A computer aided mine planning model and energy efficiency ratios of four mining systems

Thomas S. Colvin
Iowa State University

Follow this and additional works at: https://lib.dr.iastate.edu/rtd
Part of the Mining Engineering Commons, and the Oil, Gas, and Energy Commons

Recommended Citation
https://lib.dr.iastate.edu/rtd/6062

This Dissertation is brought to you for free and open access by the Iowa State University Capstones, Theses and Dissertations at Iowa State University Digital Repository. It has been accepted for inclusion in Retrospective Theses and Dissertations by an authorized administrator of Iowa State University Digital Repository. For more information, please contact digirep@iastate.edu.
INFORMATION TO USERS

This material was produced from a microfilm copy of the original document. While the most advanced technological means to photograph and reproduce this document have been used, the quality is heavily dependent upon the quality of the original submitted.

The following explanation of techniques is provided to help you understand markings or patterns which may appear on this reproduction.

1. The sign or “target” for pages apparently lacking from the document photographed is “Missing Page(s)”. If it was possible to obtain the missing page(s) or section, they are spliced into the film along with adjacent pages. This may have necessitated cutting thru an image and duplicating adjacent pages to insure you complete continuity.

2. When an image on the film is obliterated with a large round black mark, it is an indication that the photographer suspected that the copy may have moved during exposure and thus cause a blurred image. You will find a good image of the page in the adjacent frame.

3. When a map, drawing or chart, etc., was part of the material being photographed the photographer followed a definite method in “sectioning” the material. It is customary to begin photoing at the upper left hand corner of a large sheet and to continue photoing from left to right in equal sections with a small overlap. If necessary, sectioning is continued again — beginning below the first row and continuing on until complete.

4. The majority of users indicate that the textual content is of greatest value, however, a somewhat higher quality reproduction could be made from “photographs” if essential to the understanding of the dissertation. Silver prints of “photographs” may be ordered at additional charge by writing the Order Department, giving the catalog number, title, author and specific pages you wish reproduced.

5. PLEASE NOTE: Some pages may have indistinct print. Filmed as received.

University Microfilms International
300 North Zeeb Road
Ann Arbor, Michigan 48106 USA
St. John's Road, Tyler's Green
High Wycombe, Bucks, England HP10 8HR
COLVIN, Thomas S., 1947-
A COMPUTER AIDED MINE PLANNING MODEL AND
ENERGY EFFICIENCY RATIOS OF FOUR MINING
SYSTEMS.

Iowa State University, Ph.D., 1977
Engineering, mining

University Microfilms International, Ann Arbor, Michigan 48106

© 1977
Thomas S. Colvin
All Rights Reserved
A computer aided mine planning model and energy efficiency ratios of four mining systems

by

Thomas S. Colvin

A Dissertation Submitted to the Graduate Faculty in Partial Fulfillment of The Requirements for the Degree of DOCTOR OF PHILOSOPHY

Major: Agricultural Engineering

Approved:

Signature was redacted for privacy.

In Charge of Major Work

Signature was redacted for privacy.

For the Major Department

Signature was redacted for privacy.

For the Graduate College

Iowa State University
Ames, Iowa

1977

Copyright © Thomas S. Colvin, 1977. All rights reserved.
TABLE OF CONTENTS

PART I. INTRODUCTION

Objectives of the Research 4

PART II. COMPUTER AIDED MINE PLANNING MODEL (CAMP)

Introduction 6a

Application of the Model to the Iowa Coal Project
Demonstration Mine #1 7b

Input Data Required 17

Description of Program 22b

Main program 22b
Subroutine CUTVOL 22b
Subroutine FINVOL 29
Subroutine MINE 31
Subroutine RDFNVL 31
Subroutine SEQNC 34
Subroutine UNCVOL 38
Subroutine MONEY 38

Comparisons of CAMP to Other Estimating Systems Used
at ICPDM#1 40

PART III. ENERGY EFFICIENCY COMPARISON OF FOUR MINING SYSTEMS
FOR A HYPOTHETICAL MINE 46

Reason for Comparison 46

Review of Literature 46

Description of the Hypothetical Mine Site 49

Description of the Four Mining Systems in This Analysis 49

Results of Energy Efficiency Calculations 51

PART IV. SUMMARY AND CONCLUSIONS 63a

LIST OF REFERENCES 64

APPENDIX A. ORIGINAL MINE PLAN FOR ICPDM#1 68
APPENDIX B. LISTING OF CAMP AND SAMPLE OUTPUT 87

APPENDIX C. LISTING OF CALCULATIONS FOR ENERGY EFFICIENCY INFORMATION IN PART III 141

APPENDIX D. EXPLANATION OF SECONDARY SUBROUTINES 150

APPENDIX E. GLOSSARY 158
PART I. INTRODUCTION

In order to justify the continued strip mining of coal, at least two questions must be answered:

1. Is it economically profitable, considering both immediate and delayed costs of both a direct and indirect nature?
2. Does it produce more energy than it consumes?

The first question can be approached using cost accounting techniques with the addition of social costs and benefits such as changes in productivity of land and social disruption. It is certainly not true that all changes to land result only in costs. For example, the addition of tile to agricultural land can change the naturally occurring drainage patterns, but these changes have in many cases improved the crop production potential of that land.

The second question requires an analysis of energy inputs and outputs before an answer can be given. In some cases the market system can provide most of the answer in an indirect manner and it may not be feasible to make comparisons between alternate energy production systems such as tertiary crude oil recovery and coal strip mining. However, because the operation is energy production, it is well to at least have an indication of the energy input-output relationship.

Strip mining or surface mining techniques have been used to produce coal, metallic and nonmetallic ores, clay, sand, and many other products. Early in man's history rocks and clay were gathered or dug from near the surface of the earth in what might loosely be called surface mines.
In the United States, commercial coal strip mines have been in production since at least the middle of the nineteenth century. These early coal strip mines relied on animal and human power to uncover shallow seams of coal in Illinois and Kansas. Machinery for strip mining has evolved in several directions with some mines currently using draglines with up to 200 yd\(^3\) buckets and others using conventional roadbuilding equipment.

In 1936 six percent of the coal that was produced in the United States was mined from the surface. This had grown to 40 percent by 1970 and was estimated to be at least 50 percent by 1977. The majority of coal produced in Iowa in the late nineteenth and early twentieth centuries came from underground mines, but roof conditions were a major problem. Many of the mines were shallow with similar areas being mined from the surface after the middle of the twentieth century.

Strip mines have achieved a higher output per man-hour with a better safety record than many underground mines as shown in Cummins and Given (1973). The development or lead time for strip mines is shorter than for underground mines with specialized equipment not as much of a necessity.

The United States Bureau of Mines has estimated that between 1.8 and \(10 \times 10^8\) short tons of strippable coal within 100 feet of the surface remain in Iowa. Much of this coal does not have good material for a mine roof over the coal or is in deposits that are not large enough to justify underground mining.

It appears that strip mining will grow in importance if coal production is to be expanded on short notice but because the majority of reserves are deeper than can be economically strip
mined at this time, the distant future will see underground mining or some type of in situ use for coal grow in importance.

C and K Coal Company, Clarion County, Pennsylvania produced much of its coal in 1975 from many small separated pits with some leased areas containing less than 20 acres. Mobile equipment such as front end loaders and bulldozers was used in many of these small pits.

The Iowa Coal Research project was funded by the State of Iowa in 1974 in an attempt to revitalize the coal mining industry in Iowa. At that time no coal preparation plants were in use in Iowa and the primary stripping tools were small draglines.

The coal project had two major parts with activities in coal mining and coal refining. The coal mining part of the project developed a surface mine called the Iowa Coal Project Demonstration Mine #1 (ICPD#1) between Oskaloosa and Bussey, Iowa. A major goal at the mine was to restore the surface to a configuration that would allow post mining crops to equal or exceed the productivity of crops grown on the site before it was mined.

Contracts were signed in August, 1977 between an Iowa electric utility and three coal companies that should double Iowa's coal production. The companies will probably use mining methods and coal refining techniques similar to those used by the Iowa Coal Project which indicates that the project may have had an impact on the Iowa coal industry.
Objectives of the Research

Specific objectives included:

1. To develop a computer model that would aid mine planners in developing mining plans for small mines including estimates of overburden volumes, coal amounts, cash flows, storage area volumes, mobile equipment routes, and rehandle percentages for various pit configurations.

2. To apply the model to ICPDM#1 to verify the approach and assumptions of the model.

3. To determine the energy efficiency ratios of four specific mining systems for a hypothetical mine similar to ICPDM#1.

Engineers planning the earthmoving operations at ICPDM#1 often faced questions of "What if?" such as "What if we don't rent additional area for stockpiles?" or "What if we change the sizes of the pits?"

Providing the answers to these questions proved to be very time consuming and the massive number of calculations required allowed cases of human error to develop with little opportunity for double checking.

This spawned the need for a computer model framework that could be used to quickly and accurately develop answers to management questions during mining as well as being a tool for mine planning. An inexpensive model was required so that owners of small mines could either use it directly or have access to it through an organization like the Cooperative Extension Service.
Part II of this dissertation explains how the computer aided mine planning model (CAMP) can be applied to a mine site to make volume calculations that will help answer some of the "What if?" questions during initial mine planning or when changes to the plan must be considered after the mine has been started.

Another use for the model is to generate output which can be used as input for other computer model systems. CAMP can be used to simulate mining of a pit in stages and generate a description of the path that scrapers or other mobile equipment would travel in removing the overburden.

To provide some answers to the question, "What is the energy efficiency of the mine?", energy efficiency calculations were made. ICPEM#1 used roadbuilding equipment to remove the material over the coal rather than the traditional dragline used almost exclusively in Iowa strip mines before 1974.

Part III of this dissertation presents the results of energy efficiency calculations made for four mining systems at a hypothetical mine site. These calculations were made before CAMP was completed because the majority of the calculations were equipment oriented.

Part IV is a combined summary of CAMP and the energy efficiency ratio calculations.
PART II. COMPUTER AIDED MINE PLANNING MODEL (CAMP)

Introduction

The currently active surface coal mines in Iowa are primarily small operations with several being family businesses. Sendlein and Johnson (1976) reported that the average size surface mine site in Iowa was 38.3 acres. The primary overburden handling system has included bulldozers and draglines with some use of tractor-pulled or self-propelled scrapers on unconsolidated material. In at least one of the recently opened mines, the primary stripping machines are rubber-tired scrapers which allow much more freedom in choosing a mine plan than the dragline. The dragline can move material a fixed maximum distance while the scrapers, or mobile equipment, have much more flexibility because they can move material a variable distance with much longer distances than draglines.

Stefanko et al. (1973) reported on procedures available to size draglines and shovels to specific overburden sections and to calculate the output of the chosen machine. By using the same procedures the maximum capability of a given machine can be calculated.

Weis (1973a, 1973b) are examples of a dragline and shovel manufacturer's information that can be used by prospective customers to aid in choosing the proper equipment for their operations.

Mobile equipment manufacturers provide information such as TEREX (1974) and IH (ca. 1970) that can be used to determine the production rate of mobile equipment. Many mobile equipment manufacturers also
have computer programs to help analyze specific jobs given information on haulroads and loading and dumping conditions.

All of the above planning procedures require that information about the site such as the volume of material in the site or various parts of the site be known. For the mobile equipment the route information such as the length of segments of the haulroad and their slopes must be known.

Computer aided systems have been developed to calculate volumes and stripping ratios of large mining projects. Hodge (1976) has a commercially available service to do subsurface analysis that has been used on areas ranging in size from a few acres to many square miles. Cummins and Given (1973) give an example of a computer aided system particularly suited to metal mines.

Most Iowa miners have done little overall mine planning from the standpoint of designing specific pits. The variability of the coal, the lack of strict reclamation laws, and the lack of stability without long-term contracts have tended to cause this lack of formal planning. In the future more planning will be required to meet state and federal mine reclamation laws and to help meet coal supply contracts.

This computer aided mine planning model was originally developed based on the planning requirements of the scraper-dozer operation at ICPDM#1. The volume calculation portion of the model can be used with many other equipment sets ranging from conveyor belts to draglines to bulldozers as an aid to production planning.
Another application for the volume calculation segments is the
determination of the feasibility of planned final topographies. This
is done by calculating the initial volume and the volume under a pro-
posed final surface and comparing them to see if the final surface
must be adjusted to provide the correct final volume for overburden
materials.

The development of CAMP (Computer Aided Mine Planning Model) was
not completed until the mining operation at ICPDM#1 was nearly finished
so the examples of applications of the model to ICPDM#1 show how the
model could have been used if it had been available.

CAMP would have been used to generate the original mine plan
for the ICPDM#1. The actual plan is shown in Appendix A as developed
by hand. The model would also have helped to change the original
plan as the mining progressed and changing conditions were encountered
such as a change in the production rate of coal.

CAMP was designed so that it could be used as a complete unit or
split up and only needed subroutines executed. The subroutines can be
separated and combined with a different main control program to form a
smaller system if desired. Complete descriptions of the subroutines
are provided in the description of the program section. Appendix E
is a glossary of variables and subroutine names and gives a brief
description of each.

CAMP was primarily designed to calculate the amount of overburden
above coal for user specified pits, either as a total amount or split
into two volumes for layers such as unconsolidated and consolidated
material. From a user defined coal seam thickness, the model calculates
the amount of coal present by pit and then calculates the strip ratio by pit. CAMP can calculate the volume of fill material needed between a user designed final topography and the bottom of the coal seam. Secondary subroutines were developed to work with storage areas and equipment paths.

Application of the Model to the Iowa Coal Project Demonstration Mine #1

After it was determined that there was coal at the proposed site of ICPDM#1 the first task was to estimate the volume in bank cubic yards (BCY) of overburden over the coal or volume of overburden in place, the amount of coal, and the strip ratio. The strip ratio is the ratio of the cubic yards of overburden to the tons of recoverable coal. The information required to make these estimates was a map of the surface topography and information from bore holes to determine the thickness of the overburden and the location and thickness of the coal seam. CAMP is designed to handle one seam of coal but overall information can be obtained for multiple seam situations by using the thickness of overburden down to the bottom seam and estimating the amount of coal in upper seams by hand or by defining the overburden above the top seam as the top layer of material.

The surface map that was used for original estimates was developed with a plane table and alidade by coal project personnel. A map of the original surface of ICPDM#1 is shown in Figure 2.1. For some areas
United States Geological Survey maps have sufficient surface detail and can be used directly.

To prepare the surface information for the model the grid points must be located on the surface map. This can be done by drawing lines at 50 ft intervals both horizontally and vertically on the map directly or on an overlay for the map. The lines must be spaced so that they represent 50 ft intervals at the map scale. The elevations at the points where the lines cross (grid points) are then written on sheets to be keypunched or punched directly.

The origin of all the maps is the northwest or upper left hand corner and there should be 25 grid points across and 25 rows of grid points from top to bottom to completely fill a 40 acre site with strips on the edges to provide stability to adjacent property. This size was based on the average mine size in Iowa. Each row of values is punched on a separate card. When contour lines do not fall directly under a grid point, the elevation must be estimated by interpolation. An example of the first row of information on the map, on a coding sheet, and on a card is shown in Figure 2.2.

When a two-layer system is to be run, the unconsolidated layer must be defined. Figure 2.3 gives an example of developing bore hole information to the point where it can be input as a matrix of information on an unconsolidated layer. Care must be taken to make sure that corresponding grid points on all maps are at the same location. This can be done by working from a base line on all of the maps.
Figure 2.1. Original contours at ICPIM#1
Sample of the original surface at ICPDM#1

Portion of an 80 column data sheet

last 4 columns for card ID

Sample input cards

Figure 2.2. Preparation of surface map information for computer input on cards
At the ICPEM#1 the amount of unconsolidated material (soil) above the consolidated material (rock) over the coal seam was originally determined by extrapolating information from 8 cored holes distributed through the area underlain by coal. It was decided that an attempt would be made to keep all of the unconsolidated material separated from the consolidated material and return it in a layer over the consolidated material. The volume of unconsolidated material available before mining was estimated to allow the placement of a uniform 10 foot deep layer of unconsolidated material over the site during pit filling.

Information from bore holes can be used to prepare a map of the surface of the coal. The detail of the coal surface will depend on the required accuracy as well as the amount of information available.

The model only requires information for the coal surface at grid points where the pit location map indicates that there is coal. The grid points that are not over coal can be left with no information in the coal surface matrix.

The other information required to make an initial volume of overburden calculation is a matrix defining which points are over coal and which are not. Integers from 1 to 99 can be used to designate pits, and 0 or integers from 101 to 199 can be used for points not over coal. Points that are given a 0 will be ignored in all calculations while points that are in storage areas off of the coal that are used to store overburden during mining can be designated with numbers from 101 to 199.
The depth in hole 1 from the surface to consolidated rock material is 20 ft.

The depth in hole 2 from the surface to consolidated material is 16 ft.

Figure 2.3. Development of bore hole information by interpolation to give the thickness of unconsolidated material.
All points that are in a single pit which will be handled as a unit should have the same number as shown in Figure 2.4. The pits should be numbered sequentially starting with 1. When only a total volume for an area is desired all the points to be included should be given a 1 in the pit location map.

The variable CVOLUM, which controls the selection of subroutines, should be input as a 1 when the basic information is to be used to indicate that there is information for only one layer above the coal. When information is available to locate the boundary between unconsolidated and consolidated material (soil one layer above the coal). When information is available to locate the boundary between unconsolidated and consolidated material (dirt and rock) and the two layers are to be calculated separately CVOLUM should be input as a 2. The information on the boundary should be prepared like the information for the surface or coal surface (see the description of UNCVOL for further information).

Figure 2.5 shows the output of CAMP when it was used to estimate the total volume over coal at ICPDM#1. The output for the single layer was within three tenths of one percent of the volume calculated by a method using cross sections and the average end area method of volume calculation.

These volume calculations are not dependent on the type of machines to be used to mine the property. The pit sequencing subroutines MINE and SEQNC are primarily designed for scrapers, loaders, or similar mobile machinery which can load overburden and carry it to...
Figure 2.4. A portion of a pit map showing the use of pit numbers to define pit areas
TOTAL SITE VOLUME OVER COAL IN CUBIC YARDS 1769755.00

VOLUME IN PIT 1 IN CUBIC YARDS IS 25647.42
VOLUME IN PIT 2 IN CUBIC YARDS IS 102581.80
VOLUME IN PIT 3 IN CUBIC YARDS IS 129532.80
VOLUME IN PIT 4 IN CUBIC YARDS IS 150272.80
VOLUME IN PIT 5 IN CUBIC YARDS IS 166566.40
VOLUME IN PIT 6 IN CUBIC YARDS IS 198697.30
VOLUME IN PIT 7 IN CUBIC YARDS IS 235733.10
VOLUME IN PIT 8 IN CUBIC YARDS IS 364062.40
VOLUME IN PIT 9 IN CUBIC YARDS IS 188789.80
VOLUME IN PIT 10 IN CUBIC YARDS IS 86941.69
VOLUME IN PIT 11 IN CUBIC YARDS IS 54165.10
VOLUME IN PIT 12 IN CUBIC YARDS IS 66664.75

Figure 2.5. Sample cut volume output
a designated location. The rehandle that is calculated is the volume of material that must be handled more than once divided by the original volume of material.

When a final surface is to be designed for the mine the volume under that surface can be calculated using the subroutine FINVOL for one layer and FINVOL and RDFNVL for two layers of fill. The information is prepared for keypunching the same as the information for the original surface except that the information will probably be taken from an engineering design.

At ICPEM#1 it was decided that a system of bench terraces shown in Figure 2.6 would provide for good agricultural production potential while limiting the erosion hazard of the site following mining. A preliminary final surface plan was drawn, and then the volume between that final surface and the bottom of the coal seam was calculated and compared to the estimated cut volume. Using the results of the comparison, another plan was drawn and the process repeated until an acceptable plan that provided the proper volume was developed. The subroutine FINVOL would have been used in this process because more time was required to check a plan than to develop it particularly for major revisions.

If the final surface is to be similar to the original surface, the matrix for the original surface can be modified and used as a separate input to define the final surface. A definition of a final surface is required when the pits are to be cycled and rehandle calculated.

Appendix B contains a complete run of the model for two layers of overburden.
Figure 2.6. Cross section of bench terrace system designed for ICPCM#1
Input Data Required

Table 2.1 defines the input stream of cards required by the CAMP system. Different choices of options require portions of the total stream to be included. The base of information is a 50 ft by 50 ft square centered on a grid point within the site being considered. The grid points are locations on a 25 by 25 two dimensional matrix.

Table 2.2 shows numbers and their location in the input stream of control variables that could be used to obtain a cut volume estimate when the planner wants to consider all overburden above coal as one layer. The groups of 25 cards are the map information discussed in the previous section. Table 2.3 shows the input stream needed to obtain cut volume estimates when the overburden is to be split into two layers.

Table 2.4 is an example of input that could be used to obtain an estimate of volume under a final surface plan with one layer. A two-layer system would be the same except that the value of CVOLUM would be 2.

Each card of map information holds the information for the 25 points in one row of the matrix. Three digits are permitted for each point. Elevations over 999 must be scaled by a constant factor to fall under 1000. If all three spaces are not required then blanks must be left as shown in Figure 2.7. No decimal points are used. Elevations are checked by the program to make sure that they are not zero or negative.
Table 2.1. Designation of input stream for all primary options

First card                      Format free
/ CVOLUM ISURF DIRT COAL COALDP SWELL COVDEP NUMPIT NUMWAY

Second card
/ HRPROD YDCOST COALPR WXHRS FVOLUM

Starting with third card, use number of cards called for by NUMWAY with maximum use of 10 cards in that specific group, format free. One card for each mining order.
/ Number string to indicate order of pits

Next 25 cards                  Format 25F3.0
/ Surface elevations

Next 25 cards                  Format 25F3.0
/ Coal top elevations

Next 25 cards                  Format 25I3
/ Pit designation

Next 25 cards                  Format 25F3.0
/ Unconsolidated layer definition if required

Next 25 cards                  Format 25F3.0
/ Final surface elevations
Table 2.2. Example of input required for cut volume estimates with no additional output for one layer

<table>
<thead>
<tr>
<th>First card in input</th>
<th>Format free</th>
<th>(decimals are important)</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 10 0 0. 0. 0. 0. 0. 1 1</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Second card</th>
<th>Format free</th>
<th>(decimals are important)</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 0. 0. 0. 0. 0.</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Third card</th>
<th>Format free</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 1</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Next 25 cards</th>
<th>Format 25F3.0</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ Surface elevations</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Next 25 cards</th>
<th>Format 25F3.0</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ Coal top elevations</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Next 25 cards</th>
<th>Format 25I3</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ Pit designations</td>
<td></td>
</tr>
</tbody>
</table>
Table 2.3. Example of input required for cut volume estimates with no additional output for two layers

<table>
<thead>
<tr>
<th>First card in input</th>
<th>Format free (decimals are important)</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 10 0 0. 0. 0. 0. 0. 1 1</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Second card</th>
<th>Format free (decimals are important)</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 0. 0. 0. 0. 0.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Third card</th>
<th>Format free</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 1</td>
<td></td>
</tr>
</tbody>
</table>

Next 25 cards Format 25F3.0
/ Surface elevations

Next 25 cards Format 25F3.0
/ Coal top elevations

Next 25 cards Format 25I3
/ Pit designations

Next 25 cards Format 25F3.0
/ Unconsolidated layer definition if required

^See UNCVOL for specific details.
Table 2.4. Example of input required for final volume estimate only with one layer

<table>
<thead>
<tr>
<th>First card</th>
<th>Format free (decimals are important)</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 1 0 0. 0. 0. 0. 0. 1 1</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Second card</th>
<th>Format free (decimals are important)</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 0. 0. 0. 0. 1</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Third card</th>
<th>Format free</th>
</tr>
</thead>
<tbody>
<tr>
<td>/ 1</td>
<td></td>
</tr>
</tbody>
</table>

Next 25 cards Format 25F3.0

/ Coal top elevations

Next 25 cards Format 25I3

/ Pit designations

Next 25 cards Format 25F3.0

/ Final surface elevations
Figure 2.7. Examples of map data cards showing the required format
Pit locations are input as integer numbers from 1 to 99. The number zero on the pit location map causes the model to ignore the point in all calculations. Storage locations are indicated on the pit location map by numbers from 101 to 199.

If all locations over coal are given the same pit number, the total volume over coal will be calculated.

The model will accept map information that indicates the top of the consolidated or rock material in the profile. When this information is input the control variable CVOLUM must be set equal to 2 so that the model will calculate output based on 2 layers of overburden over the coal.

The user has 3 choices for indicating the boundary between the unconsolidated material and consolidated material. The first choice is digitized elevations of the bottom of the unconsolidated layer. These elevations must be on the same scale as the elevations of the surface so that when the elevations are subtracted they will give the proper thickness of material. The variable DIRT must equal 0 for this choice.

Another choice is to input the thickness of unconsolidated material at each of the grid points over coal. Use of this choice causes the model to use thickness information directly when the variable DIRT is given the real value of 1.0.

The final choice is to define a uniform thickness from the surface to be considered as the unconsolidated layer with everything below in the overburden column lumped together. This is done by setting DIRT
equal to the depth desired such as 10. or 5. (a positive real number greater than 1).

The thickness of the coal seam at the site is set by inputing the value of COALDP as a single real number. By using the top of coal elevations and bottom of coal elevations the computer could calculate the volume of an irregular coal seam as a separate operation. This should be an average value for the site under consideration. COALDP is used to calculate the amount of coal and to calculate the volume of additional fill required due to the removal of the coal.

The value of SWELL which is input is used to adjust the volume of overburden due to handling. A positive value indicates swell while a negative value indicates shrink. From experience at ICPDM#1 a value of 10% swell, input as .1, is reasonably close for a scraper coal mining operation having a ratio of unconsolidated to consolidated material of .5. The amount of swell is dependent on the type of mining equipment and the ratio of consolidated material to unconsolidated material (Caterpillar, 1974).

COVDEP is depth of unconsolidated material to be put back on top of the consolidated fill. This value is input as a positive real number such as 5.0.

COAL is the variable that is used to input the density of coal in pounds per bank cubic yard. The coal at ICPDM#1 did not have a constant density but a reasonable estimate would be somewhere between 1500 and 1800 lb/BCY.
Variable NUMPIT is used to input the number of pits over coal. NUMWAY indicates the number of mining sequences to be tried on the pits during one run of the model.

HRPROD is used to indicate the average rate of overburden removal expected for the mine fleet. At ICPEM#1, the scraper fleet of 14 and 18 cubic yard machines averaged near 150 bank cubic yards per hour per machine over the life of the mine. The production ranged from less than 100 to more than 200 BCY per hour per machine. This information is used to estimate a time schedule for the mine. The entire fleet production should be used to determine HRPROD.

YDCOST is the estimated average cost of moving each BCY of overburden. This is used with the price and amount of coal to give a rough estimate of cash flows during the life of the mine. The average cost at ICPEM#1 was near $0.60/BCY, but the survey conducted by Colvin (1975) indicated that there might be a range from 50¢ to more than $2.00 per BCY. As with all other input estimates, experience is a valuable guide to reasonable numbers.

COALPR is the estimated average selling price of coal. This value is strictly current market or contract dependent.

WKHRS is the number of hours that the overburden removal equipment will be operated per week. The value should be clock hours for the mine and not a summation of individual machine hours.

The additional cost of running the model to obtain the information such as stripping ratio and estimates of time is not great. This information may prove useful even with rough estimates for the in-
puts because the output can at least indicate the order of magnitude of the operation.

The matrix LPIT allows the user to try several sequences of mining during one run of CAMP using one set of pits. All the pit numbers are placed on each card in the order of mining desired. The number of cards must equal NUMWAY, and the number of pits on each card must equal NUMPIT.

SURFC is the matrix that is used for original ground surface elevations. CSURFC is the matrix of intermediate elevations during mining. IPITS is the pit location matrix. RSURFC is the matrix of consolidated material surface elevations or the depths of unconsolidated material.

Figure 2.8 shows a basic flow chart for the model.

Description of Program

Main program

The main program is designed to choose data processing subroutines that the user requests and read most of the input data. Table 2.5 gives a skeleton listing of CAMP. Table 2.6 lists the main program variables and the effect of their values on program control.

Subroutine CUTVOL

This subroutine calculates volumes of individual 50' x 50' columns centered on the location of the grid points represented by values in the matrices. The volumes for the individual points are
Figure 2.8. Overview of Computer Aided Mine Planning Model (CAMP)
Figure 2.8. continued

- **K**: ARE OTHER OPTIONS REQUIRED
  - **L**: CALL MONEY
    - **M**: IS THERE ONE LAYER
      - **N**: CALL MINE CYCLE PITS
    - **O**: ARE MORE PIT ORDERS REQUIRED
      - **P**: CALL SEQNC CYCLE PITS
    - **Q**: ARE MORE PIT ORDERS REQUIRED
      - **R**: BASIC COMPUTER RUN IS COMPLETE

  - **L**: CALL MONEY
    - **M**: IS THERE ONE LAYER
      - **N**: CALL MINE CYCLE PITS
    - **O**: ARE MORE PIT ORDERS REQUIRED
      - **P**: CALL SEQNC CYCLE PITS
    - **Q**: ARE MORE PIT ORDERS REQUIRED
      - **R**: BASIC COMPUTER RUN IS COMPLETE
### Table 2.5. Skeleton listing of the Computer Aided Mine Planner (CAMP)

**Key to location in Figure 2.7**

<table>
<thead>
<tr>
<th>Description</th>
<th>Code</th>
</tr>
</thead>
<tbody>
<tr>
<td>READ, CVOLUM, ISURF, DIRT, COAL, COALDP, SWELL, NUMPIT, NUMWAY (1 card)</td>
<td>A</td>
</tr>
<tr>
<td>IF (FVOLUM = 1.), go to 50</td>
<td>A</td>
</tr>
<tr>
<td>READ, HRPROD, YDCOST, COALPR, WKhrs, FVOLUM (1 card)</td>
<td>A</td>
</tr>
<tr>
<td>READ, LPIT (1 card for each mining sequence)</td>
<td>A</td>
</tr>
<tr>
<td>READ, SURFC (25 cards)</td>
<td>A</td>
</tr>
<tr>
<td>50 READ, COALT (25 cards)</td>
<td>A</td>
</tr>
<tr>
<td>READ, IPITS (25 cards)</td>
<td>A</td>
</tr>
<tr>
<td>IF (FVOLUM = 1.) and (CVOLUM = 1), go to 205</td>
<td>G</td>
</tr>
<tr>
<td>IF (FVOLUM = 1.), go to 25</td>
<td>A</td>
</tr>
<tr>
<td>IF (CVOLUM = 1 or 10), system has 1 layer; go to 35</td>
<td>B</td>
</tr>
<tr>
<td>IF (CVOLUM = 2 or 20), system has 2 layers</td>
<td>A</td>
</tr>
<tr>
<td>25 READ, RSURF (25 cards)</td>
<td>A</td>
</tr>
<tr>
<td>IF (FVOLUM = 1), go to 205</td>
<td>C</td>
</tr>
<tr>
<td>CALL CUTVOL, calculate cut volumes, areas, strip ratios, and coal amounts by pit and sum</td>
<td>C</td>
</tr>
<tr>
<td>IF (CVOLUM = 2 or 20), CALL UNCVOL to split volume into consolidated and unconsolidated</td>
<td>D,E</td>
</tr>
<tr>
<td>IF (CVOLUM = 0) go to 100</td>
<td>F,G,H</td>
</tr>
<tr>
<td>205 CALL FINVOL</td>
<td></td>
</tr>
<tr>
<td>READ, FSURF (25 cards)</td>
<td>H</td>
</tr>
<tr>
<td>Calculate volume under final surface over base of coal</td>
<td>H</td>
</tr>
</tbody>
</table>
Table 2.5. Continued

Key to location in Figure 2.7

| I    | IF (CVOLUM = 2), CALL RDFNVL to split volume into consolidated and unconsolidated |
| K    | IF (FVOLUM = 1.), go to 100 |
| L    | CALL MONEY |
| L    | Calculate cash position and time required for mining |
| L    | Order pits according to information in LPIT matrix |
| M, N | IF (CVOLUM = 1), CALL MINE to cycle mine |
| M, P | IF (CVOLUM = 2), CALL SEQNC to cycle mine |
| O, Q | If (NUMPIT > 1) repeat MONEY |
| R    | 100 STOP |
Table 2.6. Control variables in main program

<table>
<thead>
<tr>
<th>Variable</th>
<th>Value</th>
<th>Effect</th>
</tr>
</thead>
<tbody>
<tr>
<td>CVOLUM</td>
<td>10</td>
<td>Cut volume for one layer only</td>
</tr>
<tr>
<td></td>
<td>20</td>
<td>Cut volume for two layers only</td>
</tr>
<tr>
<td></td>
<td>1</td>
<td>Complete run one layer</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>Complete run two layers</td>
</tr>
<tr>
<td>ISURF</td>
<td>0</td>
<td>Skip surfal</td>
</tr>
<tr>
<td></td>
<td>1</td>
<td>Execute surfal</td>
</tr>
</tbody>
</table>

then summed first by individual pit then the sums for the pits are added together to get the total volume over coal.

Variables used in CUTVOL are listed in Table 2.7 with brief explanations.

The output generated by CUTVOL includes a map of the original ground surface, a map of the top of the coal, a map of pit locations, a listing of individual pit volumes and total volume over coal, a listing of individual pit areas and total area over coal, and an isopach map of the overburden over the coal.

The subroutine checks for zero or negative values of SURFC and COALT and ignores points with those values and prints out a message indicating the bad point location.

The calculation of the volume is based on borrow pit calculations with the difference that the elevation difference is used directly instead of being averaged with its neighbors as shown in Figure 2.9.
Table 2.7. Variables in CUTVOL

<table>
<thead>
<tr>
<th>Variable</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>SURFC (I, J)</td>
<td>Elevations of the original ground at the grid points</td>
</tr>
<tr>
<td>COALT (I, J)</td>
<td>Elevations of the top of the coal at the grid points</td>
</tr>
<tr>
<td>IPITS (I, J)</td>
<td>Designation of pits with numbers 1-99 representing pits over coal and 101-199 representing storage areas</td>
</tr>
<tr>
<td>FSURFC (I, J)</td>
<td>Designated elevation of the reclaimed surface</td>
</tr>
<tr>
<td>ELDIP (I, J)</td>
<td>The elevation difference between SURFC (I, J) and COALT (I, J)</td>
</tr>
<tr>
<td>CVOL</td>
<td>The total cut volume over coal</td>
</tr>
<tr>
<td>VOL(K)</td>
<td>Individual pit and storage area volumes in cubic yards</td>
</tr>
<tr>
<td>AREA (K)</td>
<td>Individual pit and storage area areas in square feet</td>
</tr>
<tr>
<td>TAREA</td>
<td>Total area over coal in square feet</td>
</tr>
<tr>
<td>K</td>
<td>The number of pits over coal internally determined from reading IPITS. Default K = 1.</td>
</tr>
</tbody>
</table>
Modified Borrow pit

The volume of the column centered at point \( a \) is the elevation difference times 92.59 Yd.\(^3\)/ft.

---

Standard Borrow Pit

The volume of the column centered at created point \( c \) is the average elevation of points \( a, b, d, e \) times 92.59 Yd.\(^3\)/ft.

Figure 2.9. Comparison of borrow pit calculation methods
A one-foot increment of the 50 foot square column has the volume

\[(1 \text{ ft})(50 \text{ ft} \times 50 \text{ ft})/27 \text{ ft}^3/\text{yd}^3 = 92.59 \text{ yd}^3\]

The area of the pits is determined by counting the number of grid points in the individual pits and multiplying by 2500 sq ft per point.

This system employs vertical walled pits to allow for complex pit outline. For estimating purposes, using vertical sided pits is reasonable when complex outlines are possible. The entire system is designed with vertical walled pits so that the volumes from original surface to the final surface are comparable as long as the pit designations are not changed.

**Subroutine FINVOL**

This subroutine calculates the amount of fill required from the bottom of coal to the planned final elevation. This volume can be used to check the feasibility of the proposed surface and must be used as an input to the MINE subroutine.

The calculation method is the same as used in the CUTVOL subroutine with vertical sided pits and the modified borrow pit calculation approach.

The initial pit is less accurately described by the vertical-walled pits used by CAMP because the initial pit is trapezoidal. Succeeding pits tend to be parallelograms and their volumes can be more readily estimated with the vertical-walled or rectangular pits used by CAMP. Figure 2.10 compares the type of pit used by CAMP to an initial pit as it would actually be excavated.
Initial pit as dug

Initial pit volume calculated by the straight sided pit assumption if the grid were centered in the pit.

Relationship of straight sided blocks to much wider pits.

Figure 2.10. Comparison of pit cross sections
Table 2.8 lists the variables used in the FINVOL subroutine. The pit numbers must start sequentially from a positive one (+1) so that the correct number of pits can be calculated.

Subroutine MINE

This subroutine sequentially excavates the pits and fills pits already mined. It attempts to fill the last pit mined, first. If that pit will hold all of the cut then the subroutine excavates the next pit. If there is too much fill for the closest pit, then MINE checks the first pit to see if there is any storage volume available and if fill still remains the remaining pits are checked. Figure 2.11 illustrates the logic of the subroutine. Material that will not fit in any of the pits is moved to a storage area.

The subroutine updates the status of the pits after excavating each additional pit. This can be used to determine when an area would be ready for final reclamation and seeding.

Table 2.9 lists the variables in MINE and identifies them.

Subroutine RDFNVL

This subroutine splits the fill volume available under the final surface into the volume of consolidated fill and the volume of unconsolidated fill that can be accommodated under the final plan surface. Only the volume that is over coal is calculated. The volumes are calculated for each of the pits.

The volume of unconsolidated material is calculated to the specified depth of unconsolidated cover and the area of the pit as

\[ FUVOL (I) = \frac{(COVDEP \times AREA (I))}{27}. \]
Table 2.8. Variables used in FINVOL

<table>
<thead>
<tr>
<th>Variable</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>FSURFC (I, J)</td>
<td>Elevation of final surface at grid point</td>
</tr>
<tr>
<td>M</td>
<td>Dummy to get rid of extraneous data</td>
</tr>
<tr>
<td>K</td>
<td>Number of pits in mine plan</td>
</tr>
<tr>
<td>TFVOL</td>
<td>Total final volume over coal</td>
</tr>
<tr>
<td>FELDIP</td>
<td>Final elevation difference between FSURFC and COALT</td>
</tr>
<tr>
<td>N</td>
<td>Pit designation</td>
</tr>
<tr>
<td>FVOL (I)</td>
<td>Individual pit volume</td>
</tr>
</tbody>
</table>

Table 2.9. Variables in subroutine MINE

<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>FILPIT</td>
<td>Volume in pits that are not completely full</td>
</tr>
<tr>
<td>STORVL</td>
<td>Amount of fill in storage areas</td>
</tr>
<tr>
<td>L</td>
<td>Number of pits minus one</td>
</tr>
<tr>
<td>FVOL</td>
<td>Final pit volume</td>
</tr>
<tr>
<td>FILL</td>
<td>Amount of material remaining after a pit is filled</td>
</tr>
<tr>
<td>AVLSTR</td>
<td>Difference between FVOL and FILPIT</td>
</tr>
<tr>
<td>REHANDL</td>
<td>Ratio of STORVL to CVOL</td>
</tr>
</tbody>
</table>
Pit 4 has enough volume to fill pit 3 and finish filling pit 2, so fill pit 3

Pit 1 is full, check pit 2, pit 2 will hold the remainder

Excess would be taken to a storage area if any material was left after filling pit 2

Figure 2.11. Method of placing fill in pits in subroutine MINE
The volume of consolidated material is calculated by subtracting the volume of unconsolidated material from the total final volume and adding the volume of the coal that is to be removed as

\[ FRVOL (I) = FVOL (I) - FUVOL (I) = + \left[ \frac{(COALDP \times AREA (I))}{27} \right] \]

Table 2.10 identifies the variable used in RDFNVL.

Table 2.10. Variables in RDFNVL

<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>K</td>
<td>Number of pits in mine plan</td>
</tr>
<tr>
<td>FUVOL (I)</td>
<td>Unconsolidated volume in individual pit under final surface</td>
</tr>
<tr>
<td>FRVOL (I)</td>
<td>Consolidated volume in individual pit under final surface</td>
</tr>
<tr>
<td>FVOL (I)</td>
<td>Total individual pit volume under final surface</td>
</tr>
</tbody>
</table>

**Subroutine SEQNC**

This subroutine cycles the pits when two layers of material above the coal have been specified in both the original profile and under the final surface. The logic is similar to subroutine MINE with the additional capability of handling the two types of material.

The subroutine excavates the pits in the order specified and attempts to place the consolidated material in the most recently
excavated pit. If all the material will not fit, the subroutine starts with the original pit and checks for any available storage. Any material that cannot be accommodated in that way is added to the consolidated stockpile.

The unconsolidated material is moved to the pit excavated before the most recent pit if the consolidated material has been brought to final grade. If that pit is not available, it checks all of the previously mined pits starting with the original one to find room for the material. Any material that cannot be placed directly is moved to the unconsolidated stockpile.

Figure 2.12 shows the operation of the logic of placement of the unconsolidated material. See Figure 2.11 for the logic of placement of the consolidated fill.

The subroutine reports the amount of consolidated and unconsolidated fill in all previous pits after each new pit is excavated.

The rehandle that is reported at the end of the mining includes the material that must be hauled back to the final pit. The amount of material in the unconsolidated and consolidated stockpiles is added together and divided by the total original volume. The stockpiles have material that has been adjusted by the swell factor while the original volume is in bank cubic yards, but this reflects what must be rehandled and gives a common base for the comparison of plans.

Table 2.11 identifies the variables used in SEQNC.
Status of pits after excavation of pit 3

Pit 4 has enough consolidated material to fill pit 3 and fill the consolidated volume remaining in pit 2. Pit 2 does not have all its consolidated material and pit 1 is full of consolidated material so all of the remaining consolidated material from pit 4 will be moved to stockpile.

Pit 10 has all of its consolidated fill so the unconsolidated material from pit 12 will be placed in pit 10 because pit 10 will hold it all.

Figure 2.12. Method of unconsolidated fill placement in subroutine SEQNC
Table 2.11. Variables in subroutine SEQNC

<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>L</td>
<td>Number of pits minus one</td>
</tr>
<tr>
<td>RSTRVL</td>
<td>Amount of fill in the consolidated stockpile</td>
</tr>
<tr>
<td>USTRVL</td>
<td>Amount of fill in the unconsolidated stockpile</td>
</tr>
<tr>
<td>RFLPT</td>
<td>Consolidated fill in pits that are partially full</td>
</tr>
<tr>
<td>RFILL</td>
<td>Amount of consolidated fill after a pit is filled</td>
</tr>
<tr>
<td>RVLSTR</td>
<td>Difference between FRVOL and RFLPT</td>
</tr>
<tr>
<td>UVLSTR</td>
<td>Fill volume remaining in a pit</td>
</tr>
<tr>
<td>UFILL</td>
<td>Amount of unconsolidated fill after a pit is filled</td>
</tr>
<tr>
<td>UFLPT</td>
<td>Unconsolidated fill in partially filled pits</td>
</tr>
</tbody>
</table>
Subroutine **UNCVOL**

This subroutine splits the volume of the original profile into consolidated and unconsolidated segments. **UNCVOL** is called by **CUTVOL** when a two layer system is specified.

There are three routes for calculating the volume of the unconsolidated material depending on the value of DIRT. When DIRT equals 0., the subroutine assumes that elevations for the top of the consolidated material have been input through RSURF. This causes **UNCVOL** to subtract the value in RSURF from SURFC to get the thickness of the unconsolidated layer.

If DIRT equals 1., the value in RSURF is assumed to be thickness of the unconsolidated layer and is used to calculate volume directly.

A constant depth can be assumed for the unconsolidated layer by getting DIRT equal to thickness as a positive real number greater than 1.

When the thickness of the unconsolidated layer has been determined for a point it is multiplied by a constant to get the volume at that location.

Table 2.12 lists the variables used in **UNCVOL**.

Subroutine **MONEY**

This subroutine performs some intermediate calculations and provides control to allow a single map of pit locations to be mined in several ways.
### Table 2.12. Variables in subroutine UNCVOL

<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>CONVOL</td>
<td>Total consolidated volume over coal</td>
</tr>
<tr>
<td>DVOL</td>
<td>Total unconsolidated volume over coal</td>
</tr>
<tr>
<td>K</td>
<td>Number of pits</td>
</tr>
<tr>
<td>IPITS</td>
<td>Pit locations</td>
</tr>
<tr>
<td>SURFC</td>
<td>Original surface elevations</td>
</tr>
<tr>
<td>RSURF</td>
<td>Consolidated surface elevations or thickness of unconsolidated material</td>
</tr>
</tbody>
</table>
| DIRT      | Control variable  
|           | 0. means consolidated elevation in RSURF  
|           | 1. means unconsolidated thickness in RSURF |
| UVOl      | Unconsolidated volume in pits |
| RVOL      | Consolidated volume in pits |

The time in weeks required to mine the individual pits is calculated based on the volume in the pit and the input values for hourly production and hours worked per week.

A net cash position is estimated for each pit from the amount of coal, the coal price, the pit yardage and estimated cost per cubic yard.
The volume determined by CUTVOL and UNCVOL is adjusted by the swell factor and either MINE or SEQNC is called to cycle through the mining sequence.

The variables used in MONEY are listed in Table 2.13.

Comparisons of CAMP to Other Estimating Systems Used at ICPDM#1

The CAMP system was run with several sets of data from ICPDM#1. A comparison of results that could be compared with either the hand calculations or the computer-aided average end area volume calculations is presented in Table 2.14. The two computer aided estimates agree very closely as would be expected because they shared the same data base. The hand calculation of cut volume was completed before the other two estimates and had a less complete data base and may have included some human error although the calculations were checked thoroughly.

The estimates of rehandle calculated by CAMP and by hand agree very closely which indicates that the methods of calculation are similar which is reasonable because the method of rehandle calculation used by CAMP was based on the method used in determining the rehandle in the original mine plan for ICPDM#1.

Table 2.15 presents a comparison between total site volume calculation with the computer aided average end area method and CAMP. The average end area method requires an elapsed time of about one week because of the individual computer run turn around time that must occur in sequence with hand developed input, but the estimator would
<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>NUMPIT</td>
<td>Number of pits</td>
</tr>
<tr>
<td>COALU</td>
<td>Permanent record of coal in individual pits</td>
</tr>
<tr>
<td>COALWT</td>
<td>Coal weight in various pits</td>
</tr>
<tr>
<td>VOLU</td>
<td>Permanent record of original pit volumes</td>
</tr>
<tr>
<td>VOL</td>
<td>Volume in various pits</td>
</tr>
<tr>
<td>UVOLU</td>
<td>Permanent record of unconsolidated volume</td>
</tr>
<tr>
<td>UVOL</td>
<td>Unconsolidated volume in various pits</td>
</tr>
<tr>
<td>RVOLU</td>
<td>Permanent record of consolidated volume</td>
</tr>
<tr>
<td>RVOL</td>
<td>Consolidated volume in various pits</td>
</tr>
<tr>
<td>TIMWK</td>
<td>Time to mine a pit in weeks</td>
</tr>
<tr>
<td>HRPROD</td>
<td>Overburden movement rate per hour</td>
</tr>
<tr>
<td>WKHRS</td>
<td>Number of hours worked per week</td>
</tr>
<tr>
<td>CSHNET</td>
<td>Estimate cash generation for a pit</td>
</tr>
<tr>
<td>COALWT</td>
<td>Weight of coal in a pit</td>
</tr>
<tr>
<td>COALPR</td>
<td>Price of coal</td>
</tr>
<tr>
<td>YDCOST</td>
<td>Estimated average cost of moving one BCY</td>
</tr>
<tr>
<td>SWELL</td>
<td>Amount of change in volume due to handling + is swell - is shrink</td>
</tr>
</tbody>
</table>
Table 2.14. Summary of CAMP runs that could be compared with other methods of calculation using ICPI#1 data

<table>
<thead>
<tr>
<th>Type of estimate</th>
<th>CAMP Results</th>
<th>% difference from comparative information</th>
<th>Comparative information Source</th>
<th>Results</th>
</tr>
</thead>
<tbody>
<tr>
<td>Original estimate of total site volume over coal</td>
<td>1,799,100 BCY</td>
<td>+18%</td>
<td>Original hand calculations</td>
<td>1,525,418 BCY</td>
</tr>
<tr>
<td>Same</td>
<td>1,799,100 BCY</td>
<td>-0.3%</td>
<td>Average end area computer calculations</td>
<td>1,804,513 BCY</td>
</tr>
<tr>
<td>Original estimate of rehandle using pit boundaries shown in Appendix A</td>
<td>26%</td>
<td>—</td>
<td>Original hand calculations</td>
<td>26%</td>
</tr>
</tbody>
</table>

Table 2.15. Comparison of time and cost required to compute the total amount of overburden at ICPI#1 after a surface map was completed

Average end area method using a computer program developed for the Iowa Coal Project

<table>
<thead>
<tr>
<th>CAMP</th>
<th>Current Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Time</td>
<td>1 week</td>
</tr>
<tr>
<td>Keypunch and computer cost</td>
<td>$85.00</td>
</tr>
</tbody>
</table>
not have to work exclusively on the estimates during the entire week. The elapsed time for CAMP would be about one day if one person were assigned to the project full time for the day. Estimates of the time and cost involved in developing the original mine plan in Appendix A as it was done by hand are shown in Table 2.16. Three people were assigned to develop the mine plan with only one working full time, however the time and cost given are based on full time equivalents to indicate time actually worked.

Table 2.17 is an estimate of the time and cost required to develop a similar mine plan to that shown in Appendix A using CAMP. A comparison between Table 2.16 and Table 2.17 shows the contribution that CAMP can make to the process of preparing a mine plan for a small mine.

A comparison of the time required to prepare various types of estimates for a mine plan is shown in Table 2.18 for the three calculating systems that were used with ICPM data. CAMP is shown as being fast and versatile which is important because the questions that require answers during mine planning and the life of the mine can be quite varied and require quick answers to allow informed management decisions.
Table 2.16. Summary of time and cost involved in preparing the mining plan shown in Appendix A after a surface map was available

<table>
<thead>
<tr>
<th>Personnel</th>
<th>Equivalent time spent</th>
<th>Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Project engineer</td>
<td>2 months</td>
<td>$3,500.00</td>
</tr>
<tr>
<td>Designer</td>
<td>4 months</td>
<td>$5,000.00</td>
</tr>
<tr>
<td>Technicians</td>
<td>1 month</td>
<td>$1,000.00</td>
</tr>
<tr>
<td>Total</td>
<td>7 man-months</td>
<td>$9,500.00</td>
</tr>
</tbody>
</table>

Table 2.17. Estimate of time and cost involved in preparing a mine plan similar to the one shown in Appendix A with CAMP

<table>
<thead>
<tr>
<th>Personnel</th>
<th>Equivalent time spent</th>
<th>Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Project engineer</td>
<td>1 week</td>
<td>$450.00</td>
</tr>
<tr>
<td>Designer</td>
<td>2 weeks</td>
<td>$650.00</td>
</tr>
<tr>
<td>Technicians</td>
<td>1 week</td>
<td>$500.00</td>
</tr>
<tr>
<td>Total</td>
<td>1 man-month</td>
<td>$1,600.00</td>
</tr>
<tr>
<td>Computer (10 runs)</td>
<td></td>
<td>$60.00</td>
</tr>
<tr>
<td></td>
<td></td>
<td>$1,660.00</td>
</tr>
</tbody>
</table>
Table 2.18. Comparison of capabilities of three estimating systems for ICPEM#1

<table>
<thead>
<tr>
<th>Estimate required</th>
<th>Hand calculation</th>
<th>System Average end area computer program</th>
<th>CAMP</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total site volume</td>
<td>Slow</td>
<td>Fast</td>
<td>Fast</td>
</tr>
<tr>
<td>Individual pit volumes</td>
<td>Slow</td>
<td>Not available</td>
<td>Fast</td>
</tr>
<tr>
<td>Rehandle</td>
<td>Slow</td>
<td>Not available</td>
<td>Fast</td>
</tr>
<tr>
<td>Time required for mining</td>
<td>Slow</td>
<td>Not available</td>
<td>Fast</td>
</tr>
<tr>
<td>Cash flow estimate</td>
<td>Slow</td>
<td>Not available</td>
<td>Fast</td>
</tr>
<tr>
<td>Volume under final plan surface</td>
<td>Slow</td>
<td>Fast</td>
<td>Fastest</td>
</tr>
</tbody>
</table>
PART III. ENERGY EFFICIENCY COMPARISON OF FOUR MINING SYSTEMS FOR A HYPOTHETICAL MINE

Reason for Comparison

The normal criterion used to determine whether or not a mining venture should be started or continued is its financial profitability. This criterion serves a firm well in a free enterprise system and although some mining operations would not stand as separate businesses they may benefit a parent organization, allowing the continuation of the mining.

When society must make a decision such as allocation of research funds or passage of laws to favor certain activities there are other criteria which should be considered. The energy discussion in the United States has helped to define energy efficiency as a criterion that certainly should receive attention when dealing with public energy decisions.

This study was undertaken to determine the energy efficiency of the mining method used at the Iowa Coal Project Demonstration Mine #1 (ICPD#1) and to compare it to several alternatives. During the life of the mine the question of energy efficiency was asked repeatedly by visitors.

Review of Literature

Almost no published information is available which attempts to compare surface mining systems based on an energy input-output or efficiency relationship. Colvin et al. (1975) made calculations to
compare the energy efficiency of two different mining operations in western Pennsylvania. The mines were located within 20 miles of each other with one using a scraper-ripper operation and the other a 44 yd$^3$ dragline to remove overburden. Using the operating and capital energy of the systems but ignoring blasting and support energy for the dragline gave an energy input-output ratio of 48.1:1 for the scrapers and 50.8:1 for the dragline.

Clark and Varisco (1975) reported a range of 34:1 to 2.6:1 depending on energy flow boundary definitions for shale oil production in Colorado. Leach (1975) comments on Clark and Varisco's conclusions and makes strong criticisms of their boundary assumptions and lumping of energy of different qualities such as coal and electrical energy. Leach's criticism of net energy analysis as a tool for examining energy problems included the charge that it provides little useful additional knowledge not provided by other analyses.

Leach's criticisms have been considered and can be disregarded because this analysis of four hypothetical mining systems is looking at alternate but similar methods of recovering a single resource. No previous energy analysis has been made of the ICPEM#1 as of this writing.

All the systems studied in this analysis use machines which could have been manufactured by the same industrial plant and all but the 40 yd$^3$ dragline used diesel fuel for an energy source. The boundaries shown in Figure 3.1 were chosen to attempt to give an equivalent basis for comparison of the systems among themselves.
Figure 3.1. Boundary definition used in the net energy analysis of four mining systems.
Description of the Hypothetical Mine Site

The hypothetical mine site used in this analysis is based on the topographic features of ICPDM#1 shown in Figure 3.2 and the estimated volumes of consolidated and unconsolidated material for approximately 20 acres underlain by coal before mining began.

The land use prior to mining was rotation pasture and hay production with some row cropping. The area was used to winter beef animals with the pond as a water source.

The final topography after mining is assumed to be approximately the original contour for all systems, with no unconsolidated material planned for placement below or among consolidated material. The overburden depth is assumed to vary from 15 to 80 feet from the side to the top of the hill. This is a one seam operation.

Description of the Four Mining Systems in This Analysis

The scraper system is based on the fleet of machines used at ICPDM#1. This included one 14 yd$^3$ twin engine scraper, two 18 yd$^3$ twin engine scrapers and three 300 hp crawlers equipped with dozers and rippers.

The scrapers move most of the material with the dozer ripping the consolidated material and handling material that could not be loaded in the scrapers. The scrapers are normally push loaded by the dozers.

The 40 yd$^3$ dragline system uses scrapers to move all the unconsolidated material and to fill the last pit with consolidated material. The scrapers must be of an appropriate size so that the fleet can meet the production requirement at the estimated price per yd$^3$. The pit is
Figure 3.2. Original topography at ICPEM#1
assumed to average 60 ft deep by 100 ft across. Dozers would rough grade the dragline spoil piles before the scrapers replace the unconsolidated material, all of which is assumed to be stockpiled. The dragline would be scheduled for 20 hr per day, everyday except perhaps Christmas Day.

The 5 yd$^3$ dragline system operates the same as the 40 yd$^3$ dragline system except that the pit would be 40 ft wide and the dragline would not be scheduled for more than 8 hr/day. The consolidated material for both dragline systems would be drilled and shot with ammonium nitrate.

The large bulldozer system has scrapers to remove the unconsolidated material. The single large bulldozer then rips and dozes the consolidated material to uncover the coal. The dozer pushes the material back to approximate original contours before the unconsolidated material from the stockpile is replaced by the scrapers.

Results of Energy Efficiency Calculations

Table 3.1 presents a summary of the results of the four systems studied. The 40 yd$^3$ dragline has the lowest cost, the shortest time and the best energy ratio. The scraper system has the lowest overburden rehandle percentage. The amount of rehandle might be lowered for the other systems but probably would not be below 40% for this site.

The 40 yd$^3$ dragline would not be used on this site unless the site were part of a larger mine because the 40 yd$^3$ machine is expensive to move. The cost per yard might be reduced by half and rehandle should approach that of the scrapers if this site were included in a large mine.
Table 3.1. System comparison based on $1.3 \times 10^6$ bank cubic yards of overburden

<table>
<thead>
<tr>
<th>System</th>
<th>Cost/ yd$^3$ of overburden</th>
<th>Rehandle of overburden</th>
<th>Time required for mining</th>
<th>Energy ratio, out/in</th>
</tr>
</thead>
<tbody>
<tr>
<td>Scrapers</td>
<td>79¢</td>
<td>34%</td>
<td>2 years</td>
<td>45:1</td>
</tr>
<tr>
<td>40 yd$^3$ dragline</td>
<td>69¢</td>
<td>57%</td>
<td>2 months</td>
<td>58:1</td>
</tr>
<tr>
<td>5 yd$^3$ dragline</td>
<td>99¢</td>
<td>40%</td>
<td>4.4 years</td>
<td>51:1</td>
</tr>
<tr>
<td>Large dozer</td>
<td>77¢</td>
<td>60%</td>
<td>4 years</td>
<td>51:1</td>
</tr>
</tbody>
</table>

Figure 3.3 shows the derivation of the cost for the scraper system. This was obtained by multiplying the rehandle (34%) times the original volume ($1.3 \times 10^6$ yd$^3$) to get total yd$^3$ moved. The cost per yd$^3$ moved at ICFTM#1 was then used to develop the total cost. The original volume was then used to calculate the 79¢/yd$^3$ shown in Table 3.1.

The calculation for the large bulldozer system shown in Figure 3.4 is based on Fiat-Allis handbook data as well as discussions with company application engineers in May of 1975. Based on site examinations and a seismic analysis conducted by coal project personnel, the engineers felt that a 400 hp crawler would have the capability to rip the rock. The operation would require only one man on one machine to handle the consolidated overburden.

Figure 3.5 shows the cost summary for the 5 yd$^3$ dragline. The basic difference between the 40 yd$^3$ and 5 yd$^3$ machines is the cost of moving the consolidated material. The cost of both owning and operating the small dragline is high on a per yd$^3$ basis. The drilling cost is
Machine: 1-14, 2-18, yd$^3$ scrapers and 3-300 hp crawlers

Site: ICPEM#1. Original volume estimates with actual cost and rehandle figures.

Method: Move most of the material with scrapers. Rip the shale and push load the scrapers with the crawlers. Move some material with the dozers.

Volumes: 
1.0 x 10$^6$ yd$^3$ consolidated 
0.3 x 10$^6$ yd$^3$ unconsolidated 
0.44 x 10$^6$ total yd$^3$ rehandle (34%) 
1.74 x 10$^6$ total yd$^3$ moved

Cost summary: 
(1.74 x 10$^6$)yd$^3$ x ($0.59$/yd$^3$) = $1,026,600 
1.3 x 10^6$ BCY at 79c/yd 
135,000 T coal @ $7.60/ton

Time: 2 years

Figure 3.3. Summary of the cost estimates for the scraper, ripper system based on operations at the ICPEM#1
Machine: Fiat-Allis HD 41B w/full U-blade and ripper

Site: ICPDM#1 original volume estimates

Method: The dozer working alone will rip the shale and other rock overburden and doze it up a 10% ramp away from the original topography. All unconsolidated material will be stockpiled ahead of the dozer with scrapers and replaced following the rehandling of 60% of the material not in final position at the finish of mining.

Volumes:  
- $1.0 \times 10^6 \text{ yd}^3$ consolidated 
- $0.3 \times 10^6 \text{ yd}^3$ unconsolidated 
- $0.78 \times 10^6 \text{ yd}^3$ 60% rehandle 
- $2.08 \times 10^6 \text{ yd}^3$ total

Cost summary: 
- Ripping $108,333$
- Moving consolidated material to uncover coal $400,000$
- Removing and replacing unconsolidated material $300,000$
- 60% rock rehandle $187,500$
- $995,833$

135,000 T coal @ $7.38/\text{ton}$
$1.3 \times 10^6 \text{ BCY} @ 77c/\text{yd}^3$

Assuming dozer is limiting machine:

- Dozing $4,000 \text{ hr}$
- Ripping $833$
- Reclaim $3,125$

$7,958 \text{ hr}$

Time: 4 years

Figure 3.4. Summary of the cost estimates for the large bulldozer, scraper system
Machine: Marion 111 M-D with 5 yd$^3$ bucket, 80 ft boom

Site: ICPDM #1 original volume estimates

Method: Scrapers will remove unconsolidated material and stockpile or place on knocked down shale piles. Shale piles to be knocked down with a bulldozer. Pit to be 40 ft wide based on boom length. With a 60 ft high wall this machine will work.

Volumes:
- $1.0 \times 10^6$ yd$^3$ consolidated
- $0.3 \times 10^6$ yd$^3$ unconsolidated
- $0.52 \times 10^6$ yd$^3$ rehandle
- $1.82 \times 10^6$ yd$^3$ total

Cost summary:

- Unconsolidated
  - (Jennerjohn) drilling 80¢/ft
    - 3552 holes 60 ft deep
  - (Otte) blasting $0.077$/yd$^3$
  - Removing consolidated $0.638$/yd$^3$
  - Bulldozer grading $0.40$/yd$^3$
  - 3.7 yd$^3$/lineal ft
  - 17,765 ft at ICPDM
  - 155,556 yd$^3$ last pit rehandle @ 50¢

    | Description                                      | Cost   |
    |--------------------------------------------------|--------|
    | (Jennerjohn) drilling                            | $170,543 |
    | (Otte) blasting                                  | 77,000  |
    | Removing consolidated                            | 638,000 |
    | Bulldozer grading                                | 26,000  |
    | 3.7 yd$^3$/lineal ft                             |         |
    | 17,765 ft at ICPDM                               | 77,778  |
    | 155,556 yd$^3$ last pit rehandle @ 50¢           |        |

    **Total:** $1,289,321

- 135,000 tons of coal @ $9.55/ton
- $1.3 \times 10^6$ BCY @ 99¢/yd$^3$

Time estimate: $1.0 \times 10^6$ yd

\[ \frac{115,000 \text{ yd/yr}}{} = 4.4 \text{ years} \]

Figure 3.5. Summary of cost estimates for the 5 yd$^3$ dragline, scraper system
based on a quote for the ICPEM#1 site. The blasting cost is based on infor-
mation gathered by Otte and Boehlje (1975) for an ISU Coal Project economic
study. The 40 yd$^3$ dragline system shown in Figure 3.6 would not be
feasible on a 40 acre site. The costs shown assume the machine would
work through the area. The amount of rehandle indicated assumes that
the area being mined is no wider than the ICPEM#1.

Figure 3.7 shows the energy calculations for the scraper system.
The diesel fuel consumption is based on estimates from the Caterpillar
Handbook as the records of fuel consumption at ICPEM#1 have not been
analyzed at this time. The thickness of the coal seam will have a
direct impact on the energy ratio but since these four systems are all
based on the same site, they can be compared with each other.

The energy calculations shown in Figure 3.8 for the large dozer
use the rate of fuel consumption calculated for the scraper systems
for moving the unconsolidated material.

The 40 yd$^3$ dragline system shown in Figure 3.9 requires drilling
and blasting. The energy for drilling was based on manufacturers'
estimates of drilling rates and associated fuel consumption. The
energy represented by the ammonium nitrate is based on a report by Davis and
Blouin (1976) at the Tennessee Valley Authority which was checked with
information from an Iowa nitrogen plant. The energy for the dragline
operation was based on manufacturers' estimates. The dragline uses
electricity which was taken back to the power plant with an estimated
power system efficiency of 33%.

Figure 3.10 shows the 5 yd$^3$ dragline system which has the basic
components of the 40 yd$^3$ dragline system.
Machine: Marion 7820-17, 40 yd$^3$ bucket, 225 ft boom

Site: ICPDM#1 original volume estimate

Method: Scrapers for unconsolidated. Dragline on consolidated. Cut to average 60 ft deep. Pit width 100 ft 21.7 yd$^3$/lineal foot of cut will have to be rough graded before placement of unconsolidated. Dragline is limiting factor. Dragline can handle 60 ft deep at 100 ft wide. Three shifts, 20 hr/day.

Volume: $1.0 \times 10^6$ yd$^3$ consolidated
0.3 x $10^6$ yd$^3$ unconsolidated
0.44 x $10^6$ yd$^3$ 57% rehandle
1.74 x $10^6$ yd$^3$ total

Cost summary: Unconsolidated $300,000$
Blasting $0.077$/yd$^3$ 77,000
Drilling 70,543
Dozer grading 40c/yd$^3$ 60,000
Consolidated $0.131$/yd$^3$ 143,000
Last pit rehandle 291,666 yd$^3$ @ 50c 145,833
$896,376$

135,000 tons of coal @ $6.64$/ton
1.3 x $10^6$ BCY @ 69c/yd$^3$

Time: $1.0 \times 10^6$ yd$^3$ = 1.6 mo
631,759 yd$^3$/mo

Figure 3.6. Summary of the cost estimates for the 40 yd$^3$ dragline, scraper system
1.74 \times 10^6 \text{ yd}^3 \text{ total moved} @ \frac{150 \text{ yd}^3}{18 \text{ gal}} = 8.34 \text{ yd}^3/\text{gal}

\frac{1.74 \times 10^6 \text{ yd}^3}{8.34 \text{ yd}^3/\text{gal}} = 208,633 \text{ gal diesel fuel for primary}

Total capital energy 4.2 \times 10^9 \text{ BTU} \left(\frac{5000 \text{ hr on project}}{1000 \text{ hr in life}}\right) = 2.1 \times 10^9 \text{ BTU}

1 \text{ hr support/hr primary} = 11,600 \text{ hr support}

support 12 \text{ gal/hr} (11,600) = 139,200

total fuel 347,833 \text{ gal}

(7.1 \text{ lb/gal}) (18,000 \times \text{ BTU/gal}) = 127,800 \text{ BTU/gal}

347,833 \text{ gal} (1.278 \times 10^5 \text{ BTU/gal}) = 4.45 \times 10^{10} \text{ BTU in}

coal out (1.03 \times 10^5 \text{ tons of coal})(1.02 \times 10^4 \text{ BTU/lb}) = 2.10 \times 10^{12} \text{ BTU out}

\frac{\text{out}}{\text{in}} = \frac{2.10 \times 10^{12} \text{ BTU}}{4.45 \times 10^{10} \text{ BTU}} = 0.47 \times 10^2 = 47 \text{ to uncover coal without capital energy}

\frac{2.10 \times 10^{12} \text{ BTU}}{(4.45 \times 10^{10} + 0.21 \times 10^{10}) \text{ BTU}} = 45 \text{ with capital energy}

Figure 3.7. Summary of the energy ratio estimates for the scraper, ripper system
7,958 hr bulldozer time
__25 gal/hr
198,950 gal diesel

scrapers in uncon. \( \frac{0.6 \times 10^6 \text{yd}^3}{150 \text{BCY/hr}} = 4,000 \text{ hr} \)

4,000 hr (18 gal/hr) = 72,000 gal diesel

Support 0.8(4000) = 3200(12 gal/hr) = 38,400

198,950 large dozer
72,000 scrapers

38,400 support
309,350 gal diesel

309,350 \((1.278 \times 10^5)\text{ BTU/gal} = 3.93 \times 10^{10}\)

\(\frac{\text{out}}{\text{in}} = \frac{2.10 \times 10^{12} \text{ BTU}}{3.93 \times 10^{10} \text{ BTU}} = \frac{53}{1} \) without capital energy

capital energy = 0.17 \times 10^{10} see appendix

\(\frac{2.10 \times 10^{12} \text{ BTU}}{3.93 \times 10^{10} + 0.17 \times 10^{10}} \text{ BTU} = \frac{51.2}{1} \) with capital energy

Figure 3.8. Summary of the energy ratio estimates for the large bulldozer, scraper system
scrapers in unconsolidated 72,000
support for scrapers 38,400
110,400 gal bulldozers
drilling rate 50 ft/hr (Hammond, 1964)
3,552 holes 60 ft deep = 213,120 ft hole
213,120/50 ft/hr = 4262.5 hr
4262.5 hr (5 gal/hr) = 21,312 gal fuel
Blasting 1/2 lb ANFO/yd³
  90% NH₄NO₃ 10% fuel oil
  450,000 lb (8500 BTU/lb) = 3.82 x 10⁹ BTU
  50,000 lb fuel oil at 7 lb/gal = 7142 gal = 9.1 x 10⁹ BTU
Total blasting 4.74 x 10⁹
Dragline (0.73 KWH/BCY) 1.0 x 10⁶ BCY = 0.73 x 10⁶ KWH
  = 2.5 x 10⁹ BTU
  at dragline
33% efficient gives 7.5 x 10⁹ at power plant
reclamation grading 400 BCY/hr
  0.15 x 10⁶ BCY
  400 BCY/hr = 375 hr (12 gal/hr) = 4500 gal
last pit backfill scrapers
  0.29 x 10⁶ BCY
  150 BCY/hr = 1.9 x 10³ hr 18 gal/hr = 3.4 x 10⁴ gal
support 1 x 10⁴ gal
  44,000 gal
1 gal = 1.278 x 10⁵ BTU

Figure 3.9. Summary of the energy ratio estimates for the 40 yd³ dragline, scraper system
<table>
<thead>
<tr>
<th>Operation</th>
<th>Fuel</th>
<th>BTU</th>
</tr>
</thead>
<tbody>
<tr>
<td>Unconsolidated</td>
<td>110,400 gal</td>
<td>$14 \times 10^9$</td>
</tr>
<tr>
<td>Drilling</td>
<td>21,312 gal</td>
<td>$2.7 \times 10^9$</td>
</tr>
<tr>
<td>Blasting</td>
<td></td>
<td>$4.74 \times 10^9$</td>
</tr>
<tr>
<td>Dragline</td>
<td></td>
<td>$7.5 \times 10^9$</td>
</tr>
<tr>
<td>Reclamation grading</td>
<td>4,500 gal</td>
<td>$0.48 \times 10^9$</td>
</tr>
<tr>
<td>Backfill last hole</td>
<td>44,000 gal</td>
<td>$5.6 \times 10^9$</td>
</tr>
</tbody>
</table>

\[ \text{out} = \frac{2.10 \times 10^{12}}{3.5 \times 10^{10}} = 60 \text{ without capital energy} \]

\[ \text{Capital energy} \]

\[ = \frac{2.10 \times 10^{12} \text{ BTU}}{(3.5 \times 10^{10} + 0.14 \times 10^{10}) \text{BTU}} = 57.7 \text{ with capital energy} \]

Figure 3.9. continued
scrapers in unconsolidated support
Drilling (same as 40 yd)
Blasting (same as 40 yd)
Drilling 13 gal/118.8 BCY = 9.14 BCY/gal
1.0 x 10^6/9.14 = 1.09 x 10^5 gal
reclamation grading
0.06 x 10^6 yd/400 BCY/hr = 1.5 x 10^2 hr (12 gal/hr) = 1800 gal
Filling last cut with scrapers
0.16 x 10^6 yd/150 BCY/hr = 1.1 x 10^3 hr (18 gal/hr) = 0.5 x 10^4 gal
2.4 x 10^4 gal

<table>
<thead>
<tr>
<th>Operation</th>
<th>Fuel</th>
<th>BTU</th>
</tr>
</thead>
<tbody>
<tr>
<td>Unconsolidated</td>
<td>110,400</td>
<td>140 x 10^8</td>
</tr>
<tr>
<td>Drilling</td>
<td>21,312</td>
<td>27 x 10^8</td>
</tr>
<tr>
<td>Blasting</td>
<td></td>
<td>47.4 x 10^8</td>
</tr>
<tr>
<td>Dragline</td>
<td>1.09 x 10^5 gal</td>
<td>140 x 10^8</td>
</tr>
<tr>
<td>Reclamation grading</td>
<td>1800 gal</td>
<td>2.3 x 10^8</td>
</tr>
<tr>
<td>Filling last cut</td>
<td>2.4 x 10^4 gal</td>
<td>( \frac{30.7 \times 10^8}{387.0 \times 10^8} )</td>
</tr>
</tbody>
</table>

\[
\frac{\text{out}}{\text{in}} = \frac{2.10 \times 10^{12}}{3.87 \times 10^{10}} = \frac{54}{1} \quad \text{without capital energy}
\]
\[
\frac{2.10 \times 10^{12} \text{ BTU}}{(3.87 \times 10^{10} + 0.22 \times 10^{10}) \text{BTU}} = \frac{51.3}{1} \quad \text{with capital energy}
\]

Figure 3.10. Summary of the energy ratio estimates for the 5 yd³ dragline, scraper system
PART IV. SUMMARY AND CONCLUSIONS

A computer aided mine planning model (CAMP) was developed which has the capability of assisting mine planners in designing mining plans for small mines. A planner could develop a mine plan similar to that developed for the Iowa Coal Project Demonstration Mine #1 (ICPM#1) for less than half the cost of developing the plan by hand as it was originally done for ICPM#1.

CAMP can help to estimate overburden volumes and coal amounts for the total area and for individual pits. The model also has provisions for calculating strip ratios, cash flows, and rehandle percentages for various pit configurations. Secondary subroutines were developed to compute storage area volumes and mobile equipment route information.

ICPM#1 topographic information was used for verification runs of CAMP. Using the same data base, CAMP and an average end area volume calculation computer program estimated the total site volume with a difference of less than 1%.

The reclamation laws which have been adopted for the coal industry are causing many of the small coal mines in Iowa to add mobile equipment to their traditional dragline operations or, in at least one commercial mine, the dragline has been replaced by self-propelled scrapers as the primary stripping machines. The challenge of meeting the requirements of the law and working with different equipment has shown the necessity of more complete mine site planning than has previously been done in Iowa.
Energy efficiency comparisons were made for four hypothetical mining systems that could have been employed at the ICPDM#1. The results showed that for a large site (more than 10 year life of mine site with medium to large sized equipment) an appropriately sized dragline would be the most energy efficient and would uncover coal for the least cost.

Iowa Geological Survey officials are conducting an extensive drilling program to more fully define Iowa coal deposits, but as of late 1977, the general opinion seems to be that the deposits are small and relatively scattered. This will tend to cause the application of mobile equipment to the small sites as has been done in western Pennsylvania. For the small hypothetical mine site based on ICPDM#1, either the large bulldozer or scraper system of overburden handling would have been 20c/yd$^3$ cheaper than the small 5 yd$^3$ dragline, with the energy efficiency of the mobile equipment being the same or only slightly less than the energy efficiency of the small dragline system.
LIST OF REFERENCES


Energy and Mineral Resources Research Institute (Iowa State University, Ames, Iowa) EMRRI No. IS-ICP-14.


Hodge, Charles. 1976. STRATA a computerized subsurface analysis system. VTN Corporation, 2301 Campus Drive, Irvine, Calif.


Reiss, I. H. 1974. We are farmers not miners. Coal Mining and Processing 11(5): 50-52, 63.


APPENDIX A. ORIGINAL MINE PLAN FOR ICPIM#1

This appendix is the plan that was developed for the ICPIM#1 by Thomas S. Colvin and J. Martin Briggs by hand in June 1975. The ICPIM#1 is also called the Scott Site.
SCOTT SITE

SCALE: 1 IN = 200 FT

D  2.75 a

C  4.13 a

B  4.13 a

A  4.96 a

S1  724 a

S 2  186 a

E  2.55 a

F  4.63 a
CROSS SECTION OF AREAS 'A' THRU 'D'

- UNCONSOLIDATED
- CONSOLIDATED
- COAL
- UNCONSOLIDATED AT FINISH GRADE
- CONSOLIDATED AT FINISH GRADE
- SPOIL
- ROAD

NOT TO SCALE
## AREA A

<table>
<thead>
<tr>
<th></th>
<th>A1</th>
<th>A2</th>
<th>A3</th>
<th>A (Total)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Area $\text{Ft}^2$</td>
<td>42,375</td>
<td>63,000</td>
<td>110,775</td>
<td>216,150</td>
</tr>
<tr>
<td>Initial $\text{Yd}^3$ Consol.</td>
<td>40,860</td>
<td>64,167</td>
<td>112,826</td>
<td>217,853</td>
</tr>
<tr>
<td>Final $\text{Yd}^3$ Consol.</td>
<td>20,200</td>
<td>30,032</td>
<td>52,807</td>
<td>103,039</td>
</tr>
<tr>
<td>Initial $\text{Yd}^3$ Unconsol.</td>
<td>16,040</td>
<td>24,500</td>
<td>43,079</td>
<td>83,619</td>
</tr>
<tr>
<td>Final $\text{Yd}^3$ Unconsol.</td>
<td>15,695</td>
<td>23,333</td>
<td>41,028</td>
<td>80,056</td>
</tr>
</tbody>
</table>

COAL, TONS

$\text{Yd}^3$

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>14,715</td>
</tr>
<tr>
<td></td>
<td>15,833</td>
</tr>
</tbody>
</table>
### AREA 'B'

<table>
<thead>
<tr>
<th></th>
<th>B1</th>
<th>B2</th>
<th>B (Total)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Area Ft²</td>
<td>90,000</td>
<td>90,000</td>
<td>180,000</td>
</tr>
<tr>
<td>Initial Yd³ Consol.</td>
<td>106,666</td>
<td>106,666</td>
<td>213,333</td>
</tr>
<tr>
<td>Final Yd³ Consol.</td>
<td>103,670</td>
<td>103,670</td>
<td>207,340</td>
</tr>
<tr>
<td>Initial Yd³ Unconsol.</td>
<td>42,333</td>
<td>42,334</td>
<td>84,667</td>
</tr>
<tr>
<td>Final Yd³ Unconsol.</td>
<td>33,333</td>
<td>33,334</td>
<td>66,667</td>
</tr>
<tr>
<td>COAL TONS Yd³</td>
<td></td>
<td></td>
<td>19,965</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>21,467</td>
</tr>
</tbody>
</table>

SCALE: 1 IN = 100 FT.
### AREA 'C'

<table>
<thead>
<tr>
<th></th>
<th>C 1</th>
<th>C 2</th>
<th>C (Total)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Area $ft^2$</td>
<td>90,000</td>
<td>90,000</td>
<td>180,000</td>
</tr>
<tr>
<td>Initial $yd^3$</td>
<td>137,667</td>
<td>137,666</td>
<td>275,333</td>
</tr>
<tr>
<td>Consol.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Final $yd^3$</td>
<td>155,145</td>
<td>155,145</td>
<td>310,290</td>
</tr>
<tr>
<td>Consol.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Initial $yd^3$</td>
<td>31,000</td>
<td>31,000</td>
<td>62,000</td>
</tr>
<tr>
<td>Unconsol.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Final $yd^3$</td>
<td>33,334</td>
<td>33,333</td>
<td>66,667</td>
</tr>
<tr>
<td>Unconsol.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>COAL TONS</strong></td>
<td></td>
<td></td>
<td>38,745</td>
</tr>
<tr>
<td>$yd^3$</td>
<td></td>
<td></td>
<td>41,667</td>
</tr>
</tbody>
</table>

**SCALE:**

1 IN = 100 FT.
<table>
<thead>
<tr>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Area ${\text{ft}}^2$</td>
<td>120,000</td>
</tr>
<tr>
<td>Initial Consol. ${\text{Yd}}^3$</td>
<td>183,555</td>
</tr>
<tr>
<td>Final Consol. ${\text{Yd}}^3$</td>
<td>255,180</td>
</tr>
<tr>
<td>Initial Unconsol. ${\text{Yd}}^3$</td>
<td>41,333</td>
</tr>
<tr>
<td>Final Unconsol. ${\text{Yd}}^3$</td>
<td>44,444</td>
</tr>
<tr>
<td>COAL TONS</td>
<td>33,000</td>
</tr>
<tr>
<td>COAL ${\text{Yd}}^3$</td>
<td>35,555</td>
</tr>
</tbody>
</table>
## AREA 'E'

<table>
<thead>
<tr>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Area $ft^2$</td>
<td>97,900</td>
</tr>
<tr>
<td>Initial Consol. $yd^3$</td>
<td>121,469</td>
</tr>
<tr>
<td>Final Consol. $yd^3$</td>
<td>127,986</td>
</tr>
<tr>
<td>Initial Unconsol. $yd^3$</td>
<td>25,381</td>
</tr>
<tr>
<td>Final Unconsol. $yd^3$</td>
<td>36,259</td>
</tr>
<tr>
<td>COAL TONS $yd^3$</td>
<td>14,020</td>
</tr>
<tr>
<td>Yd$^3$</td>
<td></td>
</tr>
<tr>
<td></td>
<td>F_1</td>
</tr>
<tr>
<td>----------------</td>
<td>--------</td>
</tr>
<tr>
<td>Area Ft$^2$</td>
<td>82,125</td>
</tr>
<tr>
<td>Initial Yd$^3$</td>
<td>66,004</td>
</tr>
<tr>
<td>Consol.</td>
<td></td>
</tr>
<tr>
<td>Final Yd$^3$</td>
<td>62,300</td>
</tr>
<tr>
<td>Consol.</td>
<td></td>
</tr>
<tr>
<td>Initial Yd$^3$</td>
<td>30,417</td>
</tr>
<tr>
<td>Unconsol.</td>
<td></td>
</tr>
<tr>
<td>Final Yd$^3$</td>
<td>30,417</td>
</tr>
<tr>
<td>Unconsol.</td>
<td></td>
</tr>
<tr>
<td>COAL TONS</td>
<td></td>
</tr>
<tr>
<td>Yd$^3$</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**AREA 'F'**
STORAGE AREA 'S1'
NO COAL

A = 315,250 Ft$^2$
STORAGE AREA S2
NO COAL

SCALE: 1 IN = 100 FT

S 2  A = 80,850 FT²
SCOTT SITE

PLAN AND CROSS SECTION OF
UNCONSOLIDATED MATERIAL
MOVEMENT

SECTION X-X
MINING TIME FOR EACH AREA

TIME IN WEEKS

<table>
<thead>
<tr>
<th>AREA</th>
<th>MINING TIME</th>
</tr>
</thead>
<tbody>
<tr>
<td>A1</td>
<td></td>
</tr>
<tr>
<td>A2</td>
<td>CONSOLIDATED STOCK PILE</td>
</tr>
<tr>
<td>A3</td>
<td></td>
</tr>
<tr>
<td>B1</td>
<td></td>
</tr>
<tr>
<td>B2</td>
<td></td>
</tr>
<tr>
<td>C1</td>
<td></td>
</tr>
<tr>
<td>C2</td>
<td></td>
</tr>
<tr>
<td>D</td>
<td></td>
</tr>
<tr>
<td>E</td>
<td></td>
</tr>
<tr>
<td>F1</td>
<td>ACTIVE CONSOLIDATED STOCK PILE MINING</td>
</tr>
<tr>
<td>F2</td>
<td>ACTIVE CONSOLIDATED STOCK PILE MINING</td>
</tr>
<tr>
<td>F3</td>
<td></td>
</tr>
<tr>
<td>S1</td>
<td>UNCONSOLIDATED STOCK PILE</td>
</tr>
<tr>
<td>S2</td>
<td>UNCONSOLIDATED DORMANT STOCK PILE</td>
</tr>
</tbody>
</table>
SCOTT SITE
PLAN AND CROSS-SECTION OF CONSOLIDATED MATERIAL MOVEMENT

SECTION Y-Y

MATERIAL MOVEMENT
<table>
<thead>
<tr>
<th>Overburden Operations</th>
<th>YD$^3$ to be moved</th>
<th>Hrs at 225 YD/hr unit</th>
<th>Hrs for 4,14 YD$^3$ units</th>
<th>Overburden TOTAL TIME</th>
<th>Coal YD$^3$/hr to be moved</th>
<th>Coal Operation Upper &amp; Lower</th>
</tr>
</thead>
<tbody>
<tr>
<td>Clear Trees</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>A1u → S2</td>
<td>16,040</td>
<td>71.3</td>
<td>17.8</td>
<td>17.8</td>
<td></td>
<td>A1CL</td>
</tr>
<tr>
<td>A2u → S2</td>
<td>24,500</td>
<td>108.9</td>
<td>27.2</td>
<td>45.0</td>
<td></td>
<td>A2CL</td>
</tr>
<tr>
<td>A1c → F2</td>
<td>40,860</td>
<td>181.6</td>
<td>45.4</td>
<td>90.4</td>
<td></td>
<td></td>
</tr>
<tr>
<td>A3u → S1</td>
<td>43,079</td>
<td>191.5</td>
<td>47.9</td>
<td>138.3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>A2c → A1c</td>
<td>64,167</td>
<td>285.2</td>
<td>71.3</td>
<td>209.6</td>
<td></td>
<td></td>
</tr>
<tr>
<td>A3c → A2c</td>
<td>112,826</td>
<td>501.4</td>
<td>125.4</td>
<td>325</td>
<td></td>
<td>A3CL</td>
</tr>
<tr>
<td>B1u → A1u</td>
<td>42,333</td>
<td>188.1</td>
<td>47.0</td>
<td>382</td>
<td></td>
<td></td>
</tr>
<tr>
<td>B1c → A3c</td>
<td>106,666</td>
<td>474.1</td>
<td>118.5</td>
<td>666.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>B2u → A3u</td>
<td>42,333</td>
<td>188.1</td>
<td>47.0</td>
<td>547.5</td>
<td></td>
<td>B1CL</td>
</tr>
<tr>
<td>B2c → B1c</td>
<td>106,666</td>
<td>474.1</td>
<td>118.5</td>
<td>666.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>C1u → B1u</td>
<td>33,334</td>
<td>148.2</td>
<td>37.0</td>
<td>703.0</td>
<td></td>
<td>B2CL</td>
</tr>
<tr>
<td>C1c → B2c</td>
<td>137,667</td>
<td>611.9</td>
<td>153.0</td>
<td>856.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>C2u → B2u</td>
<td>33,334</td>
<td>148.2</td>
<td>37.0</td>
<td>893.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>C2c → C1c</td>
<td>155,145</td>
<td>689.5</td>
<td>172.4</td>
<td>1056.4</td>
<td></td>
<td></td>
</tr>
<tr>
<td>D1u → C1c</td>
<td>41,333</td>
<td>183.7</td>
<td>45.9</td>
<td>1111.3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>D2c → C2c</td>
<td>183,555</td>
<td>815.8</td>
<td>203.95</td>
<td>1315.2</td>
<td></td>
<td></td>
</tr>
<tr>
<td>E1u → C2u</td>
<td>33,334</td>
<td>148.2</td>
<td>37.0</td>
<td>1352.2</td>
<td></td>
<td></td>
</tr>
<tr>
<td>E2c → Dc</td>
<td>255,180</td>
<td>1134.1</td>
<td>283.5</td>
<td>1635.7</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Note: The table above lists the overburden operations with the corresponding YD$^3$ to be moved, hours at 225 YD/hr unit, hours for 4,14 YD$^3$ units, overburden total time, coal YD$^3$/hr to be moved, and coal operation upper & lower.
<table>
<thead>
<tr>
<th>Mine Area</th>
<th>Finish Grade</th>
<th>Spoil YD$^3$</th>
<th>Running Spoil YD$^3$</th>
<th>Spoil YD$^3$</th>
<th>Finish Grade YD$^3$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Alu → S2</td>
<td>0</td>
<td>+16,040</td>
<td>0</td>
<td>16,040</td>
<td>16,040</td>
</tr>
<tr>
<td>A2u → S2</td>
<td>0</td>
<td>+24,500</td>
<td>0</td>
<td>40,540</td>
<td>40,540</td>
</tr>
<tr>
<td>A3u → S1</td>
<td>0</td>
<td>+43,079</td>
<td>43,079</td>
<td>40,540</td>
<td>83,619</td>
</tr>
<tr>
<td>Blu → Alu</td>
<td>15,695</td>
<td>+26,638</td>
<td>69,716</td>
<td>40,540</td>
<td>110,257</td>
</tr>
<tr>
<td>B2u → A3u</td>
<td>41,028</td>
<td>+1,305</td>
<td>71,022</td>
<td>40,540</td>
<td>111,562</td>
</tr>
<tr>
<td>C1u → Blu</td>
<td>33,334</td>
<td>-2,334</td>
<td>66,354</td>
<td>40,540</td>
<td>111,562</td>
</tr>
<tr>
<td>C2u → B2u</td>
<td>33,334</td>
<td>-2,334</td>
<td>66,354</td>
<td>40,540</td>
<td>111,562</td>
</tr>
<tr>
<td>Du → Clu</td>
<td>33,334</td>
<td>+7,999</td>
<td>66,354</td>
<td>40,540</td>
<td>119,561</td>
</tr>
<tr>
<td>Eu → C2u</td>
<td>33,334</td>
<td>-7,953</td>
<td>66,354</td>
<td>40,540</td>
<td>119,561</td>
</tr>
<tr>
<td>F1u → Du</td>
<td>44,444</td>
<td>-14,027</td>
<td>52,327</td>
<td>40,540</td>
<td>119,561</td>
</tr>
<tr>
<td>F2u → Eu</td>
<td>36,259</td>
<td>-13,898</td>
<td>38,429</td>
<td>40,540</td>
<td>119,561</td>
</tr>
<tr>
<td>F3u → F1u</td>
<td>30,417</td>
<td>-8,565</td>
<td>29,852</td>
<td>40,540</td>
<td>119,561</td>
</tr>
<tr>
<td>S1 → F2u</td>
<td>22,361</td>
<td>-22,361</td>
<td>7,503</td>
<td>40,540</td>
<td>119,561</td>
</tr>
<tr>
<td>S1 &amp; S2 → F3u</td>
<td>21,852</td>
<td>-21,852</td>
<td>0</td>
<td>26,191</td>
<td>119,561</td>
</tr>
<tr>
<td>S2 → A2u</td>
<td>23,333</td>
<td>-23,333</td>
<td>0</td>
<td>2,858</td>
<td>119,561</td>
</tr>
</tbody>
</table>

**Total Rehandle (u & c) = 26.4%**

- **Error = 0.8%**
- **32% of total u**
<table>
<thead>
<tr>
<th>Mine Area</th>
<th>Finish Grade Yd³</th>
<th>Spoil Grade Yd³</th>
<th>Running Spoil Yd³</th>
<th>Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>A1c → F</td>
<td>0</td>
<td>+40,860</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>A2C → A1C</td>
<td>20,200</td>
<td>+43,967</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>A3C → A2C</td>
<td>0</td>
<td>+44,758</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>A3C → A2C</td>
<td>30,032</td>
<td>+38,036</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>B1C → A3C</td>
<td>52,807</td>
<td>+53,859</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>B2C → B1C</td>
<td>103,670</td>
<td>+2,996</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>C1C → B2C</td>
<td>103,670</td>
<td>+33,997</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>C2C → C1C</td>
<td>155,145</td>
<td>-17,479</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>DC → C2C</td>
<td>155,145</td>
<td>+28,410</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>EC → DC</td>
<td>255,180</td>
<td>-133,711</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>F1C → EC</td>
<td>127,986</td>
<td>-61,982</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>F2C → F1C</td>
<td>62,300</td>
<td>-13,776</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>F3C → F2C</td>
<td>45,800</td>
<td>+1,619</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

Spoil
A2C → F3C 44,758

Finish Grade Yd³

Error = 25% of 1.4% total

CONSOLIDATED
<table>
<thead>
<tr>
<th>Overburden Operations</th>
<th>Yd$^3$ to be moved</th>
<th>Hrs at 225 Yd$^3$/hr unit</th>
<th>Hrs for 4,14 Yd$^3$ units</th>
<th>Overburden TOTAL TIME</th>
<th>Coal TOTAL TIME</th>
<th>Hrs at 160 Yd$^3$/hr (2 trucks)</th>
<th>Yd$^3$ to be Moved</th>
<th>Coal Operation Upper &amp; Lower</th>
</tr>
</thead>
<tbody>
<tr>
<td>Flu → Du</td>
<td>44,444</td>
<td>197.5</td>
<td>49.5</td>
<td>1685.1</td>
<td>860.3</td>
<td>57.0</td>
<td>9125</td>
<td>F1cL</td>
</tr>
<tr>
<td>F1c → Ec</td>
<td>127,986</td>
<td>560.8</td>
<td>142.2</td>
<td>1827.3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>F2u → Eu</td>
<td>36,259</td>
<td>161.2</td>
<td>40.3</td>
<td>1867.6</td>
<td>902.2</td>
<td>41.9</td>
<td>6708</td>
<td>F2cL</td>
</tr>
<tr>
<td>F2c → F1c</td>
<td>62,300</td>
<td>276.9</td>
<td>69.2</td>
<td>1936.8</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>F3u → F1u</td>
<td>30,417</td>
<td>135.2</td>
<td>33.8</td>
<td>1970.6</td>
<td>943.2</td>
<td>41.0</td>
<td>6556</td>
<td>F3cL</td>
</tr>
<tr>
<td>F3c → F2c</td>
<td>47,400</td>
<td>210.7</td>
<td>52.7</td>
<td>2023.3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**Spoil**

|                      |                     |                |                             |                      |                |                               |                 |                             |
|                      |                     |                |                             |                      |                |                               |                 |                             |
| A2c → F3c            | 44,758              | 198.9          | 49.7                        | 2073.0                |               |                               |                 |                             |
| S1 → F2u             | 22,361              | 99.4           | 24.8                        | 2097.8                |               |                               |                 |                             |
| S16S2 → F3u          | 21,852              | 97.1           | 24.3                        | 2122.1                |               |                               |                 |                             |
| S2 → A2u             | 23,333              | 103.7          | 25.6                        | 2148.0                |               |                               |                 |                             |

53.7 wks 23.6 wks coal
APPENDIX B. LISTING OF CAMP AND SMALL OUTPUT

This appendix gives a complete listing of the Computer Aided Mine Planning Model, input data, and output information that could have been used to develop the original mine plan shown in Appendix A.
COMMON /CONT/ DI,COAL,COALDP,SWELL,COVDEP,NUMPIT,NUMWAY
COMMON /MCNY/ HRPROD,YDCOST,COALPR,LPIT(10,100),WKHRS
COMMON K
C),RSURF(25,25),STRIPR(25,25)
COMMON/ALTER/CURFC(25,25)

C SET CVOLUM=1 FOR VOLUME OF SITE ONLY,=2 FOR 2 LAYERS,=0 TO SKIP CUTVOLL
DIMENSION UFLPT(100), RFLPT(100)
C SET ISURF EQUAL TO 0 TO SKIP SURFACE ALTERATIONS
INTEGER CVOLUM(1),ISURF(0)
C
C VOLUM IS AN INTEGER
C DIRT IS AN REAL NUMBER 0 FOR A CONTOUR MAP OF THE BOTTOM OF UNCONS.
C MATERIAL, 1 FOR AN ISOPACH MAP OF THE UNCONS., AND ANY POSITIVE
C NUMBER FOR A SPECIFIED AMOUNT OF UNCONS TO BE CONSIDERED OUT OF THE
C ORIGINAL PROFILE OF MATERIAL
C ISURF IS AN INTEGER TO CONTROL THE EXECUTION OF SURFAL
C COAL IS A REAL NUMBER WHICH GIVES THE DENSITY OF COAL IN LBS/CUBIC YARD
C SWELL IS A REAL NUMBER 0 FOR SWELL - FOR SHRINK
C COVDEP IS A REAL NUMBER WHICH IS THE DEPTH OF UNCONS. ON THE FINAL PLAN
READ, CVOLUM,ISURF,DIRT,COAL,COALDP,SWELL,COVDEP,NUMPIT,NUMWAY
READ, HRPROD,YDCOST,COALPR,WKHRS,FDVOLUM
IF(FD VOLUM,EQ.1.) GO TO 50
PRINT 40, CVOLUM,ISURF,DIRT
40 FORMAT('1','CVOLUM HAS BEEN SET AT ',I3/,'0ISURF HAS BEEN SET AT ',I3/,'0DIRT HAS BEEN SET AT ',F4.0)
PRINT 41, COAL,COALDP,SWELL
41 FORMAT('DENSITY OF THE COAL HAS BEEN SET AT ',F6.3,'LBS./BANK CUB
CIC YARD/','0THE THICKNESS OF THE COAL SEAM HAS BEEN SET AT ',F4.0,
C'FEET/','0 THE OVERBURDEN SWELL HAS BEEN SET AT ',F4.2,'%')
PRINT 42, COVDEP,NUMPIT,NUMWAY
42 FORMAT('THE DEPTH OF UNCONSOLIDATED COVER IN THE FINAL PROFILE OF
C TWO LAYER SYSTEM HAS BEEN SET AT ',F4.0,' FEET', 'O THE NUMBER OF
C PITS TO BE MINED IS ',I3/,' O THE NUMBER OF SEQUENCES OF MINING WILL
C BE', I3)

PRINT 43, HRPROD, YOCOST, COALPR, WKhrs

43 FORMAT ('O THE HOURLY PRODUCTION RATE FOR THE MINE OVERBURDEN REMOVA
CL FLEET IS ',F6.0,' BANK CUBIC YARDS PER HOUR ', 'O THE AVERAGE EST
CIMATED COST OF MOVING A BANK CUBIC YARD OF OVERBURDEN IS $',F5.2/,
C 'O THE AVERAGE ESTIMATED COAL SELLING PRICE PER TON AT THE MINE I
CS $',F6.2/,' O THE NUMBER OF HOURS THAT THE OVERBURDEN FLEET WILL WO
CRK PER WEEK IS ',F5.1)

DO 120 J=1, NUMWAY

120 READ, (LPIT(J,1), I=1, NUMPIT)

DO 2 I=1,25

READ 3, (SURFC(I, J), J=1, 25)

DO 2 J=1, 25

2 CSURFC(I,J)=SURFC(I,J)

3 FORMAT (25F3.0)

50 DO 4 I=1,25

4 READ 3, (COALT(I, J), J=1, 25)

DO 11 I=1,25

11 READ 16, (IPITS(I, J), J=1, 25)

16 FORMAT (25(13))

IF((FVOLUM.EQ.1.).AND.(CVOLUM.EQ.1)) GO TO 205

IF(FVOLUM.EQ.1.) GO TO 25

IF (CVOLUM.EQ.0) GO TO 10

IF (CVOLUM.EQ.2) GO TO 25

GO TO 35

25 DO 30 I=1,25

30 READ 3, (RSURFC(I, J), J=1, 25)

IF(FVOLUM.EQ.1.) GO TO 205

CALL CUTVOL (CVOLUM)

10 IF(CVOLUM.EQ.1) GO TO 20

IF (CVOLUM.EQ.10) GO TO 150

IF (CVOLUM.EQ.20) GO TO 150

20 IF(ISURF.EQ.0) GO TO 205
CALL SURFAL
205 CALL FINVCL(CVOLUM)
   IF(FVOLUM.EQ.1.) GO TO 100
   CALL MONEY(CVOLUM)
   CALL PATH
   CALL SQRPLE
   CALL TRPILE
   CALL ISITIN
150 PRINT 200
200 FORMAT('1 END')
STOP
END

BLOCK DATA
COMMON / DAREA / AREA(200)
COMMON/ VOLUM/ FVOL(100),CVOL,VOL(200),DVOL,UVOL(200),RVOL(200),
        CONVOL,FUVOL(100),FRVOL(100),COALWT(100)
REAL AREA/200*0.0/
REAL FVOL/100*0.0/
REAL VOL/200*0.0/
REAL UVOL/200*0.0/
REAL RVOL/200*0.0/
REAL COALWT/100*0.0/
END

SUBROUTINE CUTVOL(CVOLUM)
C A SUBROUTINE TO CALCULATE THE VOLUME OF MATERIAL OVER COAL AND BY PITS
C THIS PROGRAM USES STRAIGHT SISED PITS AND A 50 FT GRID
C THE PIT LOCATIONS ARE INDICATED BY NUMBERS AT THE GRID POINTS
COMMON / CONTRL/ DIRT,COAL,COALDP,SWELL,COVDEP,NUMPIT,NUMWAY
        ,RSURF(25,25),STRIPR(25,25)
COMMON/ VOLUM/ FVOL(100),CVOL,VOL(200),DVOL,UVOL(200),RVOL(200),
        CONVOL,FUVOL(100),FRVOL(100),COALWT(100)
COMMON / DAREA / AREA(200)
INTEGER CVOLUM
DIMENSION ELDIF (25,25)
REAL STRPRT(100)/100*0./
TCOAL=0.
TAREA=0
CVOL=0
K=1
COALAM=(COALDP*92.59)*(COAL/2000.)
DO 5 I=1,25
  DO 5 J=1,25
    IF(IPITS(I,J).EQ.0) GO TO 5
    IF((SURFC(I,J).GT.0.)AND.(COALT(I,J).GT.0.))GO TO 70
    IF(SURFC(I,J).LE.0.) PRINT75,SURFC(I,J),I,J
      75 FORMAT(' INPUT VALUE FOR ORIGINAL GROUND SURFACE IS',F4.0,'AT I=',C13,'AND J= ',13,' VOLUME OF THIS POINT IS NOT INCLUDED')
    IF(COALT(I,J).LE.0.) PRINT 76, COALT(I,J), I,J
      76 FORMAT(' INPUT VALUE FOR TOP OF THE COAL IS', F4.0, 'AT I= ',13,' AND J= ',13,' VOLUME OF THIS POINT IS NOT INCLUDED')
    GO TO 5
    70 ELDIF (I,J)= SURFC(I,J)-COALT(I,J)
    IF (IPITS(I,J).GE.100) GO TO 5
    N=IPITS(I,J)
    IF(N.GT.K) K=N
    AREA(N)=AREA(N)+2500.
    VOL(N)=VOL(N)+ELDIF(I,J)*92.59
    COALWT(N)=COALWT(N)+COALAM
    STRIPR(I,J)=(ELDIF(I,J)*92.59)/COALAM
  5 CONTINUE
  DO 12 I=1,K
    CVOL=CVOL+VOL(I)
    TAREA=TAREA+AREA(I)
  12 CONTINUE
  PRINT 21
   21 FORMAT('1','/ ',',53X,'ORIGINAL GROUND SURFACE',/')
DO 6 I=1,25
6 PRINT 7, (SURFC(I,J), J=1,25)
7 FORMAT(4X,25(F4.0,IX),/,/,/)
PRINT 28
28 FORMAT('I',/,/,54X,'TOP OF COAL ELEVATION',/,/,/)
DO 8 I=1,25
8 PRINT 7, (COALT(I,J), J=1,25)
PRINT 35
35 FORMAT('I',/,/,59X,'PIT LOCATIONS',/,/,/)
DO 13 I=1,25
13 PRINT14, (IPITS(I,J), J=1,25)
14 FORMAT(4X,25(I3,2X),/,/,/)
PRINT 42, CVOL
42 FORMAT('I', 'TOTAL SITE VOLUME OVER COAL IN CUBIC YARDS', F10.2,/) DO 15 I=1,K
15 PRINT 49, I, VOL(I)
49 FORMAT('VOLUME IN PIT ', 'IN CUBIC YARDS IS', F10.2,/) PRINT 80, TAREA
80 FORMAT('I', 'TOTAL AREA OVER COAL IN SQUARE FEET IS', F10.2,/) DO 60 I=1,K
60 PRINT65, I, AREA(I)
65 FORMAT('AREA IN PIT ', 'IN SQUARE FEET IS', F10.2,/) TSTRIP=CVOL/TCOAL
PRINT 170, TSTRIP
170 FORMAT('I', 'STRIP RATIO FOR THE AREA WITH COAL UNDERNEATH IS', C F10.2,/) DO 180 I=1,K
180 PRINT 190, I, STRPRT(I)
190 FORMAT('STRIP RATIO IN PIT ', 'IN CUBIC YARDS PER TON OF COAL C IS ', F10.2,/) PRINT 56
56 FORMAT('I', '/',/,51X,'ISCPACK MAP OF COAL OVERTBURDEN',/,/,/)
DO 9 I=1,25
9 PRINT 7, (ELDIF(I,J), J=1,25)
PRINT 150
150 FORMAT ('1',/,'25X,' MAP OF STRIP RATIO, VALUE SHOWN IS CUBIC YARDS OF OVERBURDEN PER TON OF COAL ','/','/','/')
   DO 160 I=1,25
160 PRINT 7,(STRIPR(I,J),J=1,25)
   PRINT 200, TCOAL
200 FORMAT ('THE TOTAL AMOUNT OF COAL FOR THE DEFINED AREA IS',F10.0,' TONS')
   DO 210 I=1,K
210 PRINT 220,I,COALWT(I)
220 FORMAT (' PIT',I3,' HAS ',F10.0,' TONS OF COAL')
   IF (CVOLUM.EQ.20) GO TO 100
   IF (CVOLUM.EQ.2) GO TO 100
   GO TO 110
100 CALL UNCVOL (ELDIF)
110 RETURN
END
SUBROUTINE SURFAL
C A SUBROUTINE TO DIG IN A PIT INDICATED BY ILOC TO A DEPTH ICUTMX
   COMMON/ALTER/ CSURFC(25,25)
   ICUTMX=5
   ICUTMX=30
   ILOC=1
   IDEEP=0
   CUTDEX=0.0
5 DO 10 I=1,25
   DO 10 J=1,25
   IF (IPITS(I,J).NE.ILOC) GO TO 10
   CSURFC(I,J)=CSURFC(I,J)-1.0
   IF (CSURFC(I,J).LT.COALT(I,J)) GO TO 9
   CUTDEX=CUTDEX+1.0
   GO TO 10
9 CSURFC(I,J)=COALT(I,J)
10 CONTINUE
IDEEP = IDEEP + 1
IF (IDEEP .LT. ICUTMX) GO TO 5
CUTVOL = CUT0EX * 92.59
PRINT 20
20 FORMAT ('1', '/ ', 59X, 'CUT SURFACE', ' / ', '/ ', '/ ')
DO 22 I = 1, 25
22 PRINT 25, (CSURFC(I, J), J = 1, 25)
25 FORMAT (4X, 25(F4.0, 1X), ' / ', ' / ', '/ ')
PRINT 30, I, LOG, CUTVOL
30 FORMAT (' VOLUME CUT FROM PIT ', '13', ' IN CUBIC YARDS IS ', F10.2, '/ ')
RETURN
END

SUBROUTINE FINVOL (CVOLUM)
COMMON / CONTRL / DIRT, COAL, COALDP, SWELL, CCVDEP, NUMPIT, NUMWAY
COMMON K
C), RSURF(25, 25), STRIPR(25, 25)
COMMON / VOLUM / FVOL(100), CVOL, VOL(200), DVOL, UVCL(200), RVOL(200),
C CONVOL, FUVOL(100), FRVOL(100), COALWT(100)
INTEGER CVOLUM
DIMENSION FELDIF (25, 25)
ISWIT = 1
K = 1
TFVOL = 0
DO 30 I = 1, 25
IF (ISWIT .EQ. 0) GO TO 200
READ 4, (FSURFC(I, J), J = 1, 25)
4 FORMAT (25F3.0)
IF (ISWIT .EQ. 1) GO TO 30
200 READ, M
READ 3, (FSURFC(I, J), J = 1, 18)
READ 3, (FSURFC(I, J), J = 19, 25)
3 FORMAT (18(1X, F3.0))
30 CONTINUE
DO 5 I = 1, 25
DO 5 J = 1, 25
IF(IPITS(I,J), EQ, 0) GO TO 5
IF(IPITS(I,J), GT, 99) GO TO 5
IF(FSURFC(I,J), LE, 0.) FSURFC(I,J) = SURFC(I,J)
FELDIF(I,J) = FSURFC(I,J) - COALT(I,J) + COALDP
IF(COALT(I,J), LE, 0.) FELDIF(I,J) = 0.
32 N = IPITS(I,J)
IF(N, GT, K) K = N
FVOL(N) = FVOL(N) + FELDIF(I,J) * 92.59
5 CONTINUE
PRINT 42
42 FORMAT ('!')
DO 40 I = 1, K
TFVOL = TFVOL + FVOL(I)
40 PRINT 35, I, FVOL(I)
PRINT 45, TFVOL
45 FORMAT (' THE TOTAL FILL AVAILABLE UNDER THE FINAL SURFACE IN CUBIC YARDS IS ', F10.0, '/')
35 FORMAT (' FILL VOLUME IN PIT ', I3, ' IN CUBIC YARDS IS ', F10.0, '/')
PRINT 10
10 FORMAT ('!', '/

DO 20 I = 1, 25
20 PRINT 25, (FELDIF(I,J), J = 1, 25)
25 FORMAT (4X, 25(F4.0, 1X), '/

IF(CVOLUM, EQ, 2) GO TO 100
GO TO 110
100 CALL RDFNVL
110 RETURN
END

SUBROUTINE MINE(A)

THIS PROGRAM GOES THROUGH THE MINING SEQUENCE AND FIGURES THE REHANDLE

COMMON /MCNY/ HRPROD, YDOST, COALPR, LPIT(10, 100), WKHRS

COMMON /VOLUM/ FVOL(100), CVOL, VOL(200), DVOL, UVOL(200), RVCL(200),
C CONVOL, FUVOL(100), FVOL(100), CGALW(100)
COMMON K
REAL FILPIT(200)/200.0/,
INTEGER A, B
STORVL=0.
C L IS THE NUMBER OF PITS TO BE CYCLED MINUS ONE
L=K-1
STORVL=STORVL+VOL(1)
PRINT 50
DO 20 I=1,L
IF(VOL(I+1),GT,FVOL(I)) GO TO 10
FILPIT(V)=VOL(I+1)
GO TO 75
10 FILPIT(I)=FVOL(I)
FILL=VOL(I+1)-FVOL(I)
DO 15 N=1,1
IF (FILL .LE. 0) GO TO 15
IF (FILPIT(N).GE.FVOL(N)) GO TO 16
AVLSTR=FVOL(N)-FILPIT(N)
IF (AVLSTR.LE.FILL) GO TO 30
FILPIT(N)=FILPIT(N)+FILL
FILL =0.
GO TO 15
30 FILPIT(N)= FVOL(N)
FILL=FILL-AVLSTR
16 IF (N.GE.1) STORVL=STORVL+FILL
15 CONTINUE
75 DO 45 J=1,1
M=LPIT(A,I+1)
B=LPIT(A,J)
45 PRINT 60, B,FILPIT(J), M
50 FORMAT('PIT ' ,I3, ' HAD ' ,F10.0, ' CUBIC YARDS OF FILL AFTER THE EXCAVATION OF PIT ' ,I3, ' WAS COMPLETED ' )
20 CONTINUE
REHNDL=(STORVL/CVOL)*100.
PRINT 40, STORVL
40 FORMAT(' THE TOTAL VOLUME IN STORAGE AFTER THE EXCAVATION OF THE
CPIT IS ',F10.0,' CUBIC YARDS')
PRINT 70, REHNDL
70 FORMAT (' THE REHANDLE FOR THIS PIT SEQUENCE IS ',F4.0,'%')
RETURN
END
SUBROUTINE RDFNVL
C THIS SUBROUTINE SPLITS THE FINAL VOLUME OF PITS INTO ROCK AND DIRT
C COVDEP IS THE DEPTH OF THE UNCONSOLIDATED MATERIAL THAT WILL BE PLACED
COMMON /CCNTRL/ DIRT,COAL,COALDP,SWELL,CCVDEP,NUMPIT,NUMWAY
COMMON/ VOLUM/ FVOL(IOO) ,CVCL,VOL(200),DVOL,UVCL(200),RVOL(200),
C CONVOL,FUVO1(100),FRVOL(100),COALWT(100)
COMMON /DAREA/ AREA(200)
COMMON K
C COALDP IS THE DEPTH OF THE COAL SEAM
PRINT 20
20 FORMAT ('VOLUME OF UNCONSOLIDATED AND CONSOLIDATED MATERIAL UN
CER THE FINAL PLAN SURFACE')
DO 10 I=1,K
FUVOL(I)=(COVDEP*AREA(I))/27.
FRVOL(I)=FVOL(I)-FUVOL(I)+((COALDP*AREA(I))/27.)
PRINT 25, I,FUVOL(I)
25 FORMAT(' PIT ',13,' HAS', F10.0,' CUBIC YARDS OF UNCONSOLIDATED FILL
C VOLUME AVAILABLE')
PRINT 30, I,FRVOL(I)
30 FORMAT(' PIT ',13,' HAS', F10.0,' CUBIC YARDS OF CONSOLIDATED FIL
CL VOLUME UNDER THE FINAL PLAN SURFACE')
10 CONTINUE
RETURN
END
SUBROUTINE SEQNC (A)
C THIS SUBROUTINE MINES THE COAL IN THE SEQUENCE INPUT AND
C HANDLES TWO LAYERS CALLED ROCK AND UNCONSOLIDATED OR DIRT
COMMON /MCNY/* HRPROD,YDCOST,COALPR,LPIT(10,100),WKHRS
COMMON/ VCLUM/ FVOL(100),*CVOL, VOL(200),DVL,UVOL(200),RVOL(200),
 C CONVOL, FUVOL(100),FRVOL(100),COALWT(100)
COMMON K
INTEGER A,B
REAL UFLPT(100)/100*0.,,RFLPT(100)/100*0./
50 FORMAT('I',' STATUS OF PIT FILLING')
RSTRVL=0.
USTRVL=0.
L=K-1
RSTRVL=RSTRVL+RVOL(1)
USTRVL=USTRVL+UVOL(1)+UVOL(2)
PRINT 50
DO 20 I=1,L
IF (RVOL(I+1).GT.,FRVOL(I)) GO TO 10
RFLPT(I)=RVOL(I+1)
GO TO 75
10 RFLPT(I)=FRVOL(I)
RFILL=RVOL(I+1)-FRVOL(I)
DO 15 N=1,I
IF (RFILL.EQ.0.) GO TO 15
IF (RFLPT(N).GE.,FRVOL(N)) GO TO 16
RVLSTR=FRVOL(N)-RFLPT(N)
IF (RVLSTR.LE.,RFILL) GO TO 30
RFLPT(N)=RFLPT(N)+RFILL
RFILL=0.
GO TO 15
30 RFLPT(N)=FRVOL(N)
RFILL=RFILL-RVLSTR
16 IF (N.GE.,1) RSTAVL=RSTAVL+RFILL
15 CONTINUE
75 DO 45 J=1,I
B=LPIT(A,J)
M=LPIT(A,I+1)
45 PRINT 60,B,RFLPT(J),M
60 FORMAT(' PIT',I3,' HAD ',F10.0,' CUBIC YARDS OF ROCK FILL AFTER THE
C EXCAVATION OF ROCK IN PIT ',I3,' WAS COMPLETED ')
   IF (I.EQ.I) GO TO 20
   IF (RFLPT(I)*GE*FRVOL(I)) GO TO 200
   UFLPT(UVCL(I+2))
   GO TO 300
200 IF(UVCL(I+2)*GT*FUVOL(I)) GO TO 100
   UFLPT(I)=UVCL(I+2)
   GO TO 175
100 UFLPT(I)=FUVOL(I)
   UFLPT(UVCL(I+2)-FUVOL(I))
300 DO 115 N=1,I
   IF(UFILL.EQ.0.) GO TO 115
   IF(UFLPT(N)*GE*FUVOL(N)) GO TO 116
   UFLPT(UVCL(N)-UFLPT(N))
   IF(UVCL(N)*LE*UFILL) GO TO 130
   UFLPT(N)=UFLPT(N)+UFILL
   UFILL=0.
   GO TO 115
130 UFLPT(N)=FUVOL(N)
   UFILL=UFILL-UVLST
116 IF(N*GE*I) USTRVL=USTRVL+UFILL
115 CONTINUE
175 B=LPIT(A,J)
   M=LPIT(A,1+2)
   DO 145 J=1,I
145 PRINT 160 ,B,UFLPT(J),M
160 FORMAT(' PIT',I3,' HAD ',F10.0,' CUBIC YARDS OF UNCONSOLIDATED FIL
C CL AFTER THE EXCAVATION OF UNCONS. IN PIT ',I3,' WAS COMPLETED ')
20 CONTINUE
   REHNDL=((RSTRVL+USTRVL)/CVOL)*100.
   PRINT 80,RSTRVL,USTRVL
80 FORMAT (' THE TOTAL CONSOLIDATED VOLUME IN STORAGE FOR THIS SEQUE
CCE IS ',F10.0,' CUBIC YARDS '//' THE TOTAL UNCONSOLIDATED VOLUME
C IN STORAGE FOR THIS SEQUENCE IS ',F10.0,' CUBIC YARDS ')
SUBROUTINE UNCVOL(ELD, F)

COMMON /CCNTRL/ DIRT, COAL, COALDP, SWELL, CCVDEP, NUMPIT, NUMWAY
COMMON/ VOLUM/ FVOL(100), CVOL, VOL(200), DVOL, UVCL(200), RVCL(200),
C CONVOL, F UVOL(100), FRVOL(100), COALWT(100)
COMMON K
DIMENSION EVDIF(25,25), ELDIF(25,25)

C SWELL IS THE DECIMAL AMOUNT OF SWELL OR SHRINK OF THE EXCAVATED
CONVOL=0
DVOL=0.
K=1
DO 5 I=1,25
  DO 5 J=1,25
    IF(IPITS(I,J).EQ.0) GO TO 5
    IF((SURFC(I,J).GT.0.) .AND.(RSURF(I,J).GT.0.))GO TO 70
    IF(SURFC(I,J).LE.0.) PRINT75,SURFC(I,J),I,J
      75 FORMAT(* INPUT VALUE FOR ORIGINAL GROUND SURFACE IS', F4.0, 'AT I= ',
C I3,' AND J= ', I3, ' VOLUME OF THIS POINT IS NOT INCLUDED')
      IF(RSURF(I,J).LE.0.) PRINT 76, RSURF(I,J), I, J
      76 FORMAT(* INPUT VALUE FOR TOP OF THE ROCK IS', F4.0, ' AT I= ', I3,
C AND J= ', I3, ' VOLUME OF THIS POINT IS NOT INCLUDED')
  GO TO 5
  70 IF(DIRT.EQ.0.) GO TO 100
  IF(DIRT.EQ.1.) EVDIF(I,J)=RSURF(I,J)
  IF(DIRT.GT.1.) GO TO 115
  GO TO 110
  115 EVDIF(I,J)=DIRT
GO TO 110
  100 EVDIF(I,J)= SURFC(I,J)-RSURF(I,J)
  110 N=IPITS(I,J)
IF (N.GE.100) GO TO 5
IF (N.GT.K) K=N
UVOL(N)=UVOL(N)+EVDIF(I,J)*92.59
RVOL(N)=RVOL(N)+((ELDIF(I,J)-EVDIF(I,J))*92.59)
5 CONTINUE
DO 12 I=1,K
DVOL=DVOL+UVOL(I)
CONVOL=CONVOL+RVOL(I)
12 CONTINUE
IF (DIRT.GE.1.) GO TO 120
PRINT 21
21 FORMAT(1,/,53X,'ORIGINAL ROCK SURFACE',/,/,/,)
DO 6 I=1,25
6 PRINT 7,(RSURF(I,J),J=I,25)
7 FORMAT(4X,25(F4.0,1X),/,/,/)
120 PRINT 42, DVOL
42 FORMAT('1',TOTAL UNCONSOLIDATED CN COAL IN CUBIC YARDS ',F10.2,/
C )
DO 15 I=1,K
15 PRINT 49, I,UVOL(I)
49 FORMAT(' UNCONSOLIDATED VOLUME IN PIT ',I3,' IN CUBIC YARDS IS ',
C F10.2,/
PRINT 80,CONVOL
80 FORMAT('1',TOTAL ROCK OVER COAL IN CUBIC YARDS IS ',F10.2,/
DO 60 I=1,K
60 PRINT65,1,RVOL(I)
65 FORMAT(' ROCK IN PIT ',I3,' IN CUBIC YARDS IS ',F10.2,/
PRINT 90
90 FORMAT('1',/51X,'ISOPACK MAP CF UNCONSOLIDATED MATERIAL OVER
CCOAL ',/,/,/)
DO 81 I=1,25
81 PRINT7,(EVDIF(I,J),J=1,25)
RETURN
END
SUBROUTINE MONEY (CVCLUM)
COMMON /CONTRL/ DIRT, COAL, COALDP, SWELL, CCVDP, NUMPIT, NUMWAY
COMMON /VOLUM/ FVOL(100), CVOL, VOL(200), DVOL, UVCL(200), RVOL(200)
C CONVOL, FUVOL(100), FRVOL(100), COALWT(100)
COMMON /MCNY/ HRPROD, YDCOST, COALPR, LPIT(10, 100), WKHRS
REAL TIMEWK(100), VOLU(100), COLU(100), UVCLU(100), RVOLU(100)
C CSHNET(100)
INTEGER CVOLUM
DO 20 K=1, NUMPIT
COALU(K)=COALWT(K)
20 VOLU(K)=VCL(K)
IF (CVOLUM.EQ.2) GO TO 80
GO TO 90
80 DO 50 K=1, NUMPIT
UVOLU(K)=UVOL(K)
50 RVOLU(K)=RVOL(K)
90 DO 70 N=1, NUMWAY
CSHTOT=0.
TIMTOT=0.
PRINT 120
120 FORMAT ('ESTIMATED TIME AND CASHFLOW FOR THE PIT SEQUENCE THAT FO
CllowS')
DO 30 L=1, NUMPIT
COALWT(L)=COALU(LPIT(N,L))
VOL(L)=VOLU(LPIT(N,L))*(SWELL+1.)
TIMEWK(L)=VOL(L)/(HRPROD*WKHRS)
CSHNET(L)=(COALWT(L)*COALPR)-(VOL(L)*YDCOST)
TIMTOT=TIMTOT+TIMEWK(L)
CSHTOT=CSHTOT+CSHNET(L)
PRINT 130, LPIT(N,L), TIMEWK(L), TIMTOT, CSHNET(L), CSHTOT
130 FORMAT ('PIT ', 13, ', THE ESTIMATED TIME TO EXCAVATE THIS PIT IS ', 
1F5.1, ' WEEKS', ', THE TOTAL TIME USED IN THIS MINE TO DATE IS ', 
1F5.1, ' WEEKS', ', THE ESTIMATED NET CASH FLOW FOR THIS PIT IS ', 
3F10.0, ' DOLLARS', ', THE TOTAL ESTIMATED CASH FLOW TO DATE IS ', 
4F10.0, ' DOLLARS')
30 CONTINUE
IF (CVOLUM.EQ.2) GO TO 110
CALL MINE(N)
GO TO 70
110 DO 60 L=1,NUMPIT
    UVOL(L)=UVOLU(LPIT(N,L))*(SWELL+1.)
60    RVOL(L)=RVOLU(LPIT(N,L))*(SWELL+1.)
    CALL SEQNC(N)
70 CONTINUE
RETURN
END

SUBROUTINE PATH

C DEFINITION OF VARIABLES
C
C N IS THE NUMBER OF POINTS ON THE PATH
C
C SROUTE(N+1,2) IS A MATRIX CONTAINING THE DISTANCE TO THE NEXT POINT
C ON THE ROAD. COLUMN 1 IS NORTH-SOUTH DISTANCE COLUMN 2 IS
C EAST-WEST DISTANCE
C
C ROUTE(N+1,2) IS THE COORDINATES OF THE ROAD TO BE FOLLOWED ON EMAP.
C
C RROUTE(N) IS THE HORIZONTAL DISTANCE FROM 1 ROAD POINT TO THE NEXT
C ROAD POINT. RROUTE IS CALCULATED USING SROUTE DISTANCES.
C
C ROUTE(N) IS THE 3 DIMENSIONAL DISTANCE FROM 1 ROADPOINT TO
C THE NEXT ROAD POINT.
C
C FMAP(N) IS THE ELEVATION OF THE ROAD POINTS.
C THIS VALUE IS IN THE EMAP MATRIX.
C
C SLOPE(N) IS (FMAP/RROUTE)100
C
COMMON/ MAPS/ SURFC(25,25),COALT(25,25),IPITS(25,25),FSURFC(25,25
C), RSURF(25,25),STRIPR(25,25)
COMMON/ALTER/ CSURFC(25,25)
DIMENSION ROUTE (51,2), SROUTE (51,2), TRROUTE (50), 
FMAP(50), RROUTE (50), SLOPE (50)
READ, N
M=N-1
DO 4 K=1,N
  4 READ 5,(ROUTE(K,L),L=1,2)
5 FORMAT (F2.0,2X,F2.0)
DO 6 K=1,M
  6 SROUTE(K,1)= (ROUTE(K,1)-ROUTE(K+1,1))*50
CONTINUE
DO 7 K=1,M
  7 SROUTE(K,2)= (ROUTE(K,2)-ROUTE(K+1,2))*50
DO 8 K=1,M
  8 RROUTE(K)=(SROUTE(K,1)**2+SROUTE(K,2)**2)**(1./2.)
DO 9 K=1,M
  9 FMAP(K)=CSURFC(ROUTE(K,1),ROUTE(K,2))-CSURFC(ROUTE(K+1,1),ROUTE(K+1,2))
  10 TRROUTE(K)=(RROUTE(K)**2+FMAP(K)**2)**(1./2.)
  11 SLOPE(K)=(FMAP(K)/RROUTE(K))*100
PRINT 15 , ROUTE(1,N), ROUTE(1,2)
15 FORMAT ('SEGMENT LENGTH SEGMENT SLOPE FOR PATH BEGINNING AT ROW', F2.0, ' COLUMN', F2.0)
DO 20 K=1,M
  20 PRINT 11, TROUTE(K), SLOPE(K)
11 FORMAT ('', F10.2, 9X, F10.2)
RETURN
END
SUBROUTINE ISITIN
C THIS PROGRAM CALCULATES THE PERPENDICULAR DISTANCE TO THE SIDES OF A TR
C ANGLE AND THEN INCREMENTS THE DEPTH OF THE PILE TO SEE IF THE POINT IN QU
C ESTION IS COVERED BY FILL TO THE NEW DEPTH
READ, I,J,X1,Y1,X2,Y2,X3,Y3,ANOREP,FILDEP,AREA
ANREPO=ANDREP*,01745329
C

ANREPOIS IN RADIANS

X1 = 50.0 * X1
X2 = 50.0 * X2
X3 = 50.0 * X3
Y1 = 50.0 * Y1
Y2 = 50.0 * Y2
Y3 = 50.0 * Y3
II = 50.0 * I
JJ = 50.0 * J

IF (Y1 .EQ. Y2) Y2 = Y2 + .001
IF (X1 .EQ. X2) X2 = X2 + .001
IF (Y3 .EQ. Y1) Y3 = Y3 + .001
IF (X3 .EQ. X1) X3 = X3 + .001

SA = (Y2 - Y1) / (X2 - X1)
SB = (Y3 - Y2) / (X3 - X2)
SC = (Y3 - Y1) / (X3 - X1)

BA = Y1 - (SA * X1)
BB = Y2 - (SB * X2)
BC = Y3 - (SC * X3)

BPERA = (50.0 * J) - (((-1) / SA) * 50.0 * I)
BPERB = (50.0 * J) - (((-1) / SB) * 50.0 * I)
BPERC = (50.0 * J) - (((-1) / SC) * 50.0 * I)

XA = (BPERA - BA) / (SA + (1 / SA))
XB = (BPERB - BB) / (SB + (1 / SB))
XC = (BPERC - BC) / (SC + (1 / SC))

YA = (SA * XA) + BA
YB = (SB * XB) + BB
YC = (SC * XC) + BC

DISTA = SQRT(((XA - 50.0 * I) ** 2) + ((YA - 50.0 * J) ** 2))
DISTB = SQRT(((XB - 50.0 * I) ** 2) + ((YB - 50.0 * J) ** 2))
DISTC = SQRT(((XC - 50.0 * I) ** 2) + ((YC - 50.0 * J) ** 2))

CHANGE = FILDEP / TAN(ANREPO)
CHECKA = DISTA - CHANGE
CHECKB=DISTB-CHANGE
CHECKC=DISTC-CHANGE
IF((CHECKA.LE.0.0).AND.(CHECKB.LE.0.0).AND.(CHECKC.LE.0.0))GO TO C 100
PRINT 20,I,J,IAREA,FILDEP
20 FORMAT(*1 POINT Y= ',13,' X= ',13,' IS IN THE STORAGE PILE ON AREA C*',14,' WHEN THE PILE IS ',F4.0,' FEET HIGH*)
GO TO 110
100 CONTINUE
PRINT 10,I,J,IAREA,FILDEP
10 FORMAT(*1 POINT Y= ',13,' X= ',13,' IS NOT IN THE STORAGE PILE ON AREA C*',14,' WHEN THE PILE IS ',F4.0,' FEET HIGH*)
110 RETURN
END
SUBROUTINE SQRPLE
THIS PROGRAM FIGURES THE VOLUME OF A SQUARE OR RECT. BASED PILE
READ, X1,Y1,X2,Y2,X3,Y3,ANREPO,IAREA
ANREPO=ANREPO*.01745329
VOLPRS=0.
SIDE12=SQRT((X2-X1)**2+(Y2-Y1)**2)
SIDE23=SQRT((X3-X2)**2+(Y3-Y2)**2)
IF(SIDE12.GT.SIDE23) GO TO 10
B=SIDE23
A=SIDE12
PRINT 30,IAREA
30 FORMAT(*1 STORAGE AREA ',13,' IS SQUARE, VOLUME CALCULATED AS A PYRA CMID*)
GO TO 11
10 A=SIDE23
   B=SIDE12
   PRINT 35, IAREA
35 FORMAT('1 STORAGE AREA ', I3, ' IS RECTANGULAR, VOLUME CALCULATED AS
   C A PYRAMID PLUS A PRISM')
11 PILEHI=(TAN(ANREPO))*(A/2)
   IF (A.EQ.B) GO TO 20
   VOLPRS=(A*PILEHI/2.)*(B-A)
20 VOLTOT=(VOLPYR+VOLPRS)/27.
   PRINT 40, VOLTOT
40 FORMAT(' MAXIMUM STORAGE VOLUME IN THIS AREA IS ', F10.0, ' CUBIC YARD')
RETURN
END
SUBROUTINE TRPILE
   THIS PROGRAM FIGURES THE VOLUME OF A TRIANGLE BASED PYRAMID
READ, XI, Y1, X2, Y2, X3, Y3, ANOREP, IAREA
   ANREPO=ANOREP*.017453293
   XI=50.*XI
   X2=50.*X2
   X3=50.*X3
   Y1=50.*Y1
   Y2=50.*Y2
   Y3=50.*Y3
   IF (Y1.EQ.Y2) Y2=Y2+.001
   IF (X1.EQ.X2) X2=X2+.001
   IF (Y3.EQ.Y1) Y3=Y3+.001
   IF (X3.EQ.X1) X3=X3+.001
   IF (Y3.EQ.Y2) Y3=Y3+.001
   IF (X3.EQ.X2) X3=X3+.001
   SIDE12=SQRT(((X1-X2)**2)+((Y1-Y2)**2))
   SIDE23=SQRT(((X2-X3)**2)+((Y2-Y3)**2))
   SIDE31=SQRT(((X3-X1)**2)+((Y3-Y1)**2))
ANGLE3 = \arccos\left(\frac{(\text{SIDE23}^2 + \text{SIDE31}^2 - \text{SIDE12}^2)}{2\text{SIDE23}\text{SIDE31}}\right)
\text{ANGLE2} = \arccos\left(\frac{(\text{SIDE12}^2 + \text{SIDE23}^2 - \text{SIDE31}^2)}{2\text{SIDE23}\text{SIDE12}}\right)
\text{HALF3} = \frac{\text{ANGLE3}}{2}.
\text{HALF2} = \frac{\text{ANGLE2}}{2}.
\text{THETA} = 3.14159 \times (\text{HALF3} + \text{HALF2})
\text{BIS2} = (\sin(\text{HALF3}) \times \text{SIDE23}) / (\sin(\text{THETA}))
\text{FLATD} = (\sin(\text{HALF2}) \times \text{BIS2})
\text{PILEH1} = \text{FLATD} \times (\tan(\text{ANREPO}))
\text{TRIHI} = \sin(\text{ANGLE2}) \times \text{SIDE23}
\text{AREA} = (\text{TRIHI} \times \text{SIDE12}) / 2.0
\text{VOLUME} = ((\text{AREA} \times \text{PILEH1}) / 3.0) / 27.
\text{PRINT 10, AREA, VOLUME, PILEH1}
\text{10 FORMAT ('AREA', 14, 'HAS A MAXIMUM STORAGE VOLUME OF ', F10.0, ' CUBIC YARDS WITH A PILE HEIGHT OF ', F5.0)}
\text{RETURN}
\text{END}
CVOLUM HAS BEEN SET AT 1
ISURF HAS BEEN SET AT 1
DIRT HAS BEEN SET AT 1.
DENSITY OF THE COAL HAS BEEN SET AT 1000 LBS/BANK CUBIC YARD
THE THICKNESS OF THE COAL SEAM HAS BEEN SET AT 3 FEET
THE OVERBURDEN SWELL HAS BEEN SET AT 0.10%
THE DEPTH OF UNCONSOLIDATED COVER IN THE FINAL PROFILE OF TWO LAYER SYSTEM HAS BEEN SET AT 10 FEET
THE NUMBER OF PITS TO BE MINED IS 12
THE NUMBER OF SEQUENCES OF MINING WILL BE 2
THE HOURLY PRODUCTION RATE FOR THE MINE OVERBURDEN REMOVAL FLEET IS 450 BANK CUBIC YARDS PER HOUR
THE AVERAGE ESTIMATED COST OF MOVING A BANK CUBIC YARD OF OVERBURDEN IS $0.60
THE AVERAGE ESTIMATED COAL SELLING PRICE PER TON AT THE MINE IS $15.00
THE NUMBER OF HOURS THAT THE OVERBURDEN FLEET WILL WORK PER WEEK IS 50.0
ORIGINAL GROUND SURFACE

804, 806, 809, 809, 806, 801, 795, 789, 782, 776, 770, 768, 768, 769, 769, 767, 766, 755, 764, 761, 757, 753, 749, 745, 738,

797, 798, 803, 804, 803, 797, 793, 787, 779, 772, 767, 764, 763, 763, 762, 759, 758, 758, 754, 752, 750, 745, 742, 737,

791, 792, 792, 793, 791, 786, 782, 777, 774, 768, 764, 762, 760, 752, 751, 750, 749, 748, 748, 746, 744, 741, 737, 733,

784, 787, 792, 793, 792, 787, 782, 778, 769, 762, 758, 758, 756, 746, 746, 747, 746, 746, 744, 742, 739, 735, 735, 733,

782, 784, 786, 789, 787, 783, 778, 769, 763, 756, 755, 751, 747, 744, 741, 742, 741, 738, 737, 734, 732, 730, 726,

788, 778, 782, 784, 786, 781, 777, 774, 769, 763, 753, 750, 751, 746, 739, 737, 737, 736, 734, 732, 730, 727, 723,

777, 774, 776, 780, 780, 776, 771, 767, 763, 760, 756, 749, 747, 747, 745, 741, 737, 732, 727, 725, 723, 722, 722,

772, 770, 771, 775, 777, 777, 775, 769, 763, 760, 757, 753, 748, 742, 742, 743, 743, 739, 733, 730, 726, 723, 722, 721, 719,

770, 766, 766, 767, 770, 771, 770, 767, 761, 756, 752, 748, 746, 741, 734, 738, 739, 739, 737, 734, 732, 728, 724, 719, 717,

770, 763, 762, 763, 764, 766, 766, 764, 760, 755, 748, 745, 741, 738, 732, 731, 732, 736, 736, 734, 731, 728, 723, 719, 715,

770, 763, 758, 759, 760, 761, 762, 760, 758, 754, 748, 743, 738, 733, 732, 730, 730, 735, 732, 730, 728, 726, 723, 715,

769, 764, 759, 752, 753, 754, 754, 755, 754, 752, 749, 744, 741, 734, 732, 731, 738, 728, 728, 727, 726, 724, 722, 719, 715,
<table>
<thead>
<tr>
<th>Pit</th>
<th>Volume in Cubic Yards</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>25647.42</td>
</tr>
<tr>
<td>2</td>
<td>102681.80</td>
</tr>
<tr>
<td>3</td>
<td>129532.00</td>
</tr>
<tr>
<td>4</td>
<td>150272.00</td>
</tr>
<tr>
<td>5</td>
<td>166862.40</td>
</tr>
<tr>
<td>6</td>
<td>193697.30</td>
</tr>
<tr>
<td>7</td>
<td>235733.10</td>
</tr>
<tr>
<td>8</td>
<td>364062.40</td>
</tr>
<tr>
<td>9</td>
<td>100709.00</td>
</tr>
<tr>
<td>10</td>
<td>6941.69</td>
</tr>
<tr>
<td>11</td>
<td>54165.10</td>
</tr>
<tr>
<td>12</td>
<td>86664.75</td>
</tr>
</tbody>
</table>

**Total Site Volume Over Coal in Cubic Yards:** 1769755.00
<table>
<thead>
<tr>
<th>TOTAL AREA OVER COAL IN SQUARE FEET IS 1022500.00</th>
</tr>
</thead>
<tbody>
<tr>
<td>AREA IN PIT 1 IN SQUARE FEET IS 52500.00</td>
</tr>
<tr>
<td>AREA IN PIT 2 IN SQUARE FEET IS 75000.00</td>
</tr>
<tr>
<td>AREA IN PIT 3 IN SQUARE FEET IS 90000.00</td>
</tr>
<tr>
<td>AREA IN PIT 4 IN SQUARE FEET IS 90000.00</td>
</tr>
<tr>
<td>AREA IN PIT 5 IN SQUARE FEET IS 90000.00</td>
</tr>
<tr>
<td>AREA IN PIT 6 IN SQUARE FEET IS 90000.00</td>
</tr>
<tr>
<td>AREA IN PIT 7 IN SQUARE FEET IS 90000.00</td>
</tr>
<tr>
<td>AREA IN PIT 8 IN SQUARE FEET IS 120000.00</td>
</tr>
<tr>
<td>AREA IN PIT 9 IN SQUARE FEET IS 117500.00</td>
</tr>
<tr>
<td>AREA IN PIT 10 IN SQUARE FEET IS 82500.00</td>
</tr>
<tr>
<td>AREA IN PIT 11 IN SQUARE FEET IS 60000.00</td>
</tr>
<tr>
<td>AREA IN PIT 12 IN SQUARE FEET IS 65000.00</td>
</tr>
<tr>
<td>Pit</td>
</tr>
<tr>
<td>-----</td>
</tr>
<tr>
<td>1</td>
</tr>
<tr>
<td>2</td>
</tr>
<tr>
<td>3</td>
</tr>
<tr>
<td>4</td>
</tr>
<tr>
<td>5</td>
</tr>
<tr>
<td>6</td>
</tr>
<tr>
<td>7</td>
</tr>
<tr>
<td>8</td>
</tr>
<tr>
<td>9</td>
</tr>
<tr>
<td>10</td>
</tr>
<tr>
<td>11</td>
</tr>
<tr>
<td>12</td>
</tr>
</tbody>
</table>

Strip ratio for the area with coal underneath is 17.31
ISOPACK MAP OF COAL OVERBURDEN
HAP OF STRIP RATIO, VALUE SHOWN IS CUBIC YARDS OF OVERDUREN PER TON OF COAL

<table>
<thead>
<tr>
<th>37</th>
<th>38</th>
<th>39</th>
<th>38</th>
<th>36</th>
<th>33</th>
<th>31</th>
<th>28</th>
<th>26</th>
<th>24</th>
<th>23</th>
<th>22</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
<th>21</th>
</tr>
</thead>
<tbody>
<tr>
<td>35</td>
<td>35</td>
<td>37</td>
<td>37</td>
<td>34</td>
<td>34</td>
<td>30</td>
<td>27</td>
<td>24</td>
<td>23</td>
<td>21</td>
<td>21</td>
<td>21</td>
<td>20</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td></td>
</tr>
<tr>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>32</td>
<td>30</td>
<td>29</td>
<td>27</td>
<td>25</td>
<td>23</td>
<td>21</td>
<td>21</td>
<td>20</td>
<td>17</td>
<td>16</td>
<td>16</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>30</td>
<td>31</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>32</td>
<td>30</td>
<td>27</td>
<td>25</td>
<td>23</td>
<td>21</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>18</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>29</td>
<td>30</td>
<td>30</td>
<td>31</td>
<td>31</td>
<td>29</td>
<td>27</td>
<td>26</td>
<td>24</td>
<td>23</td>
<td>21</td>
<td>19</td>
<td>19</td>
<td>18</td>
<td>16</td>
<td>15</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>28</td>
<td>29</td>
<td>30</td>
<td>30</td>
<td>29</td>
<td>27</td>
<td>25</td>
<td>23</td>
<td>21</td>
<td>17</td>
<td>16</td>
<td>16</td>
<td>15</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>28</td>
<td>28</td>
<td>29</td>
<td>30</td>
<td>29</td>
<td>27</td>
<td>25</td>
<td>23</td>
<td>21</td>
<td>17</td>
<td>16</td>
<td>16</td>
<td>15</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>26</td>
<td>26</td>
<td>27</td>
<td>26</td>
<td>27</td>
<td>24</td>
<td>24</td>
<td>23</td>
<td>21</td>
<td>20</td>
<td>19</td>
<td>18</td>
<td>16</td>
<td>15</td>
<td>15</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>26</td>
<td>25</td>
<td>25</td>
<td>26</td>
<td>27</td>
<td>26</td>
<td>24</td>
<td>24</td>
<td>21</td>
<td>20</td>
<td>19</td>
<td>18</td>
<td>16</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>27</td>
<td>26</td>
<td>24</td>
<td>23</td>
<td>24</td>
<td>25</td>
<td>25</td>
<td>21</td>
<td>23</td>
<td>21</td>
<td>19</td>
<td>17</td>
<td>15</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>25</td>
<td>25</td>
<td>25</td>
<td>26</td>
<td>27</td>
<td>27</td>
<td>26</td>
<td>26</td>
<td>22</td>
<td>23</td>
<td>21</td>
<td>19</td>
<td>17</td>
<td>15</td>
<td>14</td>
<td>13</td>
<td>11</td>
<td>10</td>
<td>9</td>
<td>8</td>
<td>7</td>
<td>6</td>
<td></td>
</tr>
</tbody>
</table>

MAP OF STRIP RATIO, VALUE SHOWN IS CUBIC YARDS OF OVERDUREN PER TON OF COAL
The total amount of coal for the defined area is **102247** tons.

<table>
<thead>
<tr>
<th>Pit</th>
<th>Tons of Coal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pit 1</td>
<td>5250</td>
</tr>
<tr>
<td>Pit 2</td>
<td>7500</td>
</tr>
<tr>
<td>Pit 3</td>
<td>9000</td>
</tr>
<tr>
<td>Pit 4</td>
<td>9000</td>
</tr>
<tr>
<td>Pit 5</td>
<td>9000</td>
</tr>
<tr>
<td>Pit 6</td>
<td>9000</td>
</tr>
<tr>
<td>Pit 7</td>
<td>9000</td>
</tr>
<tr>
<td>Pit 8</td>
<td>12000</td>
</tr>
<tr>
<td>Pit 9</td>
<td>11750</td>
</tr>
<tr>
<td>Pit 10</td>
<td>8250</td>
</tr>
<tr>
<td>Pit 11</td>
<td>6000</td>
</tr>
<tr>
<td>Pit 12</td>
<td>6500</td>
</tr>
</tbody>
</table>
The volume cut from pit 1 in cubic yards is 22221.60
<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>32962</td>
<td>112126</td>
<td>146569</td>
<td>161846</td>
<td>170272</td>
<td>190179</td>
<td>212771</td>
<td>339804</td>
<td>215733</td>
<td>115644</td>
<td>74350</td>
<td>78146</td>
</tr>
</tbody>
</table>

The total fill available under the final surface in cubic yards is 1850400.
ISOPACH MAP OF FINAL PIT FILL
ESTIMATED TIME AND CASHFLOW FOR THE PIT SEQUENCE THAT FollowS

PIT 12
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 3.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 3.3 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 53490.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 53490.00 DOLLARS

PIT 11
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 2.6 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 5.9 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 54240.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 107747.00 DOLLARS

PIT 10
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 4.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 10.2 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 66365.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 174111.00 DOLLARS

PIT 9
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 9.2 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 19.4 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 51643.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 225735.00 DOLLARS

PIT 8
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 17.0 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 37.2 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS -60287.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 165468.00 DOLLARS

PIT 7
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 11.5 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 48.7 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS -20508.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 144000.00 DOLLARS

PIT 6
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 9.7 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 58.4 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 3056.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 140736.00 DOLLARS

PIT 5
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 0.1 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 66.6 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 25061.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 173797.00 DOLLARS

PIT 4
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 7.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 73.9 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 35816.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 209613.00 DOLLARS

PIT 3
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 6.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 00.2 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 49504.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 259117.00 DOLLARS

PIT 2
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 5.0 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 05.3 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 44727.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 303044.00 DOLLARS

PIT 1
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 1.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 06.5 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 61020.00 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 365664.00 DOLLARS
<table>
<thead>
<tr>
<th>Status of Pit Filling</th>
<th>Cubic Yards of Fill After the Excavation of Pit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 95632</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 10HAD 146569</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 11HAD 112126</td>
<td>Pit 10HAD 146569</td>
</tr>
<tr>
<td>Pit 12HAD 32962</td>
<td>Pit 10HAD 146569</td>
</tr>
</tbody>
</table>
Pit 1 had 142486 cubic yards of fill after the excavation of Pit 2 was completed.
Pit 3 had 112950 cubic yards of fill after the excavation of Pit 2 was completed.
Pit 12 had 32962 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 11 had 112126 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 10 had 146569 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 9 had 161546 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 8 had 170272 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 7 had 190179 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 6 had 163225 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 5 had 165300 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 4 had 142486 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 3 had 112950 cubic yards of fill after the excavation of Pit 1 was completed.
Pit 2 had 28212 cubic yards of fill after the excavation of Pit 1 was completed.

The total volume in storage after the excavation of the pit is 500604 cubic yards.
The rehandle for this pit sequence is 26.8.
ESTIMATED TIME AND CASHFLOW FOR THE PIT SEQUENCE THAT FOLLOWS

PIT 1
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 1.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 1.3 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 61020.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 61020.0 DOLLARS

PIT 2
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 5.0 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 6.3 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 44727.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 106547.0 DOLLARS

PIT 3
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 6.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 12.6 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 49504.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 156051.0 DOLLARS

PIT 4
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 7.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 20.0 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 35816.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 191867.0 DOLLARS

PIT 5
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 8.1 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 28.1 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 25061.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 216928.0 DOLLARS

PIT 6
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 9.7 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 37.0 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 3056.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 220704.0 DOLLARS

PIT 7
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 11.5 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 49.3 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS -20588.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 200196.0 DOLLARS

PIT 8
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 17.0 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 67.1 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS -60287.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS 139909.0 DOLLARS

PIT 9
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 9.2 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 76.4 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS 51643.0 DOLLARS
THE TOTAL ESTIMATED CASH FLOW TO DATE IS $191,553.00

PIT 10
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 4.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 80.6 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS $66,365.00
THE TOTAL ESTIMATED CASH FLOW TO DATE IS $257,917.00

PIT 11
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 2.6 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 83.3 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS $54,248.00
THE TOTAL ESTIMATED CASH FLOW TO DATE IS $312,166.00

PIT 12
THE ESTIMATED TIME TO EXCAVATE THIS PIT IS 3.3 WEEKS
THE TOTAL TIME USED IN THIS MINE TO DATE IS 86.5 WEEKS
THE ESTIMATED NET CASH FLOW FOR THIS PIT IS $53,498.00
THE TOTAL ESTIMATED CASH FLOW TO DATE IS $365,664.00
STATUS OF PIT FILLING

PIT 1 had 32,962 cubic yards of fill after the excavation of pit 2 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 3 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 3 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 4 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 4 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 5 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 5 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 5 was completed
PIT 4 had 161,046 cubic yards of fill after the excavation of pit 5 was completed
PIT 5 had 170,272 cubic yards of fill after the excavation of pit 6 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 6 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 6 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 6 was completed
PIT 4 had 161,046 cubic yards of fill after the excavation of pit 6 was completed
PIT 5 had 170,272 cubic yards of fill after the excavation of pit 6 was completed
PIT 6 had 190,179 cubic yards of fill after the excavation of pit 7 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 7 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 7 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 7 was completed
PIT 4 had 161,046 cubic yards of fill after the excavation of pit 7 was completed
PIT 5 had 170,272 cubic yards of fill after the excavation of pit 7 was completed
PIT 6 had 190,179 cubic yards of fill after the excavation of pit 8 was completed
PIT 7 had 212,771 cubic yards of fill after the excavation of pit 8 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 9 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 9 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 9 was completed
PIT 4 had 161,046 cubic yards of fill after the excavation of pit 9 was completed
PIT 5 had 170,272 cubic yards of fill after the excavation of pit 9 was completed
PIT 6 had 190,179 cubic yards of fill after the excavation of pit 9 was completed
PIT 7 had 212,771 cubic yards of fill after the excavation of pit 9 was completed
PIT 8 had 207,669 cubic yards of fill after the excavation of pit 9 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 10 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 10 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 10 was completed
PIT 4 had 161,046 cubic yards of fill after the excavation of pit 10 was completed
PIT 5 had 170,272 cubic yards of fill after the excavation of pit 10 was completed
PIT 6 had 190,179 cubic yards of fill after the excavation of pit 10 was completed
PIT 7 had 212,771 cubic yards of fill after the excavation of pit 10 was completed
PIT 8 had 207,669 cubic yards of fill after the excavation of pit 10 was completed
PIT 9 had 95,036 cubic yards of fill after the excavation of pit 11 was completed
PIT 1 had 32,962 cubic yards of fill after the excavation of pit 11 was completed
PIT 2 had 112,126 cubic yards of fill after the excavation of pit 11 was completed
PIT 3 had 146,569 cubic yards of fill after the excavation of pit 11 was completed
PIT 4 had 161,046 cubic yards of fill after the excavation of pit 11 was completed
PIT 5 had 170,272 cubic yards of fill after the excavation of pit 11 was completed
PIT 6 had 190,179 cubic yards of fill after the excavation of pit 11 was completed
PIT 7 had 212,771 cubic yards of fill after the excavation of pit 11 was completed
PIT 8 had 207,669 cubic yards of fill after the excavation of pit 11 was completed
<table>
<thead>
<tr>
<th>PIT</th>
<th>Volume (cubic yards)</th>
</tr>
</thead>
<tbody>
<tr>
<td>9HAO</td>
<td>95636.1</td>
</tr>
<tr>
<td>10HAO</td>
<td>59582.5</td>
</tr>
<tr>
<td>1HAO</td>
<td>32962.1</td>
</tr>
<tr>
<td>2HAO</td>
<td>112126.0</td>
</tr>
<tr>
<td>3HAO</td>
<td>145569.0</td>
</tr>
<tr>
<td>4HAO</td>
<td>161846.0</td>
</tr>
<tr>
<td>5HAO</td>
<td>179272.0</td>
</tr>
<tr>
<td>6HAO</td>
<td>190179.0</td>
</tr>
<tr>
<td>7HAO</td>
<td>212771.0</td>
</tr>
<tr>
<td>8HAO</td>
<td>207669.0</td>
</tr>
<tr>
<td>9HAO</td>
<td>95636.1</td>
</tr>
<tr>
<td>10HAO</td>
<td>59582.5</td>
</tr>
<tr>
<td>11HAO</td>
<td>73331.0</td>
</tr>
</tbody>
</table>

The total volume in storage after the excavation of the pit is 483709.9 cubic yards. The rehandle for this pit sequence is 27.4%.
<table>
<thead>
<tr>
<th>Segment Length</th>
<th>Segment Slope</th>
</tr>
</thead>
<tbody>
<tr>
<td>250.10</td>
<td>2.80</td>
</tr>
<tr>
<td>100.00</td>
<td>4.00</td>
</tr>
<tr>
<td>141.40</td>
<td>2.03</td>
</tr>
<tr>
<td>150.65</td>
<td>9.33</td>
</tr>
<tr>
<td>70.74</td>
<td>-2.03</td>
</tr>
<tr>
<td>112.02</td>
<td>-6.26</td>
</tr>
<tr>
<td>158.92</td>
<td>-10.12</td>
</tr>
<tr>
<td>207.43</td>
<td>-11.16</td>
</tr>
<tr>
<td>70.82</td>
<td>5.66</td>
</tr>
<tr>
<td>111.92</td>
<td>4.47</td>
</tr>
<tr>
<td>70.82</td>
<td>-5.66</td>
</tr>
<tr>
<td>112.68</td>
<td>12.52</td>
</tr>
</tbody>
</table>
Storage area 102 is rectangular. Volume calculated as a pyramid plus a prism.

Maximum storage volume in this area is 31527 cubic yards.
Area 101 has a maximum storage volume of 23,072 cubic yards with a pile height of 105.
POINT Y = 17 X = 17 IS IN THE STORAGE PILE ON AREA 101 WHEN THE PILE IS 3 FEET HIGH.
APPENDIX C. LISTING OF CALCULATIONS FOR ENERGY EFFICIENCY INFORMATION IN PART III

This appendix provides more detail on the calculations presented in Part III. The methods used are similar to those presented in the manufacturers' literature cited.
PART 1
Large Bulldozer, Scrapers

Plan view of proposed mine plan at ICPM#1 for consolidated material.

Cross section of pit 1 with coal uncovered.

\[
\text{pit is } 50 \text{ ft } \times 1600 \text{ ft } \times 6 \text{ ft shale } = 18,000 \text{ yd}^3
\]

Ripping 1200 BCY/hr \(\overline{\text{15 hr}}\)
Dozing 924 BCY/hr \(\overline{\text{19.5 hr}}\)
\(\overline{\text{34.5 hr}}\)

Cross section of final pit showing stockpile before reclamation
Production Summary:

Material moves 383 ft centroid of undisturbed to centroid of stockpile

- average production rate 250 BCY/hr dozing (initial)
- Fiat-Allis average production rate 1200 BCY/hr ripping
- Information average production rate 320 BCY/hr radius

$1.0 \times 10^6 \text{ yd}^3$ of rock to move $4000$ hr

\[ \text{Fiat-Allis est.} \]
\[ \begin{array}{c}
\text{hr} \\
\$100/
\end{array} \\
\$400,000 \]

$1.0 \times 10^6 \text{ yd}^3$ of rock to rip $833$ hr

\[ \begin{array}{c}
\text{hr} \\
\$130/
\end{array} \\
\$108,333 \]

$0.6 \times 10^6 \text{ yd}^3$ rock reclaim $1875$ hr

\[ \begin{array}{c}
\text{hr} \\
\$100/
\end{array} \\
\$187,500 \]

Handling unconsolidated

$300,000$ BCY unconsolidated will be handled for $50c/\text{yd}^3$ to stockpile and $50c/\text{yd}^3$ back to final position.

$300,000$ BCY @ $1.00 \quad \underline{\text{-----}} \quad \$300,000$

Total cost = $995,833$

Comments

1. Rental rate for dozer includes profit
2. Unconsolidated handling is on contract
PART 2

40 yd$^3$ Dragline, Scrapers

Plan view of proposed mine plan showing pit orientation for unconsolidated material.

Cross section of a pit at completion of coal uncover cycle with dragline.

Cost estimating for 40 yd$^3$ dragline

Drilling (Jennerjohn)
4.75 in. holes for blasting @ 80c/ft
15 x 15 ft spacing gives 3552 holes 60 ft deep, total cost = $170,543

Blasting (Otte) $0.07/yd$^3$

$(1.0 \times 10^6 \text{ yd}^3) \times 0.07 = $70,000$
Dozer grading

Take half of the peak and doze it into the adjacent valley

21.7 yd³ per lineal feet of pit. 7000 lineal ft of pit. 15 x 10⁴ yd³
@ $0.40 gives $60,000

Backfilling final pit

vol 7875 ft³/lineal feet of last pit
total vol (7875) 1000 ft =
7,875,000 ft³
7,875,000/27 ft³/yd³ = 291,667 yd³

This calculation assumes that the last pit is parallel to one edge of the property.

Dragline Owning and Operating Costs

Machine cost, 1974 (Marion Power Shovel Co.) $3,614,342
Machine life 20 yr 7200 hr/yr total hr 144,000
Depreciation/hr 25.10
Average investment 1,897,529.50
Int. 8%, tax 3.5%, ins. 1.5% = 13% 246,678/yr
Int., tax, ins. cost/hr 34.26
Total hourly ownership cost 59.36
Hourly production 1,053 yd³
Ownership cost/yd³ $0.06
Maintenance and supply cost/hr $30.54
Electricity cost/hr $27.07
Operator cost $8.00/hr
Oiler cost $6.00/hr
Total operating cost/hr $71.61/hr
Total operating cost/yd³ $0.068
Owning and operating cost/yd³ $0.13
Production estimation 40 yd³ bucket
Required dumping height dragline 36.8 ft
Required operating radius dragline 190.2 ft
Bank yd factor .635
Average swing 90° cycle time 57 sec
Cycles/hr 63
Propel time factor .94
Operating efficiency .70
Bank cubic yd, per hour 100% efficiency
\[ 40 \times 0.635 \times 63 \times 0.94 = 1504 \]
Bank cubic yd at 70% efficiency 1053
Capital energy for hypothetical site
Dragline 30.8 x 10⁹ BTU \( \frac{2 \text{ month use}}{240 \text{ month life}} \) = 0.26 x 10⁹ BTU
Unconsolidated (same as large dozer system) = 0.60 x 10⁹ BTU
Reclamation grading 1.4 x 10⁹ BTU \( \frac{150 \text{ hr use}}{10,000 \text{ hr life}} \) = 0.02 x 10⁹ BTU
Backfilling last pit 1.4 x 10⁹ BTU \( \frac{2,000 \text{ hr use}}{10,000 \text{ hr life}} \) = 0.28 x 10⁹ BTU
Drilling 0.6 x 10⁹ BTU \( \frac{4,000 \text{ hr use}}{10,000 \text{ hr life}} \) = 0.24 x 10⁹ BTU
Total = 0.14 x 10¹⁰ BTU
PART 3

5 yd$^3$ Dragline, Scrapers

Plan view of proposed mine plan showing pit orientation for consolidated material.

Cross section of pit at completion of coal uncovering cycle with dragline.

Cost Estimating

Drilling and blasting same as for 40 yd machine

Dozer grading

Take half of peak and doze it into the adjacent valley.

\[
3.7 \text{ yd}^3/\text{lineal feet} \times (17,765 \text{ ft}) = 65,000 \text{ yd}^3
\]

\[
65,000 \text{ yd}^3 \times (\$0.40/\text{yd}^3) = \$26,000
\]
Backfilling final pit

Dragline Owning and Operating Costs

**Machine cost, 1974** $375,000

Sales tax 3% 11,250
Extras 10% 37,500
Freight 1,938

Subtotal $429,601

Ballast 10 tons 800
Erection 3,000

Total $434,401

Life of machine 10 yr 2000 hr/yr

Depr./hr $ 21.67/hr

Avg. invest. $238,370

Int. 8.5%, tax 3.5%, ins. 1.5% $ 32,180
Int. 8.5%, tax 3.5%, ins. 1.5%/hr $ 16./hr

Total hourly ownership cost $ 37.67/hr

Operating hr/yr at 68% eff. 1,360.

Maintenance and supply/hr $ 22./hr

Fuel cost/hr 13 gal at $0.40 $ 5.20/hr

Operator rate/hr $ 5.00/hr

Total operating cost/hr $ 32.20/hr

vol 4200 ft³/lineal feet
total (4200)(1000 ft)
(27 ft³/yd³) =
155,556 yd³
Total owning and operating cost/hr $69.87/hr
Total owning and operating cost/yard^3 $0.58/yard

Machine Production

Bank yard factor 0.56
Cycle time 55.4 sec
Cycles/hr 65
Propel time factor 0.96
BCY/cycle 2.8

BCY/hr at 100% eff. 65 cycles (0.96) 2.8 = 174.7
Operating eff. fair to good 0.68

BCY/hr 174.7 (0.68) = 118.8

Capital energy use

Dragline $2.8 \times 10^9$ BTU (4.4 yr use) (10 yr life) = $1.2 \times 10^9$ BTU

Unconsolidated (same as large dozer) system = $0.6 \times 10^9$ BTU

Reclamation grading $0.9 \times 10^9$ BTU

(150 hr use) (10,000 hr life) = $0.01 \times 10^9$ BTU

Backfilling last pit $1.4 \times 10^9$ BTU

(1,100 hr use) (10,000 hr life) = $0.15 \times 10^9$ BTU

Drilling (same as 40 yard^3 system) = $0.24 \times 10^9$ BTU

Total for Site = $0.22 \times 10^{10}$ BTU
APPENDIX D. EXPLANATION OF SECONDARY SUBROUTINES

This appendix provides an explanation of the secondary subroutines that can be used in conjunction with CAMP but are not required for the basic mine planning calculations.
Subroutine SURFAL

This subroutine lowers the surface in a designated pit, in one foot increments, parallel to the original surface. The variable ICUTMX controls the maximum depth of cut for one call to SURFAL. The elevation of each cut is checked so that the cut is not made deeper than the top of the coal.

When the pit has been excavated to the depth specified, the subroutine calculates the volume excavated using the factor 92.59 yd³/ft of column.

Table D.1 lists the variables used in the SURFAL subroutine.

Table D.1. Variables in SURFAL

<table>
<thead>
<tr>
<th>Variable</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>CSURFC (I, J)</td>
<td>Elevation of cut surface at grid points</td>
</tr>
<tr>
<td>ICUTMX</td>
<td>Maximum depth of cut for one execution of SURFAL</td>
</tr>
<tr>
<td>ILOC</td>
<td>Pit to be excavated</td>
</tr>
<tr>
<td>IDEEP</td>
<td>Index of the depth of the excavation</td>
</tr>
<tr>
<td>CUTDEX</td>
<td>Summation of the 1 ft increment cuts</td>
</tr>
<tr>
<td>IPITS (I, J)</td>
<td>Designation of pits</td>
</tr>
<tr>
<td>COALT (I, J)</td>
<td>Top of coal</td>
</tr>
</tbody>
</table>
Subroutine PATH

This subroutine calculates the distance along a map surface and the slope of the line of travel from horizontal. The calculations require a surface map defined by elevations at grid points and a list of grid points at the end of straight line segments of the route.

The information generated by this subroutine can be used as input to several of the construction equipment companies' cycle time programs which estimate the travel time of a machine for a route. Otte and Boehlje (1975), incorporated a cycle time program in their model.

The distance between specified grid points on the map is calculated by determining the X and Y distances between the points and then the straight line horizontal distance between the points. The difference in elevation between the points is then used to get the straight line distance on the surface between the points and then the slope of the path connecting the grid points.

The subroutine can handle 50 individual points as written but the number can be changed by changing the dimensions of the appropriate matrices.

The origin of the map is in the northwest or top left corner of the map with numbers increasing positively down and to the right.

The variables are listed in Table D.2.
Table D.2. Variables used in subroutine PATH

<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>EMAP</td>
<td>The matrix of elevations of the surface to be traveled</td>
</tr>
<tr>
<td>ROUTE</td>
<td>The coordinates of the ends of segments on the path to be followed</td>
</tr>
<tr>
<td>SROUTE</td>
<td>The distance to the next point in ROUTE. Column 1 has the North-South distance. Column 2 has the East-West distance.</td>
</tr>
<tr>
<td>RROUTE</td>
<td>The horizontal distance from one point on the path to the next.</td>
</tr>
<tr>
<td>TROUTE</td>
<td>The distance on the map surface from one point to the next.</td>
</tr>
<tr>
<td>FMAP</td>
<td>The difference in elevation between two succeeding points in ROUTE.</td>
</tr>
</tbody>
</table>

Subroutine ISITIN

This subroutine calculates the distance from a point in a triangle to the three sides of a triangle. The fill depth is then increased and the point is checked again to see if the point at the new level would be inside or outside of the piled material.

ISITIN is particularly useful when surface alterations are being performed with the excavated material hauled to a triangular spoil area. The output of this subroutine can be used to help define the surface of a stockpile for either contour or three dimensional plotting.

Figure D.1 defines the variables used in this subroutine.
$X_1, Y_1$ are the coordinates of point 1.

$X_2, Y_2$ are the coordinates of point 2.

$X_3, Y_3$ are the coordinates of point 3.

A, B, C are the lines between points as shown.

SA, SB, SC are the slopes of A, B, C respectively.

BA, BB, BC are the intercepts of A, B, C on the X axis.

BPERA, BPERB, BPERC are the intercepts of lines perpendicular to A, B, C.

$(X_A, Y_A)$ are the coordinates of a point on A and a line perpendicular to A.

$(X_B, Y_B)$ are the coordinates of a point on B and a line perpendicular to B.

$(X_C, Y_C)$ are the coordinates of a point on C and a line perpendicular to C.

DISTA is the distance from the point being considered to line A.

DISTB, DISTC are similar for lines B and C.

CHANGE is the amount that line A has shifted inward due to piling material.

CHECKA, B, and C are the amounts that the distances would be altered.

If CHECKA, B, or C goes negative, the grid point at the new elevation would be outside the new smaller triangle.

Figure D.1. Variables used in subroutine ISITIN
**Subroutine SQRPLE**

This subroutine calculates the maximum volume of storage areas which have square or rectangular bases. The volume of a square based pile is calculated as the volume of a pyramid and the rectangular based pile is split into a pyramid and a prism.

SQRPLE requires the coordinates of three corner points and the angle of repose of the material to be stockpiled. The shale and unconsolidated material in stockpiles at ICPEM had a maximum angle of repose of 31° to the horizontal.

The subroutine assumes that the base of the pile is flat. The calculation of a volume with a square or rectangular base is determined by a comparison of the lengths of the sides.

Table D.3 lists the variables used in SQRPLE.

**Subroutine TRPILE**

This subroutine calculates the maximum volume that can be piled in a triangular storage area to make pyramid shaped piles.

Figure D.2 shows the relationship of the variables used in TRPILE.

The subroutine requires the coordinates at the corners of the triangular base and the angle of repose of the pile in degrees as inputs. The volume that is calculated is based on a flat bottom. Unless the site is very uneven or has a slope over 10% this will probably not cause a large error. The user must remember that these are estimates.
<table>
<thead>
<tr>
<th>Variables</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>((X_1, X_1))</td>
<td>Coordinates of point one</td>
</tr>
<tr>
<td>((X_2, Y_2))</td>
<td>Coordinates of point two</td>
</tr>
<tr>
<td>((X_3, Y_3))</td>
<td>Coordinates of point three</td>
</tr>
<tr>
<td>SIDE 12</td>
<td>Length of the side between points 1 and 2</td>
</tr>
<tr>
<td>SIDE 23</td>
<td>Length of the side between points 2 and 3</td>
</tr>
<tr>
<td>ANOREP</td>
<td>Angle of repose of the pile in degrees</td>
</tr>
<tr>
<td>ANREPO</td>
<td>Angle of repose in radians</td>
</tr>
<tr>
<td>PILEHI</td>
<td>Maximum height of storage pile</td>
</tr>
<tr>
<td>VOLPYR</td>
<td>Volume of pyramid</td>
</tr>
<tr>
<td>VOLPRS</td>
<td>Volume of prism</td>
</tr>
<tr>
<td>VOLTOT</td>
<td>Total volume of storage area</td>
</tr>
</tbody>
</table>
PILEHI is the height of the pyramid.

TRIHI is the height of the triangular base.

AREA is the area of the base.

VOLUME is the volume of the pyramid.

Figure D.2. Variables used in subroutine TRPILE
APPENDIX E. GLOSSARY

This appendix is the glossary of subroutine names, abbreviations and variable names used in the Computer Aided Mine Planner.
### Listing of subroutines

<table>
<thead>
<tr>
<th>Subroutine</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>CUTVOL</td>
<td>Primary subroutine that calculates cut volumes, areas, and strip ratios</td>
</tr>
<tr>
<td>FINVOL</td>
<td>Primary subroutine that calculates volumes under a final surface to the base of the coal seam</td>
</tr>
<tr>
<td>ISITIN</td>
<td>Secondary subroutine that can be used to aid in plotting storage areas</td>
</tr>
<tr>
<td>MINE</td>
<td>Primary subroutine that cycles through a set of pits for a one-layer system</td>
</tr>
<tr>
<td>MONEY</td>
<td>Primary subroutine that computes cash position and mining time and prepares input for MINE or SEQNC</td>
</tr>
<tr>
<td>PATH</td>
<td>Secondary subroutine that calculates slopes and distances for a defined path on a map</td>
</tr>
<tr>
<td>RDFNVL</td>
<td>Primary subroutine that splits volumes under the final surface into consolidated and unconsolidated volumes</td>
</tr>
<tr>
<td>SEQNC</td>
<td>Primary subroutine that cycles through a set of pits for a two-layer system</td>
</tr>
<tr>
<td>SQRPLE</td>
<td>Secondary subroutine that can calculate the maximum volume that can be piled on a square or rectangular area</td>
</tr>
<tr>
<td>SURFAL</td>
<td>Secondary subroutine that can excavate a pit in stages</td>
</tr>
<tr>
<td>TRPTILE</td>
<td>Secondary subroutine that can calculate the maximum volume that can be piled on a triangular area</td>
</tr>
<tr>
<td>UNCVOL</td>
<td>Primary subroutine that splits volumes in CUTVOL into consolidated and unconsolidated volumes</td>
</tr>
</tbody>
</table>

### Listing of abbreviations and special terms

<table>
<thead>
<tr>
<th>Abbreviation</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>BCY</td>
<td>Bank Cubic Yard</td>
</tr>
<tr>
<td>CAMP</td>
<td>Computer Aided Mine Planner</td>
</tr>
<tr>
<td>Fortran</td>
<td>Computer language used to develop the Computer Aided Mine Planner</td>
</tr>
<tr>
<td>ICPDM#1</td>
<td>Iowa Coal Project Demonstration Mine #1</td>
</tr>
</tbody>
</table>
TEREX Division of General Motors that manufactures earthmoving equipment

WATFIV Compiler used in the development of CAMP

**Listing of variable specifications and definitions**

<table>
<thead>
<tr>
<th>Variable</th>
<th>Type</th>
<th>Dimensions</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>Integer</td>
<td>1</td>
<td>internal variable to allow proper pit number to be printed in MINE and SEQNC</td>
</tr>
<tr>
<td>AREA</td>
<td>Real</td>
<td>200</td>
<td>computed array that contains the area of pits and storage areas in square feet</td>
</tr>
<tr>
<td>AVLSTR</td>
<td>Real</td>
<td>1</td>
<td>internal variable in MINE to indicate status of storage remaining in pits</td>
</tr>
<tr>
<td>B</td>
<td>Integer</td>
<td>1</td>
<td>internal variable to allow proper pit number to be printed in MINE and SEQNC</td>
</tr>
<tr>
<td>COAL</td>
<td>Real</td>
<td>1</td>
<td>input variable for the density of the coal in pounds per cubic foot</td>
</tr>
<tr>
<td>COALAM</td>
<td>Real</td>
<td>1</td>
<td>tons of coal per 50 ft square</td>
</tr>
<tr>
<td>COALDP</td>
<td>Real</td>
<td>1</td>
<td>input variable that defines the thickness of the coal seam in feet</td>
</tr>
<tr>
<td>COALPR</td>
<td>Real</td>
<td>1</td>
<td>input variable that sets the price of coal to be used for cash position estimating in $/ton</td>
</tr>
<tr>
<td>COALT</td>
<td>Real</td>
<td>(25,25)</td>
<td>input matrix of the top of coal elevations in feet</td>
</tr>
<tr>
<td>COALWT</td>
<td>Real</td>
<td>100</td>
<td>tons of coal by pit</td>
</tr>
<tr>
<td>CONVOL</td>
<td>Real</td>
<td>1</td>
<td>total volume of consolidated material over coal in BCY computed</td>
</tr>
<tr>
<td>COVDEP</td>
<td>Real</td>
<td>1</td>
<td>input variable in feet that defines the depth of unconsolidated cover over consolidated material after mining</td>
</tr>
<tr>
<td>CSHNET</td>
<td>Real</td>
<td>100</td>
<td>net cash flow for an individual pit</td>
</tr>
<tr>
<td>Variable</td>
<td>Type</td>
<td>Dimensions</td>
<td>Comments</td>
</tr>
<tr>
<td>------------</td>
<td>----------</td>
<td>------------</td>
<td>-------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>CSHTOT</td>
<td>Real</td>
<td>1</td>
<td>accumulated total cash position</td>
</tr>
<tr>
<td>CSURFC</td>
<td>Real</td>
<td>(25,25)</td>
<td>calculated matrix of surface elevations after alterations</td>
</tr>
<tr>
<td>CVOL</td>
<td>Real</td>
<td>1</td>
<td>computed total overburden volume over cost in BCY</td>
</tr>
<tr>
<td>CVOLUM</td>
<td>Integer</td>
<td>1</td>
<td>input variable that controls amount of program execution and whether one or two layer system is desired, default CVOLUM = 1</td>
</tr>
<tr>
<td>DIRT</td>
<td>Real</td>
<td>1</td>
<td>input variable that defines RSURF 0 for contour map of bottom of unconsolidated 1 for isopach map of the unconsolidated any positive number higher than 1 for a constant layer over the property</td>
</tr>
<tr>
<td>DVOL</td>
<td>Real</td>
<td>1</td>
<td>computed total unconsolidated volume over coal in BCY</td>
</tr>
<tr>
<td>ELDIF</td>
<td>Real</td>
<td>100</td>
<td>computed matrix of elevation difference between original surface and top of coal</td>
</tr>
<tr>
<td>FELDIF</td>
<td>Real</td>
<td>(25,25)</td>
<td>computed elevation difference between the final surface and the bottom of the coal seam in feet</td>
</tr>
<tr>
<td>FILL</td>
<td>Real</td>
<td>1</td>
<td>internal variable in MINE to transfer fill while hunting for room for placement</td>
</tr>
<tr>
<td>FILPIT</td>
<td>Real</td>
<td>200</td>
<td>computed amount of fill in individual pits during mining in yd$^3$</td>
</tr>
<tr>
<td>FRVOL</td>
<td>Real</td>
<td>100</td>
<td>computed volume of consolidated material under final surface by pit in yd$^3$</td>
</tr>
<tr>
<td>FSURFC</td>
<td>Real</td>
<td>(25,25)</td>
<td>input matrix of final surface elevations in feet, not required for cut volume calculations</td>
</tr>
<tr>
<td>FUVOL</td>
<td>Real</td>
<td>100</td>
<td>volume of unconsolidated material under final surface by pit in yd$^3$</td>
</tr>
<tr>
<td>Variable</td>
<td>Type</td>
<td>Dimensions</td>
<td>Comments</td>
</tr>
<tr>
<td>----------</td>
<td>-------</td>
<td>------------</td>
<td>--------------------------------------------------------------------------</td>
</tr>
<tr>
<td>FVOL</td>
<td>Real</td>
<td>100</td>
<td>computed final volumes of pits for one layer</td>
</tr>
<tr>
<td>FVOLUM</td>
<td>Real</td>
<td>1</td>
<td>control variable used to get final volume estimate only</td>
</tr>
<tr>
<td>HRPROD</td>
<td>Real</td>
<td>1</td>
<td>input variable that defines the estimated hourly production rate of the over-burden removal equipment fleet in BCY/hour</td>
</tr>
<tr>
<td>IPITS</td>
<td>Integer</td>
<td>(25,25)</td>
<td>input matrix of the locations of pits, pit numbers must start with one and increase sequentially</td>
</tr>
<tr>
<td>ISURF</td>
<td>Integer</td>
<td>1</td>
<td>input variable that controls the call of the surface alteration subroutine SURFAL</td>
</tr>
<tr>
<td>L</td>
<td>Integer</td>
<td>1</td>
<td>internal variable in MINE and SEQNC to end execution</td>
</tr>
<tr>
<td>LPIT</td>
<td>Integer</td>
<td>(10,100)</td>
<td>input list that specifies the order of mining of the pits with a minimum of one sequence and maximum of 10 sequences</td>
</tr>
<tr>
<td>M</td>
<td>Integer</td>
<td>1</td>
<td>internal variable to allow proper pit number to be printed in MINE and SEQNC</td>
</tr>
<tr>
<td>MDUM</td>
<td>Integer</td>
<td>1</td>
<td>dummy variable to dispose of extraneous data in alternate input stream for FINVOL</td>
</tr>
<tr>
<td>NUMPIT</td>
<td>Integer</td>
<td>1</td>
<td>input variable that defines the number of pits to be handled</td>
</tr>
<tr>
<td>NUMWAY</td>
<td>Integer</td>
<td>1</td>
<td>input variable that defines the number of ways that the pits will be cycled with a minimum of one</td>
</tr>
<tr>
<td>RFILL</td>
<td>Real</td>
<td>1</td>
<td>internal variable to transfer consolidated material between pits in SEQNC</td>
</tr>
<tr>
<td>RFLPT</td>
<td>Real</td>
<td>(100)</td>
<td>calculated amount of consolidated fill in pits during mining for two layer system in yd$^3$</td>
</tr>
<tr>
<td>Variable</td>
<td>Type</td>
<td>Dimensions</td>
<td>Comments</td>
</tr>
<tr>
<td>------------</td>
<td>------</td>
<td>------------</td>
<td>--------------------------------------------------------------------------</td>
</tr>
<tr>
<td>RSTRVL</td>
<td>Real</td>
<td>1</td>
<td>computed volume in consolidated stockpile in yd³</td>
</tr>
<tr>
<td>RSURF</td>
<td>Real</td>
<td>(25,25)</td>
<td>input matrix to define the unconsolidated layer</td>
</tr>
<tr>
<td>RVLSTR</td>
<td>Real</td>
<td>1</td>
<td>internal variable to indicate pit storage status in SEQNC</td>
</tr>
<tr>
<td>RVOL</td>
<td>Real</td>
<td>200</td>
<td>computed volume of consolidated material in individual pits over coal in BCY</td>
</tr>
<tr>
<td>RVOLU</td>
<td>Real</td>
<td>100</td>
<td>permanent record of original RVOL</td>
</tr>
<tr>
<td>STORVL</td>
<td>Real</td>
<td>1</td>
<td>calculated amount of stockpile in one layer system</td>
</tr>
<tr>
<td>STRIPR</td>
<td>Real</td>
<td>(25,25)</td>
<td>calculated matrix of strip ratios at grid points in cubic yards/ton</td>
</tr>
<tr>
<td>STRPRT</td>
<td>Real</td>
<td>100</td>
<td>calculated strip ratio by pit in yd³/ton</td>
</tr>
<tr>
<td>SURFC</td>
<td>Real</td>
<td>(25,25)</td>
<td>input matrix of the original ground elevations in feet</td>
</tr>
<tr>
<td>SWELL</td>
<td>Real</td>
<td>1</td>
<td>input variable that defines the average swell of the material for the site in decimal form</td>
</tr>
<tr>
<td>TAREA</td>
<td>Real</td>
<td>1</td>
<td>calculated total area over coal in ft²</td>
</tr>
<tr>
<td>TCOAL</td>
<td>Real</td>
<td>1</td>
<td>calculated total coal for the site in tons</td>
</tr>
<tr>
<td>TPVOL</td>
<td>Real</td>
<td>1</td>
<td>total final volume over coal in yd³</td>
</tr>
<tr>
<td>TIMEWK</td>
<td>Real</td>
<td>100</td>
<td>time to mine an individual pit</td>
</tr>
<tr>
<td>TIMTOT</td>
<td>Real</td>
<td>1</td>
<td>accumulated total time calculated</td>
</tr>
<tr>
<td>UFILL</td>
<td>Real</td>
<td>1</td>
<td>internal variable to transfer unconsolidated material between pits in SEQNC</td>
</tr>
<tr>
<td>UFLPT</td>
<td>Real</td>
<td>(100)</td>
<td>calculated amount of unconsolidated fill in pits during mining for two-layer system in yd³</td>
</tr>
<tr>
<td>Variable</td>
<td>Type</td>
<td>Dimensions</td>
<td>Comments</td>
</tr>
<tr>
<td>-----------</td>
<td>------</td>
<td>------------</td>
<td>-------------------------------------------------------------------------</td>
</tr>
<tr>
<td>USTRVL</td>
<td>Real</td>
<td>1</td>
<td>computed volume in unconsolidated stockpile in yd$^3$</td>
</tr>
<tr>
<td>UVOL</td>
<td>Real</td>
<td>200</td>
<td>computed volumes of unconsolidated material in individual pits over coal in BCY</td>
</tr>
<tr>
<td>UVLSTR</td>
<td>Real</td>
<td>1</td>
<td>internal variable to indicate pit storage status of unconsolidated storage in SEQNC</td>
</tr>
<tr>
<td>UVOLU</td>
<td>Real</td>
<td>100</td>
<td>permanent record of original UVOL</td>
</tr>
<tr>
<td>VOL</td>
<td>Real</td>
<td>200</td>
<td>computed cut volumes of individual pits over coal in BCY</td>
</tr>
<tr>
<td>VOLUM</td>
<td>Real</td>
<td>100</td>
<td>permanent record of original VOL</td>
</tr>
<tr>
<td>WKHRS</td>
<td>Real</td>
<td>1</td>
<td>input variable that defines the number of hours that the overburden removal fleet works per week</td>
</tr>
<tr>
<td>YDCOST</td>
<td>Real</td>
<td>1</td>
<td>input variable that defines the estimated cost of moving a cubic yard of overburden in $/BCY</td>
</tr>
</tbody>
</table>