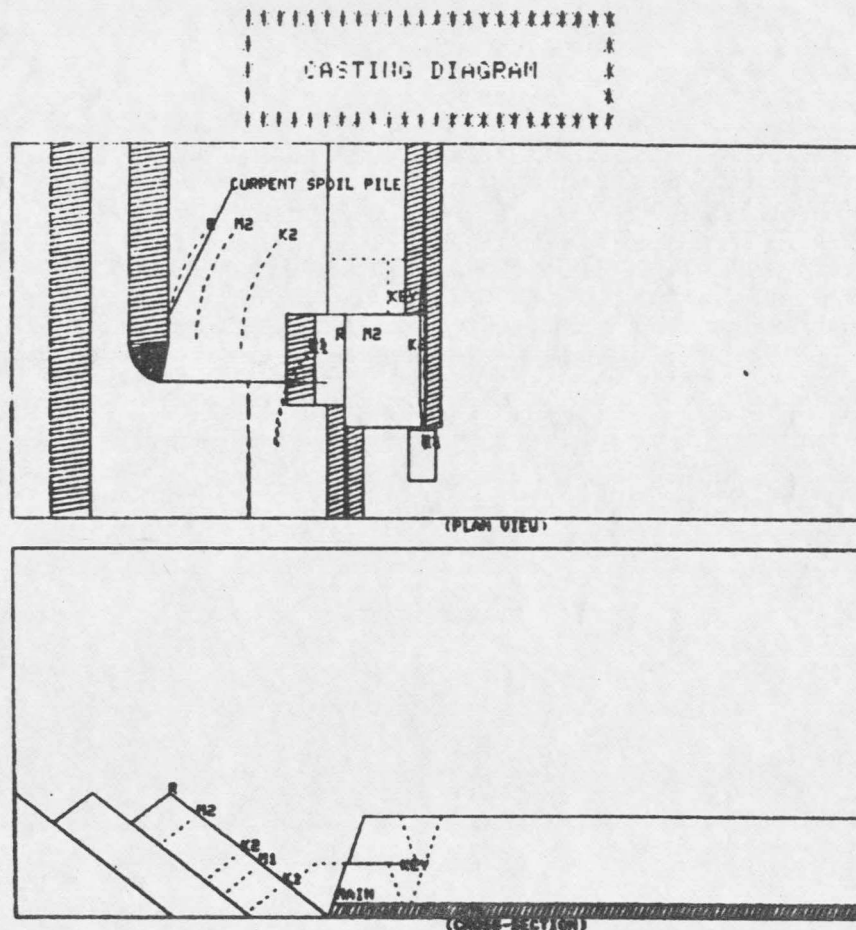


Figure 14. Example I Minimum Cost

Table VII. Final Decision Vector

Variable	Description	Value
w	the pit width	132
d ₁	the first pass depth	82
b ₁	the boom length of first pass dragline	273
b ₂	the boom length of second pass dragline	318
B ₁	the bucket size of first pass dragline	56
B ₂	the bucket size of second pass dragline	95

Employing this strategy caused the cost per ton of coal for the initial plan to increase from \$.712 for the initial plan to \$.796.

The optimization yielded the plan shown in Figure 15. Notice the objective value increased to \$.899 per ton in order to gain design feasibility. This design still slightly violates the productivity constraint. However, for the plan to become totally feasible means a further increase in the objective function. The results of the optimization are summarized in Table VIII.

Example III. The design criteria for this example require an annual production rate of 4 million tons of coal within 7000 annual operating hours. Recall as in Example III for the minimum time objective the dragline buckets are resized for the initial plan. A cost per ton of coal of \$.684 is derived by the model for the initial plan and the constraint set indicates the plan is feasible.

***** SITE CHARACTERISTICS *****

LENGTH OF MINE PLAN IS 2000. FT.
 WIDTH OF MINE PLAN IS 2000. FT.
 DEPTH OF OVERBURDEN IS 150. FT.
 COAL SEAM THICKNESS IS 25. FT.
 SPOIL ANGLE OF REPOSE IS 39. DEG.
 HIGHWALL ANGLE IS 72. DEG.
 SOIL SWELL FACTOR IS .25

***** DRAGLINE CHARACTERISTICS *****

1ST BUCKET SIZE IS 58. CU. YDS.
 2ND BUCKET SIZE IS 83. CU. YDS.
 1ST BOOM LENGTH IS 275. FT.
 2ND BOOM LENGTH IS 319. FT.
 BOOM ANGLE IS 35. DEG.
 TUB RADIUS IS 25. FT.
 ANNUAL OPERATING HOURS IS 3629.
 FIXED TIME TO MOVE DRAGLINE IS .5 HRS.
 DRAGLINE WALKING SPEED IS .0013 HR./FT.

***** COMPUTED PIT GEOMETRY *****

1ST WIDTH OF KEY CUT IS 71. FT.
 2ND WIDTH OF KEY CUT IS 59. FT.
 1ST WIDTH OF MAIN CUT IS 62. FT.
 2ND WIDTH OF MAIN CUT IS 74. FT.
 WIDTH OF PREVIOUS PIT IS 133. FT.
 1ST LENGTH OF CUT IS 88. FT.
 2ND LENGTH OF CUT IS 94. FT.
 WIDTH OF SAFETY BENCH IS 3. FT.
 DEPTH OF FIRST PASS IS 84. FT.
 WIDTH OF FILL BENCH IS 62. FT.

***** DRAGLINE LOCATIONS *****

1ST PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 45. FT.
 FROM OLD HIGHWALL EDGE IS 136. FT.
 1ST PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 40. FT.
 FROM OLD HIGHWALL EDGE IS 127. FT.
 2ND PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 55. FT.
 FROM OLD HIGHWALL EDGE IS 111. FT.
 2ND PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 40. FT.
 FROM OLD HIGHWALL EDGE IS 36. FT.
 2ND PASS REHANDLE LOCATION IS -
 FROM CUT FACE IS 40. FT.
 FROM OLD HIGHWALL EDGE IS -13. FT.

***** DESIGN PRODUCTIVITY *****

TIME TO REMOVE OVERBURDEN IS 7213. HRS.
 COST PER TON OF COAL IS \$.899
 COAL UNCOVERED PER YEAR IS 4000. K TONS

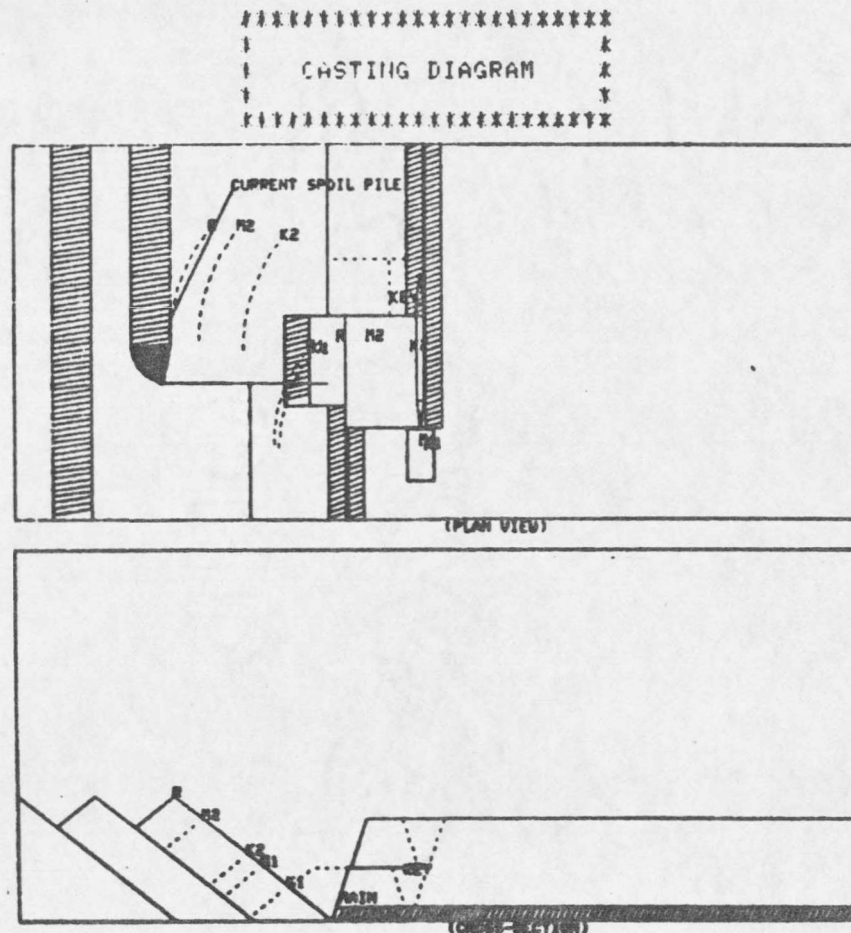


Figure 15. Example II Minimum Cost

Table VIII. Results of Example II for the Minimum Cost Objective

Variable	Description	Value
w	the pit width	133 ft.
d ₁	the first pass depth	84 ft.
b ₁	the dragline boom length on the first pass	275 ft.
b ₂	the dragline boom length on the second pass	319 ft.
B ₁	the dragline bucket size on the first pass	58 cu. yd.
B ₂	the dragline bucket size on the second pass	82 cu. yd.

The optimization generates the plan shown in Figure 16. Notice once again the objective function increased from the initial value in order to gain design feasibility. The results of the optimization are summarized in Table IX.

Table IX. Results for Example III

Variable	Description	Value
w	the pit width	128 ft.
d ₁	the first pass depth	90 ft.
b ₁	the boom length of first pass dragline	293 ft.
b ₂	the boom length of second pass dragline	213 ft.
B ₁	the bucket size of first pass dragline	36
B ₂	the bucket size of second pass dragline	37 yd. ³

***** SITE CHARACTERISTICS *****

LENGTH OF MINE PLAN IS 2000. FT.
 WIDTH OF MINE PLAN IS 2000. FT.
 DEPTH OF OVERBURDEN IS 150. FT.
 COAL SEAM THICKNESS IS 25. FT.
 SPOIL ANGLE OF REPOSE IS 39. DEG.
 HIGHWALL ANGLE IS 72. DEG.
 SOIL SWELL FACTOR IS .25

***** DRAGLINE CHARACTERISTICS *****

1ST BUCKET SIZE IS 36. CU. YDS.
 2ND BUCKET SIZE IS 37. CU. YDS.
 1ST BOOM LENGTH IS 293. FT.
 2ND BOOM LENGTH IS 311. FT.
 BOOM ANGLE IS 35. DEG.
 TUB RADIUS IS 25. FT.
 ANNUAL OPERATING HOURS IS 7000.
 FIXED TIME TO MOVE DRAGLINE IS .5 HRS.
 DRAGLINE WALKING SPEED IS .0013 HR./FT.

***** COMPUTED PIT GEOMETRY *****

1ST WIDTH OF KEY CUT IS 73. FT.
 2ND WIDTH OF KEY CUT IS 55. FT.
 1ST WIDTH OF MAIN CUT IS 54. FT.
 2ND WIDTH OF MAIN CUT IS 72. FT.
 WIDTH OF PREVIOUS PIT IS 127. FT.
 1ST LENGTH OF CUT IS FT.
 2ND LENGTH OF CUT IS 93. FT.
 WIDTH OF SAFETY BENCH IS FT.
 DEPTH OF FIRST PASS IS 90. FT.
 WIDTH OF FILL BENCH IS 99. FT.

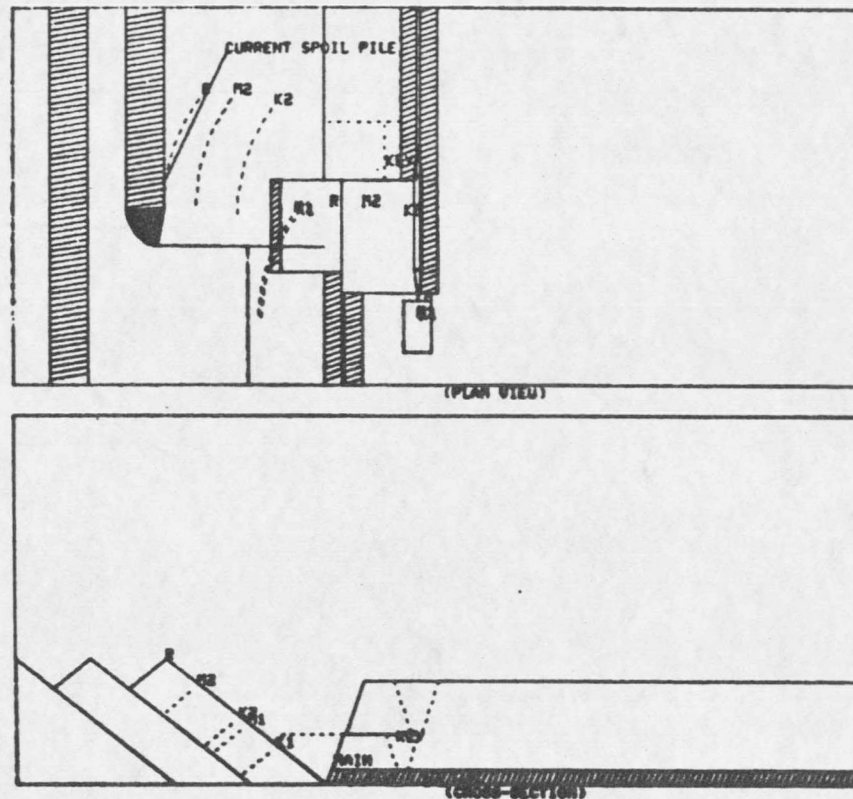
***** DRAGLINE LOCATIONS *****

1ST PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 46. FT.
 FROM OLD HIGHWALL EDGE IS 132. FT.
 1ST PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 41. FT.
 FROM OLD HIGHWALL EDGE IS 127. FT.
 2ND PASS KEY CUT LOCATION IS -
 FROM CUT FACE IS 57. FT.
 FROM OLD HIGHWALL EDGE IS 107. FT.
 2ND PASS MAIN CUT LOCATION IS -
 FROM CUT FACE IS 41. FT.
 FROM OLD HIGHWALL EDGE IS 38. FT.
 2ND PASS REHANDLE LOCATION IS -
 FROM CUT FACE IS 41. FT.
 FROM OLD HIGHWALL EDGE IS -16. FT.

***** DESIGN PRODUCTIVITY *****

TIME TO REMOVE OVERBURDEN IS 13950. HRS.
 COST PER TON OF COAL IS \$.793
 COAL UNCOVERED PER YEAR IS 4000. K TONS

 * CASTING DIAGRAM *
 *



55

Figure 16. Example III Minimum Cost

All example results are summarized in Table X. From these examples it should be apparent that dragline productivity is a function of the design options selected. Simply, the design criteria constrain the optimization to yield less productive plans had they not been employed. The examples also indicate that satisfying the productivity constrain provides some feasibility problems and loss of productivity. Feasibility problems result from the model trying to satisfy the fill bench requirements and productivity constraint, simultaneously. To fully explore the loss of productivity encountered by the productivity constraint, one should contrast the above examples to a tandem system without the constraint and also to a single dragline operation.

Table X. Summary of Examples

Variables	<u>Time</u>			<u>Cost</u>		
	I	II	III	I	II	III
w	153	151	144	132	133	127
d ₁	85	85	91	82	84	90
b ₁	373	370	341	273	275	293
b ₂	374	373	341	318	319	311
B ₁	78	68	35	56	58	36
B ₂	79	76	35	95	82	37
Objective Value	6466	6827	14016	.643	.899	.793

SUMMARY

Conclusions

As stated in the Introduction, the national energy plan calls for a tremendous increase in coal production to alleviate our dependency on foreign oil. It was noted that much of this coal would come from the West, since the western states contain 88 percent of all hypothetical and known coal reserves. In order to extract this vast amount of coal in the most efficient manner, a means for analyzing system productivity must be established, since what might appear on the surface to be an insignificant cost component of the operation might involve millions of dollars because of the large tonnages produced. Thus, any planning methodology that enables an improvement in the efficiency of a mining operation is warranted.

The topic of the SEAM research at MSU is to develop such a planning methodology for Western Surface Mines. Recall Level II of the Production Analysis Module of the mine planning system contains a planning methodology for productivity improvement relative to dragline stripping operations.

As stated in the section entitled Computerized Mine Planning System, the two pass model resides at this level of the planning system, as indicated in Figure 17 by the outlined block.

¹Refer to the section entitled Computerized Mine Planning System for an explanation of Production Analysis Module.

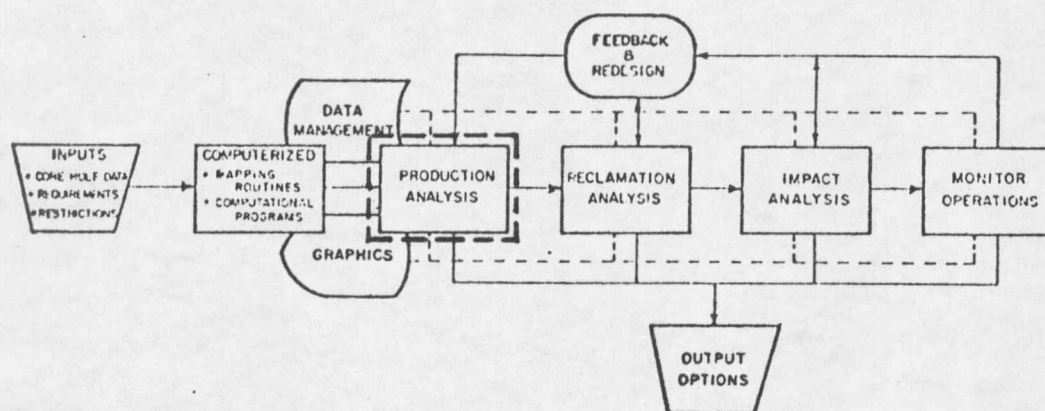


Figure 17. Model Location

When designing dragline stripping operations in deep overburden, the two pass model provides the capability for obtaining the near optimum plan thus yielding the capability of maximizing dragline productivity. The structure of the model allows for interactive computer aided design with results displayed on a CRT via computer drawn casting diagrams. This technology provides the engineer/designer with a new tool for rapidly and efficiently planning pit geometry for two pass strip mine operations.

Although the model responds well under all design conditions, its application should be limited to deep overburden design problems with reasonable stripping ratios. The overburden depth should fall between

90 and 160 feet. Stripping ratios below 6 to 1 are most appropriate the model's application. However, it is advisable to employ the model when designing dragline operations under worse conditions, since the optimization strives for feasibility as well as optimality. Thus, even if no feasible plan exists, the model will yield the most feasible plan.

In order to obtain the best results from the optimization, a good initial plan is required. A rule of thumb to follow for initial inputs is to specify a pit width of approximately 150 feet, a first pass depth of about a third of the total overburden depth, and significantly large values for the boom length and bucket size. It is easier for the optimization to decrease the dragline variables than to increase them in order to gain design optimality and feasibility.

Since the true optimum plan is not guaranteed by the model formulation or direct search technique, a series of better plans can be obtained via successive applications of the model with different initial conditions. Then the best plan can be selected. This type of design is extremely efficient, since the designer can view many more plans relative to a vast number of design alternatives in a reasonably short period of time.

Recommendations

As it stands, the model is a useful tool; it is by no means complete in its own right. Many alterations and enhancements can be made

to improve its operation. For instance, the spoiling sequence should be made optimal. This option would make the model less restrictive in terms of fulfilling fill bench requirements. Another useful modification would be to increase the number of dragline positions per lift. To further increase the versatility of the model, loss of spoil room due to haul roads should be considered. These and many other modifications could easily become an integral part of the mathematical model, because of its structured development. Also, certain physical characteristics that are currently not considered in the mathematical formulation may be recognized by simply adding them to the constraint set.

APPENDIX

APPENDIX I

This appendix contains parameter definitions. Unmarked variables are to be considered as user inputs.

<u>Parameter</u>	<u>Definition</u>
* x_{ij}	distance from the cut face to the center of dragline on the i^{th} pass and j^{th} position.
* y_{ij}	distance from the old highwall to the center of dragline on the i^{th} pass and j^{th} position.
* A_{ij}	spoil are on the i^{th} pass and j^{th} position.
* A_{ijs}	the area removed on the i^{th} pass and j^{th} position.
* A_{rij}	rehandle attributed to the i^{th} pass and j^{th} position.
* A_{rtot}	total rehandle area.
* $\bar{y}_{ij}$	is the centroid of the area removed on the i^{th} pass and at the j^{th} position.
* G_{ij}	distance to the spoil peak from old highwall brow on the i^{th} pass and at the j^{th} position.
* d	total depth of overburden.
** d_i	depth of cut on the i^{th} pass; is a decision variable when $i = 1$.
* t	coal seam thickness
* W'_{ij}	the length of the base of the isosceles triangle formed by the spoil pile configuration on the i^{th} pass and j^{th} position.

**the parameter is a decision variable.

*the value is derived in the model.

$\hat{a}_{ij}$	is the base of the isosceles triangle formed by current spoil resting on previous spoil bank.
w_{ij}	width of the cut on the i^{th} pass and at the j^{th} position.
l_i	length of the cut on the i^{th} pass.
w	width of the pit.
α_{ij}	the average swing angle on the i^{th} pass and at the j^{th} position.
D_{ij}	the dig time on the i^{th} pass and at the j^{th} position (hr).
E_{ij}	walk time between the j^{th} position on the i^{th} pass (hr).
TP_i	time to walk between pits on the i^{th} pass (hr).
$TDIG_i$	the dig time on the i^{th} pass.
$TWALK_i$	the walk time on the i^{th} pass.
T_{ij}	total time to mine the j^{th} position on i^{th} pass (hr).
$safe$	safety bench width (ft.).
b_i	the boom length of the dragline on the i^{th} pass (ft.).
B_i	the bucket size of the dragline on the i^{th} pass (yd.).
F	the bucket fill factor.
BW_i	the bucket width of the dragline on i^{th} pass (ft.).

****the parameter is a decision variable.**

***the value is derived in the model.**

c_i	the cab radius of the dragline on i^{th} pass (ft.).
β	boom angle of both the first and second pass draglines (degrees).
r	vertical distance from ground to bottom of boom (ft.).
v	dragline walking speed (hr./ft.).
CC2	fixed time to move the dragline (hr.)
f	soil swell factor.
Castd	coasting distance used to approximate a straight spoil peak (ft.).
*Reach _i	the dragline reach (ft.).
†Aprod	annual production rate (tons of coal).
*APRDTP	tons of coal within mine site.
Dprice	dragline purchase price.
†Ophrs _i	maximum annual operating hours for i^{th} dragline.
OP _i	hourly operating cost.
OW _i	hourly ownership cost.
θ	spoil angle of repose (degrees).
ϕ	highwall angle (degrees).
ρ	density of coal in tons/ft. ³

**the parameter is a decision variable.

*the value is derived in the model.

†may be derived in the model depending on the design criteria selected.

APPENDIX II

This appendix contains the procedure for determining cut length and dragline positions from the cut face. The model first determines the maximum cut length possible given a dragline reach and casting distance. Then following the computational procedures outlined in the following appendices, the dragline distance from the cut face is computed. If the computed distance is too close, the cut length on that pass is shortened until the dragline is safely located from the cut face.

Figure 18 illustrates the geometry associated with the maximum cut length on both lifts.

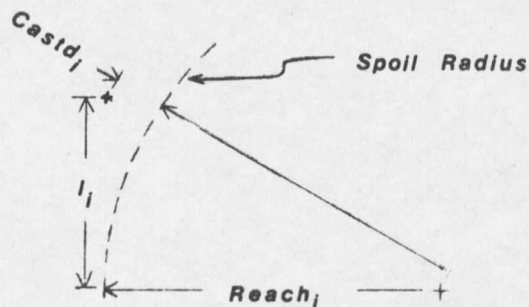


Figure 18. Maximum Cut Length

Referring to the above figure, the following equations can be formulated to determine the maximum cut lengths.

$$l_1 = ((Reach_1 + Castd)^2 - Reach_1^2)^{1/2} \quad [1]$$

$$l_2 = ((\text{Reach}_2 + \text{Castd})^2 - \text{Reach}_2^2)^{1/2} \quad [2]$$

Once the maximum cut length is determined, the procedure for locating the dragline from the cut face can be initiated. The procedure is based on obtaining the maximum cut length such that all dragline positions for that pass are feasible. A feasible location allows the dragline to reach the farthest point in the cut at that position without hanging over the cut face. The following procedure is iterative in the sense that it shortens the cut length on each pass until all dragline positions are feasible.

In order to assure the dragline can reach the farthest point in the cut, the distance y'_{ij} must be determined. This is the maximum perpendicular distance from either end of the cut to the dragline centerline. The distance is diagrammed in Figure 19 for the first pass main cut position. Notice the distance y'_{ij} just happened to be on the dragline's left. It could have been on the right if the dragline was located closer to the old highwall. Similar diagrams can be drawn for the other dragline positions. Mathematically, the distance y'_{ij} can be found as follows for all dragline positions:

$$y'_{11} = \text{Maximum}(y_{11} - (W_{12} + \text{safe} + d_1 \cdot \cot\phi), \\ W + \text{safe} + d_1 \cdot \cot\phi - y_{11}) \quad [3]$$

$$y'_{12} = \text{maximum}(y_{12} - \text{safe}, W_{12} + \text{safe} + d_1 \cdot \cot\phi - y_{12}) \quad [4]$$

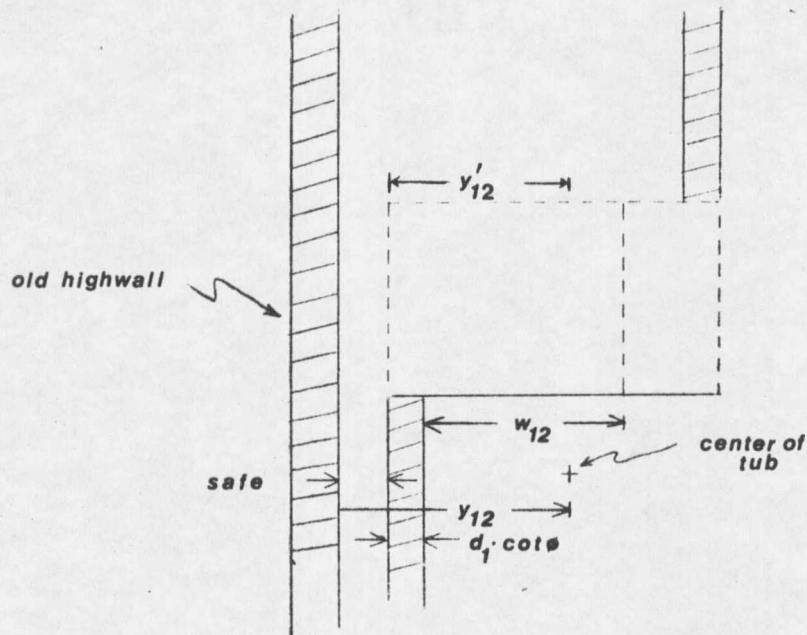


Figure 19. Maximum Perpendicular Distance to Edge of the Cut

$$y'_{21} = \text{maximum}(y_{21} - W_{22}, W - y_{21}) \quad [5]$$

$$y'_{22} = \text{maximum}(y_{22}, W_{22} - y_{22}) \quad [6]$$

$${}^1 y'_{23} = \bar{y}_{23} + y_{23} \quad [7]$$

Once the y'_{ij} are evaluated, the distance from the cut face x_{ij} can be found. This distance is shown in Figure 20 along with its relationship to the pit geometry. Once the maximum cut length is determined, the procedure for locating the dragline from the cut face can be initiated.

${}^1 \bar{y}_{23}$ is the centroid of the handle area computed with reference to the highwall brow. It is calculated in Appendix XI.

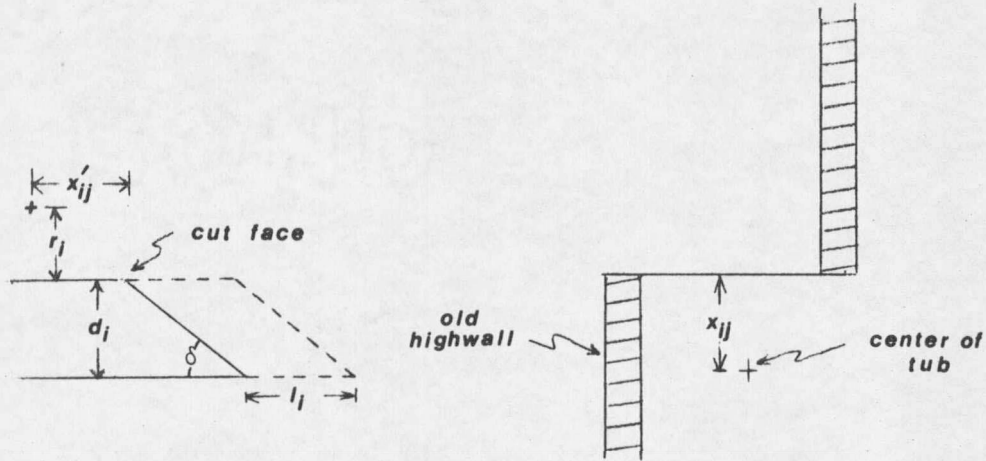


Figure 20. Dragline Distance from the Cut Face

Define:

$$\delta = \tan^{-1} \left(\frac{r_i}{y'_{ij}} \right) \quad [8]$$

The distance R' in the above figure can be expressed in the following two ways:

$$R' = x'_{ij} + (l_i + d_i \cdot \cot \delta) \quad [9]$$

$$R' = ((\text{Reach}_i - c_i)^2 - y_{ij}^2)^{1/2} \quad [10]$$

Since the Reach_i , y_{ij} , c_i , l_i , d_i , and r_i are known, we can solve for x'_{ij} from Equations [9] and [10] as follows:

$$x'_{ij} = R' - (l_i + d_i \cdot \frac{x'_{ij}}{r_i}) \quad [11]$$

Substitute [10] for R' and rewrite [11] to yield

$$x'_{ij} = (\text{Reach}_i - c_i)^2 - y_{ij}^2)^{1/2} - (l_i + d_i \cdot \frac{x'_{ij}}{r_i}) \quad [12]$$

Now solving for x'_{ij} and adding c_i yields:

$$x_{ij} = \frac{(\text{Reach}_i - c_i)^2 + y_{ij}^2)^{1/2} - l_i}{(1 + \frac{d_i}{r_i})} + c_i \quad [13]$$

If any x_{ij} is less than c_i , the cut length must be shortened to keep the dragline from overhanging the cut face. This can easily be done by setting the particular x_{ij} equal to zero and solving Equation [13] for the cut length.

$$l_i = ((\text{Reach}_i - c_i)^2 - y_{ij}^2)^{1/2} \quad [14]$$

Now the other positions on that pass must be reevaluated with the new cut length. Simply, recompute the x_{ij} using expression [13].

APPENDIX III

This appendix describes the computational procedure for determining whether or not a fill bench is necessary. All figures depict spoil at the bottom of coal. Therefore, one should keep in mind the distance cc is greater than zero only when a coal fender has been specified.¹ Figure 21 illustrates the final spoil configuration and yields a physical interpretation of the parameters used. (Notice the first pass overburden is removed.)

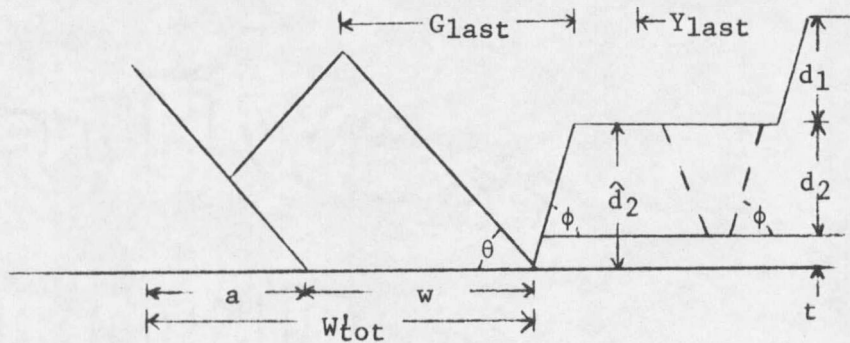


Figure 21. Final Spoil Configuration

Define the following areas:

$$A_1 = w \cdot d_1 \cdot (1 + f) \quad [1]$$

$$A_2 = w \cdot d_2 \cdot (1 + f) \quad [2]$$

$$A_{rtot} = A_1 + A_2 + 1/2 \cdot cc \cdot t \quad [3]$$

¹See page 25 for a description of the coal fender.

Notice A_{tot} is the dark outlined area.

$$\text{recall, } cc = \begin{cases} 0, & \text{if no coal fender} \\ t \cdot (\cot\phi + \cot\theta), & \text{if coal fender} \end{cases} \quad [4]$$

Now compute the total base of the spoil pile triangle as follows:

$$W'_{tot} = (4 \cdot A_{tot} \cdot \cot\theta + a^2)^{1/2} \quad [5]$$

$$\text{Where } \hat{a} = \begin{cases} 0, & \text{if } A_{tot} \leq A_{temp} \\ 2 \cdot A_{tot} \cdot \cot\theta / wc - 1/2wc, & \text{if } A_{tot} > A_{temp} \end{cases} \quad [6]$$

Define the following terms:

$$A_{temp} = 1/4 \cdot wc^2 \cdot \tan\theta \quad [7]$$

$$\text{where } wc = w + cc \quad [8]$$

Now the distance to the spoil peak from the second pass old highwall brow.

$$G_{last} = 1/2 \cdot W'_{tot} + \hat{a}_2 \cdot \cot\theta - cc \quad [9]$$

The distance from the old highwall brow to the dragline center at the last position on the second pass is denoted y_{last} . This distance is shown in Figure 21.

$$y_{last} = (b_2 \cdot \cos\beta) - G_{last} \quad [10]$$

$$\text{if } Y_{\text{last}} \equiv \begin{cases} \geq 0, \text{ no fill is necessary} \\ < 0, \text{ fill bench is necessary} \end{cases} \quad [11]$$

Now compute the amount of rehandle necessary to form the bench. The situation is illustrated in Figure 22.

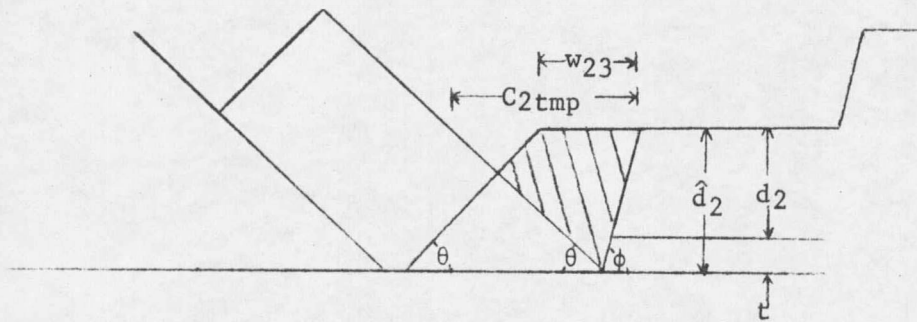


Figure 22. Fill Bench

Define the following terms:

$$d'_2 \equiv \begin{cases} \hat{d}_2, & \text{if no coal fender} \\ d_2, & \text{if coal fender} \end{cases} \quad [12]$$

$$C_{2\text{tmp}} = d'_2 \cdot (\cot\theta + \cot\phi)$$

The width of the fill bench is

$$w_{23} = \begin{cases} y_{\text{last}} , & \text{if } y_{\text{last}} \geq 2 \cdot c_2 \\ 2 \cdot c_2 , & \text{if } y_{\text{last}} < 2 \cdot c_2 \\ C_{2\text{tmp}} , & \text{if either of above} > C_{2\text{tmp}} \end{cases} \quad [13]$$

Therefore, amount of rehandle required to build the bench is;

$$A_{\text{rreq}} = 1/2 \cdot C_{2\text{tmp}} \cdot d_2' - 1/4 \cdot (C_{2\text{tmp}} - w_{23})^2 \cdot \tan \theta \quad [14]$$

APPENDIX IV

This appendix describes the computational procedure for determining the spoil configuration for the first pass key cut material. The following figures reference the spoil situation without a coal fender.¹ Therefore, the distance cc is equal to zero.

The procedure begins by trying to spoil the minimum key cut area, A_{llmin} , without rehandle, as shown in Figure 23 by the shaded region. If this is not possible, rehandle is then accumulated until the minimum key cut area can be spoiled. This rehandle area is then compared to the maximum available rehandle area at the key cut position before the area to be moved is finalized.

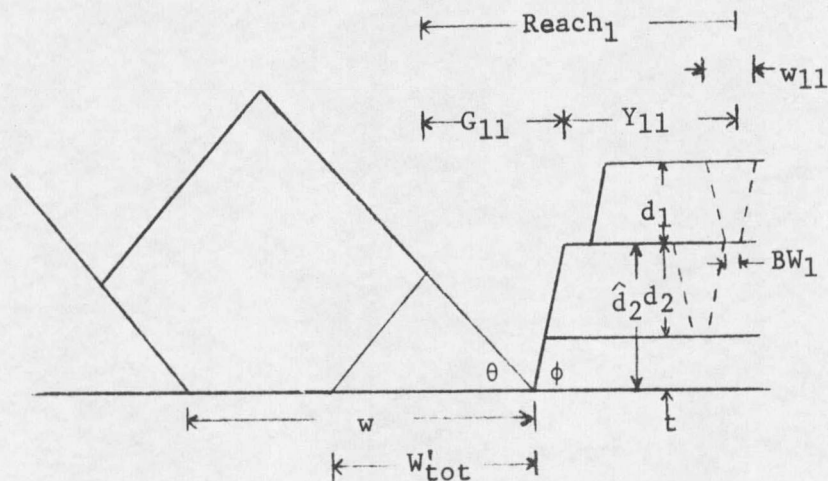


Figure 23. Key Cut Spoil Area Without Rehandle

¹See page 25 for an explanation of a coal fender.

First, locate the dragline and calculate the distance from the old high-wall brown on the second pass to the key cut spoil peak as follows:

$$Y_{11} = w + \text{safe} \quad [1]$$

$$G_{11} = \begin{cases} \text{Reach}_1 - Y_{11}, & \text{if this value is } > 0 \\ 0, & \text{if the above value is } \leq \end{cases} \quad [2]$$

$$\text{where } \text{Reach}_1 = b_1 \cdot \cos + c_1 \quad [3]$$

Now compute the base of the spoil pile as follows:

$$W'_{11t} = 2 \cdot (G_{11} - d_2 \cdot \cot \phi + cc) \quad [4]$$

and let

$$\hat{a}_{11t} = \begin{cases} W'_{11t} - (w + cc), & \text{if difference } > 0 \\ 0, & \text{otherwise} \end{cases} \quad [5]$$

The key cut spoil area without rehandle is denoted A_{11t} . This is the shaded region in Figure 18.

$$A_{11t} = 1/4 \cdot W'_{11t}^2 \cdot \tan \theta - 1/4 \cdot \hat{a}_{11t}^2 \cdot \tan \theta - 1/2 \cdot cc \cdot t \quad [6]$$

This area is tentative since we must be sure it is large enough to satisfy the minimum key cut requirements. If it is greater than the minimum key cut area, we will then remove the minimum key cut area. Calculate the minimum key cut area on the first pass and set it equal

to the cut area as follows:

$$A_{11s} = A_{11min} = (BW_1 + d_1 \cdot \cot \phi) \cdot d_1 \quad [7]$$

Now check A_{11t} against $(A_{11min}) \cdot (1 + f)$: If A_{11t} is greater than $(A_{11min}) \cdot (1 + f)$ then compute the key cut width, w_{11} , as follows:

$$w_{11} = A_{11} / d_1 \cdot (1 + f) + d_1 \cdot \cot \phi \quad [8]$$

Therefore, the first pass main cut width is calculated from

$$w_{12} = w - w_{11} \quad [9]$$

Had A_{11t} been less than A_{11min} , we must compute the spoil configuration since some rehandle will be accumulated. This situation is shown in Figure 24.

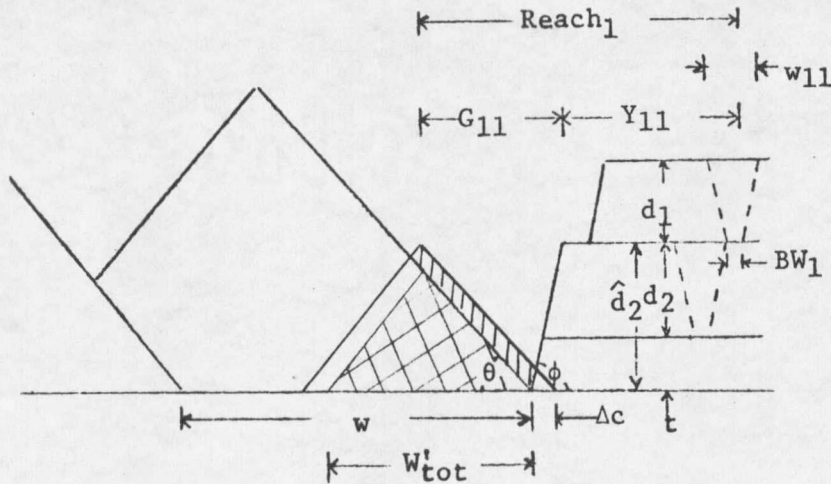


Figure 24. Key Cut Spoil with Rehandle

In reference to the figure, the doubly cross hatched area depicts A_{11t} and the single cross hatch is the rehandle accumulated in order to spoil A_{11s} , or equivalently A_{11min} .

Define:

$$W'_{11} = 2 \cdot (G_{11} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c) \quad [10]$$

Therefore, the key cut spoil area can be calculated as follows:

$$\begin{aligned} A_{11} = A_{11min} \cdot (1 + f) = & 1/4 \cdot W'_{11t}{}^2 \cdot \tan\theta - 1/2(cc + \Delta c) \cdot d_F \\ & - 1/4 \cdot \hat{a}_{11}^2 \cdot \tan\theta \end{aligned} \quad [11]$$

The distance d_F , the vertical distance from the bottom of coal to where the spoil fender intersects the old highwall, is calculated as follows:

$$d_F = \frac{(cc + \Delta c)}{\cot\theta + \cot\phi} \quad [12]$$

To obtain the key cut spoil configuration, Δc must be determined.

First, a substitution is made for W'_{11t} , and rewriting [11] yields

$$\begin{aligned} A_{11min} \cdot (1 + f) = & (G_{11} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c)^2 \tan\theta \\ & - 1/2(cc + \Delta c) \cdot d_F - 1/4 \cdot \hat{a}_{11}^2 \tan\theta \end{aligned} \quad [13]$$

Substituting [12] into [13] yields:

$$\begin{aligned} A_{11min} \cdot (1 + f) = & (G_{11} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c)^2 \tan\theta \\ & - \frac{1/2(cc + \Delta c)^2}{\cot\theta + \cot\phi} - 1/4 \cdot \hat{a}_{11}^2 \tan\theta \end{aligned} \quad [14]$$

When solving for Δc we first set $\hat{a}_{11}=0$ under the assumption the key spoil pile doesn't rest against the previous spoil pile. This assumption will then be checked.

Setting $\hat{a}_{11} = 0$, and letting

$$S_1 = G_{11} - \hat{d}_2 \cdot \cot\phi + cc \quad [15]$$

$$S_2 = 1/2 \cdot \frac{1}{\cot\phi + \cot\theta} \quad [16]$$

we can rewrite [14] as follows:

$$A_{11min} \cdot (1 + f) = (S_1 + \Delta c)^2 \tan\theta - S_2 \cdot (cc + \Delta c)^2 \quad [17]$$

Squaring out the terms in [17] yields:

$$\begin{aligned} A_{11min} (1 + f) &= (S_1^2 + 2 \cdot S_1 \cdot \Delta c + \Delta c^2) \tan\theta \\ &\quad - S_2 \cdot (cc^2 + 2 \cdot cc \cdot \Delta c + \Delta c^2) \end{aligned} \quad [18]$$

Now collect like terms and rewrite [18] as follows:

$$\begin{aligned} A_{11min} \cdot (1 + f) &= \Delta c^2 \cdot (\tan\theta - S_2) + \Delta c(2 \cdot S_1 \cdot \tan\theta \\ &\quad - 2 \cdot S_2 \cdot cc) + S_1^2 \cdot \tan\theta - S_2 \cdot cc^2 \end{aligned} \quad [19]$$

Now use the quadratic formula to solve for Δc , and take the positive root. At this point compute the distance d_F , which is the vertical height at which the key spoil makes contact with the old highwall as follows:

$$d_F = \frac{cc + \Delta c}{\cot\theta + \cot\phi} \quad [20]$$

Also, calculate the exact value of W'_{11} as follows:

$$W'_{11} = 2 \cdot (G_{11} - \hat{d}_2 \cdot \cot \phi + cc + \Delta c) \quad [21]$$

Now the assumption of setting $\hat{a}_{11} = 0$ must be checked as follows:

$$\hat{a}_{11} = \begin{cases} 0, & \text{if } 1/2 \cdot W'_{11} + G_{11} \leq d_2 \cdot \cot \phi + w \\ > 0, & \text{otherwise} \end{cases} \quad [22]$$

If the value of $\hat{a}_{11} > 0$, then a new value for Δc must be found using the following procedure. The situation is illustrated in Figure 25.

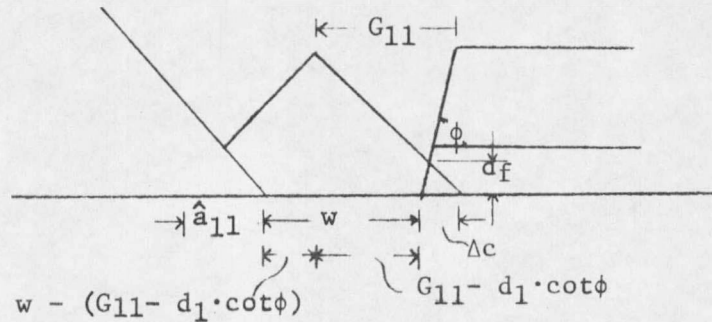


Figure 25. Key Cut Spoil Rests on Previous Spoil Bank

Note from the figure that

$$\hat{a}_{11} = (G_{11} - \hat{d}_2 \cdot \cot \phi + cc + \Delta c) - (w - (G_{11} - \hat{d}_2 \cdot \cot \phi)) \quad [23]$$

Substituting [23] into [14] and rewriting yields

$$\begin{aligned}
A_{llmin} \cdot (1 + f) &= (G_{11} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c)^2 \tan\theta \\
&- 1/2 \cdot \frac{(cc + \Delta c)^2}{\cot\theta + \cot\phi} \\
&- 1/4 \cdot (2 \cdot G_{11} - 2 \cdot \hat{d}_2 \cdot \cot\phi + cc + \Delta c - w)^2 \tan\theta
\end{aligned} \tag{24}$$

Make the following substitutions in [24]:

$$S_1 = G_{11} - \hat{d}_2 \cdot \cot\phi + cc \tag{25}$$

$$S_2 = 2 \cdot G_{11} - 2 \cdot \hat{d}_2 \cdot \cot\phi + cc - w \tag{26}$$

$$S_3 = 2 \cdot (\cot\theta + \cot\phi) \tag{27}$$

Rewriting [24] with the above substitution yields

$$A_{llmin} \cdot (1 + f) = (S_1^2 + \Delta c)^2 \cdot \tan\theta - \frac{(cc + \Delta c)^2}{S_3} - 1/4 (S_2 + \Delta c)^2 \tan\theta \tag{28}$$

Square out the terms in [28] and rewrite as follows:

$$\begin{aligned}
A_{llmin} \cdot (1 + f) &= (S_1^2 + 2 \cdot S_1 \cdot \Delta c + \Delta c^2) \cdot \tan\theta \\
&- \frac{1}{S_3} \cdot (cc^2 + 2 \cdot cc \cdot \Delta c + \Delta c^2) \\
&- 1/4 \cdot (S_2^2 + 2 \cdot S_2 \cdot \Delta c + \Delta c^2) \cdot \tan\theta
\end{aligned} \tag{29}$$

Combining like terms in [29] yields

$$\begin{aligned}
A_{llmin} \cdot (1 + f) &= \Delta c^2 \cdot (\tan\theta - \frac{1}{S_3} - 1/4 \cdot \tan\theta) + \Delta c \cdot (2 \cdot S_1 \cdot \tan\theta \\
&- \frac{2 \cdot cc}{S_3} - 1/2 \cdot S_2 \cdot \tan\theta) + S_1^2 \cdot \tan\theta - \frac{cc^2}{S_3} - 1/4 \cdot S_2^2 \cdot \tan\theta
\end{aligned} \tag{30}$$

Now use the quadratic formula to solve for Δc , and select the positive root.

Recompute the values for $\hat{a}_{11}$ and W'_{11} as follows:

$$a_{11} = 2 \cdot G_{11} - 2 \cdot \hat{d}_2 \cot \phi + cc + \Delta c - w \quad [31]$$

$$W'_{11} = 2 \cdot (G_{11} - \hat{d}_2 \cdot \cot \phi + cc + \Delta c) \quad [32]$$

The distance d_F is revised also as follows:

$$d_F = \frac{(cc + \Delta c)}{\cot \phi + \cot \theta} \quad [33]$$

Next, for this key cut area to be actually removed, it must be compared to the maximum available area to be spoiled in. This maximum area is shown in Figure 26 by the cross hatched region.

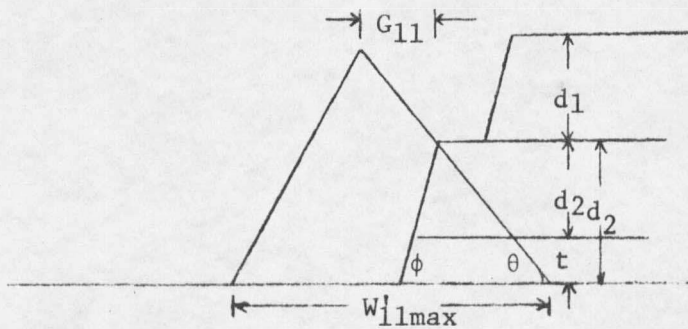


Figure 26. Maximum Key Cut Spoil Area

First, derive the base of the maximum key cut spoil as follows:

$$W'_{11max} = 2 \cdot (G_{11} + \hat{d}_2 \cdot \cot \theta) \quad [34]$$

Note if the maximum key cut spoil rests against the previous spoil pile

then $\hat{a}_{11\max} > 0$

$$\hat{a}_{11\max} = \begin{cases} 0, & \text{if } 1/2 \cdot W'_{11\max} + G_{11} \leq w + \hat{d}_2 \cdot \cot\phi \\ 1/2 \cdot W'_{11\max} - ((\hat{d}_2 \cdot \cot\phi - G_{11}) + w), & \text{otherwise} \end{cases} \quad [35]$$

Now we can calculate the maximum key spoil area as follows:

$$\begin{aligned} A_{11\max} &= 1/4 \cdot W'^2_{11\max} \cdot \tan\theta - 1/2 \cdot d_2 \cdot (\cot\phi + \cot\theta) \cdot d_2 \\ &\quad - 1/4 \cdot \hat{a}_{11\max} \cdot \tan\theta \end{aligned} \quad [36]$$

Compare this maximum available area to the minimum key cut area;

select the area that is the smallest to determine the key cut area to be removed. Note the area selected is swollen.

$$A_{11} = \text{minimum}(A_{11\min} \cdot (1 + f), A_{11\max}) \quad [37]$$

If $A_{11\max}$ was selected recalculate the following parameters:

$$W'_{11} = W'_{11\max} \quad [38]$$

$$\Delta c = \hat{d}_2 \cdot (\cot\theta + \cot\phi) - cc \quad [39]$$

$$\hat{a}_{11} = \hat{a}_{11\max} \quad [40]$$

$$d_F = \frac{(cc + \Delta c)}{(\cot\phi + \cot\theta)} \quad [41]$$

Next the main cut and key cut widths can be calculated for the first pass. These are shown in Figure 27.

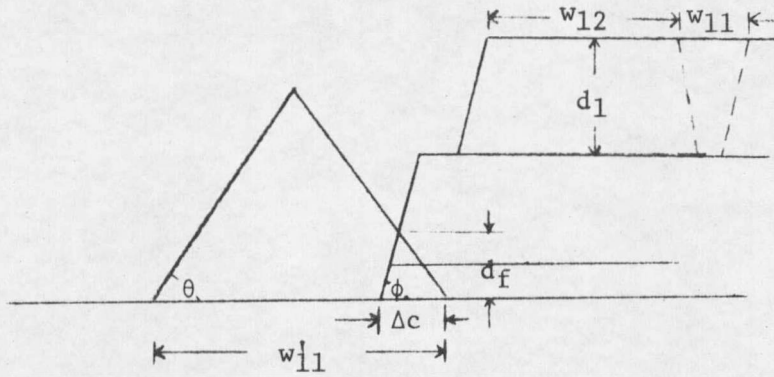


Figure 27. Main Cut and Key Cut Widths

The key cut width can be calculated as follows:

$$w_{11} = \begin{cases} A_{11}/(d_2 \cdot (1 + f)) + d_1 \cdot \cot \phi, & \text{if } A_{11s} = A_{11min} \\ \frac{(4 \cdot A_{11} \cdot \cot \phi)^{1/2}}{(1 + f)}, & \text{if } A_{11s} < A_{11min} \end{cases} \quad [42]$$

[43]

Therefore, the first pass main cut width is

$$w_{12} = w - w_{11} \quad [44]$$

Finally, the rehandle on the first pass key cut can be evaluated.

This is the shaded region in Figure 27. It can be calculated as follows:

$$A_{r11} = 1/4 \cdot w'_{11} \cdot \tan \theta - 1/4 \cdot (w'_{11} - \Delta c)^2 \cdot \tan \theta - 1/2 \cdot (cc + \Delta c) \cdot d_f + 1/2 \cdot cc \cdot t \quad [45]$$

APPENDIX V

This appendix contains the procedure for determining the spoil configuration at the first pass main cut position. Figure 28 depicts the typical situation for spoiling the main cut material over the previously spoiled key cut overburden. Different spoil configurations are possible depending on the amount of rehandle accumulated at the main cut position. These situations will be dealt with in the following procedure. Again, the top for coal spoiling option is not shown in the figures.¹ However, it is considered within the mathematical development.

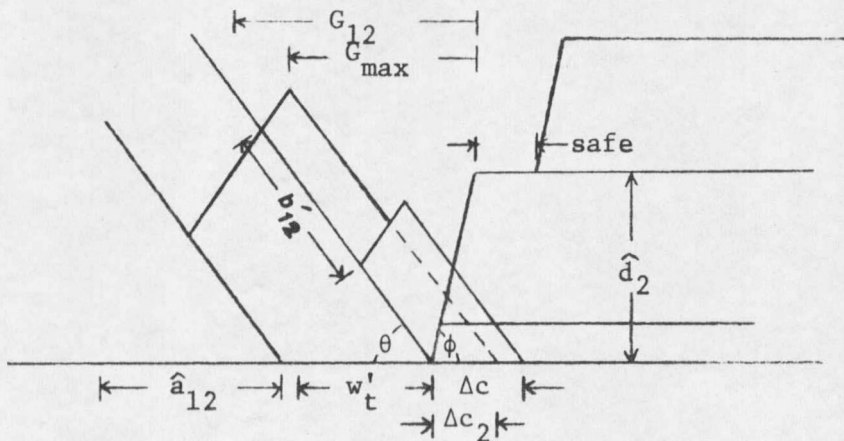


Figure 28. Typical Main Cut Spoil Situation

The area to be spoiled at the first pass main cut is

$$A_{12s} = \frac{A_1}{(1 + f)} - A_{11s} \quad [1]$$

¹See page 25 for a description of the coal fender.

Excluding first pass key cut rehandle material, the total spoil area on the first pass without any main cut rehandle becomes

$$A_{12} = A_1 - A_{r11} + 1/2 \cdot cc \cdot t \quad [2]$$

Notice allowance has been made for the top of the coal spoil option.

Next determine the distance from the old highwall to the spoil peak for this spoil area. First, calculate $\hat{a}_{12}$ as follows:

$$\hat{a}_{12} = \begin{cases} 0, & \text{if } A_{temp} > A_{12} \\ 2 \cdot A_{12} \cdot \cot\theta / wc - 1/2 \cdot wc, & \text{if } A_{temp} < A_{12} \end{cases} \quad [3]$$

$$\text{Recall } wc = w + cc \quad [4]$$

$$\text{and } A_{temp} = 1/4 \cdot wc^2 \cdot \tan\theta \quad [5]$$

The base of the first pass spoil is

$$W'_{12} = (4 \cdot A_{12} \cdot \cot\theta + \hat{a}_{12}^2)^{1/2} \quad [6]$$

Thus, the distance to the spoil peak can be found as follows:

$$G_{12} = 1/2 \cdot W'_{12} + \hat{d}_2 \cdot \cot\phi - cc \quad [7]$$

Now check if there exists rehandle requirements at the first pass main cut position. These requirements could come from one or both of the following situations:

- i. remaining fill bench requirements
- ii. the dragline reach not being sufficient to reach the spoil peak at G_{12}

Both requirements if they exist are analyzed in the following procedure. If both requirements need to be fulfilled simultaneously, the largest one prevails; thus, automatically satisfying the other. To satisfy the first of these requirements, the necessary rehandle area would be

$$A_{r12} = \begin{cases} A_{rreq} - A_{r11}, & \text{if the difference} > 0 \\ 0, & \text{otherwise} \end{cases} \quad [8]$$

Now find the maximum amount of rehandle possible at the first pass main cut position. This area is shown in Figure 29.

First, find a_{12max} if the spoil pile rests against the previous spoil bank as follows:

$$\text{Let } wc_{tmp} = w + \hat{d}_2 \cdot (\cot\theta + \cot\phi) \quad [9]$$

$$\text{and } A_{12max} = A_1 + 1/2 \cdot \hat{d}_2^2 \cdot (\cot\theta + \cot\phi) \quad [10]$$

Therefore,

$$\hat{a}_{12max} = \begin{cases} 0, & \text{if } A_{tmp} > A_{12max} \\ 2 \cdot A_{12max} \cdot \cot\theta / wc_{tmp} \cdot \tan\theta - 1/2 \cdot wc, & \text{if } A_{tmp} < A_{12max} \end{cases} \quad [11]$$

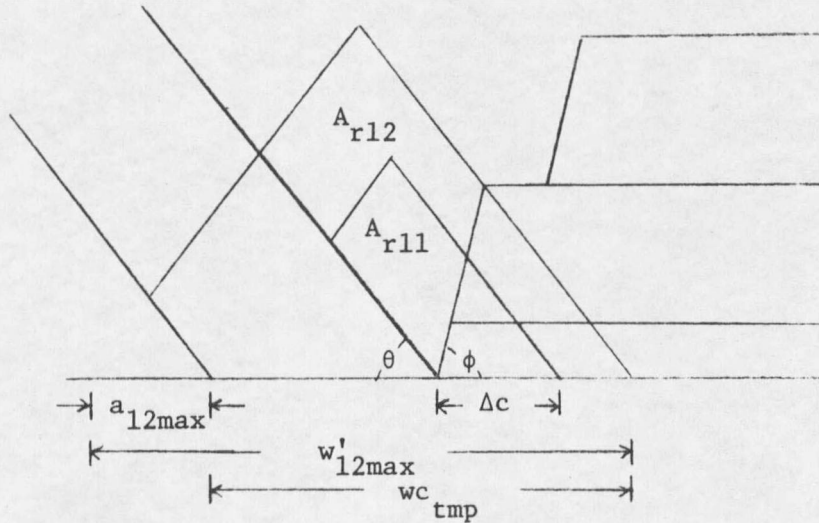


Figure 29. Maximum Rehandle Situation for First Pass Overburden

where

$$A_{tmp} = 1/4 \cdot wc_{tmp}^2 \cdot \tan \theta \quad [12]$$

Now the base of the spoil pile can be calculated as follows:

$$w'_{12max} = (4 \cdot A_{tmp} \cdot \cot \theta - \hat{a}_{12max}^2)^{1/2} \quad [13]$$

Calculate the maximum amount of rehandle that is possible to spoil from the first pass main cut position.

$$A_{r12max} = 1/4 \cdot w'_{12}^2 \cdot \tan \theta \quad 1/4 (wc + \hat{a}_{12max})^2 \cdot \tan \theta \quad [14]$$

$$- 1/2 \cdot \hat{d}_2^2 \cdot (\cot \theta + \cot \phi) + 1/2 \cdot cc \cdot t - A_{r11}$$

To enable the model to respond correctly when the first pass overburden is shallow, the following procedure is used to compute A_{r12max} . Specifically, when d_1 is shallow the first pass overburden swollen, namely A_1 , may be less than $1/2 \cdot \hat{d}_2 \cdot \cot\theta - 1/2 \cdot \hat{d}_2^2 \cdot \cot\phi$. This situation is shown in Figure 30. The following procedure will give an approximation to the maximum rehandle possible from the first pass main cut position.

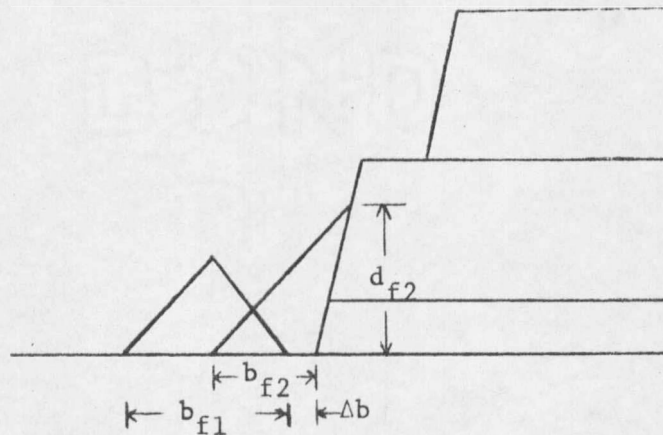


Figure 30. Possible Rehandle Configuration for Shallow First Pass

$$\text{Define } \sigma = \begin{cases} 1, & \text{on first computation} \\ 0, & \text{if } b_{F2} > b_{F1} + \Delta b \end{cases} \quad [15]$$

Compute the height and base of the spoil pile after main cut spoil is deposited as follows:

$$d_{f2} = \left[\frac{2 \cdot (A_1 - \sigma \cdot A_{11})}{\cot\theta - \cot\phi} \right]^{1/2} \quad [16]$$

$$b_{F2} = d_{F2} \cdot (\cot\theta - \cot\phi) + cc \quad [17]$$

Compute the base of the key cut spoil as follows:

$$b_{F1} = [2 \cdot (A_{11} + 1/2 \cdot cc \cdot t) \cdot \cot\theta]^{1/2} \quad [18]$$

$$\Delta b = [G_{11} - \hat{d}_2 \cdot \cot\phi] - 1/2 \cdot b_{F1} \quad [19]$$

Now if b_{F2} is greater than $b_{F1} + \Delta b$ reset σ accordingly and recompute the above procedure. Then approximate the maximum rehandle as follows:

$$A_{r12max} = A_1 - \sigma \cdot A_{11} - 1/4 \cdot b_{F2}^2 \cdot \tan\theta - A_{r11} + \sigma \cdot 1/4 \cdot (b_{F2} - \Delta b)^2 \tan\theta \quad [20]$$

The distance to the main cut spoil peak can be arbitrarily set as follows:

$$G_{12} = \hat{d}_2 \cdot \cot\phi \quad [21]$$

Note if the overburden is shallow and $A_{rreq} > 0$ the remaining procedure can be by-passed. Simply set

$$A_{r12} = A_{r12max} \quad [22]$$

The following procedure calculates the amount of rehandle necessary to satisfy the second condition stated above. (Note if G_{12} is less than $G_{\max}$, this computation can be omitted.) Define the maximum distance from the old highwall brow to the spoil peak to be

$$G_{\max} = \text{Reach}_1 - Y_{\min} \quad [23]$$

$$\text{where } Y_{\min} = \text{Safe} + d_1 \cdot \cot \phi + c_1 \quad [24]$$

Now reset the distance from the highwall brow to the spoil peak as follows:

$$G_{12} = G_{\max} \quad [25]$$

The following computation enables the dragline to create the spoil peak at $G_{\max}$; thus eliminating unnecessary rehandle.

The sigma functions, σ_1 and σ_2 , are employed to handle the case where the main cut rehandle would overhang the key cut spoil. (Naturally this situation can only arise in the mathematical development.) The geometry of the situation is illustrated in Figure 31.

The sigma function, σ_3 , considers whether or not the spoil pile after main cut overburden is removed, rests on the previous spoil bank. Specifically, it monitors whether $\hat{a}_{12t}$ is greater than zero. The sigma functions are explicitly defined as follows:

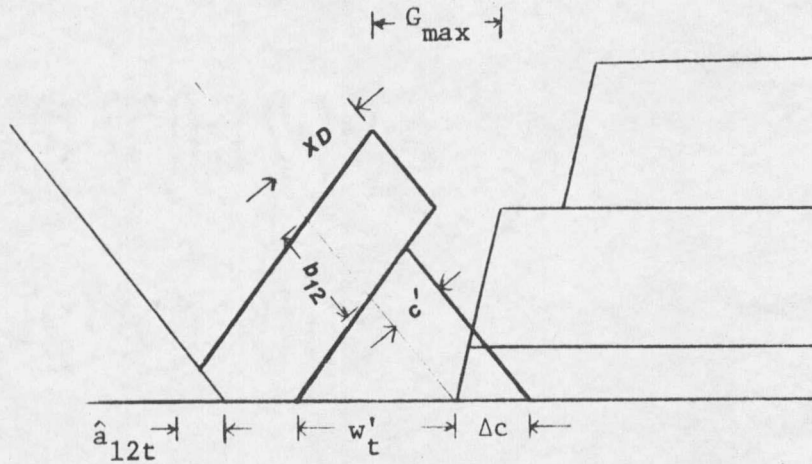


Figure 31. Main Cut Rehandle Overhangs Key Cut Spoil

$$\sigma_1 = \begin{cases} 0, & \text{on the first computation of } \Delta c_2 \\ 1, & \text{if } XD > c' \end{cases} \quad [26]$$

$$\sigma_2 = \begin{cases} 0, & \text{if } XD > c' \\ 1, & \text{on the first computation of } \Delta c_2 \end{cases} \quad [27]$$

$$\sigma_3 = \begin{cases} 0, & \text{assume } \hat{a}_{12} = 0 \text{ on first computation of } \Delta c_2 \\ 1, & \text{if } \hat{a}_{12t} > 0 \end{cases} \quad [28]$$

Define the following terms:

$$W'_{12t} = 2 \cdot (G_{\max} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2) \quad [29]$$

$$\hat{a}_{12t} = W'_{12t} - (w + cc + \Delta c_2) \quad [30]$$

$$= 2 \cdot G_{\max} - 2 \cdot \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2 - w \quad [31]$$

$$W'_t = W'_{11} - \Delta c \quad [32]$$

The mathematical equation expressing the swollen main cut area as a function of these terms can be written as follows:

$$\begin{aligned} A_{12s}(1+f) &= (G_{\max} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2)^2 \cdot \tan\theta \\ &\quad - 1/4 \cdot (2 \cdot G_{\max} - 2 \cdot \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2 - w)^2 \\ &\quad \cdot \tan\theta - 1/4 (W'_t + \Delta c_2)^2 \cdot \tan\theta \\ &\quad + 1/4 \hat{a}_{11}^2 \tan\theta \end{aligned} \quad [33]$$

Equation [33] can now be modified to handle the various spoil configurations by using the sigma functions and adding the term A_{r11} .

$$\begin{aligned} \sigma_1 \cdot A_{r11} + A_{12s} \cdot (1+f) &= (G_{\max} - \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2)^2 \cdot \tan\theta \\ &\quad - 1/4 \cdot \sigma_3 \cdot (2 \cdot G_{\max} - 2 \cdot \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2 - 2)^2 \cdot \tan\theta \\ &\quad - 1/4 \cdot (W'_t + \sigma_2 \cdot \Delta c_2)^2 \cdot \tan\theta + 1/4 \cdot \hat{a}_{11}^2 \cdot \tan\theta \end{aligned} \quad [34]$$

Let

$$S_1 = G_{\max} - \hat{d}_2 \cdot \cot\phi + cc \quad [35]$$

$$S_2 = 2 \cdot G_{\max} - 2 \cdot \hat{d}_2 \cdot \cot\phi + cc - W \quad [36]$$

Now substitute [35] and [36] into [34] and rewrite as follows:

$$\begin{aligned} \sigma_1 \cdot A_{r11} + A_{12s} \cdot (1+f) = & (S_1 + c_2)^2 \cdot \tan\theta - 1/4 \cdot \sigma_3 \cdot (S_2 + \Delta c_2)^2 \cdot \tan\theta \\ & - 1/4 \cdot (W'_t + \sigma_2 \cdot \Delta c_2)^2 \cdot \tan\theta + 1/4 \cdot \hat{a}_{11}^2 \cdot \tan\theta \end{aligned} \quad [37]$$

Squaring the terms yields:

$$\begin{aligned} \sigma_1 \cdot A_{11} + A_{12s} \cdot (1+f) = & S_1^2 \cdot \tan\theta + 2S_1 \cdot \tan\theta \cdot \Delta c_2 + \Delta c_2^2 \cdot \tan\theta \\ & - 1/4 \cdot \sigma_3 \cdot S_2^2 \cdot \tan\theta - 1/4 \cdot \sigma_3 \cdot S_2 \cdot \tan\theta \cdot \Delta c_2 - \Delta c_2^2 \cdot 1/4 \cdot \sigma_3 \cdot \tan\theta \\ & - 1/4 \cdot \tan\theta \cdot W_t'^2 - \sigma_2 \cdot 1/2 \cdot \tan\theta \cdot W_t' \cdot \Delta c_2 \\ & - \sigma_2 \cdot 1/4 \cdot \tan\theta \Delta c_2^2 + 1/4 \cdot \hat{a}_{11}^2 \cdot \tan\theta \end{aligned} \quad [38]$$

Combining like terms in [38] and rewriting yields:

$$\begin{aligned} & (\tan\theta - 1/4 \cdot \sigma_3 \cdot \tan\theta - \sigma_2 \cdot 1/4 \cdot \tan\theta) \cdot \Delta c_2^2 + (2S_1 \cdot \tan\theta \\ & - 1/4 \cdot \sigma_3 \cdot S_2 \cdot \tan\theta - \sigma_2 \cdot 1/4 \cdot W_t' \cdot \tan\theta) \Delta c_2 + S_1^2 \cdot \tan\theta \\ & - 1/4 \cdot \sigma_3 \cdot S_2^2 \cdot \tan\theta - 1/4 \cdot W_t' \cdot \tan\theta + 1/4 \cdot \hat{a}_{11}^2 \cdot \tan\theta \\ & - \sigma_1 \cdot A_{r11} - A_{12s} \cdot (1 + f) = 0 \end{aligned} \quad [39]$$

Now use the quadratic formula to solve for Δc_2 . Take the positive root.

Now compute $\hat{a}_{12}$ as follows:

$$W'_{12} = 2 \cdot (G_{\max} - \hat{d}_2 \cdot \cot \phi + cc + \Delta c_2) \quad [40]$$

$$\hat{a}_{12} = W'_{12} - (w + cc + \Delta c_2) \quad [41]$$

If $\hat{a}_{12}$ is greater than zero, reset the sigma function σ_3 accordingly and reevaluate Δc_2 .

Define:

$$XD = \Delta c_2 \cdot \sin \theta \quad [42]$$

$$c' = \Delta c \cdot \sin \theta \quad [43]$$

If $XD > c'$, recompute Δc_2 setting the sigma functions accordingly. The rehandle area to satisfy condition ii. as given previously can be found via the following procedure.

Define:

$$\begin{aligned} b'_{12} = & (G_{\max} - \hat{d}_2 \cdot \cot \phi + cc + \Delta c_2) / \cos \theta \\ & - 1/2(W'_t + \Delta c_2) / \cos \theta \end{aligned} \quad [44]$$

Therefore, the rehandle area is

$$A_{r12t} = XD \cdot b'_{12} \quad \text{if } XD \leq c' \quad [45]$$

If $XD > c'$ the rehandle area is calculated as follows:

$$\begin{aligned}
 A_{rl2t} = & (G_{\max} - \hat{d}_2 \cot\phi + cc + \Delta c_2)^2 \tan\theta \\
 & - (G_{\max} + 1/2 \cdot \Delta c_2 - \hat{d}_2 \cdot \cot\phi + cc)^2 \tan\theta \\
 & - \frac{(cc + \Delta c_2)^2}{2 \cdot (\cot\theta + \cot\phi)} + 1/2 \cdot cc \cdot t - A_{rl1} \quad [46]
 \end{aligned}$$

Now select the appropriate rehandle area as follows:

$$A_{rl2} = \text{MAX}(\text{MIN}(A_{rl2\max}, A_{rl2}), \text{MIN}(A_{rl2\max}, A_{rl2t})) \quad [47]$$

If the selected rehandle area A_{rl2} is due to fill bench requirements or is the maximum rehandle area, a new value for G_{12} is computed via the following procedure. This will allow the dragline to be properly located from the second pass highwall brow. The geometry for this situation is shown in Figure 32. Notice the distance XD may once again be greater than c' . Thus, the procedure is capable of handling this situation.

If A_{rl2} is due to fill bench requirements, the correction for G_{12} can be made with the following procedure. Define:

$$A_{\text{tot}} = A_{12s} \cdot ((1+f) - A_{rl2} + A_{11s} \cdot (1+f) - A_{rl1} + 1/2 \cdot cc \cdot t) \quad [48]$$

(Note A_{tot} is the heavily shaded area in Figure 27.)

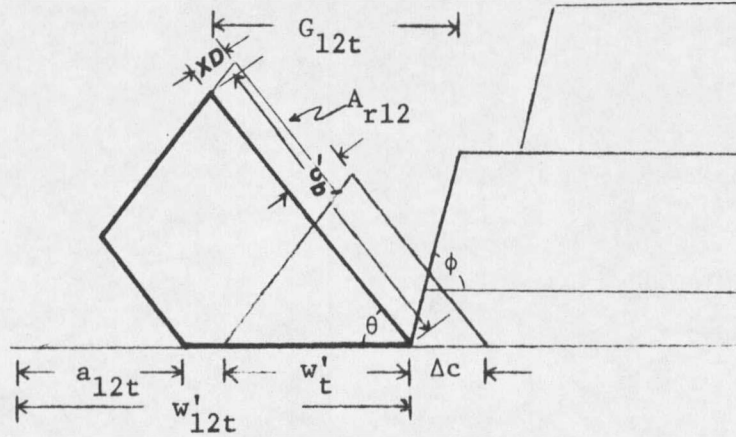


Figure 32. Spoil Configuration for Main Cut Rehandle Area

$$a_{12t} = \begin{cases} 2 \cdot A_{tot} \cdot \cot \theta / wc - 1/2 \cdot wc, & \text{if } A_{tmp} \leq A_{tot} \\ 0, & \text{otherwise} \end{cases} \quad [49]$$

$$\text{recall } wc = w + cc \quad [50]$$

$$\text{and } A_{tmp} = 1/4 \cdot wc^2 \cdot \tan \theta \quad [51]$$

$$w'_{12t} = (4 \cdot A_{tot} \cdot \cot \theta + \hat{a}_{12t}^2)^{1/2} \quad [52]$$

$$G_{12t} = 1/2 \cdot w'_{12t} + \hat{d}_2 \cdot \cot \phi - cc \quad [53]$$

Next evaluate b'_{12} as follows:

$$b'_{12} = w'_{12t} / \cos \theta - w'_t / \cos \theta \quad [54]$$

Therefore, the distance XD is

$$XD = A_{r12}/b'_{12} \quad [55]$$

Now if $XD > c'$ recompute XD by the following procedure. The following equation is an expression for the total rehandle accumulated at the key cut and main cut positions.

$$\begin{aligned} A_{r11} + A_{r12} = & (G_{12t} - 1/2 \cdot \Delta c_2 - \hat{d}_2 \cdot \cot\phi + cc + \Delta c_2)^2 \tan\theta \\ & - (G_{12t} - \hat{d}_2 \cdot \cot\phi + cc)^2 \cdot \tan\theta \\ & - \frac{1/2(cc + \Delta c_2)^2}{(\cot\phi + \cot\theta)} + 1/2 \cdot cc \cdot t \end{aligned} \quad [56]$$

$$\text{Let } S_1 = G_{12t} - \hat{d}_2 \cdot \cot\phi + cc \quad [57]$$

$$S_2 = \frac{1}{2 \cdot (\cot\theta + \cot\phi)} \quad [58]$$

Substitute [57] and [58] into [56] and rewrite as follows:

$$\begin{aligned} A_{r11} + A_{r12} = & (S_1 + 1/2 \cdot \Delta c_2)^2 \cdot \tan\theta - S_1^2 \cdot \tan\theta \\ & - S_2 \cdot (cc + \Delta c_2)^2 + 1/2 \cdot cc \cdot t \end{aligned} \quad [59]$$

Squaring out the terms in [59] yields:

$$\begin{aligned} A_{r11} + A_{r12} = & S_1^2 \cdot \tan\theta + S_1 \cdot \Delta c_2 \cdot \tan\theta + 1/4 \cdot \Delta c_2^2 \cdot \tan\theta - S_1^2 \cdot \tan\theta \\ & - S_2 \cdot cc^2 - S_2 \cdot 2 \cdot cc \cdot \Delta c_2 - S_2 \cdot \Delta c_2^2 + 1/2 \cdot cc \cdot t \end{aligned} \quad [60]$$

Combine like terms in [60] and rewrite as follows:

$$A_{r11} + A_{r12} = (1/4 \cdot \tan\theta - S_2) \cdot \Delta c_2^2 + (S_1 \cdot \tan\theta - 2 \cdot S_2 \cdot cc) \cdot \Delta c_2 - S_2 \cdot cc^2 + 1/2 \cdot cc \cdot t \quad [61]$$

Now use the quadratic formula to solve for Δc_2 . Take the positive root. The new value for the distance XD may be obtained as follows:

$$XD = \Delta c_2 \cdot \sin\theta \quad [62]$$

The following expression can be used to obtain the distance from the second pass highwall brow to the spoil peak. Figure 33 depicts the spoil peak and the angles used in calculating G_{12} .

$$G_{12} = G_{12t} - XD / \cos(|(90 - 2\theta)|) \cdot \cos\theta \quad [63]$$

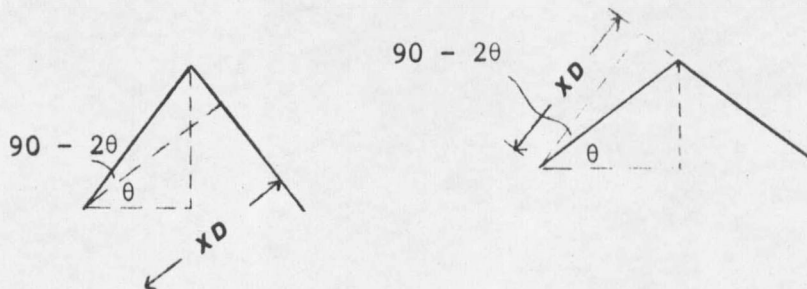


Figure 33. Spoil Peak Configurations

Had the rehandle area A_{r12} equaled the maximum, G_{12} would be

$$G_{12} = 1/2 \cdot W'_{12\max} - \hat{d}_2 \cdot (\cot\phi + \cot\theta) + \hat{d}_2 \cdot \cot\phi \quad [64]$$

Therefore, the dragline distance from the second pass highwall brow is:

$$Y_{12} = \text{Reach}_1 - G_{12} \quad [65]$$

APPENDIX VI

The following procedure determines the configuration of the second pass key cut spoil, and the dragline location from the old highwall. First, calculate the first pass spoil area less the first pass rehandle plus the area of the coal fender if the top of coal spoil option is selected.

$$A_{1LR} = A_1 - A_{r11} - A_{r12} + 1/2 \cdot cc \cdot t \quad [1]$$

$$\text{recall } A_1 = w \cdot d_1 \cdot (1 + f) \quad [2]$$

This area is depicted in Figure 34 by the shaded region.

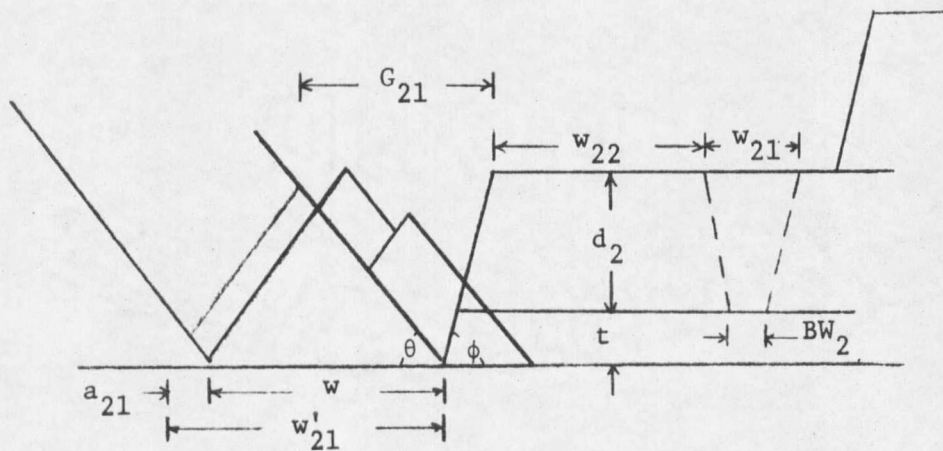


Figure 34. First Pass Area Less Rehandle

Now compute the total spoil area after the second pass key cut is spoiled from the specified dragline position.

Figure 35 illustrates the key cut spoil configuration.

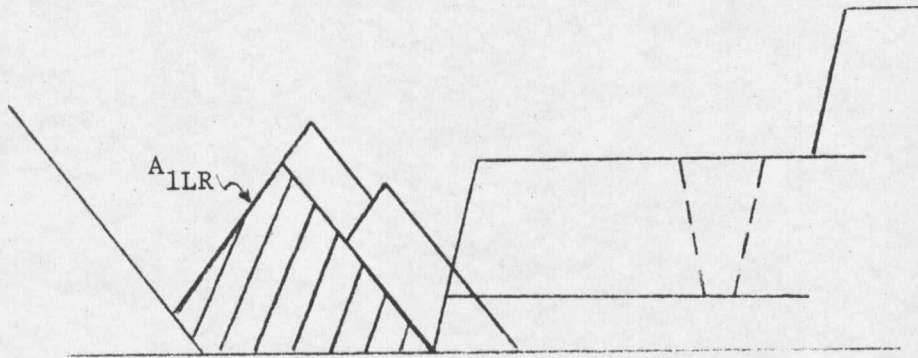


Figure 35. Second Pass Key Cut Spoil

Define the following terms:

$$Y_{21} = w - d_2 \cdot \cot \phi \quad [3]$$

$$G_{21} = \text{Reach}_2 - Y_{21} \quad [4]$$

$$W'_{21} = 2 \cdot (G_{21} - \hat{d}_2 \cdot \cot \phi + cc) \quad [5]$$

$$\hat{a}_{21} = \begin{cases} W'_{21} - wc, & \text{if difference} > 0 \\ 0, & \text{otherwise} \end{cases} \quad [6]$$

$$\text{recall } wc = w + cc \quad [7]$$

Therefore the total key cut spoil area without rehandle can be written as:

$$A_{21} = 1/4 \cdot W_{21}'^2 \cdot \tan \theta - 1/4 \cdot \hat{a}_{21}^2 \cdot \tan \theta \quad [8]$$

The area available for second pass key cut material is

$$A_{21t} = A_{21} - A_{1LR} \quad [9]$$

If this area is less than zero; consider it zero, and note the second pass key cut material will be all rehandle.

Since we only want to remove the minimum key cut area based on a bucket width at the bottom of the cut, the key cut area becomes

$$A_{21s} = A_{21min} = (d_2 \cdot \cot \phi + BW_2) \cdot d_2 \quad [10]$$

If A_{21t} is greater than zero, any rehandle accumulated can be calculated as follows:

$$A_{r21} = A_{21s} \cdot (1 + f) - A_{21t} \quad [11]$$

If this value should be less than zero, consider it zero. Now the total rehandle accumulated on both the first and second pass is

$$A_{rtot} = A_{r11} + A_{r12} + A_{r21} \quad [12]$$

and the width of the second pass key cut can be determined as follows:

$$w_{21} = A_{21s} / d_2 + d_2 \cdot \cot \phi \quad [13]$$

The main cut width is simply

$$w_{22} = w - w_{21}$$

[14]

APPENDIX VII

This appendix contains the computational procedure for determining the second pass main cut spoil configuration. Figure 36 illustrates the situation.

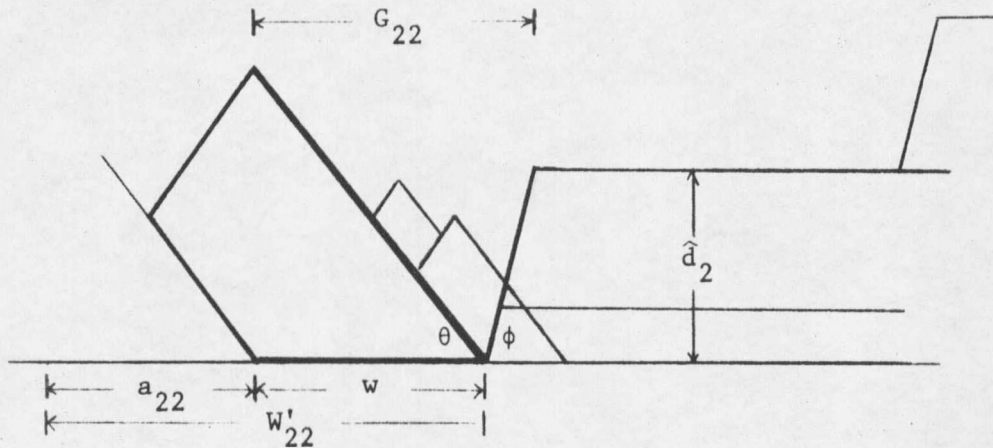


Figure 36. Main Cut Spoil Configuration

Note the amount of overburden in terms of area spoiled from the main cut position is

$$A_{22s} = \frac{A_2}{(1 + f)} - A_{21s} \quad [1]$$

Now compute the total spoil area including the second pass main cut overburden.

$$^1A_{\text{tot}} = A_{1LR} + (A_{21s} + A_{22s}) \cdot (1 + f) - A_{r21} \quad [2]$$

¹The area A_{1LR} is defined in Appendix VI.

This area is depicted in the above figure by the heavy dark line.

Now calculate the base of the spoil pile W'_{22} .

$$\hat{a}_{22} = \begin{cases} 2 \cdot A_{\text{tot}} \cdot \cot \theta / wc - 1/2 \cdot wc, & \text{if } A_{\text{tmp}} < A_{\text{tot}} \\ 0, & \text{otherwise} \end{cases} \quad [3]$$

$$\text{recall } wc = w + cc \quad [4]$$

$$\text{and } A_{\text{tmp}} = 1/4 \cdot wc^2 \cdot \tan \theta \quad [5]$$

Thus, the spoil pile base is

$$W'_{22} = (4 \cdot A_{\text{tot}} \cdot \cot \theta + \hat{a}_{22}^2)^{1/2} \quad [6]$$

Now compute the distance from the old highwall to the spoil peak.

$$G_{22} = 1/2 \cdot W'_{22} + \hat{a}_{22} \cdot \cot \phi - cc \quad [7]$$

and finally the distance the dragline center is from the old highwall is

$$Y_{22} = \text{Reach}_2 - G_{22} \quad [8]$$

$$\text{recall } \text{Reach}_2 = b_2 \cdot \cos \beta + c_2 \quad [9]$$

APPENDIX VIII

This appendix contains the procedure for determining the total spoil configuration after the rehandle is spoiled. This situation is shown in Figure 37. Note the dragline may or may not be setting on a fill bench to accomplish this task.

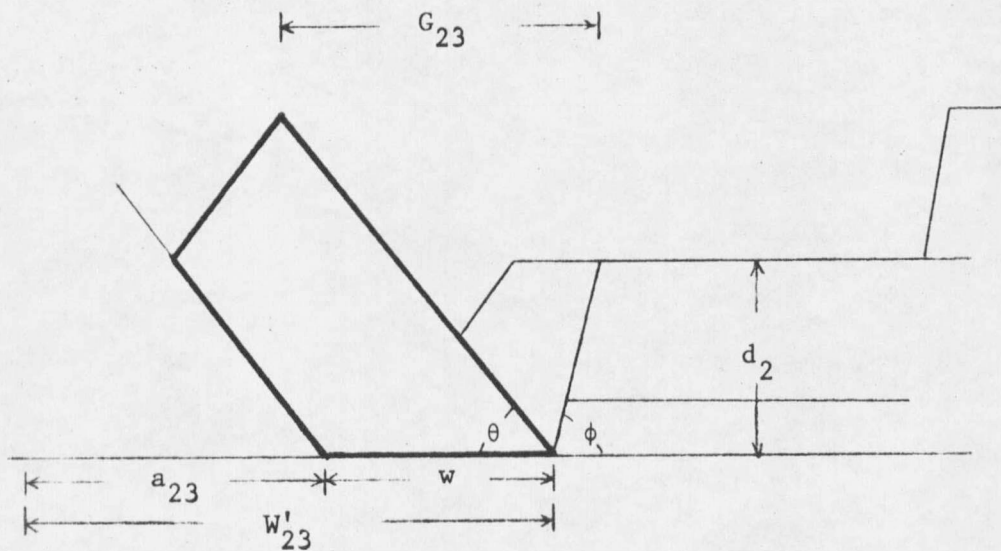


Figure 37. Final Spoil Configuration

The area outlined in dark is the total spoil including rehandle. This area can be determined as follows:

$$A_{\text{tot}} = (A_{11s} + A_{21s} + A_{12s} + A_{22s}) \cdot (1 + f) + 1/2 \cdot cc \cdot t \quad [1]$$

Notice the rehandle accumulated at each cut is now included in the total.

Find the distance across the base of the spoil as follows:

$$\hat{a}_{23} = \begin{cases} 0, & \text{otherwise} \\ 2 \cdot A_{\text{tot}} \cdot \cot\theta / wc - 1/2 \cdot wc, & \text{if } A_{\text{tmp}} < A_{\text{tot}} \end{cases} \quad [2]$$

$$\text{recall } wc = w + cc \quad [3]$$

$$\text{and } A_{\text{tmp}} = 1/4 \cdot wc^2 \cdot \tan\theta \quad [4]$$

Thus, the spoil pile base is

$$W'_{23} = (4 \cdot A_{\text{tot}} \cdot \cot\theta + \hat{a}_{23}^2)^{1/2} \quad [5]$$

Now the distance from the old highwall brow to the spoil peak

$$G_{23} = 1/2 \cdot W'_{23} + \hat{d}_2 \cdot \cot\phi - cc \quad [6]$$

and the dragline center from the old highwall brow is

$$Y_{23} = \text{Reach}_2 - G_{23} \quad [7]$$

$$\text{where } \text{Reach}_2 = b_2 \cdot \cos\beta + c_2 \quad [8]$$

(Note Y_{23} may be negative if there is a fill bench required.)

APPENDIX IX

The average swing angle is that angle confined between the two solid lines in Figure 38. One line connects the dragline tub center with the centroid of the overburden area to be removed and the other

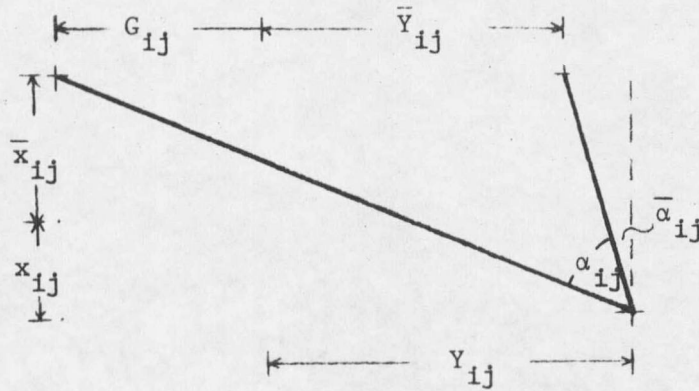


Figure 38. Average Swing Angle

connects the dragline tub center with the centroid of the spoil area. To compute the average swing angle, the following equation is used:

$$\alpha_{ij} = \tan^{-1} \left[\frac{Y_{ij} + G_{ij}}{\bar{X}_{ij} + x_{ij}} \right] - \bar{\alpha}_{ij}$$

where

$$\bar{\alpha}_{ij} = \tan^{-1} \left[\frac{Y_{ij} (+) Y_{ij}}{\bar{X}_{ij} + x_{ij}} \right]$$

and $\bar{x}_{ij} = 1/2 \cdot l_i$ for all $j, i = 1, 2$. The plus - minus in the expression for $\bar{\alpha}_{ij}$ is plus except at the rehandle position ($i = 2, j = 3$). The derivation of the centroids, $\bar{Y}_{ij}$, is given in Appendix XI.

APPENDIX X

The various design options are a useful tool for comprehensive evaluation of pit design. However, such design flexibility involves some complicated interaction between decision variables. Recall the set of decision variables is composed of both pit design variables (w, d_1) and dragline variables (B_1, b_1) . As an option, the user may exclude either subset from the set of decision variables. This may be done explicitly by answering a direct question. The question is under the control of the computer planning system executive. Thus, if the model were isolated from the system, an appropriate input routine must be supplied. The code word IVAR indicates whether or not either subset of the decision vector has been explicitly excluded from the set. Its definition is given at the head of the program listing in Appendix XII. If either the pit design variables or dragline variables are explicitly excluded from the decision set, their respective values must be user specified, and they would be held constant during the optimization.

The dragline decision subset may also have implicit alterations depending upon the design criteria selected. Naturally, the dragline subset must be explicitly included in the decision vector or no alterations would be possible. The remainder of this appendix describes how the design options affect the decision set.

Table XI indicates the composition of the dragline decision subset relative to the various combinations of the design options. Also, half the table is relative to the minimum time objective with the other half corresponding to the minimum cost per ton of coal uncovered objective.

Table XI. Contents of Dragline Decision Subset

	DRAGLINE DECISION SET	PRODUCTION RATE		PURCHASE PRICE	ANNUAL OPERATING HOURS	
		Fix	Derive		Fix	Derive
TIME	empty	yes			yes	
	b_1	yes		yes	yes	
	B_1		yes		yes	
	b_1, B_1		yes	yes	yes	
	b_1, B_1		yes		yes	
COST	b_1	yes			yes	
	b_1	yes		yes	yes	
	b_1, B_1	yes				yes
	b_1, B_1	yes		yes		yes
	b_1, B_1		yes	yes	yes	

Although the table clearly relates the composition of the dragline decision subset for the design options, an explanation of the design strategies is warranted. Referring to Table XI, consider the design strategies available when the minimum time objective is selected. The user may specify an annual production rate within a

given number of annual operating hours. To yield this production rate, the bucket size on the i^{th} pass is solved for as follows:

$$B_i = \frac{TDIG_i}{((OPHRS)(APRDTP/APROD) - TWALK_i)} \quad [1]$$

The boom length is then set to the maximum using the expression obtained from Stafenko's data.¹ This equation expresses maximum boom length as a function of the bucket size and is written as follows:

$$b_i = (351.67 - c_i - 1633.33/B_i)/\cos\beta \quad [2]$$

The user may also elect not to specify an annual production rate; that is, derive its value within the model. In this case, the bucket size is entered into the decision vector. Again the boom length is set to the maximum as in the above strategy. To derive the annual production rate, the following calculations are employed.

$$APROD = \frac{(OPHRS) \cdot (L_P) \cdot (W_P) \cdot (t) \cdot (RHO)}{T} \quad [3]$$

or if tandem draglines are in use

$$APROD = \frac{(OPHRS) (L_P) \cdot (W_P) \cdot (t) \cdot (RHO)}{\text{Maximum } (T_1, T_2)} \quad [4]$$

¹This equation is discussed in full on page 35.

When the user specifies only the annual operating hours and no production rate, the bucket size again enters the decision set and the boom length is set to its maximum.

Notice if a dragline purchase price is specified, the boom length also enters the decision vector for all combinations of the design criteria. Recall when specified, the purchase price is employed as a constraint on the boom and bucket combination.

In terms of the minimum cost per ton of coal exposed objective the design strategies are a bit more complex. In this case the user may derive either the production rate or the annual operating hours within the model. The objective function is sensitive to both strategies since it is a function of both the hourly operating cost and hourly ownership cost; the latter of which is inversely proportional to the total annual operating hours.

When only the annual operating hours are specified, that is no design options are elected, both the boom length and bucket size are in the decision vector. Also, the production rate is derived in the model via expressions [3] or [4] above. If the user elects to fix both a production rate and an annual operating hours, only boom length enters the decision set, since the bucket size can be found via expression [1] above.

The user may specify an annual production rate and wish to derive the operating hours in the model. With this strategy both the boom

length and bucket size enter the decisions set and the annual operating hours for the i^{th} dragline can be calculated as follows:

$$\text{OPHRS} = \frac{(\text{APROD}) \cdot (T)}{\text{APRDTP}} \quad [5]$$

or if tandem draglines are used

$$\text{OPHRS} = \frac{\text{APROD} \cdot \text{MAX}(T_1, T_2)}{\text{APRDTP}} \quad [6]$$

Again, if a dragline purchase price is specified, the boom and bucket combination is constrained using Flour of Utah's purchase price equation.

This discussion on the design strategies and the composition of the dragline decision subset should give an idea of the design options available and their individual effects on the decision vector. How these strategies influence model results is described in the chapter containing example output.

APPENDIX XI

This appendix contains the computational procedures for computing the centroids of the cut areas relative to the second pass highwall brow. The centroids of the two cut areas associated with the first pass overburden are shown in Figure 39.

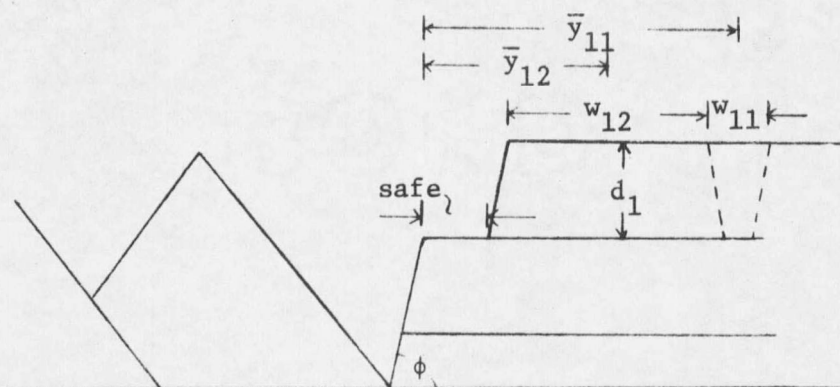


Figure 39. Centroids of Cut Areas on First Pass

The centroid of the first pass key cut is

$$\bar{y}_{11} = \text{safe} + d_1 \cdot \cot \phi + w_{12} + \frac{1}{2} w_{11} \quad [1]$$

and the centroid of the main cut is

$$\bar{y}_{12} = \text{safe} + d_1 \cdot \cot \phi + \frac{1}{2} w_{12} \quad [2]$$

The centroids for the second pass overburden removal operation are shown in Figure 40.

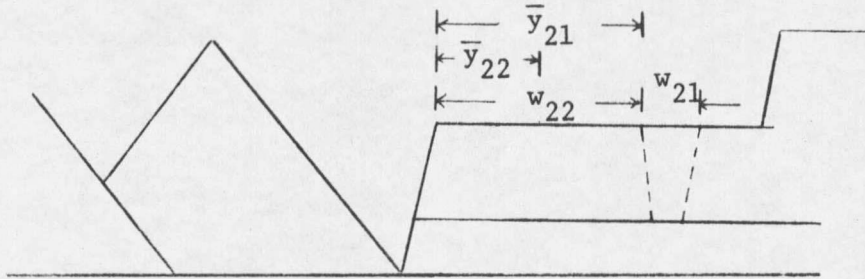


Figure 40. Centroids of Cut Areas on Second Pass

The centroid of the second pass key cut is

$$\bar{y}_{21} = w_{22} + \frac{1}{2} \cdot w_{21} \quad [3]$$

and the main cut centroid is

$$\bar{y}_{22} = \frac{1}{2} \cdot w_{22} \quad [4]$$

The remaining procedure is used to compute the centroid of the rehandle area. The rehandle configuration is shown in Figure 41.

Primarily two rehandle configurations exist:

- i. when w_{23} is greater than zero,
- ii. when w_{23} is equal to zero.

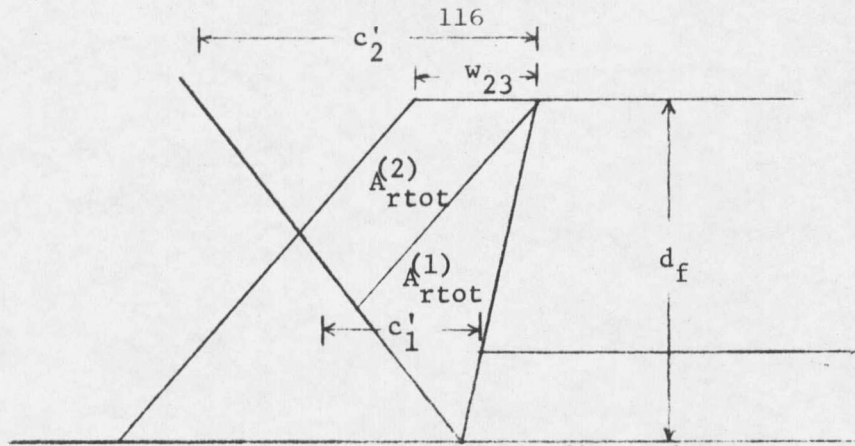


Figure 41. Rehandle Configuration

The following procedure considers both conditions as possible rehandle bench configurations.

The distance d_f is measured from top of coal if we have a coal fender and from bottom of coal if there is no coal fender.

Define:

$$c'_2 = (\cot\phi + \cot\theta) \cdot d_f \quad [5]$$

$$c'_1 = (\cot\theta - \cot\phi) \cdot d_f \quad [6]$$

Recall A_{rtot} is known from rehandle calculations and now can be expressed as follows:

$$A_{rtot} = \frac{1}{2} c'_2 d_f - \frac{1}{4} (c'_2 - w_{23})^2 \tan\theta \quad [7]$$

$$A_{rtot}^{(1)} = \frac{1}{2} c'_1 d_f - \frac{1}{4} \tan\theta c_1'^2 \quad [8]$$

$$A_{rtot}^{(2)} = A_{rtot} - A_{rtot}^{(1)} \quad [9]$$

Substitute (4) into [6] and rewriting yields

$$A_{rtot} = \frac{1}{2}(\cot\phi + \cot\theta)d_F^2 - \frac{1}{4}((\cot\phi + \cot\theta)d_F - W_{23})^3 \tan\theta \quad [10]$$

Now solve for d_F , assuming W_{23} to be equal to zero. This assumption will then be checked. Expand [9] for solution of d_F as follows:

$$A_{rtot} = \frac{1}{2}(\cot\phi + \cot\theta)d_F^2 - \frac{1}{4}((\cot\phi + \cot\theta)d_F)^2 \tan\theta \quad [11]$$

$$4A_{rtot} = [2(\cot\phi + \cot\theta) - (\cot\phi + \cot\theta)^2 \tan\theta]d_F^2 \quad [12]$$

$$d_F = \frac{4 A_{rtot}^{1/2}}{[2(\cot\phi + \cot\theta) - (\cot\phi + \cot\theta)^2 \tan\theta]} \quad [13]$$

$$\text{Now if } d_F \text{ is } > d'_2 = \begin{cases} d_2, \text{ no coal fender} \\ d'_2, \text{ coal fender} \end{cases}$$

then set $d_F = d'_2$ and solve [6] above for W_{23} .

Let:

$$A = \frac{1}{2} \cdot (\cot\phi + \cot\theta)d_F^2 \quad [14]$$

$$B = (\cot\phi + \cot\theta) \cdot d_F \quad [15]$$

Substitute [13] and [14] into [6] and rewrite as follows:

$$A_{rtot} = A - \frac{1}{4}(B - W_{23})^2 \tan\theta \quad [16]$$

$$= A - \left(\frac{1}{4} B^2 - \frac{1}{2} B \cdot W_{23} + \frac{1}{4} W_{23}^2\right) \tan\theta \quad [17]$$

Now use the quadratic formula and solve for W_{23} .

With W_{23} and d_F determined we can now derive $\bar{y}_{23}$ for computation of average swing angle

$$\begin{aligned}
 y_{23}^{(1)} = & \cot\phi(d'_2 - d_F) + [(\frac{1}{3} \cot\theta d_F)(\frac{1}{2} \cot\theta d_F^2) \\
 & - (\cot\phi d_F + \frac{1}{2} c'_1)(\frac{1}{4} \tan\theta c'_1{}^2) \\
 & - (\frac{1}{3} \cot\phi d_F)(\frac{1}{2} \cot\phi d_F^2)]/A_{rtot}^{(1)}
 \end{aligned} \tag{17}$$

$$\begin{aligned}
 \bar{y}_{23}^{(1)} = & [\frac{1}{6} \cot\phi^2 d_F^3 + \frac{1}{8} c'_2 \tan\theta c'_1{}^2 - \frac{1}{6} \cot\theta^2 d_F^3]/A_{rtot}^{(1)} \\
 & - \cot\phi(d'_2 - d_F)
 \end{aligned} \tag{18}$$

$$\bar{y}_{23}^{(2)} = \frac{\frac{1}{2} c'_2 (\frac{1}{2} c'_2 \tan\theta) + \frac{1}{2} (c'_2 - W_{23}) (-\frac{1}{4} (c'_2 - W_{23})^2)}{A_{rtot}^{(2)}} \tag{19}$$

$$\text{Finally, } \bar{y}_{23} = \frac{\bar{y}_{23}^{(1)} A_{rtot}^{(1)} + \bar{y}_{23}^{(2)} A_{rtot}^{(2)}}{A_{rtot}} \tag{20}$$

APPENDIX XII

```

FTN4,L
      PROGRAM MOD2,3
C
C THIS IS THE MAIN BODY OF THE PROGRAM
C ITS PURPOSE IS TO COMPUTE THE SPOIL
C CONFIGURATION AT EACH CUT
C
C:SPECIAL CONTROL WORDS
C
C IAND(IVAR,8) = 8; A PRODUCTION RATE HAS
C                      BEEN SPECIFIED
C IAND(IVAR,32;) = 32; AN ANNUAL OPERATING HOURS HAS
C                      BEEN SPECIFIED
C IAND(IVAR,16) = 16; A PURCHASE PRICE HAS BEEN
C                      SPECIFIED
C IAND(IVAR,1) = 1; FIT DESIGN VARIABLES ARE IN THE
C                      DECISION SET
C IAND(IVAR,2) = 2; DRAGLINE VARIABLES ARE IN THE
C                      DECISION SET
C-----
C IAND(IOBJ,1) = 1; MINIMUM TIME OBJECTIVE WAS SELECTED
C IAND(IOBJ,2) = 2; MINIMUM COST OBJECTIVE WAS SELECTED
C-----
C INQ = 0; CALL INPUT ROUTINE
C INQ = 1; CALL EQUALITY CONSTRAINT EVALUATION
C INQ = 2; CALL INEQUALITY CONSTRAINT EVALUATION
C INQ = 3; CALL OBJECTIVE EVALUATION
C INQ = 5; CALL FOR TOTAL EVALUATION
C-----
C PARM = ARRAY CONTAINING INFORMATION ON PROGRAM SWAP
C
      DIMENSION XX(16), XX1(24,16), XX2(19,16), R(40),
&SUM(24), FF(19), SR(19), ROLD(40), H(16)
      DIMENSION IBUFR(1)
C
      COMMON NX,NC,NIC,STEP,ALFA,BBETA,GAMA,IN,INF,
1FDIFER,SEQL,K1,K2,K3,K4,K5,K6,K7,K8,K9,XX,XX1,
&XX2,R,SUM,FF,SR,ROLD,SCALE,FOLD,
2LFEAS,L5,L6,L7,L8,L9,R1A,R2A,R3A,
3D, T, LP, WP, APROD, F, V, CC2, CF,
&DPRICE, PHI, THETA, BETA,

```

```

4CZ, CR, BW, W2, DELTA, C, D2, X(22),
&IXPTR(22), IMDL, RLOC(34),
5RHO, WPIT, OPHRS, IFNDR, CASTD, SAFE, IOBJ,
&IVAR, HOURS, CTON,

```

C

```

6A1A, A1B, A2, A3A, A3B, A4, COSB, COTDEL, COTPHI,
&COTT, TANT, D2P, D2HAT,
7G1B, G2, G3B, G4, REACH, XLAST, YLAST, YB4

```

C

```

COMMON A11S, A12S, A21S, A22S, ARTOT, AR11, AR12,
&DF, REACH1, REACH2,
1      C1, C2, CR1, CR2, BW1, BW2, G11, G12, G21, G22
2      , ITANDM

```

C

```

COMMON IDAT, JUNK(4), IPRM(5)
REAL LP, L1, L2

```

C

```

EQUIVALENCE (ISTRK, IPRM(1)), (ISECT, IPRM(2)),
1      (INQ, IPRM(3)), (IBL, IPRM(4))

```

C

```

EQUIVALENCE (W21, X(1)), (W, X(2)), (L2, X(3)),
1(X11, X(4)), (X12, X(5)), (Y11, X(6)), (Y12, X(7)),
2(D1, X(8)), (W3, X(9)), (W23, X(10)), (X21, X(11)),
3(Y21, X(12)), (X22, X(13)), (Y22, X(14)), (X23, X(15)),
4(Y23, X(16)), (B2, X(17)), (CB2, X(18)), (W11, X(19)),
5(L1, X(20)), (B1, X(21)), (CB1, X(22)), (W2, W22)

```

C

C

C

```

EQUIVALENCE (IBUFR, NX)

```

C

```

C RETRIEVE COMMON FROM FATHER

```

C

```

CALL RMPAR(IPRM)
CALL EXEC (1, 66, IBUFR, IBL, ISTRK, ISECT)
IBL = IPRM(4)

```

C

C:

```

IDAT = 0

```

```

C:CALL INPUT ROUTINE

```

```

IF (INQ .NE. 0) GOTO 5

```



```

C
C
C
5      IF (INQ .EQ. 5) GOTO 17
C:CHANGE X TO VALUES SELECTED BY FLEX-TOL
C
10      DO 15 I= 1,NX
15      X(IXPTR(I)) = XX(I)
C
C #####
C #                                           #
C #      INITIALIZE  VARIABLES                #
C #                                           #
C #####
C
17      NC = 0
        ALFA = 1.3
        BBETA = .5
        GAMA = 2.
C
        ALPHA = 3.14159 - (THETA+PHI)
        COTT = 1./TAN(THETA)
        COST = COS(THETA)
        COSB = COS(BETA)
        COTPHI = 1./TAN(PHI)
        COTDEL = 1./TAN(DELTA)
        TANT = TAN(THETA)
C
        SINT = SIN(THETA)
        SINP = SIN(PHI)
        SINA = SIN(ALPHA)
C:
C:RESET BOOM LENGTH IF MINIMUM TIME OBJECTIVE
C:& PURCHASE PRICE
C:
        IF (IAND(IVAR,2) .NE. 2) GOTO 265
        IF (IOBJ.EQ.2) GOTO 265
        IF (IAND(IVAR,8).EQ.8 .AND. IAND(IVAR,16).EQ.16)
16GOTO 265
        IF (IAND(IVAR,16).EQ.16) GOTO 265
C:

```

```

      CB1=(351.67-C1-1633.33/B1)/COSB
      CB2=(351.67-C2-1633.33/B2)/COSB
C:
265  CONTINUE
C:
C:SET THE REACH
C:
      REACH1=CB1*COSB + C1
      REACH2=CB2*COSB + C2
C:TANDEM
      IF (ITANDM.EQ.2) GOTO 20
      CB2=CB1
      REACH2=REACH1
      B2=B1
      BW2=BW1
      CR2=CR1
C
C #####
C #
C # APPENDIX II - CUT LENGTH COMPUTATION #
C #
C #####
20  L1=SQRT((REACH1+CASTD)**2 - REACH1**2)
    L2=SQRT((REACH2+CASTD)**2 - REACH2**2)
C:
C:TANDEM
      IF (ITANDM.EQ.2) GOTO 21
      L2=L1
C:
C
21  DELTUB = 24.0
      D2 = D - D1
      D2HAT = D2 + T
      SAFE = C2 - D2*COTPHI
      IF (SAFE .LT. 0.) SAFE = 0.
      SWELL = 1. + CF
      REACH = REACH2
      W3 = SAFE
C:
      W11=0.
      W12=0.

```

W21=0.
W22=0.
W23=0.

C

X11 = 0.
Y11 = 0.
X12 = 0.
Y12 = 0.
X21 = 0.
Y21 = 0.
X22 = 0.
Y22 = 0.
X23 = 0.
Y23 = 0.

C

A1 = W*D1*SWELL
A2 = W*D2*SWELL
A11 = 0.
A12 = 0.
A11S = 0.
A12S = 0.
A21S = 0.
A22S = 0.
ARTOT = 0.
AR11 = 0.
AR12 = 0.
AR12T = 0.
AR21 = 0.

C

CC = 0.0
IF (IFNDR .EQ. 1) CC = T*(COTT+COTPHI)
WC = W + CC
WPIT=W

C

```
C #####
C #
C # APPENDIX III - DETERMINE IF REHANDLE SPOIL #
C # IS NECESSARY TO SPOIL ALL THE #
C # OVERBURDEN #
C #
C #####
```



```

C
C
      ATOT = A1 + A2 + .5*CC*T
C
C NOW FIND GLAST (THE DISTANCE FROM THE SPOIL
C PEAK TO THE LAST DRAGLINE LOCATION)
C
      CALL AHAT (ATOT,WC,AH,WTOT)
      GLAST = .5*WTOT + D2HAT*COTPHI - CC
C
C NOW COMPUTE THE DISTANCE FROM THE OLD HIGHWALL
C TO THE DRAGLINE CENTERLINE
C
      YLAST = CB2*COSB - GLAST
C
C IF YLAST < 0 THEN WE ARE GOING TO HAVE SOME
C REHANDLE
C
      IF (YLAST .GE. 0.) GOTO 25
C
C
C COMPUTE THE VOLUME OF REHANDLE TO MAKE THE
C PLAN FEASIBLE. THIS RESULTS IN INCURRING THE
C LEAST REHANDLE POSSIBLE.
C
      W23 = ABS(YLAST)
      IF (ABS(YLAST).LT.2*C2) W23=2*C2
      D2P = D2HAT
      IF (IFNDR .EQ. 1) D2P = D2
      C2TMP = (COTT+COTPHI)*D2P
      IF (W23 .GT. C2TMP) W23 = C2TMP
      ARREQ = .5*C2TMP*D2P - .25*(C2TMP-W23)**2*TANT
C $$$$$$$$$$$$$$$$$$$$$$
25  CONTINUE
C #####
C #
C # APPENDIX IV - FIND THE KEY CUT SPOIL #
C # CONFIGURATION ON THE FIRST #
C # PASS #
C #
C #####

```

```

C
C
C
C: COMPUTE THE DISTANCE TO THE SPOIL PEAK
C
100  Y11 = W + SAFE
      G11 = REACH1 - Y11
      IF (G11 .LT. 0.) G11=0.0
C
C: COMPUTE THE AVAILABLE KEY CUT AREA
C: ASSUME AHAT11=0
C
      AHAT11 = 0.0
      WP11 = 2.*(G11-D2HAT*COTPHI+CC)
      IF (.5*WP11+G11 .GT. D2HAT*COTPHI+W)
1AHAT11=WP11-WC
      DELC = 0.0
      DF = T
C: THE ENTERTAINED SWELLED KEY CUT AREA IS
      A11T = .25*WP11**2*TANT - .25*AHAT11**2*TANT
1- .5*CC*T
      IF (G11 .LE. D2HAT*COTPHI-CC) A11T=0.0
C
C: MINIMUM KEY CUT AREA IS
      A11MIN = (BW1+D1*COTPHI)*D1*SWELL
C: SET A11 = A11MIN (REMOVE ONLY THE MINIMUM)
      A11=A11MIN
C
C: COMPARE THE COMPUTED AREA TO THE MINIMUM
C
      IF (A11T .GE. A11MIN) GOTO 165
C
C
C: OTHERWISE WE MUST PLAN SOME REHANDLE TO SATISFY
C: THE MINIMUM KEY CUT REQUIREMENTS
C
C USE THE QUADRATIC FORMULA TO SOLVE FOR DELC
C
      CALL SPCNF(A11MIN,AHATT,WPRIM,DELC)
      AHAT11 = AHATT
      WP11 = WPRIM

```

```

      DELC = DELC
C: NOW COMPUTE THE MAXIMUM AREA AVAILABLE TO SPOIL
C: IN AT THE 1ST PASS KEY CUT POSITION
C
160   WP11MX = 2.*(G11+D2HAT*COTT)
      AHT1MX = 0.0
      IF (.5*WP11MX+G11 .GT. D2HAT*COTPHI + W) AHT1MX=
1.5*WP11MX
1    - (D2HAT*COTPHI+W-G11)
      A11MAX = .25*WP11MX**2*TANT - .5*D2HAT*
1(COTT+COTPHI)*D2HAT
1    - .25*AHT1MX**2*TANT
C NOW SELECT THE APPROPRIATE SPOIL AREA
      A11 = AMIN1(A11,A11MAX)
C
C: NOW COMPUTE THE FIRST PASS KEY CUT AND MAIN
C: CUT WIDTHS
C
      IF (A11.NE.A11MAX) GOTO 165
C: CHANGE WP11 AND AHAT11 IF A11=A11MAX
      WP11 = WP11MX
      DELC = D2HAT*(COTT+COTPHI) - CC
      AHAT11 = AHT1MX
      DF = (CC+DELC)/(COTT+COTPHI)
C
165   W11=A11/(D1*SWELL) + D1*COTPHI
      IF (A11.LT.A11MIN) W11=SQRT(4.*A11*COTPHI)
      W12=W-W11
C
C: NOW DETERMINE THE KEY CUT POSITION FROM CUT FACE
C
175   CONTINUE
C
C
      IF (A11T .LT. A11MIN)
1AR11 = .25*WP11**2*TANT - .25*(WP11-DELC)**2*TANT
2    - .5*(CC+DELC)*DF + .5*CC*T
      ARTOT = AR11
C
C COMPUTE THE KEY AREA SITU * SWELL
C

```

```

      A11S = A11
C
C #####
C #
C #   APPENDIX V - FIND AREA TO BE SPOILED #
C #   FROM FIRST PASS MAIN CUT          #
C #   POSITION                             #
C #                                       #
C #####
C
C:THE AREA TO REMOVE AT THE MAIN CUT IS
C
      A12 = A1 + .5*CC*T - AR11
      A12S = A1 - A11S
C
C FIND WP12 AND AHAT12
C
      CALL AHAT (A12,WC,AHAT12,WP12)
      G12 = .5*WP12 + D2HAT*COTPHI - CC
      YMIN = SAFE + D1*COTPHI + C1
      GMAX = REACH1 - YMIN
C
C CHECK IF THERE IS ANY REHANDLE AT THE MAIN
C CUT POSITION
C
204   AR12 = ARREQ - AR11
      IF (AR12 .LT. 0.) AR12 = 0.
C
C: FIRST COMPUTE THE MAXIMUM AMT. OF REHANDLE AVAILABLE
C AT THE MAIN CUT POSITION
C
      WCTMP = W + D2HAT*(COTT+COTPHI)
      AREA = A1 + .5*D2HAT**2*(COTT+COTPHI)
      CALL AHAT (AREA,WCTMP,AHAT12,WP12MX)
      AR12MX = .25*WP12MX**2*TANT - .25*(WP12MX-D2HAT
1*(COTT+COTPHI)+CC)
1      **2*TANT - .5*D2HAT**2*(COTT+COTPHI)
2      + .5*CC*T - AR11
C:CHECK IF FIRST PASS SPOIL IS CAPABLE OF FORMING A
C:BENCH
      IF (A1 .GE. .5*D2HAT**2*(COTT-COTPHI)) GOTO 206

```



```

      SIGMA = 0.
203  DF2=SQRT(2.*(A1-SIGMA*A11)/(COTT-COTPHI))
      BF2=DF2*(COTT-COTPHI) + CC
      BF1=SQRT(2.*(A11+.5*CC*T)*COTT)
      DELB=G11-D2HAT*COTPHI-.5*BF1-CC
      IF (DELB.LE.0.0) DELB=0.0
      IF (BF2.GE.BF1+DELB .OR. SIGMA.EQ.1.) GOTO 202
      SIGMA=1.0
      GOTO 203

C:
202  AR12=A1-SIGMA*A11-.25*BF2**2*TANT-AR11 +SIGMA*.25*
1    (BF2-DELB)**2*TANT
      G12=D2HAT*COTPHI
      GOTO 130

C:
206  WPT = WP11 - DELC
      CP = DELC*SINT
      IF (G12 .LE. GMAX) GOTO 209
C:FIND THE MAIN CUT SPOIL CONFIGURATION
      SIG1=0.
      SIG2=1.
      SIG3=0.

C:
205  S1=GMAX-D2HAT*COTPHI+CC
      S2=(2.*GMAX-2.*D2HAT*COTPHI+CC-W)*SIG3
      G12=GMAX
C:FIND DELC2
207  AQ=TANT-SIG3*.25*TANT-SIG2*.25*TANT
      BQ=2.*S1*TANT - .25*S2*TANT - SIG2*.25*TANT*WPT
      CQ=S1*S1*TANT - .25*S2*S2*TANT - .25*WPT*WPT*TANT
1    + .25*AHAT11*AHAT11*TANT - SIG1*AR11 -A12S
      DELC2=FNCQD(AQ,BQ,CQ)
C:TEST THE ASSUMPTION AHAT12=0
      WP12T=2.*(GMAX-D2HAT*COTPHI+CC+DELC2)
      AHAT12=WP12T-(W+CC+DELC2)
      IF (AHAT12.LE.0. .OR. SIG3.EQ.1.0) GOTO 212
      SIG3=1.0
      GOTO 205
C:CHECK THE REHANDLE CONFIGURATION FOR OVERHANG
212  XD=DELC2*SINT
      IF (SIG1.EQ.1.) GOTO 208

```

```

      IF (XD,LE,CP) GOTO 208
      SIG1=1.
      SIG2=0.
      GOTO 207
C: COMPUTE THE NECESSARY REHANDLE
208  BP12=(GMAX-D2HAT*COTPHI+CC+DELC2)/COST
      1- .5*(WPT+DELC2)/COST
      IF (XD,LE,CP) AR12T = XD*BP12
      IF (XD,LE,CP) GOTO 209
      S1=GMAX-D2HAT*COTPHI+CC
      S2=.5/(COTPHI+COTT)
      AR12T=(.75*TANT-S2)*DELC2*DELC2 +
      1(S1*TANT-2.*S2*CC)*DELC2
      2      + .5*CC*T - AR11
C:
209  AR12 = AMAX1(AMIN1(AR12MX,AR12),AMIN1(AR12MX,AR12T))
C: CHECK IF AR12 IS FILL BENCH MATERIAL
      IF (AR12,EQ,0.0) GOTO 130
      IF (AR12,NE,ARREQ-AR11) GOTO 211
      AREA=A1-AR11-AR12+.5*CC*T
      CALL AHAT(AREA,WC,AHT12T,WP12T)
      G12T=.5*WP12T + D2HAT*COTPHI - CC
      BP12T=WP12T/COST - WPT/COST
      XD = AR12/BP12
C:
      IF (XD,LE,CP) GOTO 210
      S1=G12T-D2HAT*COTPHI+CC
      S2=.5/(COTPHI+COTT)
      AQ=.25*TANT-S2
      BQ=S1*TANT-2.*S2*CC
      CQ=S2*CC*CC + .5*CC*T -AR11 -AR12
      DELC2=FNCQD(AQ,BQ,CQ)
      XD=DELC2*SINT
C:
210  G12 = G12T - XD/COS(ABS(1.5707-2.*THETA))*COST
      GOTO 130
C: CHECK IF AR12 IS THE MAXIMUM REHANDLE AREA
211  IF (AR12,NE,AR12MX) GOTO 130
      G12=.5*WP12MX - D2HAT*COTT
C
C NOW COMPUTE THE Y LOCATION AT MAIN CUT

```

```

C
C $$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$$
130  CONTINUE
131  ARTOT=AR11 + AR12
      Y12 = REACH1 - G12
C
C: COMPUTE THE DRAGLINE LOCATION FROM THE CUT FACE
135  YP=AMAX1(Y11-(W12+SAFE+D1*COTPHI),
      1(W+SAFE+D1*COTPHI)-Y11)
      RP=SQRT((REACH1-C1)**2 - YP**2)
      X11=(RP-L1)/(1.0+D1/CR1)
      X11=X11+C1
      IF (X11.LT.24.9) GOTO 140
C:
      YP = AMAX1 (Y12-SAFE,(W12+SAFE+D1*COTPHI)-Y12)
      RP = SQRT((REACH1-C1)**2 - YP**2)
      X12 = (RP-L1)/(1.+D1/CR)
      X12=X12+C1
      IF (X12.LT.24.9) GOTO 140
      IF (X11 .LT. X12) X12 = X11
      GOTO 145
C:
140  L1=SQRT((REACH1-C1)**2 - YP**2)
      GOTO 135
C
C
C ***** 2ND PASS POSITIONS *****
C
C NOW THE SECOND PASS POSITIONS CANBE DETERMINED
C
C FIRST, THE AREA TAKEN UP BY 1ST PASS SPOIL IS
C
145  A1LR = A1 + .5*CC*T - AR11 - AR12
C
C *****
C #
C # APPENDIX VI - 2ND PASS KEY CUT POSITION #
C #
C *****
C
C
C

```

C: NOW DETERMINE THE AREA REMOVED

C

```

      Y21 = W - D2*COTPHI
      G21 = REACH2 - Y21
      WP21 = 2.*(G21-D2HAT*COTPHI+CC)
      AHAT21 = WP21 - WC
      IF (AHAT21 .LT. 0.0) AHAT21 = 0.0
      A21 = .25*WP21**2*TANT - .25*AHAT21**2*TANT
      A21T = A21 - A1LR
      IF (A21T .LT. 0.) A21T = 0.

```

C

C IF A21S < 0 SET THE AREA = THE MINIMUM AND CALL
C IT REHANDLE

C

```

      A21MIN = (D2*COTPHI+BW2)*D2*SWELL
      A21S = A21MIN
      AR21 = A21MIN - A21T
      IF (AR21 .LT. 0.) AR21 = 0.
      ARTOT = ARTOT + AR21

```

C

C FIND THE CUT WIDTHS ON THE SECOND

C

```

      W21 = A21S/(D2*SWELL) + D2*COTPHI

```

C

C #####

C #

C # APPENDIX VII - COMPUTE THE MAIN CUT AREA #

C #

C #####

C

215 W22 = W - W21

```

      A22S = A2 - A21S

```

```

      ATOT = A1LR + (A21S-AR21) + A22S

```

C

C NOW COMPUTE THE MAIN CUT 2ND PASS DRAGLINE LOCATION

C

```

      CALL AHAT (ATOT,WC,AHAT22,WP22)
      G22 = .5*WP22 + D2HAT*COTPHI - CC
      Y22 = REACH2 - G22

```

C

C #####


```

C #
C # APPENDIX VIII - REHANDLE POSTION #
C # #
C #####
C
    IF (ARTOT .EQ. 0.0) GOTO 216
C
C OTHERWISE THERE IS A REHANDLE POSITION
C
    ATOT = ATOT + ARTOT
    CALL AHAT (ATOT,WC,AHAT23,WP23)
    G23 = .5*WP23 + D2HAT*COTPHI - CC
    Y23 = REACH2 - G23
C:
C: COMPUTE THE DRAGLINE LOCATIONS FROM THE CUT FACE
216 YP=AMAX1(Y21-W22,W-Y21)
    RP=SQRT((REACH2-C2)**2 - YP**2)
    X21=(RP-L2)/(1.0+D2/CR2)
    X21=X21+C2
    IF (X21.LT.24.9) GOTO 217
C:
    YP=AMAX1(Y22,W22-Y22)
    RP=SQRT((REACH2-C2)**2 - YP**2)
    X22=(RP-L2)/(1.0+D2/CR2)
    X22=X22+C2
    IF (X22.LT.24.9) GOTO 217
    X22=AMIN1(X22,X12)
C:
    IF (ARTOT.LE.0.0) GOTO 220
    YP = FYB23(1.) + Y23
    RP = SQRT((REACH2-C2)**2 - YP**2)
    X23 = (RP-L2)/(1.+D2/CR2)
    X23=X23+C2
    IF (X23.LT.24.9) GOTO 217
    X23 = AMIN1(X23,X12)
    GOTO 220
217 L2=SQRT((REACH2-C2)**2 - YP**2)
    GOTO 216
C
C #####
C # #

```

```

C #   APPENDIX X - COMPUTES BUCKET SIZE   #
C #                                         #
C #####
C
C
220  CONTINUE
C: COMPUTE THE PRODUCTION OF THE AREA
    APRDTP = WP*LP*T*RHO
C: COMPUTE BUCKET SIZE
    IF (IAND(IVAR,8).EQ.0 .OR. IAND(IVAR,32).NE.32)
1GOTO 225
    IF (IAND(IVAR,2).NE.2 .AND. IAND(IVAR,24).EQ.0)
1GOTO 225
    B1=1.0
    B2=1.0
    TDUM=TIME(W12,W22,TIME1,TIME2,TCUTS1,TCUTS2,
1TWALK1,TWALK2)
C:
    B1=TCUTS1/(OPHRS*(APRDTP/APROD)-TWALK1)
C:
    B2=TCUTS2/(OPHRS*(APRDTP/APROD)-TWALK2)
C: ITANDM=1:
    IF (ITANDM.EQ.2) GOTO 225
    B1=(TCUTS1+TCUTS2)/(OPHRS*(APRDTP/APROD)
1-TWALK1-TWALK2)
    B2=B1
C:
225  CONTINUE
C
C CALL TIME FOR CONSTRAINTS AND OBJECTIVE
C
    TTIME=TIME(W12,W22,TIME1,TIME2,TCUT1,TCUT2,
1TWALK1,TWALK2)
C
C: FIND THE PRODUCTION RATE OR OPERATING HOURS
C
    APRDTP=LP*WP*T*RHO
    OPHRS1=OPHRS
    IF (IAND(IVAR,8).NE.8 .OR. IAND(IVAR,32).EQ.32)
1GOTO 235
    OPHRS1=APROD*TTIME/APRDTP

```

```

      OPHRS2=0.0
C:TANDEM
      IF (ITANDM.EQ.1) GOTO 230
      OPHRS1=APROD*TIME1/APRDTP
      OPHRS2=APROD*TIME2/APRDTP
230   OPHRS=AMAX1(OPHRS1,OPHRS2)
C:
C:FIND PRODUCTION RATE
C:
235   IF (IAND(IVAR,8).EQ.8 .OR. IAND(IVAR,32).NE.32)
      1GOTO 245
      APROD=OPHRS*APRDTP/TTIME
C:TANDEM
      IF (ITANDM.EQ.1) GOTO 245
      APROD=OPHRS*APRDTP/AMAX1(TIME1,TIME2)
245   CONTINUE
C:
C NOW GET VALUES FOR OPTIMIZATION ROUTINE
C
300   GOTO (1000,2000,3000,9999,1000), INQ
C
C EQUALITY CONSTRAINTS
C
1000  CONTINUE
C
      IF (INQ .EQ. 5) GOTO 2000
      GOTO 9999
C
C INEQUALITY CONSTRAINTS
C
C
2000  DO 2005 I=1,40
2005  R(I) = 1.0
C
C:PRUCHASE PRICE BOOM LENGTH CONSTRAINT
C
      NIC=1
      IF (IAND(IVAR,16).EQ.0) GOTO 2006
      R(NIC)=DPRICE-(981.71+.4106*B1*CB1)
      R(NIC+1)=DPRICE-(981.71+.4106*B2*CB2)
      NIC=NIC+1

```

```

C
C:PRODUCTIVITY CONSTRAINT
C
2006 IF (ITANDM .EQ. 1) GOTO 2010
      R(NIC+1) = DELTUB - ABS(TIME1-TIME2)
      NIC = NIC+1
C
C
C:CUT WIDTH CONSTRAINT
C
      R(NIC+1)=1.0
      IF (REACH1-Y11 .LE. 0.0) R(NIC)=-1000.
      NIC=NIC+1
C
C: THE KEY CUT AREAS MUST BE > MINIMUM KEY CUT AREA
C
2010 R(NIC+1) = -(BW1+D1*COTPHI)*D1*SWELL + A11S
      NIC = NIC + 1
      R(NIC+1) = -(BW2+D2*COTPHI)*D2*SWELL + A21S
      NIC = NIC + 1
C
C: THE TOTAL REHANDLE AREA CAN'T EXCEED THE MAXIMUM
C AREA AVAILABLE FOR REHANDLE
C
      D2P = D2HAT
      IF (IFNDR .EQ. 1) D2P = D2
      ARMAX = .5*(COTT+COTPHI)*D2P**2
      R(NIC+1) = ARMAX - ARTOT
      NIC = NIC + 1
C:THE REHANDLE ACCUMULATED MUST BE SUFFICIENT FOR BENCH
      R(NIC+1) = ARTOT - ARREQ
      NIC = NIC + 1
C
C: THE BOOM LENGTH CONSTRAINT RELATIVE TO BUCKET SIZE
C
      R(NIC+1) = 351.67 - 1633.33/B1 -REACH1
      NIC = NIC + 1
      R(NIC+1) = 351.67 - 1633.33/B2 - REACH2
      NIC = NIC + 1
C
C: NOW THE CONSTRAINTS FOR THE DRAGLINE LOCATIONS

```



```

C
C (A) THE FIRST CONSTRAINT FOR SET -- INDICATES THE
C   DRAGLINE LOCATION RELATIVE TO CUT FACE
C (B) THE SECOND CONSTRAINT FOR EACH SET -- INDICATES
C   DRAGLINE LOCATION FROM OLD HIGHWALL EDGE
C
  R(NIC+1) = X11 - C1
  NIC = NIC + 1
  R(NIC+1) = Y11 - C1
  NIC = NIC + 1
C
  IF (Y12 .EQ. 0.) GOTO 2020
  R(NIC+1) = X12 - C1
  NIC = NIC + 1
  R(NIC+1) = Y12 - C1
  NIC = NIC + 1
C
2020 IF (Y21 .EQ. 0.) GOTO 2030
  R(NIC+1) = X21 - C2
  NIC = NIC + 1
  R(NIC+1) = Y21 - C2
  NIC = NIC + 1
C
2030 R(NIC+1) = X22 - C2
  NIC = NIC + 1
  R(NIC+1) = Y22 - C2
  NIC = NIC + 1
C
  R(NIC+1) = 0.0
  NIC = NIC + 1
  R(NIC+1) = 0.0
  NIC = NIC + 1
  IF (ARTOT .EQ. 0.0) GOTO 170
  NIC = NIC - 2
  R(NIC+1) = X23 - C2
  NIC = NIC + 1
  R(NIC+1) = Y23 - (C2-W23)
  NIC = NIC + 1
C
170 DO 180 I = 1,34
180 RLOC(I) = R(I)

```

```

      IF (INQ .EQ. 5) GOTO 3000
C
      GOTO 9999
C
C GET THE OBJECTIVE FUNCTION VALUE
C
3000  CONTINUE
C
      NCT = NC + NIC + 1
      HOURS=TTIME
      TONS = LP*WP*RHO*T
C
C: GET COST PER TON OF COAL
      MUFD1 = (REACH1-C1)*B1*.001
      MUFD2 = (REACH2-C2)*B2*.001
      IF (ITANDM .EQ. 1) TIME1 = HOURS
      OPCST1 = (61.96 + 9.74*MUFD1)*TIME1/TONS
      OPCST2 = 0.
      OWCST1=(17.695+7.04*MUFD1)*(7467./OPHRS)
1*TIME1/TONS
      OWCST2=0.0
      IF(ITANDM.EQ.2) OPCST2=(61.96+9.74*MUFD2)
1*TIME2/TONS
      IF(ITANDM.EQ.2)OWCST2=(17.695+7.04*MUFD2)
&*(7467./OPHRS)*TIME2/TONS
      CTON = OPCST1 + OPCST2 + OWCST1 + OWCST2
C
C DEPENDING ON OBJECTIVE
C
      GOTO (3010,3020) IOBJ
3010  R(NCT) = HOURS
      GOTO 9999
3020  R(NCT) = CTON
C
C SET UP FOR RETURN TO EXEC1
C
      RLOC(NCT) = R(NCT)
C
9999  CONTINUE
C
C SEND COMMON BACK TO FATHER AND RETURN

```

```
C      CALL EXEC (2,66,IBUFR,IBL,ISTRK,ISECT)
C
      OPH=HOURS
      IF (ITANDM.EQ.2) OPH=AMAX1(TIME1,TIME2)
      CHPROD=TONS/OPH*OPHRS
C:
999    CONTINUE
      END
```

```

C      FUNCTION TIME(W12,W22,TIME1,TIME2,TDIG1,TDIG2,
1TWALK1,TWALK2)
C
C      DIMENSION XX(16), XX1(24,16), XX2(19,16), R(40),
&SUM(24), FF(19), SR(19), ROLD(40), H(16)
C
C      COMMON NX,NC,NIC,STEP,ALFA,BBETA,GAMA,IN,INF,
1FDIFER,SEQL,K1,K2,K3,K4,K5,K6,K7,K8,K9,XX,XX1,
&XX2,R,SUM,FF,SR,ROLD,SCALE,FOLD,
2LFEAS,L5,L6,L7,L8,L9,R1A,R2A,R3A,
3D, T, LP, WP, APROD, F, V, CC2, CF,
&DPRICE, PHI, THETA, BETA,
4CZ, CR, BW, W2, DELTA, C, D2, X(22),
&IXPTR(22), IMDL, RLOC(34),
5RHO,WPIT, OPHRS, IFNDR, CASTD, SAFE, IOBJ,
&IVAR, HOURS, CTON,
C
6A1A,A1B,A2,A3A,A3B,A4, COSB,COTDEL,COTPHI,
&COTT,TANT,D2P,D2HAT,
7G1B,G2,G3B,G4, REACH, XLAST, YLAST, YB4
C
C      COMMON A11S,A12S,A21S,A22S,ARTOT,AR11,AR12,
&DF,REACH1,REACH2,
1      C1,C2,CR1,CR2,BW1,BW2, G11,G12,G21,G22
2      , ITANDM
C
C      COMMON IDAT,JUNK(4),IPRM(5)
REAL LP, L1, L2, NC1, NC2
C
C      EQUIVALENCE (W21,X(1)),(W,X(2)),(L2,X(3)),
1(X11,X(4)),(X12,X(5)),(Y11,X(6)),(Y12,X(7)),
2(D1,X(8)),(W3,X(9)),(W23,X(10)),(X21,X(11)),
3(Y21,X(12)),(X22,X(13)),(Y22,X(14)),(X23,X(15)),
4(Y23,X(16)),(B2,X(17)),(CB2,X(18)),(W11,X(19)),
5(L1,X(20)),(B1,X(21)),(CB1,X(22))
C
C      A1 = W*D1
SWELL = 1. + CF

```



```

ALFB11 = 0.
ALF11 = 0.
ALFB12 = 0.
ALF12 = 0.
ALFB21 = 0.
ALF21 = 0.
ALFB22 = 0.
ALF22 = 0.
ALFB23 = 0.
ALF23 = 0.
D11 = 0.
D12 = 0.
D21 = 0.
D22 = 0.
D23 = 0.
E11 = 0.
E12 = 0.
E21 = 0.
E22 = 0.
E23 = 0.

```

C
C

C I. FIRST PASS TIMES

C

```

YB11=SAFE+D1*COTPHI+W12+.5*W11
YB12=SAFE+D1*COTPHI+.5*W12

```

C

```

XB11=.5*L1
XB12 = XB11

```

C

```

IF (A11S .EQ. 0.) GOTO 100
ALFB11 = ATAN((Y11-YB11)/(XB11+X11))
ALF11 = ATAN((Y11+G11)/(XB11+X11)) - ALFB11

```

C: TIME TO DRAG LEY CUT

```

D11 = (A11S*L1)/(3600.*27.*B1*F*SWELL)
1*(40.2+.228*ALF11/.01745)

```

C: TIME TO WALK FROM KEY CUT TO MAIN CUT

```

E11 = ((ABS(Y12-Y11)+ABS(X12-X11))*V) + CC2
IF (Y12 .EQ. 0.) E11 = L2*V + CC2

```

C
C

```

100  IF (A12S .EQ. 0.) GOTO 110
      ALFB12 = ATAN((Y12-YB12)/(XB12+X12))
      ALF12 = ATAN((Y12+G12)/(XB12+X12)) - ALFB12
C: TIME TO DRAG MAIN CUT
      D12 = (A12S*L2)/(3600.*27.*B2*F*SWELL)
      1*(40.2+.228*ALF12/.01745)
C: TIME TO WALK FROM MAIN CUT TO NEXT KEY CUT
      E12 = E11 + L2*V
      IF (E11 .EQ. 0.) E12 = E12 + CC2
C
C
C TIME TO WALK BETWEEN PITS
C A) ONE MACHINE DEAD HEADING OPERATION
C
110  TP1 = ((LP+(X12-X21)+(W-Y12)+Y11))*V + CC2
C
C
C
C II. SECOND PASS TIMES
C
      YB21=W22+.5*W21
      YB22=.5*W22
      YB23 = FYB23(1.)
C
      XB21 = .5*L2
      XB22 = XB21
      XB23 = XB21
C
      IF (A21S .EQ. 0.) GOTO 120
      ALFB21 = ATAN((Y21-YB21)/(XB21+X21))
      ALF21 = ATAN((G21+Y21)/(XB21+X21)) - ALFB21
C: TIME TO DRAG KEY CUT
      D21 = (A21S*L2)/(3600.*27.*B2*F*SWELL)
      1*(40.2+.228*ALF21/.01745)
C: TIME TO WALK FROM KEY CUT TO MAIN CUT
      E21 = ((ABS(Y22-Y21)+ABS(X22-X21))*V) + CC2
      IF (Y22 .EQ. 0.) E21 = L2*V + CC2
C
120  IF (A22S .EQ. 0.) GOTO 125
      ALFB22 = ATAN((Y22-YB22)/(XB22+X22))
      ALF22 = ATAN((G22+Y22)/(XB22+X22)) - ALFB22

```

```

C: TIME TO DRAG THE MAIN CUT
    D22 = (A22S*L2)/(3600.*27.*B2*F*SWELL)
    1*(40.2+.228*ALF22/.01745)
C: IF MAIN CUT POSITION
    XLAST = X22
    YLAST = Y22
C: TIME TO WALK FROM MAIN CUT TO REHANDLE
    IF (ARTOT .EQ. 0.) GOTO 125
    E22 = ((ABS(Y23-Y22)+ABS(X23-X22))*V) + CC2
C
125  IF (ARTOT .EQ. 0.) GOTO 130
    ALFB23 = ATAN((YB23+Y23)/(XB23+X23))
    ALF23 = ATAN((G23+Y23)/(XB23+X23)) - ALFB23
C: TIME TO DRAG REHANDLE
    D23 = (ARTOT*L2)/(3600.*27.*B2*F*SWELL)
    1*(40.2+.228*ALF23/.01745)
C: IF REHANDLE POSITION
    XLAST = X23
    YLAST = Y23
C
C TIME TO WALK FROM LAST POSITION TO NEXT KEY CUT
C
130  E23 = ((ABS(YLAST-Y21)+ABS(L2-XLAST+X21))*V) + CC2
C
C TIME TO WALK BETWEEN PITS -- SINGLE MACHINE DEAD HEADING
C
    TP2 = (LP+XLAST-X11+(W-YLAST)+Y11)*V + CC2
C
C COMPUTE THE TIME PER EACH PASS
C
    NCUTS1 = (LP*WF)/(L1*W)
    NCUTS2 = (LP*WF)/(L2*W)
    NPITS = WF/W
    TIME1 = NCUTS1*(D11 + D12 + E11 + E12) + NPITS*TP1
    TIME2 = NCUTS2*(D21 + D22 + D23 + E21 + E22 + E23)
    1+ NPITS*TP2
C
C: ESCAVATION TIME FOR A CUT ON EACH PASS
C
    TDIG1 = NCUTS1*(D11+D12)
    TDIG2 = NCUTS2*(D21+D22+D23)

```

143

TWALK1 = NCUTS1*(E11+E12) + NPITS*TP1
TWALK2 = NCUTS2*(E21+E22+E23) + NPITS*TP2

C

C NOW COMPUTE THE TOTAL TIME FOR THE PLAN

C

TIME = TIME1 + TIME2

135 CONTINUE

999 RETURN

END


```

C
C: COMPUTE THE CENTROID OF THE REHANDLE AREA
C
      FUNCTION FYE23 (DUM)
C
      DIMENSION XX(16), XX1(24,16), XX2(19,16), R(40),
&SUM(24), FF(19), SR(19), ROLD(40), H(16)
C
      COMMON NX, NC, NIC, STEP, ALFA, BBETA, GAMA, IN, INF,
1FDIFER, SEQL, K1, K2, K3, K4, K5, K6, K7, K8, K9, XX, XX1,
&XX2, R, SUM, FF, SR, ROLD, SCALE, FOLD,
2LFEAS, L5, L6, L7, L8, L9, R1A, R2A, R3A,
3D, T, LP, WP, APROD, F, V, CC2, CF,
&DPRICE, PHI, THETA, BETA,
4CZ, CR, BW, W2, DELTA, C, D2, X(22),
&IXPTR(22), IMDL, RLOC(34),
5RHO, WPIT, OPHRS, IFNDR, CASTD, SAFE, IOBJ,
&IVAR, HOURS, CTON,
C
6A1A, A1B, A2, A3A, A3B, A4, COSB, COTDEL, COTPHI,
&COTT, TANT, D2P, D2HAT,
7G1B, G2, G3B, G4, REACH, XLAST, YLAST, YB4
C
      COMMON A11S, A12S, A21S, A22S, ARTOT, AR11, AR12,
&DF, REACH1, REACH2,
1      C1, C2, CR1, CR2, BW1, BW2, G11, G12, G21, G22
2      , ITANDM
C
      COMMON IDAT, JUNK(4), IPRM(5)
      REAL LP, L1, L2
C
      EQUIVALENCE (ISTRK, IPRM(1)), (ISECT, IPRM(2)),
1      (INQ, IPRM(3)), (IBL, IPRM(4))
C
      EQUIVALENCE (W21, X(1)), (W, X(2)), (L2, X(3)),
1(X11, X(4)), (X12, X(5)), (Y11, X(6)), (Y12, X(7)),
2(D1, X(8)), (W3, X(9)), (W23, X(10)), (X21, X(11)),
3(Y21, X(12)), (X22, X(13)), (Y22, X(14)), (X23, X(15)),
4(Y23, X(16)), (B2, X(17)), (CB2, X(18)), (W11, X(19)),
5(L1, X(20)), (B1, X(21)), (CB1, X(22))

```

```

C
C
C
C COMPUTE THE CENTROID OF THE FILL BENCH
C
      COTTS = COTPHI + COTT
      DF = SQRT(4.*ARTOT/(2.*COTTS-COTTS**2*TANT))
      D2P = D2HAT
      IF (IFNDR .EQ. 1) D2P = D2
C
C CHECK IF DF IS > D2HAT -- IF IT IS WE MUST
C FIND W23
C
      IF (DF .LE. D2P) GOTO 100
      DF = D2P
      AQ = .25*TANT
      BQ = .5*COTTS*DF*TANT
      CQ = -.5*COTTS*DF**2 + .25*(COTTS*DF)**2*TANT
      1+ ARTOT
      RNUM=BQ - SQRT(BQ*BQ - 4.*AQ*CQ)
      W23=RNUM/(2.*AQ)
C
C NOW DERIVE YB23 FOR FILL BENCH
C
100   C2TMP = COTTS*DF
      C1TMP = (COTT - COTPHI)*DF
      ARTOT1 = .5*C1TMP*DF - .25*TANT*C1TMP**2
      ARTOT2 = ARTOT - ARTOT1
C
      YB231 = (.1666*COTPHI**2*DF**3+.125*C2TMP*TANT
1      *C1TMP**2-.1666*COTPHI**2*DF**3)/ARTOT1
2      - COTPHI*(D2P-DF)
      YB232 = (.125*(C2TMP**3*TANT-(C2TMP-W23)**3
1*TANT))/ARTOT2
      FYB23 = (YB231*ARTOT1 + YB232*ARTOT2)/ARTOT
      RETURN
      END

```

```

C
SUBROUTINE AHAT (AREA,WC,AHATT,WPRIM)
C
  DIMENSION XX(16), XX1(24,16), XX2(19,16), R(40),
&SUM(24), FF(19), SR(19), ROLD(40), H(16)
C
  COMMON NX,NC,NIC,STEP,ALFA,BBETA,GAMA,IN,INF,
1FDIFER,SEQL,K1,K2,K3,K4,K5,K6,K7,K8,K9,XX,XX1,
&XX2,R,SUM,FF,SR,ROLD,SCALE,FOLD,
2LFEAS,L5,L6,L7,L8,L9,R1A,R2A,R3A,
3D, T, LP, WP, APROD, F, V, CC2, CF,
&DPRICE, PHI, THETA, BETA,
4CZ, CR, BW, W2, DELTA, C, D2, X(22),
&IXPTR(22), IMDL, RLOC(34),
5RHO,WPIT, OPHRS, IFNDR, CASTD, SAFE, IOBJ,
&IVAR, HOURS, CTON,
C
6A1A,A1B,A2,A3A,A3B,A4, COSB,COTDEL,COTPHI,
&COTT,TANT,D2P,D2HAT,
7G1B,G2,G3B,G4, REACH, XLAST, YLAST, YB4
C
  COMMON A11S,A12S,A21S,A22S,ARTOT,AR11,AR12,
&DF,REACH1,REACH2,
1      C1,C2,CR1,CR2,BW1,BW2, G11,G12,G21,G22
2      , ITANDM
C
  COMMON IDAT,JUNK(4),IPRM(5)
  REAL LP, L1, L2
C
  EQUIVALENCE (W21,X(1)),(W,X(2)),(L2,X(3)),
1(X11,X(4)),(X12,X(5)),(Y11,X(6)),(Y12,X(7)),
2(D1,X(8)),(W3,X(9)),(W23,X(10)),(X21,X(11)),
3(Y21,X(12)),(X22,X(13)),(Y22,X(14)),(X23,X(15)),
4(Y23,X(16)),(B2,X(17)),(CB2,X(18)),(W11,X(19)),
5(L1,X(20)),(B1,X(21)),(CB1,X(22))
C
C
C
  AHATT = 0.0

```

```
      ATEMP = (WC**2*TANT)*.25
      IF (ATEMP .LT. AREA) AHATT = 2.*AREA*COTT/WC
      1- .5*WC
C
C CALCULATE WPRIM
C
      WPRIM = SQRT(4.*AREA*COTT + AHATT**2)
      RETURN
      END
```



```

C
C
C      SUBROUTINE SPCNF(AREA,AHATT,WPRIM,DELC)
C
C      DIMENSION XX(16), XX1(24,16), XX2(19,16), R(40),
&SUM(24), FF(19), SR(19), ROLD(40), H(16)
C      DIMENSION IBUFR(1)
C
C      COMMON NX,NC,NIC,STEP,ALFA,BBETA,GAMA,IN,INF,
1FDIFER,SEQL,K1,K2,K3,K4,K5,K6,K7,K8,K9,XX,XX1,
&XX2,R,SUM,FF,SR,ROLD,SCALE,FOLD,
2LFEAS,L5,L6,L7,L8,L9,R1A,R2A,R3A,
3D, T, LP, WP, APROD, F, V, CC2, CF,
&DPRICE, PHI, THETA, BETA,
4CZ, CR, BW, W2, DELTA, C, D2, X(22),
&IXPTR(22), IMDL, RLOC(34),
5RHO,WPIT, OPHRS, IFNDR, CASTD, SAFE, IOBJ,
&IVAR, HOURS, CTON,
C
C      6A1A,A1B,A2,A3A,A3B,A4, COSB,COTDEL,COTPHI,
&COTT,TANT,D2P,D2HAT,
7G1B,G2,G3B,G4, REACH, XLAST, YLAST, YB4
C
C      COMMON A11S,A12S,A21S,A22S,ARTOT,AR11,AR12,
&DF,REACH1,REACH2,
1      C1,C2,CR1,CR2,BW1,BW2, G11,G12,G21,G22
2      , ITANDM
C
C      COMMON IDAT,JUNK(4),IPRM(5)
C      REAL LP, L1, L2
C
C
C      D2HAT = D2 + T
C      CC = 0.0
C      IF (1FNDR .EQ. 1) CC = T*(COTT+COTPHI)
C
C      C:ASSUME AHAT=0.0
C      SUB1 = (G11-D2HAT*COTPHI + CC)
C      SUB2 = .5/(COTT+COTPHI)
C      AQ = TANT - SUB2

```

```

BQ = 2.*SUB1*TANT - 2.*CC*SUB2
CQ = SUB1**2*TANT - SUB2*CC**2 - AREA
DELC = FNCQD(AQ,BQ,CQ)

```

C

```

AHATT = 0.0
WPRIM = 2.*(G11-D2HAT*COTPHI+CC+DELC)
DF = (CC+DELC)/(COTT+COTPHI)

```

C

C NOW DETERMINE IF SETTING AHAT = 0.0 WAS VALID

C

```

IF (.5*WPRIM+G11 .LE. D2HAT*COTPHI+W) GOTO 999

```

C

C OTHERWISE WE MUST DETERMINE THE VALUE OF AHAT

C

```

SUB1 = G11-D2HAT*COTPHI+CC
SUB2 = 2.*G11-2*D2HAT*COTPHI+CC-W
SUB3 = (COTT+COTPHI)*2.
AQ = TANT-1./SUB3-.25*TANT
BQ = 2.*SUB1*TANT-.5*SUB2*TANT-2.*CC/SUB3
CQ = SUB1**2*TANT-.25*SUB2**2*TANT-CC**2/SUB3-AREA
DELC = FNCQD (AQ,BQ,CQ)

```

C

```

AHATT = 2.*G11-2.*D2HAT*COTPHI+CC+DELC-W
WPRIM = 2.*(G11-D2HAT*COTPHI+CC+DELC)
DF = (CC+DELC)/(COTT+COTPHI)

```

C

999 RETURN

END

```
C
C THIS FUNCTION APPLIES THE QUADRATIC FORMULA
C
C      FUNCTION FNCQD(AQ,BQ,CQ)
C
C      SQROOT = SQRT(BQ**2-4.*AQ*CQ)
C      DENOM = 2.*AQ
C      FNCQD = (SQROOT-BQ)/DENOM
C      RETURN
C      END
```

REFERENCES CITED

REFERENCES CITED

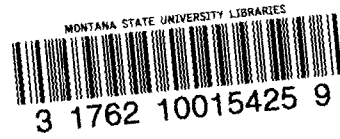
1. Barron, K. and Tomica, J., "Mineability of Saslatchewa Lignite Reserves: Report 1 - Dragline Selection, Capital and Operating Costs", Mining Research Laboratories Western Office, May 1977.
2. Cook, F. and Cohen, P., "A Mathematical Model of Dragline Side-cast Operation," Working Paper No. 3, March 1976.
3. Dunlap, J.W. and Jacobs, H.J., "How Operations Research Solved the Dragline Problem," Engineering and Mining Journal, Vol. 156, No. 8, August, 1955.
4. Finch, T.E. and Speake, C.J., "Calculating Dragline Reach Requirements for Western Surface Mines," Mining Engineering, May 1977.
5. Finch, T.E. and Dale, R.T., "Alternate Methods for Handling the Parting Between Two Coal Seams," USBM Contract Report 60254004.
6. Flour Utah, "Economics of Large Scale Surface Coal Mining Using Simulation Models," Final Report, Vol. 10, 1977.
7. Gibson, D.F. and Lehman, T.E., "Productivity Improvement of Surface Coal Mining Operations," Proceedings - AIIE Spring Annual Conference, May 1977.
8. Gibson, D.F., Lehman, T.E., and Mooney, E.L., "Interactive Computerized Planning of Surface Coal Mining,"
9. Himmelblau, D.M., Applied Nonlinear Programming, McGraw-Hill Book Company, Copyright 1972.
10. Mathematica, Inc., "Evaluation of Current Surface Coal Mining Overburden Handling Techniques and Reclamation Practices," Prepared for Bureau of Mines, December 24, 1976.
11. Mathematica, Inc., "Working Papers and Computer Program Listing for Dragline Model," Prepared for U.S. Bureau of Mines, March 1, 1976.
12. Mathematica, Inc., Results of a Study of Overburden Handling Techniques and Reclamation Practices at Western U.S. Surface Coal Mines, Princeton, New Jersey, 1975.

13. Morris, C.S. and Keller, K.J., "Development of Operational Aids for Improved Dragline Operations," Final Report to U.S. Dept. of Interior Bureau of Mines by McDonnell Douglas Corporation, Vol. 3, 1978.
14. Mudd, Seeley, W., "Surface Mining," AIME, 1968.
15. Rumfeldt, H., "Computer Methods for Estimating Proper Machinery Mass for Stripping Overburden," Mining Engineering, May 1961.
16. Sheridan, D., "A Second Coal Age Promises to Slow Our Dependence on Imported Oil," Smithsonian, August 1977.
17. Stefanko, R., Ramini, R.V., and Ferko, M.R., "An Analysis of Strip Mining Methods and Equipment Selection," Office of Coal Research, U.S. Dept. of Interior R & D Report No. 61, May 1973.
18. Winczewski, L.M., "Western North Dakota Lignite Strip Mining Processes and Resulting Subsurface Characteristics," University of North Dakota, Graduate Thesis, 1977.

BIBLIOGRAPHY

BIBLIOGRAPHY

- Butler, J.M., and Fonts, R.K., "Optimizing Open Pit Mining with Computer Simulation," Mining Congress Journal, March 1975.
- Chatterjee, P.K., Rowland, D., and Siller, K.C., "Simulation of Dragline Operations," Working Paper No. 3, March 1976.
- Finch, T.E., and Haley, D.R., "Calculating Dragline Reach Requirements for Western Surface Mines," Mining Engineering, May 1977.
- Gibson, D.F., Mooney, E.L., and Sattoriva, G.A., "A Minicomputer System for Planning Surface Mining Operations," Proceedings - AIIE Systems Engineering Conference, November 1977.
- Luke, K.W., "Functional Optimization of Open Pit Mine Design Utilizing Geologic Cross-Section Data," SME, AIME, June 1972.
- Morris, C.S. and Keller, K.J., "Development of Operational Aids for Improved Dragline Operation," Final Report to U.S. Dept. of Interior Bureau of Mines by McDonnell Douglas Corporation, Vol. 3, 1978.
- Phillips, D.A., "Optimum Design of an Open Pit," South African Institute of Mining and Metallurgy, 1973.



N378

Sa84

cop.2

Sattoriva, Glen Arthur
Productivity improve-
ment model for planning
two pass dragline opera-
tions

DATE	ISSUED TO
	William J Hornby
	Box 217 (2)
	Gralliation Center
	764-966
R	
W. J. Hornby	
Zuker	

N378
Sa84
cop 2