

SELECTIVE INTERBURDEN HANDLING TECHNIQUES



VOLUME II - APPENDICES

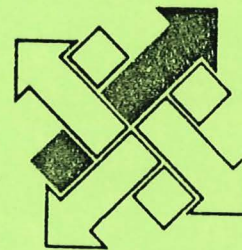
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UNITED STATES DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

by

MATHTECH, INC.
P. O. Box 2392
Princeton, New Jersey 08540



FINAL REPORT

On

Contract No. J0285025

Improved Interburden Handling Techniques
at Multiseam Dragline Operations to
Advance Reclamation Practices

September 1980

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| 16. Abstract Eight interburden handling concepts are developed for multiple seam area mine stripping operations in central and western United States. They are based on data gathered and analyzed during field surveys, literature reviews and visits to state regulatory agencies in five states. It is concluded that improvement of reclamation practices through selective interburden handling techniques is possible at lower operating costs than currently incurred. Specific multiple seam mining systems are discussed in this report. | | | |
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APPENDIX A
LITERATURE SURVEY

APPENDIX A
LITERATURE SURVEY

1. Introduction

Summary of the major findings of the literature survey are presented in this Appendix. Literature pertaining to mine practices in multiseam strip mining operations were reviewed. The survey has covered two coal regions; the Western coal region with emphasis on the Northern Great Plains coal province and the Interior Central coal region with emphasis on the Tri-State area of Illinois, Indiana and west Kentucky. Previous works on multiseam mining and selective handling techniques were identified and evaluated. The effects of the current and proposed State and Federal surface mining laws on alternative interburden handling techniques employed in the Western and Central coal regions were investigated.

2. Regional Characteristics

Physical features and mining technologies vary widely across the United States but are fairly homogeneous within each coal mining region (1). The regional compositions are summarized in Table A-1. Environmental problems vary between the Western and Central coal regions.

Brief summaries of the regional characteristics of the Western and Central coal regions with emphasis on the Northern Great Plains coal province and the tri-state area of Illinois, Indiana and west Kentucky follow.

2.1 Western Coal Region (Northern Great Plains Province)

Preliminary investigation and results from past studies have shown that interburden problems are most severe in the mining operations located in the Northern Great Plains coal province, including Montana, North Dakota and Wyoming.

The climate of the Northern Great Plains province is arid to semi-arid. Average annual precipitation is 15 inches. Most of the existing topography is rolling plains. Growing season for the province is between 60 and 120 days per year. The predominant soil types at active mines are aridsols and entisols. Subsoil

Table A-1. Composition of Geographic Regions (Ref. 1)

| Region | Coal Province(s) | States |
|---------|--|--|
| WEST | Rocky Mountain and Northern Great Plains | Arizona, Colorado, Montana, New Mexico, North Dakota, Wyoming |
| CENTRAL | Interior and Gulf | Arkansas, Illinois, Indiana, Iowa, Kansas, Missouri, Oklahoma, Texas, west Kentucky |
| EAST | Eastern | Alabama, east Kentucky, Maryland, Ohio, Pennsylvania, Tennessee, Virginia, West Virginia |

materials are clayey and often contain sodium. There is little if any acid overburden in the province. The overburden and interburden may be high in sodium, soluble minerals and trace elements. When placed as spoil these materials generally weather rapidly and produce a clayey "soil", which tends to become impermeable after a series of wet-dry cycles (1).

Another environmental problem in the Northern Great Plains coal province is the effect of mining on the groundwater. Mining may affect groundwater in shallow aquifers by lowering the water table, reducing the rate of flow, or degrading the chemical quality of the water when problem materials are placed below the water table.

2.2 Central Coal Region (Illinois, Indiana and West Kentucky)

The environment in the Central region differs markedly from that in the west. The climate is humid, with annual precipitation averaging about 40 inches. The growing season is longer than that in the Western coal region (1).

The topography of northern and southern Illinois and central Indiana is fairly flat. Southern Indiana and west Kentucky are characterized by gradually rolling terrain.

Although overburden varies greatly throughout the region, acid producing strata are present with different percentage in the overburden at most mines (1). The problem areas in Illinois are located in the far southeastern corner of the state, bordering the coal fields of southwestern Indiana and west Kentucky, where there are also acid problems. This area, in which acid drainage is a potentially severe problem, is shown in Figure A-1. In the area there is no glacial drift - calcareous or otherwise - and there is little or no limestone in the overburden at the active surface coal mines (1).

Most of the environmental problems result from failure to adequately identify and bury acid strata. This situation may be remedied in part by modifying overburden and interburden handling and placement procedures so that toxic materials are buried during spoil placement. Acid drainage may result from placement of pyritic materials on spoil surfaces in sidecast mining. The fairly heavy rainfall in the region may aggravate the problem.

Disturbance of groundwater aquifers by mining does not seem to be a major problem because surface waters provide an adequate source of water in the region. Contamination of groundwater by mine spoils may be a problem locally (11).

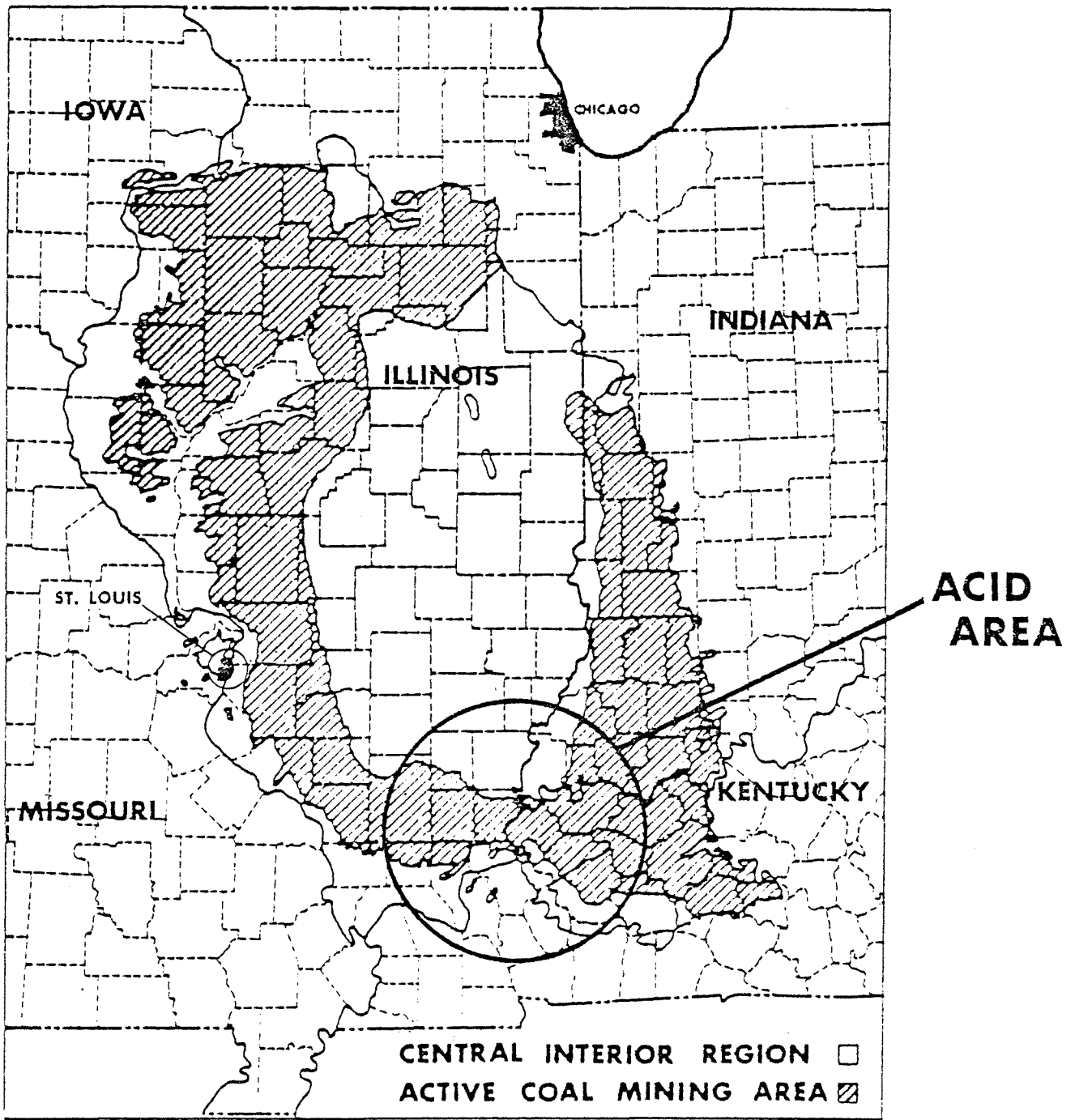


Figure A-1. Map Showing Acid Area in the Central Coal Region

3. Multiseam Surface Coal Mining Systems

Multiseam surface coal mining accounts for a large percent of the surface coal mining production in the United States. On a regional basis for 1975 an estimated 37 percent of the western surface coal mine production and an estimated 40 percent for the central was produced by multiple seam methods (1). Single machine stripping subsystems utilizing single dragline are predominant for the regions under discussion. Most mining is of the area type, with modified open pit mining being used in the west where coal is very thick.

Multiple seam surface mining methods have been studied in numerous research studies by the Government and private industry (1, 2, 3, 4, 5, 8, 13, 14). Additionally, many publications of general character, describe existing multiseam surface coal mining operations (6, 7, 9, 10). In the following subsections an attempt has been made to classify the existing information and present it in such a way so that it could be used as a reference in locating specific topics for multiseam surface coal mining systems. Table A-2 shows the distribution of multiseam surface mining operations by size and state (3).

3.1 Dragline Mining Methods

There are many types of multiseam dragline mining methods. Given a set of equipment and physical conditions, the choice of method is dependent primarily upon the following factors:

- the relative depths of the overburden and the interburden
- the dumping radius and height of the dragline
- the thickness of the coal seam being mined
(relevant only when the coal is very thick)

Detailed descriptions and classifications of the existing multiseam dragline systems are given in two previous Mathtech studies (1, 2). Various characteristics of the multiseam dragline systems are summarized in Table A-3.

Draglines are the primary stripping equipment used at existing multiseam surface coal mining operations as described by W. Porter in his master thesis (3). Detailed descriptions of existing multiseam dragline mining operations could be found in a report by the Office of Coal Research (7). Numerous professional publications and journal's articles describe multiseam dragline mining operations (6, 7, 9, 9, 10, 15).

Table A-2. Distribution of Multiseam Surface Mining Operations by Size and State
(Ref. 3)

| State | No Capacity Given | Less Than 99,999 TPY | 100,000 499,999 TPY | 500,000 999,999 TPY | Greater Than 1,000,000 TPY | Total |
|---------------|-------------------|----------------------|---------------------|---------------------|----------------------------|----------|
| Alabama | 1 | | 6 | 1 | | 8 |
| Alaska | 1 | | | | 1 | 2 |
| Colorado | | | | 1 | | 1 |
| Illinois | 1 | | | | 2 | 3 |
| Indiana | | | | | 4 | 4 |
| Kansas | | | 1 | | 1 | 2 |
| Kentucky | 2 | 3 | 6 | 8 | 8 | 27 (12) |
| Maryland | | 1 | 1 | | | 2 (1) |
| Missouri | | | 1 | | | 1 |
| Montana | | | 1 | | | 1 |
| New Mexico | | | | | 1 | 1 |
| North Dakota | | | 1 | | 1 | 2 |
| Ohio | | 8 | 10 | 4 | 3 | 25 (2) |
| Pennsylvania | 1 | 35 | 32 | 2 | 1 | 71 (8) |
| Tennessee | 1 | | 1 | | | 2 (2) |
| Virginia | | | 6 | | | 6 (6) |
| West Virginia | 2 | 7 | 15 | 2 | 2 | 28 (3) |
| Wyoming | | | | 1 | 1 | 2 |
| Total | 9 | 54 | 81 | 19 | 25 | 188 (39) |

Note: Numbers in parentheses indicate operations where a combination of strip and auger methods are used.

Table A-3. Summary of Two-Seam Dragline Mining Methods (Ref. 1)

| Name of Method | Stripping Subsystem | Regions in which Method is Used | No. of Lifts | No. of Passes | Extension of Dragline Range? | Extension of Dragline Dump Height? | Location of Bench for Stripping of Interburden? | Spoil Rehandle? | Easy to Bury Interburden Material in Spoil Pile? | Remarks |
|-------------------------|---------------------|-----------------------------------|--------------|---------------|------------------------------|------------------------------------|---|-----------------|--|--|
| Conventional | Single | West | 2 | 2 | No | No | Interburden | No | No | <ul style="list-style-type: none"> • Feasible only where dumping height is sufficient • High hoists on second pass |
| Extended Bench | Single & Tandem | All | 2 | 2 | Yes | No | Extended Bench | Yes | Relatively | <ul style="list-style-type: none"> • Tandem systems have several advantages under certain operating conditions |
| Extended Bench | Single | Central (limited use) | 2 | 1 | Yes | No | Extended Bench | Yes | Relatively | <ul style="list-style-type: none"> • Reduced dragline walking time must be traded off against possible productivity decreases due to overhanding chopping of overburden • Effect of overhand chopping on dragline productivity is subject of controversy • Method may be optimal where interburden depth exceeds overburden depth |
| Spoil Side | Single & Tandem | All | 2 | 2 | Yes | Yes | Bench in spoil pile | Site specific | Relatively, if there is re-handle | <ul style="list-style-type: none"> • Widely used method of increasing the effective dumping height of a dragline • Productivity is decreased by chopping of interburden and large swing angles |
| Elevated Extended Bench | Single | Northern Appalachia (limited use) | 2 | 1 | Yes | Yes | Elevated, extended bench | Yes | Relatively | <ul style="list-style-type: none"> • Requires placement of spoil on the interburden • Reduced dragline walking time and elimination of interburden chopping must be traded off against the need for re-handle and overhand chopping of the overburden • Method may be optimal when required bench elevation is small |

3.2 Stripping Shovel Operations

Stripping shovels were the prime movers at an estimated 45 percent of the surface coal mines in the Central region in 1975, but are not widely used outside of that region (1). Stripping shovels, when used as a single machine in multiseam operations, are positioned on top of the lower seam (8).

Active operations using stripping shovels as the primary stripping equipment are described by W. Porter (3). Most of these operations are located in the Tri-State area of Illinois, Indiana and west Kentucky. Figure A-2 shows an operation using single stripping shovel to mine two coal seams.

Also, descriptions of stripping shovel operations could be found in the report by the Office of Coal Research (7).

The stripping shovels will probably decline steadily over time, even in the Central region, because draglines are widely preferred for overburden removal and placement (1).

3.3 Tandem Systems

In two-seam situations in which tandem machine systems are used, one machine is usually used to strip overburden and the second machine follows, stripping the interburden. Situations such as these occur frequently in the Central region, but not in the Western region (1). The tandem systems as defined in previous Mathtech study (1) are of four types:

- dragline/shovel
- dragline/dragline
- shovel/shovel
- BWE/shovel

A major advantage of the tandem machine systems is that each of the stripping machines can be tailored for a specific task.

Two-seam tandem shovel/dragline systems are discussed in length in Ref. 1. Existing tandem systems are described in Porter's thesis (3). Figure A-3 shows section view of two-seam tandem shovel/dragline method.

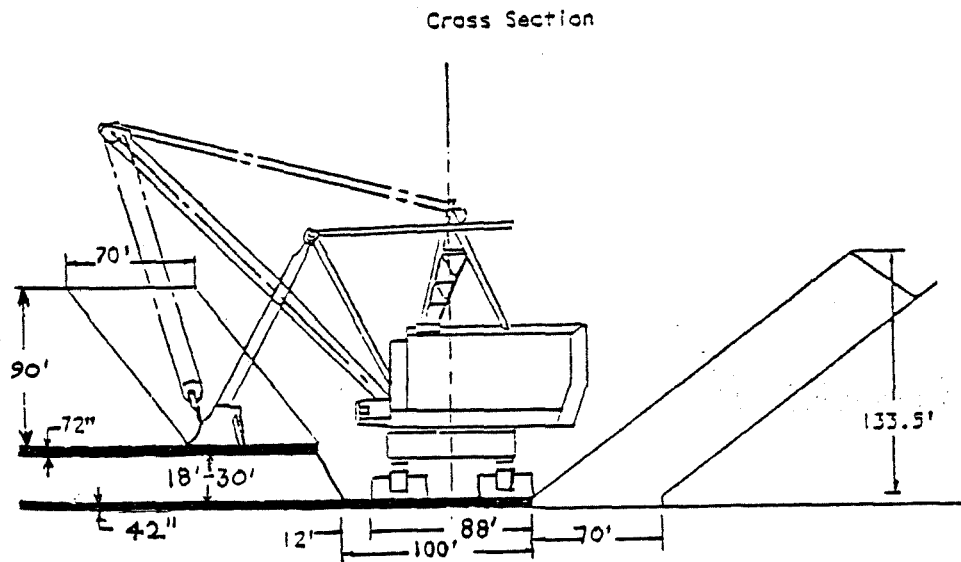
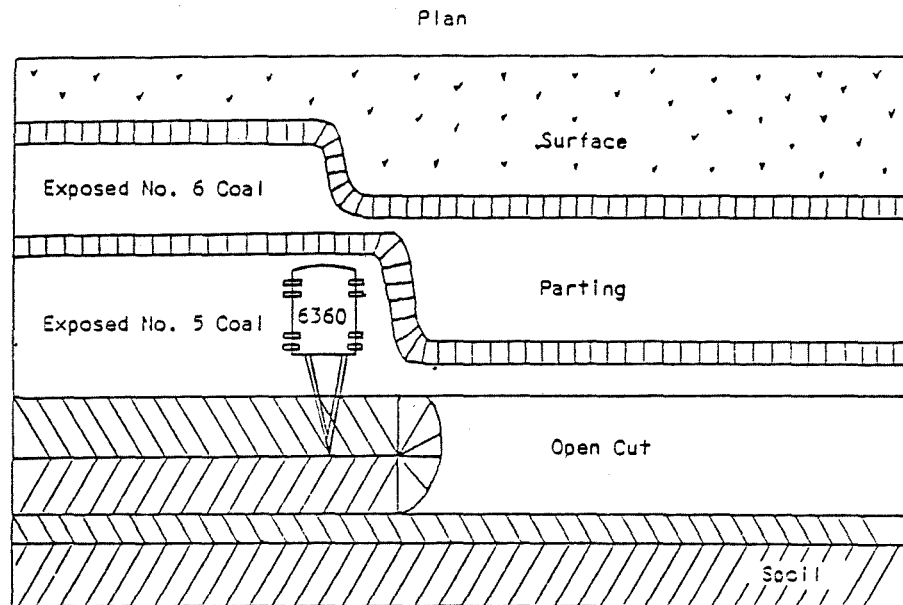
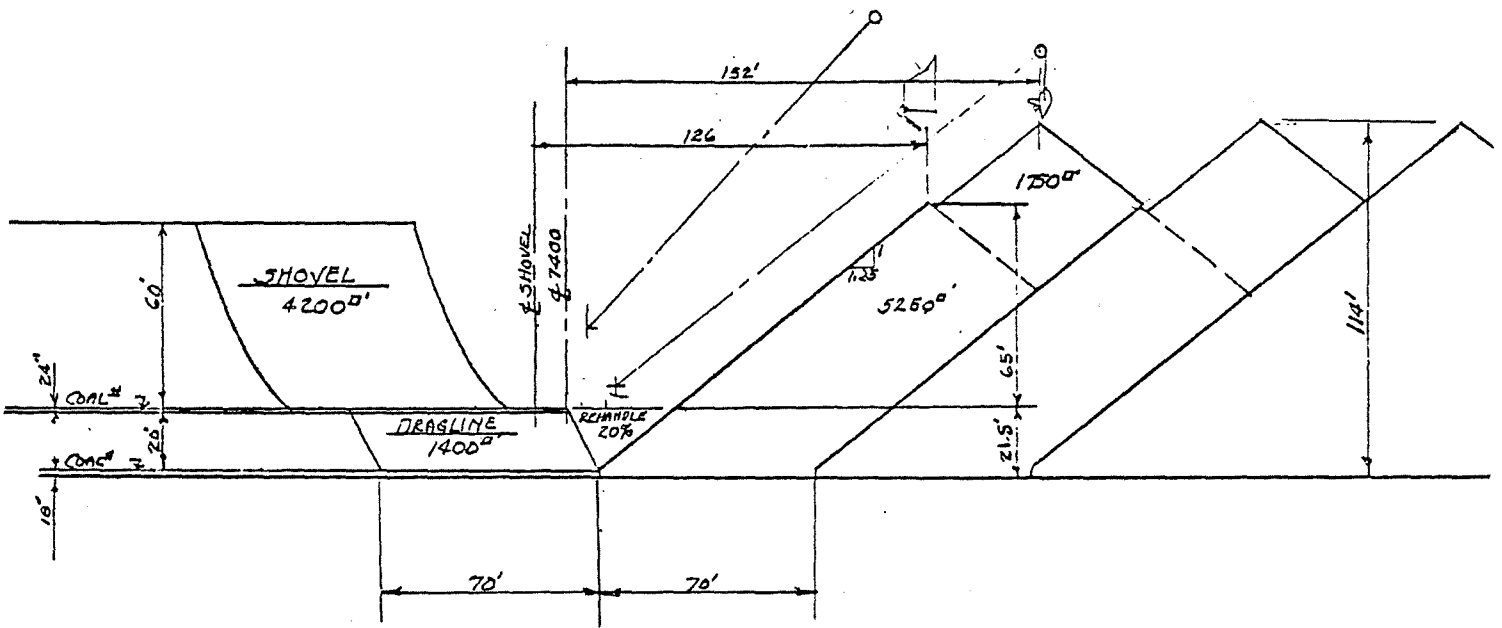


Figure A-2. Multiseam Two Lifts
One Pass Stripping Shovel Method (Ref. 3.)



55¢/SHOVEL-135' BOOM @ 47½° @ 40 CU. YD BUCKET = 1,000,000 ⁴/₁₁₄ - NOTE - SHOVEL TO USE MATTS ON SPK. SIDE.
 7400 DRAGLINE-175' " @ 38° - 12" " " = 200,000 ⁴/₁₁₄
 1,200,000

Figure A-3. Section View Showing a Two-Seam Tandem Shovel/Dragline Method (Ref. 1)

4.0 Selective Interburden Handling Techniques

4.1 The Interburden Problem

Interburden at about half of the mines in the western United States active in 1975 contained strata that were clayey in texture and high in exchangeable sodium, the latter factor indicated by a sodium absorption ration (SAR) greater than about ten (1). Where interburden is shale, if placed on the surfaces of the spoil piles, as often is the case where selective placement procedures are not used, this material soon becomes impermeable, increasing surface runoff and causing subsequent revegetation failure. Where there is a water table above the pit floor, placement on the pit floor of interburden material high in soluble salts and trace elements may lead to mineralization or pollution of groundwater.

Similarly, in acid areas of the Central coal region, exposure of toxic interburden materials may cause significant environmental damage.

Selective removal and placement of problem interburden is the only significant way to integrate the mining and reclamation practices in dragline stripping situations. Improvement in spoil placement control might enable reduction of the depths of topsoil required to achieve specified reclamation goals and reduce the environmental damage.

Selective handling of the interburden could be accomplished in two ways. First by selective handling techniques employing single stripping systems, secondly by using tandem stripping systems in which the interburden removal is done independently from the overburden. Selective handling techniques as reported in the literature are summarized in the following two subsections.

4.2 Single System Selective Handling Techniques

Using a single dragline to strip overburden and interburden with selective handling of the problem strata (interburden) has been presently applied to some western and central multiseam surface coal mining operations (1, 2, 5).

In some cases, where rehandle is required, the material could be removed last and placed on top of the interburden. Figure A-4 shows a situation where dragline range was such that an extended bench was not required for production reasons. In order to satisfy a requirement that the interburden spoil be buried, mine operators have chosen to use rehandle material. This way reclamation requirements affect production methods and costs (1).

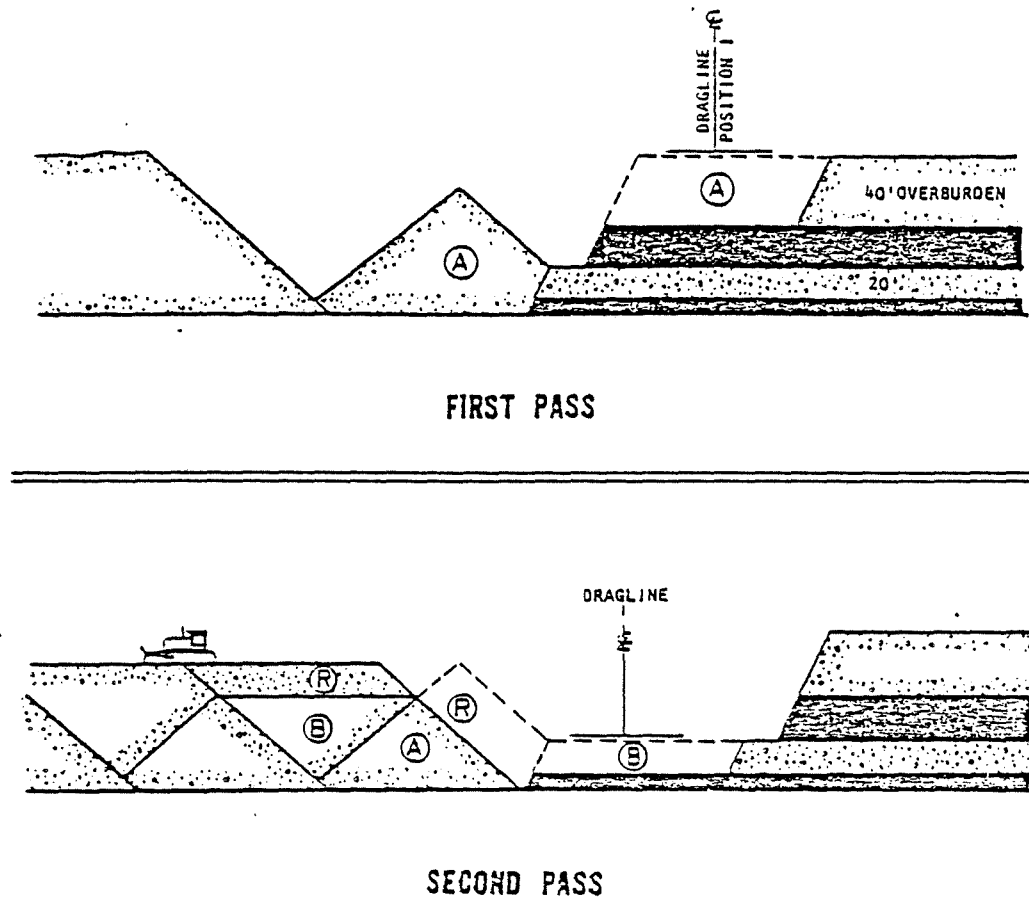


Figure A-4. Section Views Showing The Use of Rehandle Material To Bury Spoiled Interburden Material (Ref. 1)

4.3 Tandem Systems Selective Handling Techniques

Two recent studies conducted by the Bureau of Mines (13, 14), discuss a mine system which uses smaller units of equipment sited only for interburden removal. Two equipment tandem systems, one for overburden removal and the other for interburden removal, may be much more efficient in selective placement of the interburden material.

T.E. Finch and R.T. Dale, in their report (13), investigate alternative methods of handling the parting between two coal seams. Equipment selected for removing the interburden and second coal seam is removed nearly independent of the overburden and first coal seam equipment, and has its own profitability measure. The report discusses flexible (alternative) extractive techniques involving castback or haulback techniques applicable to western conditions.

The report includes field observations from a large western strip mine, operators' data on costs and availabilities, equipment manufacturers' specifications and production characteristics. This data is combined in all alternative methods to arrive at a composite production and cost that is objective and representative of the conditions assumed in the model stratigraphy (13).

Two methods of interburden handling were studied: castback and haulback techniques. Five types of machines have been considered for castback including dragline, shovel, hydraulic and conveyor unit. The haulback technique uses three types of equipment - front end loader and trucks, shovel and trucks, and scrapers.

The results of the study show that the most economical method compared on \$/ton of work produced is the conventional dragline operation. The rest of the alternative handling techniques ranked the following way:

- hydraulic backhoe
- bucket wheel excavator
- scraper, front end loader and trucks
- dragline in castback
- shovel in castback

The authors make recommendations for closer examination for each case. The study finds parting (interburden) total thickness and parting to coal thickness ratios very important. Recommendations are also made to develop a computer program to handle alternative methods - cost sensitivity analysis, updating cost input parameters.

In their second study, T.E. Finch, et al, (14) have developed two computer programs to evaluate interburden handling methods over a wide range of coal and interburden thickness ratios. The output from the programs is not optimized. The final choice has been left to the mine operators.

5.0 Other Multiseam Surface Mining Systems

5.1 Pitching Seams Mining Systems

Mining of pitching seams occurs in a number of coal mines in the Western region. Two or more pitching seams are mined per pit (1).

In southwestern Wyoming as many as 50 different seams have been deposited in places, but the subsequent displacement of the stratigraphic horizons has resulted in seams dipping at various angles.

McDonald (15) in his article discusses some of the problems related to multiple dipping seams where draglines are used. Moderately pitching coal seam mining is discussed in detail in a recently completed USBM study (16). Detailed characterization of moderately pitching coal seams found in the United States is presented in the report. Field investigations and new extracting methods are described also.

5.2 Unusual Surface Coal Mining Systems

Novel and unusual mining methods could be found in the literature. E. M. Frizzell (17) describes the application of cross-pit conveyor system in two-seam operations. D. D. Marston in his Mining Congress Journal article focuses his attention on the applications of bucket wheel excavators to thick western coal seams (18).

6.0 State and Federal Surface Mining and Reclamation Laws

State and Federal laws were reviewed under this task, in order to understand the numerous requirements of the law on multi-seam surface coal mining.

Review of twenty Eastern and Midwestern state coal surface mining laws could be found in R.W. Grubagh's paper (19). Permanent regulations of the Federal Strip Mining Law of 1977 were reviewed (20). Special emphasis was made on requirements for selective handling of problem strata and effects of mining on hydrology.

7.0 Conclusions

The following conclusions were reached as a result of the literature survey:

- Research on multiseam surface coal mining has been limited
- Selective handling techniques are not well developed and publicized in the available literature.
- Dragline operations and mining methods are well documented and developed
- Unusual situations and novel mining methods are not well studied
- Cost estimates of multiple seam surface coal mining are limited.

References

1. Mathematica, Inc. (Mathtech Division), "Evaluation of Current Surface Coal Mining Overburden Handling Techniques and Reclamation Practices, Phase I - Western U.S. and Phase II - Central U.S.", Technical Report under U.S. Bureau of Mines Contract No. S0144081, March 15, 1976, 216 pp.; available from National Technical Information Service, Springfield Va.
2. Mathematica, Inc., "Evaluation of the Environmental Effects of Western Surface Coal Mining, Volume I", Technical Report under U.S. Environmental Protection Agency Contract No. 68-03-2226, March 1978, 136 pp.
3. William E. Porter, "Multiple Seam Strip Mining: A Survey and Economic Feasibility Model", The Pennsylvania State University, Master of Engineering Thesis, December 1971.
4. "An Analysis of Strip Mining Methods and Equipment Selection", Coal Research Section, College of Earth and Mineral Sciences, The Penn State University, Office of Coal Research; U.S. Dept. of the Interior, May 1973.
5. Frank Cook and Lewis M. McNay, "Operational Constraints: Surface Mining In The Central Coal Province", Paper presented at Society of Mining Engineers, AIME Fall Meeting and Exhibit, October 19-21, 1977.
6. Dan Jackson, "Arch Mineral Puts Three on the Line", Coal Age, September 1976, pp. 128-132.
7. "Montana New Mine Reaches for 5.5 MTPY Mark", Coal Mining and Processing, December 1976, pp. 76-78.
8. J.W. Francis, D.R. Mahan, and W.W. Weaver, "Two Seam Dragline Mining Techniques of the Consolidation Coal Company's Burning Star #4 Mine", paper presented at the 1977 SME Fall Meeting and Exhibit, St. Louis, Missouri, October 19-21, 1977.
9. Carol Nolden, "Big Construction Signifies Growth at P&M's New Mexico Mine", Coal Mining and Processing, June 1976, pp 50-55.
10. Richard A. Campbell, "Multiple Seam Surface Mining", Mining Congress Journal, August 1970, pp. 71-75.
11. Francis X. Murray, Editor, "Where We Agree", Report of the National Coal Policy Project, Vol. II, Center for Strategic and International Studies, Georgetown University, 1978.

12. Gene Tuma, "Mining Methodology and Reclamation Planning in the Big Sky Mine, Montana", Reclamation of Western Surface Mined Lands, Edited by Kimery C. Vories, Workshop Proceedings, March 1-3, 1976, Ecology Consultants, Inc.
13. T.E. Finch and R.T. Dale, "Alternative Methods of Handling the Parting Between Two Coal Seams", Final Technical Report under U.S. Bureau of Mines Grant No. G0254004, Montana College of Mineral Science and Technology, August 1975. USBM Open File Report No. 90-76 available from NTIS, Springfield, Va., PB256767/AS.
14. T.E. Finch, D.R. Haley, and C.J. Speake, Jr., "Limits and Cost Sensitivity of Alternate Parting Handling Methods", Final Technical Report under U.S. Bureau of Mines Grant No. G0264014, Montana College of Mineral Science and Technology, March 1977, USBM Open File Report No. 117-(1-2)-77.
15. David B. McDonald and Duane E. Eubanks, "Western Coal Development: The Problems of Economics, Engineering and Environment", Mining Engineering, August 1976, pp. 40-44.
16. Skelly and Loy, "Development of Concepts for Surface Mining Moderately Pitching Coal Seams", Final Report, USBM Contract No. J0275023, November 1978.
17. Earl M. Frizzell, "Cross-Pit Systems", Mining Congress Journal, Vol. 63, No. 8, August 1977, pp. 67-76
18. Donald D. Marston, "Application of Bucket Wheel Excavators to Thick Western Coal Seams," Mining Congress Journal, June 1976, pp. 22-28.
19. Kraig W. Grubaugh and Lee W. Saperstein, "Review of Twenty Eastern and Midwestern State Coal Surface Mining Laws", Transactions, Vol. 262, Society of Mining Engineers, AIME, December 1977, pp. 335-341.
20. United States Department of the Interior, Office of Surface Mining Reclamation and Enforcement (SMRE), "Surface Coal Mining and Reclamation Operations, Permanent Regulatory Program", 1978.

APPENDIX B

THE HORSESHOE METHOD*
A MODEL AND CASE STUDY
(by Georgi Smrikarov)

*NOTE: Mathtech, Inc., "Operating Guidelines for Dragline Stripping Systems; Analysis of Tandem Systems," Volume II: Appendices, DOE Contract No. U.S.D.O.E. ET-77-C-01-9110, October 1979.

APPENDIX B
THE HORSESHOE METHOD:
A MODEL AND CASE STUDY
(by George Smrikarov)

1. Introduction

A model for estimating the production rate for a dragline using the horseshoe method of two-seam stripping is presented in this Appendix. The model derivation and a step-by-step computational procedure are included. Computations are illustrated for a case study based on a Marion 7400 stripping an average of 70 feet of overburden on the first pass in each cut, and 22 feet of interburden on the second. The interburden is chopped out from the spoil side. The pit cross-section for the case study is shown in Figure B-1.

2. Assumptions

The following assumptions have been made in deriving the model. Most of them are based on observed operating procedures.

- In shallow overburden, if the dragline bench is made by a dozer, the dozer does not bench down too far, so that the bench dirt does not ride up the highwall above the elevation of the upper coal seam.
- At a given point along the length of the pit, the elevation of the dragline bench is kept at approximately the elevation of the spoil bench.
- In moderate-to-deep overburden, benching is done by the dragline, from the keyway position.
- If, on the overburden pass, an extended bench is not needed, blocks are taken with two sets. From the first set, the keyway position, the side bench material -- if any -- and keyway are dug. From the second set, out near the highwall edge, the remaining bank plus any spoil that has ridden up the highwall above the upper coal seam are dug.
- On the other hand, if an extended bench is needed on the overburden pass, three sets are used. The first two are the same as before; the third is on the extended bench.

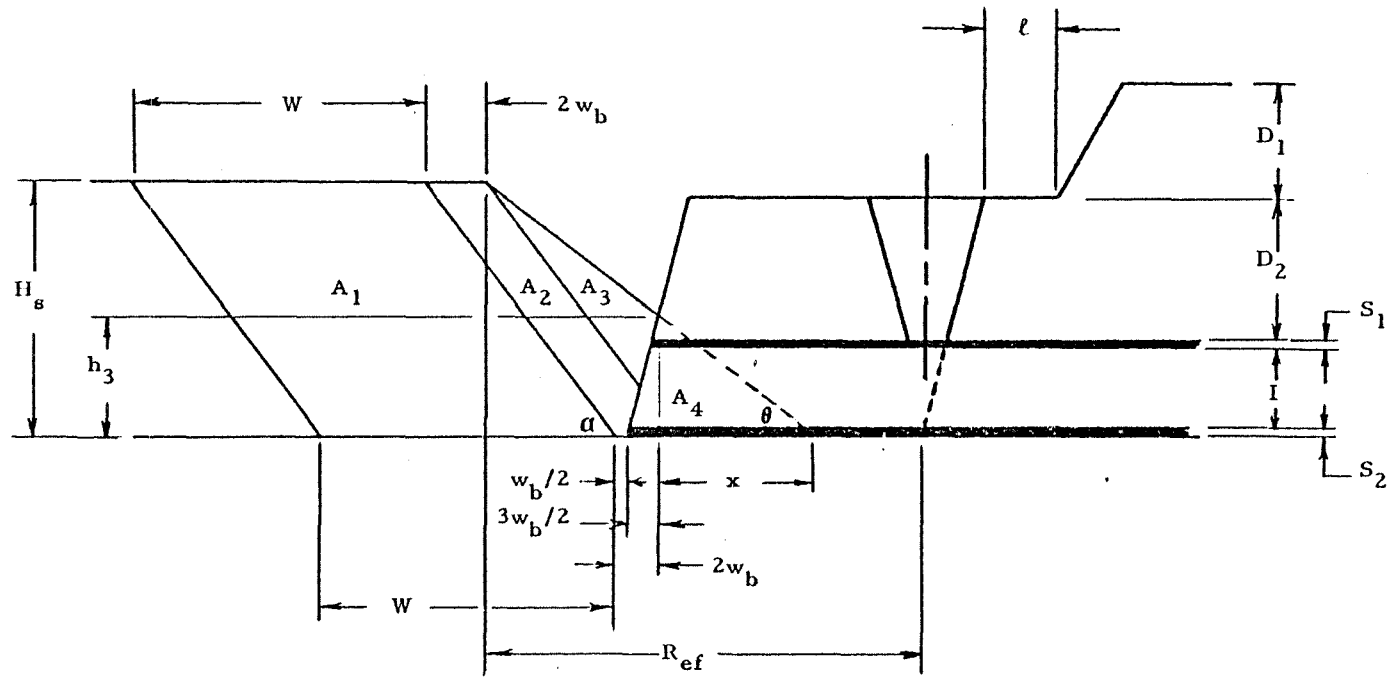


Figure B-1. Pit Geometry for Case Study

- Rehandle material on the overburden pass is dug only down to the elevation of the upper coal seam.
- The width of the spoil bench is the pit width plus two dragline bucket widths.
- On the interburden pass, a section of the spoil bench, two buckets wide at the top, must be rehandled on every move, even if the toe of the spoil pile from the overburden pass did not ride up the lower highwall.

3. Model Derivation

3.1. Input Variables

Variables representing numerical values that must be input to the model are listed below.

- W = 80 feet: Pit width.
- W_b = 100 feet: Dragline bench on overburden.
- ℓ = 20 feet: Lip.
- D_1 = 30 feet: Height of upper highwall (depth of dragline bench).
- D_2 = 39 feet: Height of overburden dragline bench.
- D_3 = 22 feet: Interburden thickness.
- S_1 = 24 inches: Upper coal seam thickness.
- S_2 = 18 inches: Lower coal seam thickness.
- x_1 = 30 feet: Thickness of overburden stratum 1 -- clay.
- x_2 = 31 feet: Thickness of overburden stratum 2 -- limey shale.
- x_3 = 8 feet: Thickness of overburden stratum 3 -- dark gray shale.
- x_4 = 22 feet: Interburden stratum -- shale.
- W_k = 10 feet: Width of keyway at the bottom.
- L = 60 feet: Length of block.

- ϕ = 75 degrees: Highwall angle.
- ϕ_1 = 75 degrees: Keyway highwall angle.
- θ = 37 degrees: Natural angle of repose.
- α = 55 degrees: Angle of oversteepened spoil.
- δ = 0.30: Spoil swell factor.
- R_d = 150 feet: Dragline dumping radius.
- R_{ef} = 125 feet: Dragline effective radius.
- H_d = 76 feet: Dragline dumping height.
- b = 14 cubic yards: Bucket capacity.
- L_1 = 30 feet: Dragline tailroom required.
- w_b = 8.3 feet: Width of bucket.

3.2. Output Variables

Variables representing numerical values that are to be calculated using the model are listed below.

- H_s = Height of the spoil bench above the pit floor, feet.
- h_1 = Height of material for filling the spoil bench, feet.
- H'_s = Total spoil pile height from the overburden pass, feet.
- h_3 = Height of the spoil pile toe from the overburden pass, feet.
- R_{nr} = Dragline radius required for no rehandle, feet.
- R_I = Dragline reach required for the interburden pass, feet.
- H_{SF} = Height of spoil pile from the interburden pass, feet.
- $V_1, V_2,$
 V_3, V_4 = Dig component volumes, bcy.
- V_R = Volume of total rehandle material, bcy.
- p_{OV} = Percent rehandle on overburden pass.
- p_{IN} = Percent rehandle on interburden pass.

- p = Percent rehandle (total).
 T_{ti} = Total cycle time for each component, seconds.
 S_i = Swing cycles per hour for each component.
 C_i = Bucket carrying capacity for each component.
 T_i = Time required to dig each component, hours.
 T = Time to complete one block, hours.
 T_S = Scheduled time to dig one block, hours.
 P_H = Dragline production rate per scheduled hour, bcy/hr.
 P_M = Dragline monthly production, bcy/month.

3.3. Equation for H_s , the Height of the Spoil Pile Bench for Handling Interburden

The height of the spoil bench after leveling is given by Eq. (4), which is found by equating the spoil section with the overburden after swelling, as shown below.

$$A_{OB}(1 + \text{swell}) = A_{OBS} \quad (1)$$

$$A_{OB} = W (D_1 + D_2) \quad (2)$$

$$\begin{aligned}
 A_{OBS} = H_s^2 & [(\tan \theta)/2 - (\tan \alpha)/2 \\
 & - (1/\tan \theta - 1/\tan \alpha)^2 \tan \phi \tan \theta/2 (\tan \theta + \tan \phi)] \\
 & + H_s [W + 2w_b - 3w_b (1/\tan \theta - 1/\tan \alpha) \tan \phi \tan \theta \\
 & \div 2 (\tan \theta + \tan \alpha)] \\
 & - 9w_b^2 \tan \phi \tan \theta/8 (\tan \theta + \tan \phi)
 \end{aligned} \quad (3)$$

By equating Eqs. (2) and (3) and then solving for H_s , Eq. (4) is derived.

$$H_s = \left(-b \pm \sqrt{b^2 - 4ac} \right) / 2a \quad (4)$$

$$\begin{aligned} \text{where } a &= [1/2(\tan \theta) - 1/2(\tan \alpha)] - (1/\tan \theta - 1/\tan \alpha)^2 \tan \phi \tan \theta \\ &\div 2(\tan \theta + \tan \phi) \end{aligned} \quad (4.1)$$

$$\begin{aligned} b &= W + 2w_b - 3w_b (1/\tan \theta - 1/\tan \alpha) \tan \phi \tan \theta \\ &\div 2(\tan \theta + \tan \phi) \end{aligned} \quad (4.2)$$

$$c = -W(D_1 + D_2)(1 + \delta) - 9w_b^2 \tan \phi \tan \theta / 8(\tan \theta + \tan \phi) \quad (4.3)$$

3.4. Equation for H'_s , the Spoil Pile Height Before Leveling the Bench

The height of the spoil pile from the overburden pass will be determined by the amount of material needed to fill the spoil bench to the desired width. The spoil height is that for which the areas A_1 and A_2 , shown in Figure B-2, are equal.

$$H'_s = H_s + h_1 \quad (5)$$

where h_1 = height of spoil peak to be leveled.

$$h_1 = \left(-b \pm \sqrt{b^2 - 4ac} \right) / 2a \quad (6)$$

$$a = 2/\tan^2 \theta (1/\tan \theta + 1/\tan \alpha) - 1/\tan \theta \quad (6.1)$$

$$b = -2(W + 2w_b)/\tan \theta (1/\tan \theta + 1/\tan \alpha) \quad (6.2)$$

$$c = (W + 2w_b)^2 / 2(1/\tan \theta + 1/\tan \alpha) \quad (6.3)$$

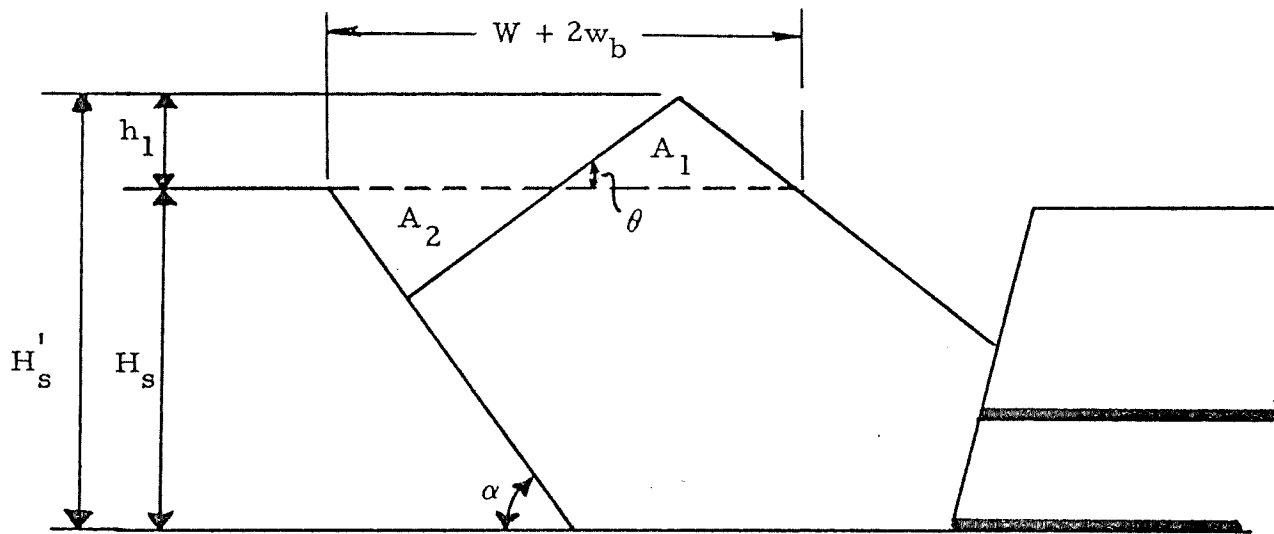


Figure B-2. Illustration of Variables Used in Spoil Height Calculations

3.5. Equation for h_3 , the Height of the Spoil Pile Toe from the Overburden Pass

$$h_3 = H_s (1/\tan \theta - 1/\tan \alpha + 3w_b/2) \tan \phi \tan \theta / (\tan \theta + \tan \phi) \quad (7)$$

Check for spoil pile toe covering the upper coal seam.

$$h_3 \leq I + S_2$$

where I = thickness of the interburden.

If $h_3 > I + S_2$, there will be rehandle on the overburden pass.

3.6. Equation for A_{rOB} , the Area of the Rehandle Section on the Overburden Pass (see Figure -3)

$$A_{rOB} = (h_3 - I - S_2)^2 \sin \phi^2 \sin [180 - (\alpha + \phi)] \sin (\theta + \phi) \div 2 \sin (\alpha - \theta) \quad (8)$$

3.7. Check for Dragline Reach Required for No Extended Bench on the Overburden Pass

$$R_{nr} = (H'_s - h_3)/\tan \theta + (D_2 + I - h_3)/\tan \phi \quad (9)$$

If $R_{nr} \leq R_{ef}$, no extended bench is needed; if $R_{nr} > R_{ef}$, the width of the extended bench will be

$$W_e = R_{nr} - R_d \quad (10)$$

The area of the rehandle section then is given by

$$A_{rOB} = (D_1 + D_2 + S_1)(W_e + V_2/2) + S_1^2/2 \tan \phi - (W_e + V_5)^2 \tan \theta/4 \quad (11)$$

$$\text{where } V_2 = (D_1 + D_2 + S_1)(1/\tan \theta - 1/\tan \phi) \quad (11.1)$$

$$V_5 = V_2 + S_1/\tan \phi \quad (11.2)$$

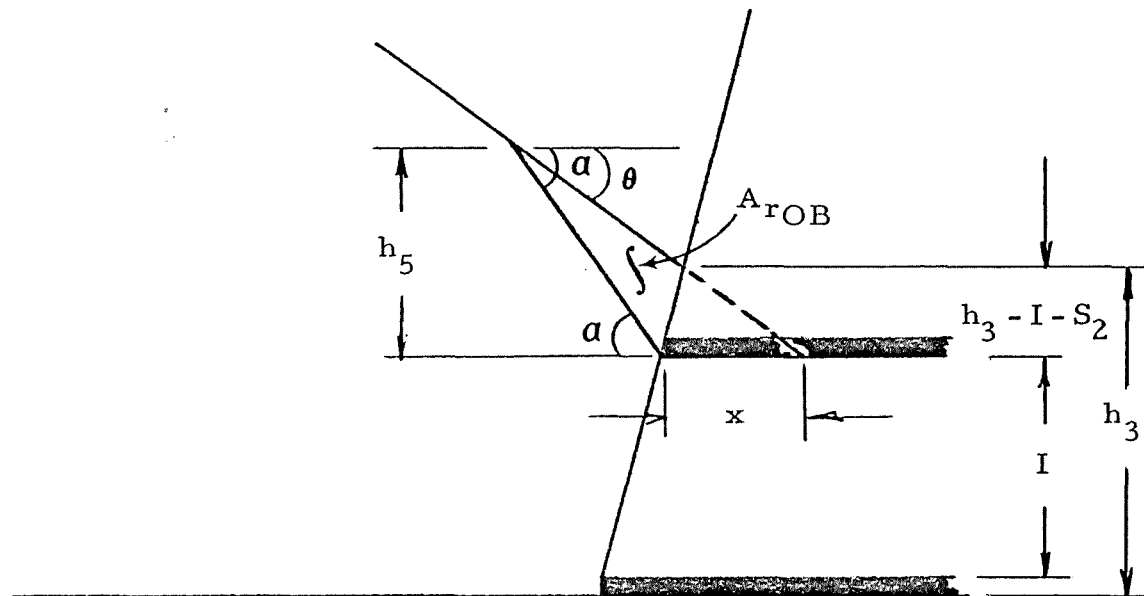


Figure B-3. Illustration of Variables Used in Calculation of Rehandle on Overburden Pass

3. 8. Check for Dragline Dump Height Required

$$(H'_s - D) \leq H_d$$

where $D = D_2 + S_1 + I + S_2$

3. 9. Formula for h_4 , the Height of the Spoil Pile while Digging the Upper Highwall (see Figure B-4)

$$h_4 = \left(-b \pm \sqrt{b^2 - 4ac} \right) / 2a \quad (12)$$

where $a = 1/\tan \theta$ (12. 1)

$$b = 2h_3/\tan \theta \quad (12. 2)$$

$$c = h_3^2 [(\tan \theta)/2 - (\tan \alpha)/2] - D_1 W(1 + \delta) \quad (12. 3)$$

3. 10. Check for Dragline Reach Required for Digging the Interburden from the Spoil Side

$$R_I = (H_s - h_3)/\tan \theta + W - h_3/\tan \phi \quad (13)$$

If $R_I > R_d$, the spoil bench should be lowered or the dragline will not have sufficient dump radius when working from the spoil side.

3. 11. Equation for H_{SF} , the Spoil Pile Height from the Interburden Pass

$$H_{SF} = A_{SP}/W + W \tan \theta/4$$

where $A_{SP} = A_I + A_{rI}$

$$A_I = \text{cross-sectional area of interburden}$$

$$A_I = IW(1 + \delta)$$

$$A_{rI} = \text{cross-sectional area of the rehandle material of the spoil bench on the interburden pass}$$

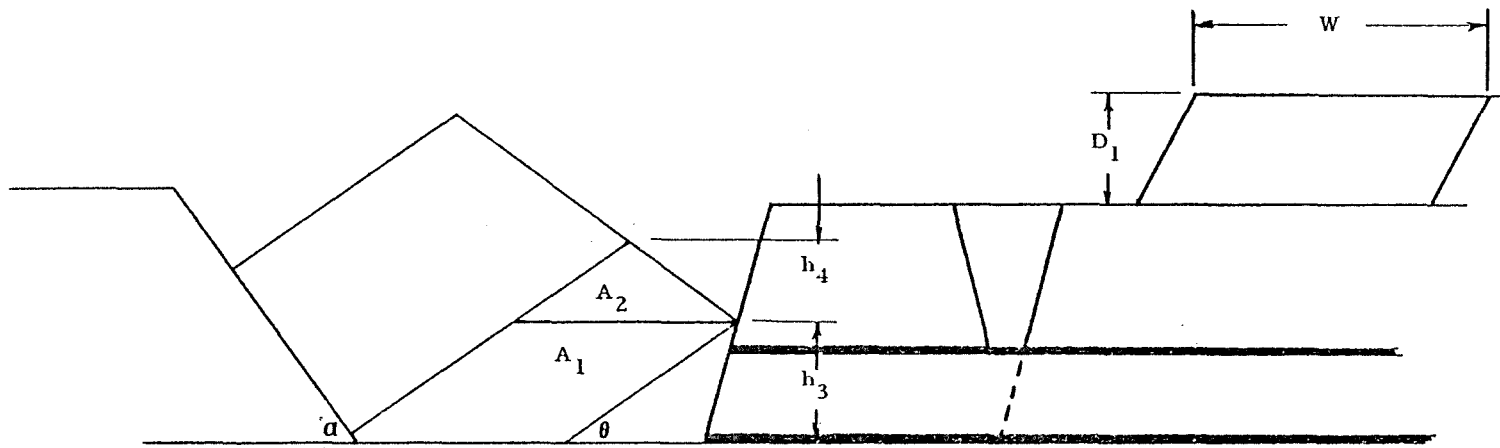


Figure B-4. Illustration of Variables Used in Calculation of Spoil Pile Height: Side Bench Spoil

$$A_{rI} = A_2 + A_3 - A_4$$

$$A_{rI} = 2w_b H_s + H_s^2 [1/2(\tan \theta) - 1/2(\tan \alpha)] - \left[H_s (1/\tan \theta - 1/\tan \alpha) + 3w_b/2 \right] h_3/2$$

where w_b = width of the bucket.

3.12. Equations for Rehandle Percentages

$$p_{OV} = \frac{A_{rOB}}{A_{OB}} \times 100, \text{ percent}$$

where $A_{OB} = W(D_1 + D_2)(1 + \delta)$, feet²

$$p_{IN} = \frac{A_{rI}}{A_I} \times 100$$

where $A_I = WI(1 + \delta)$

$$p = p_{OV} + p_{IN}$$

3.13. Check for Dragline Reach Digging Interburden for Required Spoil Pile Height

$$H_{SF} \leq H_d$$

3.14. Equation for V_1 , the Volume of Side-Bench Material

$$V_1 = A_{DL} \times L \times 0.037, \text{ bcy}$$

A_{DL} = cross-sectional area of bench material, feet²

$$A_{DL} = D_1 W, \text{ feet}^2$$

3.15. Equation for V_2 , the Volume of Keyway

$$V_2 = A_k \times L \times 0.037, \text{ bcy}$$

where A_k = cross-sectional area of keyway, feet²

$$A_k = W_k D_2 + D_2^2 [(\tan \phi)/2 + (\tan \phi_1)/2]$$

3.16. Equation for V_3 , the Volume of Remaining Bank Material

$$V_3 = V_T - V_2$$

where $V_T = A_T \times L \times 0.037, \text{ bcy}$

$$A_T = D_2 \times W, \text{ feet}^2$$

4. Production Estimate

Having defined the pit geometry, two additional steps are required to get estimates of the dragline production rates. The first is to estimate average digging depths, swing angles, and spoil pile heights for each digging component. Average digging heights are easily estimated as the midpoint elevations of the bank or rehandle sections being dug. The average swing angles for a component are estimated graphically from plan views such as those shown in Figures B-5 and B-6. Average spoil pile heights are estimated graphically from pit sections such as those shown in Figures B-5 and B-6. Estimated swing angles for the case study described in this Appendix are shown in Table B-1.

Table B-1. Average Swing Angles by Dig Component for Case Study No. 1

| Dig Component | Spoil Dump Position | Average Swing Angle (degrees) |
|--------------------|-----------------------|-------------------------------|
| (1) Dragline Bench | Overburden Spoil Pile | 90 |
| (2) Keyway | Overburden Spoil Pile | 90 |
| (3) Remaining Bank | Overburden Spoil Pile | 45 |
| (4) Interburden | Main Spoil Pile | 135 |
| (5) Rehandle | Main Spoil Pile | 110 |

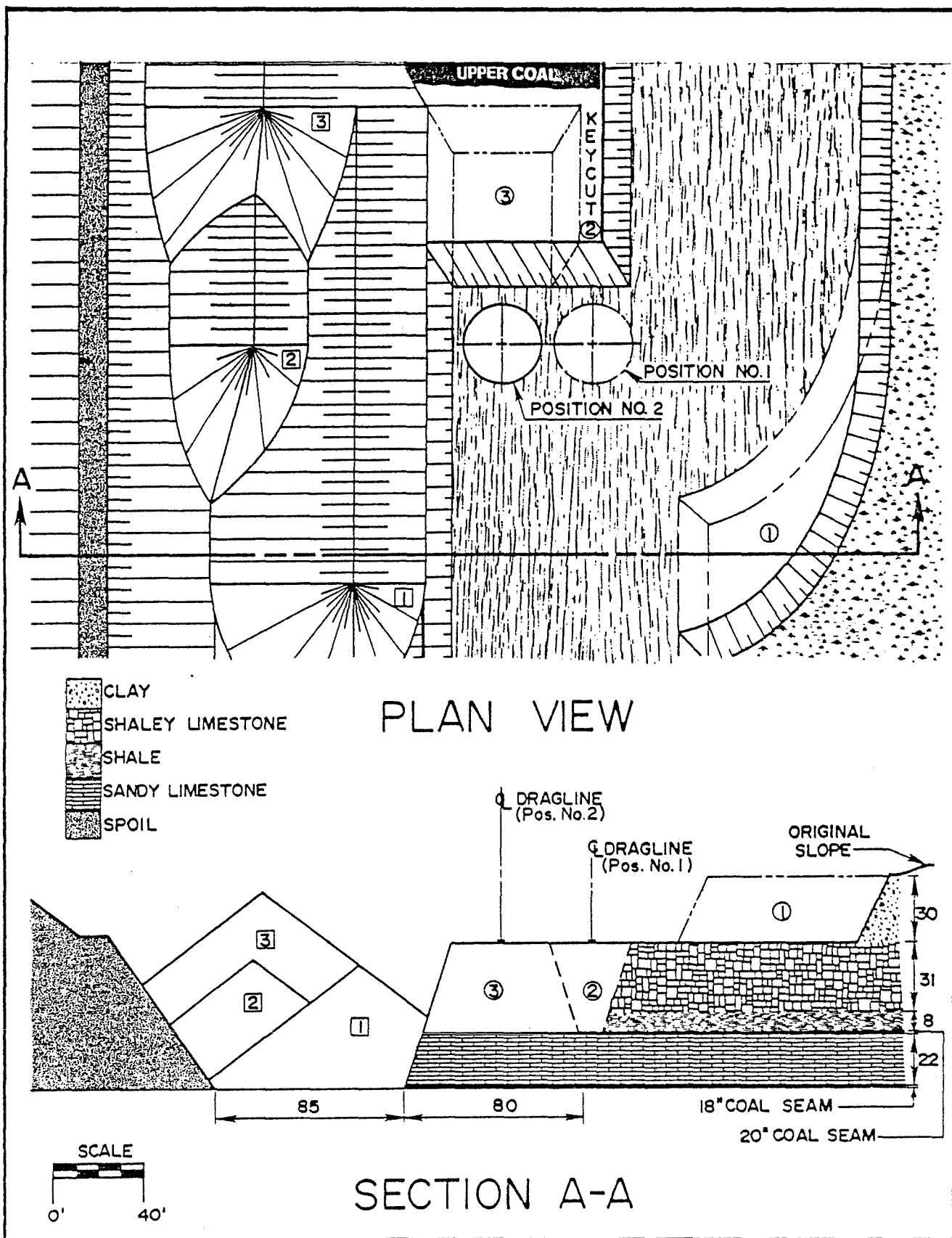


Figure B-5. Pit Geometry for Overburden Pass: Case Study

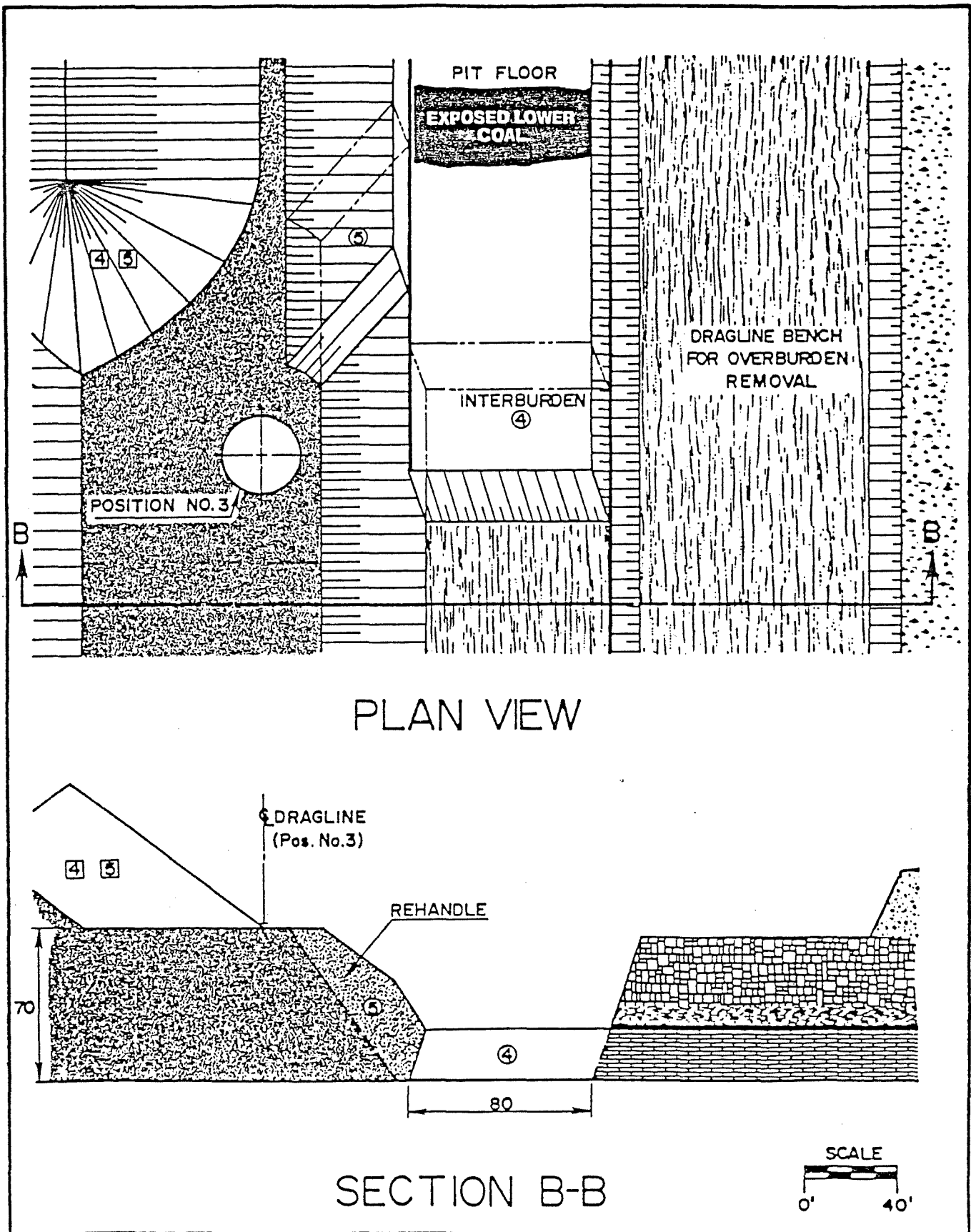


Figure B-6. Pit Geometry for Interburden Pass: Case Study

The second step is to convert the foregoing parameters into dragline cycle time estimates. In this model, this is done using cycle time equations derived from time study data. The equations for the Marion 7400 dragline that is the subject of the case study are given below.

$$T_{ti} = T_{di} + T_{Si}$$

where $T_{di} = 14.0 + 0.13d$

T_{di} = dig time, seconds

$T_{Si} = 18.7 + 0.16\Omega + 0.23d + 1.3z_2$

T_{Si} = swing time, seconds

Ω = swing angle, degrees

d = digging depth, feet

$$z_2 = \begin{cases} 1, & \text{if keyway} \\ 0, & \text{otherwise} \end{cases}$$

Now, the swings per hour for each dig component can be calculated, as follows:

$$S_i = 3600/T_{ti}$$

The bucket carrying capacity for component i is:

$$C_i = b \times f_i / (1 + \delta)$$

where, for the case study,

- Dragline benching

$$f_1 = 70$$

- Keyway and bank material

$$f_2 = 93$$

- Interburden

$$f_3 = 84$$

- Rehandle

$$f_4 = 93$$

The time required to dig each component is:

$$T_i = V_i / (C_i S_i), \text{ hours}$$

The total time to complete a block is:

$$T = \sum T_i, \text{ hours}$$

The scheduled time required to dig one block is:

$$T_S = T / pq, \text{ hours}$$

where p = dragline availability, decimal fraction.

q = dragline utilization, decimal fraction.

The dragline production rate per digging hour is:

$$P_H = V_M / T_S, \text{ bcy/hr}$$

where V_M = total volume of one block, bcy.

The estimated monthly dragline production rate is:

$$P_M = P_H \times H_M$$

where H_M = scheduled working hours per month.

5. Calculations for Case Study

The calculations for the Marion 7400 dragline are presented in this section.

Step 1. Compute the Height of the Spoil Pile Bench, H_s (feet)

$$a = (1/2 \times 0.7536 - 1/2 \times 1.428) - [(1/0.7536 - 1/1.428)^2 \times 3.732 \times 0.7536] \div 2(0.7536 + 3.732) = 0.192$$

$$b = 80 + 2 \times 8.3 - [3 \times 8.3 (1/0.7536 - 1/1.428) \times 3.732 \times 0.7536] \div 2(0.7536 + 3.732) = 91.7$$

$$c = -80(30 + 39)(1 + 0.30) - 9 \times 8.3^2 \times 0.7536 \times 3.732 \div 8(0.7536 + 3.732) = -7224.6$$

$$H_s = \left[-91.7 \pm \sqrt{91.7^2 - 4 \times 0.192 \times -7224.6} \right] \div 2 \times 0.194 = 68.85$$

$$H_s = 69 \text{ feet}$$

Step 2. Compute the Height of the Material to be Used for the Spoil Bench, h_1 (feet)

$$a = 2/0.7536^2 (1/0.7536 + 1/1.428) - 1/0.7536 = 0.42$$

$$b = -2 \times (80 + 2 \times 8.3)/0.7536 (1/0.7536 + 1/1.428) = 127.1$$

$$c = (80 + 2 \times 8.3)^2 / 2 (1/0.7536 + 1/1.428) = 2309.8$$

$$h_1 = \left[127.1 \pm \sqrt{-127.1^2 - 4 \times 0.42 \times 2309.8} \right] \div 2 \times 0.42 = 19.4$$

$$h_1 = 19.4$$

Step 3. Compute Total Spoil Pile Height for the Overburden Pass, H_s (feet)

$$H'_s = 69 + 19.4 = 88.4 \text{ feet}$$

Step 4. Compute the Height of the Spoil Pile Toe, h_3 (feet)

$$h_3 = \left\{ [69'(1/0.7536 - 1/1.428) + (3 \times 8.3)/2] \times 3.732 \times 0.7536 \right\} \\ \div (0.7536 + 3.732) = 32.7 \text{ feet}$$

Step 5. Check Toe Height for Rehandle

$$33' > 22.5'$$

The spoil pile covers the upper coal seam. The rehandle section is given by

$$A_{rOB} = (33 - 22 - 1.5)^2 \times 0.9659^2 \times 0.76604 \div 2 \times 0.309 \\ = 96.7 \text{ feet}^2$$

Step 6. Check for Dragline Reach for No Rehandle for the Overburden Pass, R_{nr} (feet)

$$R_{nr} = (88.4' - 33')/0.7536 + (39 + 22 - 33)/3.732 = 81 \text{ feet}$$

$$81' < 125'$$

Step 7. Check for Dragline Dump Height for the Overburden Pass

$$(88.4 - 64.5) \leq 76'$$

$$23.9 < 76'$$

Step 8. Check for Dragline Reach on the Interburden Pass, R_I (feet)

$$R_I = (69 - 33)/0.7536 + 80 - 33/3.732 = 118.9 \text{ feet}$$

$$118.9' < 125'$$

Step 9. Compute Spoil Pile Height for the Interburden Pass, H_{SF} (feet)

$$A_I = 22 \times 80 (1 + 0.3) = 2288 \text{ feet}^2$$

$$A_{rI} = A_2 + A_3 - A_4 = 2 \times 8.3 \times 69 + 69^2 (1/2 \times 0.7536 - 1/2 \times 1.428) \\ - [69 (1/0.7536 - 1/1.428) + 3 \times 8.3/2] \times 33/2 = 1673.4 \text{ feet}^2$$

$$A_{SP} = 2288 + 1673.0 = 3961 \text{ feet}^2$$

$$H_{SF} = 3961/80 + 80 \times 0.7536/4 = 64.5 \text{ feet}$$

Step 10. Check for Dragline Dump Height for the Interburden Pass

$$64.5' < 76'$$

Step 11. Compute Dig Component Volumes

a. Dragline Bench Material, V_1 (bcy)

$$V_1 = 30 \times 80 \times 60 \times 0.037 = 5328 \text{ bcy}$$

b. Keyway Material, V_2 (bcy)

$$V_2 = 60 \times 0.037 (10 \times 39 + 39^2/1/2 \times 3.732 + 1/2 \times 3.732) \\ = 1770 \text{ bcy}$$

c. Solid Bank Material, V_3 (bcy)

$$V_T = 39 \times 80 \times 60 \times 0.037 = 6926 \text{ bcy}$$

$$V_3 = 6926 - 1770 = 5156 \text{ bcy}$$

Step 12. Compute Percent Rehandle

$$p = \frac{(1673 + 97)}{9464} \times 100 = 18.7 \text{ percent}$$

Step 13. Compute Dragline Production Rates

a. Compute Total Cycle Time, T_{ti} (seconds)

(1) Dragline Digging Overburden

$$T_{ti} = T_{di} + T_{Si}$$

where $T_{di} = 14.0 + 0.13d$

- Dragline Benching

$$d = 0 \text{ to } 30$$

$$d_{avg} = 15'$$

$$T_{di} = 14.0 + 1.95 = 15.95 \text{ seconds}$$

- Keyway

$$d = 0 \text{ to } 39$$

$$d_{avg} = 19.5$$

$$T_{dk} = 14.0 + 0.13 \times 19.5 = 16.53 \text{ seconds}$$

- Bank Material

$$d = 0 \text{ to } 39$$

$$d_{avg} = 19.5$$

$$T_{db} = 14.0 + 0.13 \times 19.5 = 16.53 \text{ seconds}$$

$$T_{Si} = 18.7 + 0.16 \Omega + 0.23d + 1.3z_2$$

where Ω = swing angle

d = digging depth

$$z_2 = \begin{cases} 1, & \text{if keyway} \\ 0, & \text{otherwise} \end{cases}$$

- Dragline Benching

$$\Omega = 90^\circ$$

$$d = 0 \text{ to } 30'$$

$$d_{\text{avg}} = 15'$$

$$z_2 = 0$$

$$T_{sb} = 18.7 + 0.16 \times 90^\circ + 0.23 \times 15 = 36.55 \text{ seconds}$$

- Keyway

$$\Omega = 90^\circ$$

$$d = 0 \text{ to } 39'$$

$$d_{\text{avg}} = 19.5'$$

$$z_2 = 1$$

$$T_{sk} = 18.7 + 0.16 \times 90 + 0.23 \times 19.5 + 1.3 = 38.89 \text{ sec.}$$

- Bank Material

$$\Omega = 45^\circ$$

$$d = 0 \text{ to } 39'$$

$$d_{\text{avg}} = 19.5'$$

$$z_2 = 0$$

$$T_{sb} = 18.7 + 0.16 \times 45 + 0.23 \times 19.5 = 30.38 \text{ seconds}$$

Total Cycle Time

- Keyway

$$T_{tk} = 16.53 + 38.89 = 55.42 \text{ seconds}$$

- Dragline Benching

$$T_{tD} = 15.95 + 36.55 = 52.5 \text{ seconds}$$

- Bank Material

$$T_{tB} = 16.53 + 30.38 = 46.91 \text{ seconds}$$

(2) Interburden Total Cycle Time, T_{ti} (seconds)

$$T_{ti} = T_{di} + T_{Si}$$

Dig Time

- Interburden

$$T_{dI} = 23.3$$

- Rehandle

$$T_{dR} = 15.4$$

Swing Time (return, position and dump time) T_{Si} (sec.)

$$T_{Si} = 24.8 + 0.13\Omega + 0.07h + 0.03d$$

where h = dumping height

- Interburden; h = 0 to 65

$$h_{avg} = 32.5$$

$$T_{SI} = 24.8 + 0.13 \times 135 + 0.07 \times 32.5 = 44.6 \text{ sec.}$$

- Rehandle; h = 0 to 65

$$h_{avg} = 32.5$$

$$T_{SR} = 24.8 + 0.13 \times 100 + 0.07 \times 32.5 = 40.1 \text{ sec.}$$

Total Cycle Time

- Interburden

$$T_{tI} = 23.3 + 44.6 = 67.9 \text{ seconds}$$

$$T_{tR} = 15.4 + 40.1 = 55.5 \text{ seconds}$$

(3) Compute Swing Cycles/Hour

- Dragline Benching

$$S_D = 3600/52.5 = 68$$

- Keyway

$$S_k = 3600/55.4 = 65$$

- Bank Material

$$S_B = 3600/46.9 = 77$$

- Interburden

$$S_I = 3600/67.9 = 53$$

- Rehandle

$$S_I = 3600/55.5 = 65$$

(4) Compute Bucket Fill Factor (percent)

- Dragline Benching

$$f_1 = 70$$

- Keyway - Bank Wheel

$$f_2 = 93$$

- Interburden

$$f_3 = 84$$

- Rehandle

$$f_4 = 93$$

(5) Compute Bucket Carrying Factor, C_i (bcy)

- Dragline Benching

$$C_{DL} = 14 \times 0.70/1.3 = 7.5$$

- Keyway

$$C_k = 14 \times 0.93/1.3 = 10.0$$

- Bank Material

$$C_B = 14 \times 0.93/1.3 = 10.0$$

- Interburden

$$C_I = 14 \times 0.84/1.3 = 9.0$$

- Rehandle (Use with bank yardage equivalent)

$$C_k = 14 \times 0.93/1.3 = 10.0$$

(6) Time Required to Dig Component i (hours)

- Dragline Benching

$$T_{DL} = 5328/7.5 \times 68 = 10.5 \text{ hours}$$

- Keyway

$$T_k = 1770/10 \times 65 = 2.70 \text{ hours}$$

- Bank Material

$$T_B = 5156/10 \times 77 = 6.7 \text{ hours}$$

- Interburden

$$T_I = 3907/9.0 \times 53 = 8.2 \text{ hours}$$

- Rehandle

$$T_R = 3714/10 \times 65 \times 1.3 = 4.4 \text{ hours}$$

(7) Total Time to Complete Block

$$T = 10.5 + 2.7 + 6.7 + 8.2 + 4.4 = 32.5 \text{ hours}$$

(8) Scheduled Time (hours)

$$T_S = 32.5 / 0.85 \times 0.83 = 46.1 \text{ hours}$$

(9) Production Rate Per Scheduled Hour

$$P_H = 16,161 / 46.1 = 350 \text{ bcy/hr}$$

(10) Production Rate Per Month (bcy/month)

$$P_M = 350 \times 640 = 224,000 \text{ bcy/month}$$

Table B-2. Production Rate Calculation for Marion 7400 Dragline:
Multiple Coal Seam Horseshoe Method

| Output Variable | Dig Components For One Block | | | | |
|---|------------------------------|--------|---------------------|-------------|------------------------------|
| | Side Bench | Keyway | Solid Bank Material | Interburden | Rehandle on Interburden Pass |
| 1. Volume of material, bcy | 5,328 | 1,770 | 5,156 | 3,907 | 2,857* |
| 2. Average swing angles, degrees | 90 | 90 | 45 | 135 | 100 |
| 3. Average digging depth, feet | 15 | 19.5 | 19.5 | 56.5 | 34.5 |
| 4. Average dumping height, feet | 0 | 0 | 12 | 32.5 | 32.5 |
| 5. Total bucket drag (dig) time, seconds | 16 | 16.5 | 16.5 | 23.3 | 15.4 |
| 6. Return and loaded swing times, seconds | 36.5 | 39 | 30.4 | 46.6 | 40.1 |
| 7. Total cycle time, seconds | 52.5 | 55.5 | 46.9 | 67.9 | 55.5 |
| 8. Swing cycles per hour | 68 | 65 | 77 | 53 | 65 |
| 9. Bucket fill factor, percent | 70 | 93 | 93 | 84 | 93 |
| 10. Bucket carrying capacity, bcy | 7.5 | 10 | 10 | 9 | 10 |
| 11. Required digging time, hours | 10.5 | 2.7 | 6.7 | 8.2 | 4.4 |
| 12. Total time to complete one block, hours | 32.5 | | | | |
| 13. Scheduled time for one block, hours** | 46.1 | | | | |
| 14. Production per scheduled hour, bcy/hr | 497 | | | | |
| 15. Monthly production, bcy/month | 224,000 | | | | |

* Rehandle converted to bank equivalent for computational ease.

** At 85 percent availability, 83 percent utilization, and 640 scheduled hours per month.

APPENDIX C
FIELD SURVEY MINE EVALUATION
MINE NO. 1
(CENTRAL REGION)

APPENDIX C
FIELD SURVEY MINE EVALUATION
MINE NO. 1
(Central Region)

INTRODUCTION

This Appendix consists of two parts. The first part contains a quantitative engineering evaluation of the mine operation. The second part contains a qualitative evaluation of reclamation practices, effects of mining on surrounding water resources and potential for improving the existing interburden handling practices.

Estimation of production rates for the observed stripping systems are made using graphical and analytical procedures. Costs are estimated based on purchase of new stripping equipment.

I. ENGINEERING EVALUATION OF THE MINE OPERATION

1. Production Rate Calculations - Dragline
50 Cu. Yd. Bucket, 342 Foot Boom

Two Seams Horseshoe Method

1.1 Input Variables

W = 120 feet: pit width

D = 80 feet: total overburden depth

D₁ = 10 feet: depth of dragline bench - removed by dozers

D₂ = 70 feet: height of dragline bench

D₃ = 20 feet: interburden thickness

S₁ = 60 inches: thickness of upper coal seam

S₂ = 42 inches: thickness of lower coal seam

φ = 70 degrees: highwall angle

θ = 37 degrees: natural angle of repose

α = 50 degrees: angle of oversteepened spoil

δ = 30 percent: average spoil factor

L = 150 feet: length of block
R_d = 315 feet: dragline dump radius
R_{ef} = 263 feet: dragline effective radius
H_d = 128 feet: dragline dump height
b = 50 cu. yd.: dragline bucket capacity
q = 0.83: dragline utilization
p = 0.85: dragline availability
w_b = 10 feet: width of bucket

1.2 Output Variables

H_S = Height of spoil bench above the pit floor, feet
h₁ = Height of material for filling the spoil bench, feet
H'_S = Spoil pile height from the overburden pass, feet
h₃ = Height of spoil pile toe from the overburden pass, feet
R_{nr} = Dragline radius required for no rehandle, feet
R_I = Dragline reach required for the interburden pass, feet
H_{SF} = Height of spoil pile from the interburden pass, feet
V_i = Dig volumes of each dragline pass, bcy
V_{ROB} = Volume of rehandle material, highwall pass, bcy
V_{RI} = Volume of rehandle material, spoil side pass, bcy
S_i = Swing cycles per hour for each pass
C_i = Bucket carrying capacity for each pass, bcy
T_i = Time required to dig material of each pass, hours
T = Time to complete one block, hours
T_S = Scheduled time to dig one block, hours
P_H = Dragline production rate per scheduled hour, bcy
P_M = Dragline monthly production, bcy
P_Y = Dragline annual production, bcy

1.3 Step-by-Step Procedure for Computing Dragline Production Rate

1.3.1 Compute Spoil Bench Height Above the Pit Floor, H_S , ft.
(See Figure C-1)

$$H_S = (-b \pm \sqrt{b^2 - 4ac})/2a, \text{ ft.} \quad (\text{See Appendix B for derivation})$$

where

$$a = (1/2 \tan \theta - 1/2 \tan \alpha) - (1/\tan \theta - 1/\tan \alpha)^2 \times \tan \phi \tan \theta / 2(\tan \theta + \tan \phi)$$

$$b = W + 2w_b - 3w_b (1/\tan \theta - 1/\tan \alpha) \tan \phi \tan \theta \div 2(\tan \theta + \tan \phi)$$

$$c = -W(D_1 + D_2 + D_3)(1 + \delta) - 9w_b^2 \tan \phi \tan \theta \div 8(\tan \theta + \tan \phi)$$

$$a = (1/2 \times 0.7536 - 1/2 \times 1.1918) - (1/0.7536 - 1/1.1918)^2 \times 2.7475 \times 0.7536 / 2(0.7536 + 2.7475) = 0.17$$

$$b = 120 + 2 \times 10 - 3 \times 10 (1/0.7536 - 1/1.1918) \times 2.7475 \times 0.7536 / 2(0.7536 + 2.7475) = 128.2$$

$$c = -120(10 + 70 + 20)(1 + 0.30) - 9 \times 10^2 \times 2.7475 \times 0.7536 \div 8(0.7536 + 2.7475) = -15666.5$$

$$H_S = (-128.2 \pm \sqrt{128.2^2 - 4 \times 0.17 \times -15666.5}) / 2 \times 0.17 = 107.0, \text{ ft.}$$

1.3.2 Compute Spoil Pile Height from the Overburden Pass, H'_S , ft.
(See Figure C-1)

$$h_1 = (-b \pm \sqrt{b^2 - 4ac})/2a, \text{ ft.}$$

$$H'_S = h_1 + H_S$$

$$a = 2/\tan^2 \theta (1/\tan \theta + 1/\tan \alpha) - 1/\tan \theta$$

$$b = -2(W + 2w_b)/\tan \theta (1/\tan \theta + 1/\tan \alpha)$$

$$c = (W + 2w_b)^2 / 2(1/\tan \theta + 1/\tan \alpha)$$

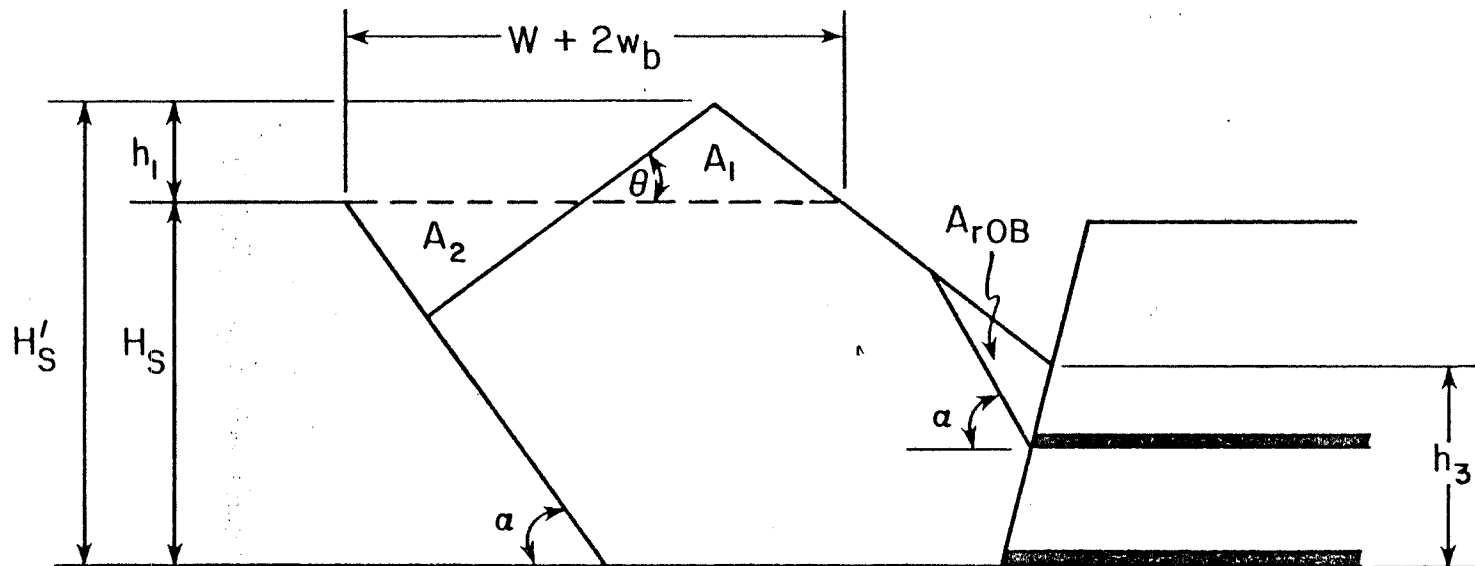


Figure C-1. Variables Used in Overburden Spoil Height Calculations

$$a = 2/0.7536^2(1/0.7536 + 1/1.1918) - 1/0.7536 = 0.31$$

$$b = -2(120 + 2 \times 10)/0.7536(1/0.7536 + 1/1.1918) = -172.1$$

$$c = (120 + 2 \times 10)^2/2(1/0.7536 + 1/1.1918) = 7424$$

$$h_1 = 172.1 \pm \sqrt{-172.1^2 - 4 \times 0.31 \times 7424/2 \times 0.31} = 47, \text{ ft.}$$

$$H'_S = 47 + 107 = 154, \text{ ft.}$$

1.3.3 Compute Height of Spoil Pile Toe, h_3 , ft.
(See Figure C-1)

$$h_3 = [H'_S(1/\tan \theta - 1/\tan \alpha) + 3w_b/2] \tan \phi \tan \theta \\ \div (\tan \theta + \tan \phi), \text{ ft.}$$

$$h_3 = [107(1/0.7536 - 1/1.1918) + 3 \times 10/2] \\ \times 2.7475 \times 0.7536/(0.7536 + 2.7475) = 40, \text{ ft.}$$

1.3.4 Check Toe Height for Rehandle, Overburden Pass

40' > 28.5' - rehandle is required

o Compute rehandle section, A_{rOB} , ft.²

$$A_{rOB} = [h_3 - (D_3 + S_1 + S_2)]^2 \sin \phi^2 \sin [180 - (\alpha + \phi)] \\ \times \sin (\theta + \phi)/2 \sin (\alpha - \theta), \text{ ft.}^2$$

$$A_{rOB} = [40 - (20 + 5 + 3.5)]^2 0.939^2 \times 0.866 \times 0.866 \\ \div 2 \times 0.2250 = 194, \text{ ft.}^2$$

1.3.5 Check for Dragline Reach for No Rehandle for the Overburden Pass, R_{nr} , ft.

$$R_{nr} = (H'_S - h_3)/\tan \theta + (D_1 + D_2 + D_3 - h_3) \div \tan \phi, \text{ ft.}$$

$$R_{nr} = (154 - 40)/0.7536 + (10 + 70 + 20 - 40)/2.7475 \\ = 173, \text{ ft.}$$

$$R_{nr} < R_{ef}$$

173' < 263', no extended bench necessary

1.3.6 Check for Dragline Reach on the Spoil Pass, R_I , ft.

$$R_I = (H_S - h_3)/\tan \theta + W - h_3/\tan \phi, \text{ ft.}$$

$$R_I = (107 - 40)/0.7536 + 120 - 40/2.7475 = 194, \text{ ft.}$$

$$R_I < R_{ef}$$

194' < 263' - dragline has sufficient reach

1.3.7 Compute Spoil Pile Height for the Spoil Pass, H_{SF} , ft.

$$H_{SF} = A_{sp}/W + W \tan \theta /4, \text{ ft.}$$

where

$$A_{sp} = A_I + A_{rI}, \text{ ft.}^2$$

$$A_I = \text{section of interburden, ft.}^2$$

$$A_I = D_3 W (1 + \delta), \text{ ft.}^2$$

$$A_I = 20 \times 120 (1 + 0.30) = 3,120, \text{ ft.}^2$$

$$A_{rI} = \text{rehandle section, ft.}^2$$

$$A_{rI} = A_2 + A_3 - A_4, \text{ ft.}^2$$

$$A_{rI} = 2w_b H_S + H_S^2 [1/2(\tan \theta) - 1/2(\tan \alpha)] \\ - [H_S(1/\tan \theta - 1/\tan \alpha) + 3w_b/2] h_3/2, \text{ ft.}^2$$

$$A_{rI} = 2 \times 10 \times 107 + 107^2 (1/2 \times 0.7536 - 1/2 \times 1.1918) \\ - [107(1/0.7536 - 1/1.1918) + 3 \times 10/2] \times 40/2 \\ = 3,573, \text{ ft.}^2$$

$$A_{sp} = 3,120 + 3,573 = 6,693, \text{ ft.}^2$$

$$H_{SF} = 6693/120 + 120 \times 0.7536/4 = 78, \text{ ft.}$$

$$H_{SF} < H_d$$

78' < 128' - dragline has sufficient dump height

1.3.8 Compute Dig Volumes of Each Dragline Pass, V_i , bcy

- Overburden pass, V_1 , bcy

$$V_1 = D_2 \times W \times L/27, \text{ bcy}$$

$$V_1 = 70 \times 120 \times 150/27 = 46,667, \text{ bcy}$$

- Interburden pass, V_2 , bcy

$$V_2 = D_3 \times W \times L/27, \text{ bcy}$$

$$V_2 = 20 \times 120 \times 150/27 = 13,333, \text{ bcy}$$

- Rehandle, bcy

$$V_{rI} = [A_{r1}/(1 + \delta)] \times L/27, \text{ bcy}$$

$$V_{rI} = [3,573/(1 + 0.30)] \times 150/27 = 15,269, \text{ bcy}$$

1.3.9 Compute Percent Rehandle, p , percent

$$p = \frac{V_{rI}}{V_T} \times 100, \text{ percent}$$

where

$$V_T = V_1 + V_2, \text{ bcy}$$

$$V_T = 46,667 + 13,333 = 60,000, \text{ bcy}$$

$$p = \frac{15,269}{60,000} \times 100 = 25.5, \text{ percent}$$

1.3.10 Estimate Average Swing Angles for Each Dragline Pass

Graphical estimate of the average swing angles for each dragline pass are shown in Table C-1.

Table C-1. Average Swing Angles for Each Dragline Pass

| Pass No. | Spoil Dump Position | Average Swing Angle (Degrees) |
|------------------------------|-----------------------|-------------------------------|
| (1) Overburden Highwall Side | Overburden Spoil Pile | 90 |
| (2) Interburden Spoil Side | Main Spoil Pile | 180 |
| (3) Rehandle Spoil Side | Main Spoil Pile | 120 |

1.3.11 Compute Swings per Hour for Each Dragline Pass, S_i

Swings per hour are computed for each pass from the graph shown in Figure C-2. Complete cycle is estimated to have 28 sec. dig time and 4 sec. delay for digging overburden and interburden material. Rehandle is estimated to have 20 sec. dig time and 4 sec. delay in one cycle. Swing cycles per hour for each pass are shown in Table C-2.

1.3.12 Compute Bucket Carrying Capacity for Each Pass, C_i , bcy

$$C_i = b \times f_i / (1 + \delta), \text{ bcy}$$

where, for this mine

$$f_1 = 0.8 \text{ sandstone}$$

$$f_2 = 0.85 \text{ shale}$$

$$f_3 = 0.9 \text{ rehandle}$$

1.3.13 Compute Time Required to Dig Material at Each Pass, T_i , hrs.

$$T_i = V_i / (C_i S_i), \text{ hrs.}$$

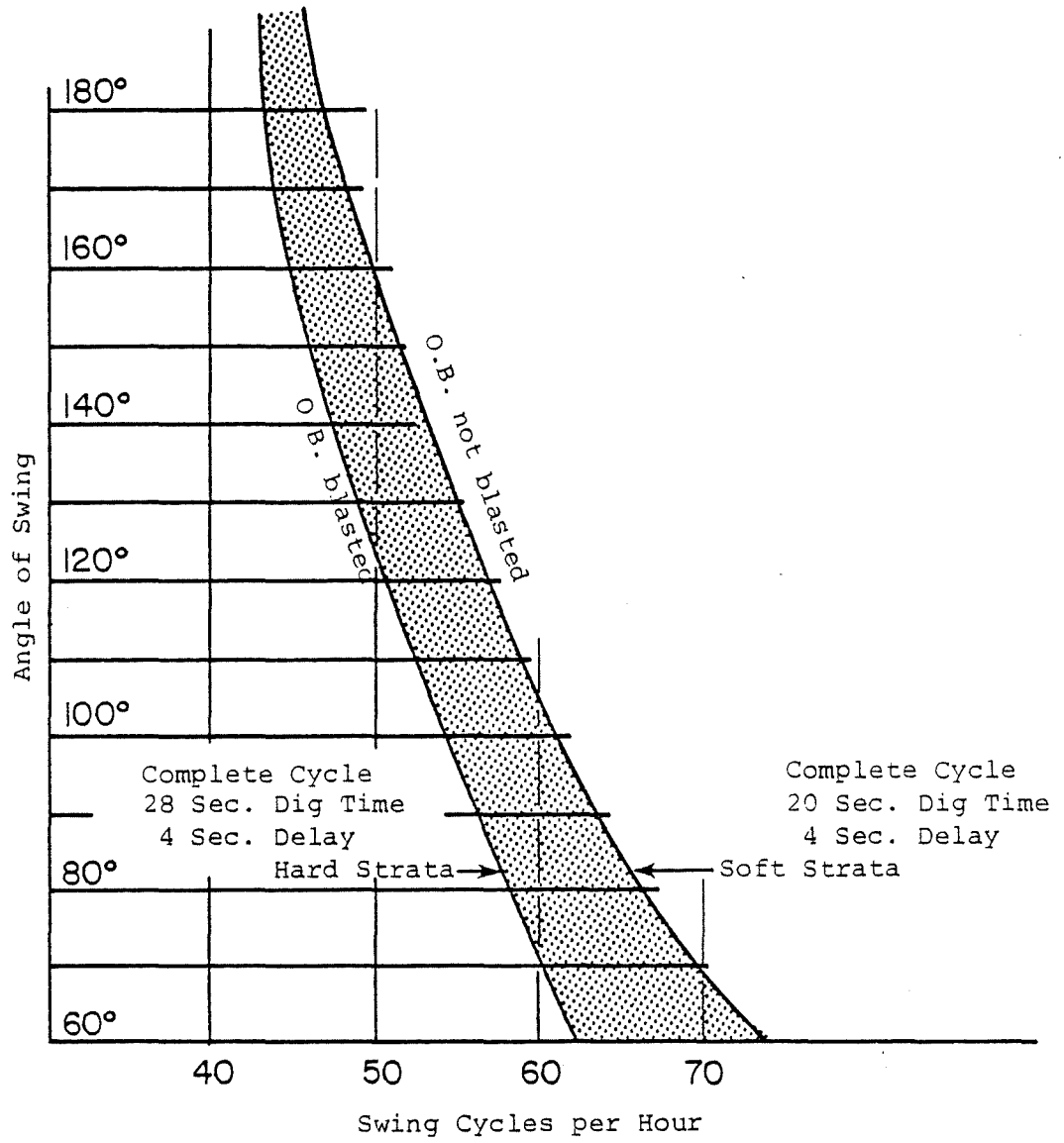


Figure C-2. Swing Cycles per Hour vs. Angle of Swing

1.3.14 Compute Total Operating Time to Complete One Block, T, hrs.

$$T = \sum T_i, \text{ hrs.}$$

1.3.15 Compute Scheduled Time to Dig One Block, T_S, hrs.

$$T_S = T/pq, \text{ hrs.}$$

1.3.16 Compute Dragline Production Rate per Scheduled Hour, P_H, bcy

$$P_H = V_M/T_S, \text{ bcy}$$

1.3.17 Compute Dragline Monthly Production, P_M, bcy

$$P_M = P_H \times H_M, \text{ bcy}$$

H_M = 660 hrs.: scheduled working hours per month

1.3.18 Compute Dragline Annual Production, P_Y, bcy

$$P_Y = 12 \times P_M, \text{ bcy}$$

Results from the computation of dragline production rate are summarized in Table C-2.

Table C-2. Production Rate Calculation for Dragline
50 Cu. Yd. Bucket

| Output Variable | Dig Component for One Block | | |
|--|-----------------------------|-----------------------|--------------------|
| | Overburden Pass 1 | Interburden Pass 2 | Rehandle Pass 2 |
| Volume of Material, bcy | 46,667 | 13,333 | 15,269 |
| Digging Time, sec. | 32 | 32 | 24 |
| Average Swing Angle, degrees | 90 | 180 | 120 |
| Swing Cycles Per Hour | 56 | 43 | 58 |
| Bucket Fill Factor, percent | 80 | 85 | 90 |
| Bucket Carrying Capacity, bcy | 46 | 49 | 68 |
| Required Digging Time, hours | 18.1 | 6.32 | 3.87 |
| Total Time to Complete One Block, hours | 28.29 | | |
| Scheduled Time for One Block, hours | 39.6 | | |
| Production Per Scheduled Hour, bcy | 1,515 | | |
| Monthly Production Rate, bcy | 1,000,000 | | |
| Annual Production Rate, bcy | 12,000,000 | | |

2. Production Rate Calculation - Dragline
110 Cu. Yd. Bucket

Two Seams One Pass Extended Bench Method

2.1 Input Variables

W = 100 feet: pit width
D = 60 feet: total overburden depth
D₁ = 20 feet: depth of dragline bench
D₂ = 40 feet: height of dragline bench
D₃ = 20 feet: interburden thickness
S₁ = 60 inches: thickness of upper coal seam
S₂ = 42 inches: thickness of lower coal seam
W_b = 100 feet: width of solid bench
φ = 70 degrees: highwall angle
θ = 37 degrees: natural angle of repose
α = 50 degrees: angle of oversteepened spoil
f = 30 percent: average spoil factor
R_d = 300 feet: dragline dump radius
R_{ef} = 235 feet: dragline effective radius
H_d = 107 feet: dragline dump height
b = 110 cu. yd.: dragline bucket capacity
q = 0.83 dragline utilization
p = 0.85 dragline availability
W_K = 20': bottom width of dragline keycut
L = 150': block width

2.2 Output Variables

| | |
|----------|--|
| H_S | = Height of final spoil pile, feet |
| W_e | = Width of the extended bench, feet |
| A_r | = Extended bench area, ft. ² |
| A_{K1} | = Overburden keyway area, ft. ² |
| A_B | = Dragline benching area, ft. ² |
| A_{K2} | = Area of additional keyway, ft. ² |
| V_i | = Volume of dig component i, bcy. |
| S_i | = Swings per hour of dig component i |
| C_i | = Bucket carrying capacity of dig component i, bcy |
| T_i | = Time to dig component i, hrs. |
| T | = Operating time to dig one block, hrs. |
| T_S | = Scheduled time to dig one block, hrs. |
| P_H | = Dragline production rate per scheduled hour, bcy |
| P_M | = Dragline monthly production, bcy/month |
| P_Y | = Dragline annual production, bcy |

2.3 Step-by-Step Procedure of Computing Dragline Production Rate

2.3.1 Compute Height of Final Spoil Pile, H_S , ft.

$$H_S = (20 + 40 + 20)(1.3) + 100 \tan 37^\circ/4 = 123, \text{ ft.}$$

2.3.2 Compute Width of the Extended Bench, W_e , ft.

- Main distance to highwall:

$$\text{Overall width over shoes: } 110.5' \div 2 = 55, \text{ ft.}$$

$$\text{Safety: } 10, \text{ ft.}$$

$$\text{Min. distance to highwall: } 55' + 10' = 65, \text{ ft.}$$

- Effective dragline reach: $300' - 65' = 235, \text{ ft.}$

- Extended bench required for reach, W_e : 54, ft. (graphically)
- Extended bench required based on positioning dragline over the middle of interburden keyway, $W_e = 65' + 33' = 98$, ft.

2.3.3 Compute Rehandle/Extended Bench area, A_r , ft.²
(See Figure C-2)

$$A_r = A_1 + A_2, \text{ ft.}^2$$

where

$$A_1 = (98 + 142)45/2 = 5,400, \text{ ft.}^2$$

$$A_2 = 42 \times 23.5/2 = 493, \text{ ft.}^2$$

$$A_r = 5,400 + 493 = 5,893, \text{ ft.}^2$$

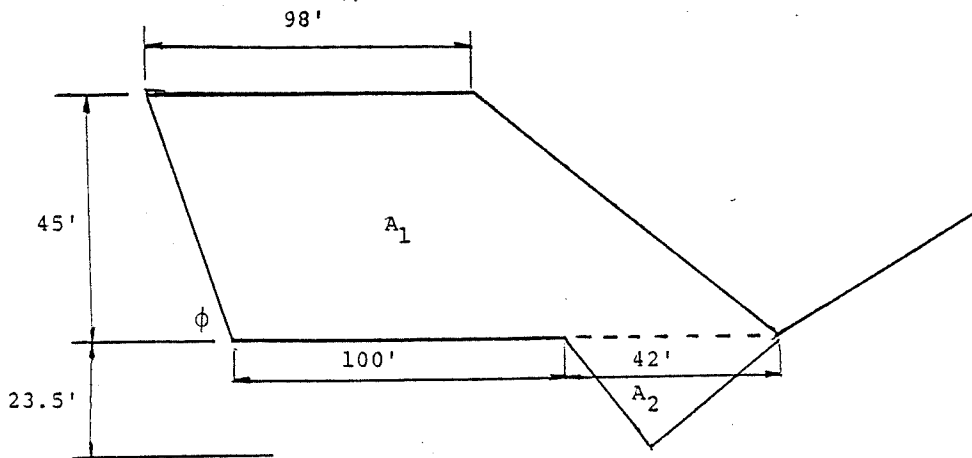


Figure C-3. Area of Extended Bench

- Material to build the extended bench overburden keyway area (min), A_{K_1} , ft.²

$$A_{K_1} = 20 \times 40 + 2 \times (40 \times 40 / \tan 70) / 2 = 1,797, \text{ ft.}^2$$

- Dragline benching area, A_B , ft.²

$$A_B = 20 \times 100 \times 1.3 = 2,600, \text{ ft.}^2$$

$$\text{Total area} = 1,797 + 2,600 = 4,397, \text{ ft.}^2$$

- Additional material to build the extended bench, A_{K_2} , ft.²

$$A_{K_2} = A_r = A_B - A_{K_1}$$

$$A_{K_2} = 5,893 - 4,397 = 1,496, \text{ ft.}^2$$

2.3.4 Compute Average Swing Angles and Volumes for Each Dig Component

- Upper keyway to extended bench

Min. swing angle: 115°

Max. swing angle: 139°

Ave. swing angle: 127°

$$A_{K_1} + A_{K_2} = 3,293.5$$

$$\text{Volume, } V_1 = 3,293.5 \times 150/17 \times 1.3 = 14,075, \text{ bcy}$$

- Dragline benching to extended bench

Min. swing angle: 102°

Max. swing angle: 132°

Ave. swing angle: 117°

$$\text{Volume, } V_2 = 2,600 \times 150/27 \times 1.3 = 11,111, \text{ bcy}$$

2.3.5 Compute Percent of Rehandle, p, percent

$$p = \frac{VR}{VT} \times 100, \text{ percent}$$

where $VT = (D_1 + D_2 + D_3)W \times L/27, \text{ bcy.}$

$$VT = (20 + 40 + 20)100 \times 150/27 = 44,444, \text{ bcy}$$

$$VR = Ar \times L/27, \text{ bcy.}$$

$$VR = 5,893.5 \times 150/27 = 32,742, \text{ bcy}$$

$$p = \frac{32,742}{44,444 (1+0.3)} \times 100 = 56.7, \text{ percent rehandle}$$

2.3.6 Compute Swings per Hour for Each Dig Component, S_1

Swings per hour are computed for each dig component from the graph shown in Figure C-2. Complete cycle is estimated to have 28 sec. dig time and 4 sec. delay for digging overburden and interburden material. Rehandle is estimated to have 20 sec. dig time and 4 sec. delay in one cycle. Swings per hour for overhead digging is computed as follows:

Increase digging time by 47%

Normal 51 swings/hr = 70.6 secs/swing

$70.6 = S + 28 + 4$, $S = 38.6$ secs. swing

$T' = S + 28 \times 1.47 + 4 = 38.6 + 4 + 41.16 = 83.76$, secs.

(or 43 swings per hr.)

Swing cycles, S_i , per hour for each pass are shown in Table C-3.

2.3.7 Compute Bucket Carrying Capacity for Each Dig Component, C_i , bcy.

$$C_i = b \times f_i / (1 + \delta), \text{ bcy.}$$

where

f_i = bucket fill factor for each dig component (see Table C-3)

2.3.8 Compute Time Required to Dig Material for Each Dig Component, T_i , hrs.

$$T_i = V_i (C_i S_i), \text{ hrs.}$$

2.3.9 Compute Total Operating Time to Complete One Block, T , hrs.

$$T = \sum T_i, \text{ hrs.}$$

2.3.10 Compute Scheduled Time to Dig One Block, T_S , hrs.

$$T_S = T/pq, \text{ hrs.}$$

2.3.11 Compute Dragline Production Rate per Scheduled Hour, P_H , bcy

$$P_H = V_M / T_S, \text{ bcy}$$

2.3.12 Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy}$$

$H_M = 660$ hrs.: scheduled working hours per month

2.3.13 Compute Dragline Annual Production, P_Y, bcy

$$P_Y = 12 \times P_M, \text{ bcy}$$

Results from the computations of dragline production rate are summarized in Table C-3.

3. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables C-4, C-5, and C-6 show detailed estimated hourly ownership and operating costs for draglines and dozer.

The total estimated hourly ownership and operating costs for a dragline and dozer used as utility machine are \$686.78 for 50 cu. yd., and \$1,075.41 for 110 cu. yd. draglines.

4. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by dragline with 400 H.P. dozer used as utility machine is \$0.46 for 50 cu. yd. and \$0.57 for 110 cu. yd. draglines.

Table C-3. Production Rate Calculation for Dragline
110 Cu. Yd. Bucket

Extended Bench Method

| Dig Component of One Block | Average Swing Angle, Degrees | Swing Cycles Per Hour | Bucket Carrying Capacity, bcy | Volume of Material, bcy | Operating Time, hrs. |
|--|------------------------------|-----------------------|-------------------------------|-------------------------|----------------------|
| Upper keyway to extend bench (from position 1) | 127° | 49 | 84 (0.99 fill factor) | 14,075 | 3.4 |
| Upper lift to extend bench (from position 1) | 117° | 43 | 64 (0.76 fill factor) | 11,111 | 4.0 |
| Rest of upper bank to spoil (from position 1) | 90° | 56 | 84 (0.99 fill factor) | 8,147 | 1.7 |
| Rehandle from position 2, to spoil | 70° | 70 | 84 (0.99 fill factor) | 10,125 | 1.7 |
| Interburden keyway, from position 3 to spoil | 90° | 56 | 84 (0.99 fill factor) | 2,889 | .6 |
| Rest of rehandle, from position 3, to spoil | 70° | 70 | 84 (0.99 fill factor) | 22,619 | 3.8 |
| Rest of interburden, from position 3, to spoil | 70° | 60 | 84 (0.99 fill factor) | 8,222 | 1.6 |
| Total time to complete one block, hrs. | 16.8 | | | | |
| Scheduled time for one block, hrs. | 23.5 | | | | |
| Production per scheduled hour, bcy. | 1,891 | | | | |
| Monthly production rate, bcy. | 1,248,000 | | | | |
| Annual production rate, bcy | 14,976,000 | | | | |

Table C-4. Estimated Hourly Ownership and Operating Costs

Dragline: 50 Cu. Yd. Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$13,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 15,000,000 |
| Salvage Value @ 20% | 3,000,000 |
| Depreciated Amount | 12,000,000 |
| Life of Machine | 20 years |
| | @ 8030 hrs/yr |
| Depreciation Cost | 74.72 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 224.16 |
| Total Ownership Costs | <u>298.88</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 92.60 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | <u>333.39</u> |
| Total O&O Costs | 632.27 |

Table C-5. Estimated Hourly Ownership and Operating Costs

Dragline: 110 Cu. Yd. Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$24,000,000 |
| Extras, Freight & Erection | 2,000,000 |
| Total Price | 26,000,000 |
| Salvage Value @ 20% | 5,200,000 |
| Depreciated Amount | 20,800,000 |
| Life of Machine | 20 years |
| | @ 8030 hrs/yr |
| Depreciation Cost | 129.50 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 388.50 |
| Total Ownership Costs | 518.00 |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 175.00 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 273.90 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | 502.90 |
| Total O&O Costs | 1,020.90 |

Table C-6. Estimated Hourly Ownership and Operating Costs

Dozer: 400 H.P.

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|----------------------------------|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$300,000 |
| Extras, Freight & Erection | - |
| Total Price | 300,000 |
| Salvage Value @ 20% | 60,000 |
| Depreciated Amount | 240,000 |
| Life of Machine | 4 years |
| | @ 8030 hrs/yr |
| Depreciation Cost | 7.47 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>4.48</u> |
| Total Ownership Costs | 11.95 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./oper. hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | |
| | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/hr. | <u>14.00</u> |
| Total Operating Costs | <u>42.56</u> |
| Total O&O Costs | 54.51 |

II. RECLAMATION AND ENVIRONMENTAL ASSESSMENT (by John Tapp)

1. Introduction

Three periods in the mining cycle of the operation are described in this part. The first is the pre-mining period. Available reports and information on new mining permits on adjacent tracts were used to reconstruct the general characteristics of the site.

The second period of the mining operation described is the during-mining period, including the reclamation that was in progress. Most of the information collected during the visit to the site related to this phase of the mining operation. The third phase described is the post-mining period. This period begins when reclamation has just been completed for the entire site and extends into the future. Since no good data were available for this period, the description is a projection based on the information available for the pre-mining and during-mining periods.

In each of the three periods the main focus is placed on the soils, geology, groundwater, and surface water.

2. Description of the Area Prior to Mining

2.1 Soils

Based on the adjacent area, the soil units occurring in the general area of the mining site are shown in Table C-7. These soils were supporting the land uses of cropland, woodland, scrubland, pasture, and idle. A description of each soil series is also shown in SCS(1978). As indicated in Table C-7, five of the soil series can be classified as prime farmland. Observations on a part of the area to be mined where the topsoil had not been stripped showed the A horizon to vary from 2 to 12 inches thick. The B horizon was several feet thick and the C horizon and underlying weathered bedrock, which was observed at an adjacent mine site, was considerably more than 10 feet thick. The entire area generally contains at least 20 feet of unconsolidated glacial till material. The results of the chemical and physical testing of the A, B, and C horizons are shown in Table C-8.

Table C-7. Soil Map Units in the Mining Area

| Map Symbol | Soil Mapping Unit | Slope Group | Percent Slope | Capability Unit | Degree of Erosion | Prime Farmland |
|------------|-----------------------|-------------|---------------|-----------------|-------------------|------------------|
| 2 | Cisne Silt Loam | A | 0-2 | IIIw | - | x ⁽¹⁾ |
| 3A | Hoyleton Silt Loam | A | 0-3 | IIw | - | x |
| 3B2 | Hoyleton Silt Loam | B | 3-6 | IIe | Eroded | x |
| 4C2 | Richview Silt Loam | C | 5-10 | IIIe | Eroded | -(2) |
| 5D3 | Blair Silty Clay Loam | D | 10-18 | VIe | Severely Eroded | - |
| 8E | Hickory Loam | E | 18-30 | VIe | - | - |
| 12 | Wynoose Silt Loam | A | 0-2 | IIIw | - | - |
| 13A | Bluford Silt Loam | A | 0-3 | IIw | - | x |
| 13B2 | Bluford Silt Loam | B | 3-6 | IIe | Eroded | - |
| 14B | Ava Silt Loam | B | 1-5 | IIe | - | x |
| 14C2 | Ava Silt Loam | C | 5-10 | IIIe | Eroded | - |
| 14C3 | Ava Silty Clay Loam | C | 5-10 | IVe | Severely Eroded | - |
| 382 | Belknap Silt Loam | A | 0-3 | IIw | - | 0 ⁽³⁾ |
| 620A | Darmstadt Silt Loam | A | 0-3 | IVe | - | - |
| 929D | Hickory-Ava Complex | D | 10-18 | IVe | - | - |
| 929D3 | Hickory-Ava Complex | D | 10-18 | VIe | Severely Eroded | - |

(1) Does qualify for prime farmland.

(2) Does not qualify for prime farmland.

(3) Qualifies for prime farmland if flooding is less frequent than once in two (2) years during the growing season.

Note: Information From SCS County Technical Guide Section IIA, February 1978.

Table C-8. Results of Chemical and Physical Testing of A, B, and C Horizons

| <u>PARAMETER</u> | <u>HORIZON</u> | | |
|----------------------------------|----------------|-----------|----------|
| | <u>A</u> | <u>B</u> | <u>C</u> |
| pH | 4.6 | 5.1 | 7.1 |
| pH (SMP Buffer) | 6.4 | 6.5 | 7.1 |
| Potential Acidity (tons/acre) | 0.1 | 0 | 0 |
| Organic Matter (%) | 1.4 | 0.3 | 0.7 |
| Total Nitrogen (lbs./acre) | 1972 | 682 | 342 |
| Available Nitrogen (lbs./acre) | 0.5 | 0.4 | 2.1 |
| Available Phosphorus (lbs./acre) | 89 | 9.4 | 2.0 |
| Available Potassium (lbs./acre) | 163 | 192 | 172 |
| Texture (Soil Binder Material) | Silt Loam | Clay Loam | Loam |
| Plant Available Water (%) | 18.3 | 18.2 | 11.3 |

These samples show that the A horizon was acidic which is generally the case for the silt loam soils. Its organic matter, total nitrogen, and plant available water contents were average for this type of soil. The available phosphorus and available potassium were high, indicating that the land had been heavily fertilized in the past. The B horizon was not quite as acidic as the A horizon. It was low in organic matter, total nitrogen, and total phosphorus content. The texture of the soil fraction indicated that the B horizon was borderline between a loam and a clay loam. The available potassium and plant available water contents were similar to the A horizon. The texture of the C horizon indicated that the soil binder fraction of this alluvial material was a loam. The C horizon had a pH greater than 7. The material was very low in total nitrogen content but had an organic matter content higher than that of the B horizon. Available phosphorus content of the C horizon was very low but the available potassium content was about the same as the A and B horizons. The plant available water content of the C horizon was lower than the A and B horizons. The potential acidity for all three horizons was essentially zero.

2.2 Geology

Lying beneath the unconsolidated glacial till material at the site are rocks of the Pennsylvanian System. These rocks are of the Carbondale Formation of the Kewanee Group and have been described by Willman, et. al. (1975). The Carbondale Formation includes strata from the base of the Colchester (No. 2) Coal Member to the top of the Danville (No. 7) Coal Member. The Carbondale Formation consists of a large number of named members, many of which are said to possess remarkable lateral persistence in thickness and lithologic character. The sandstones and gray shales, however, do not follow this pattern with rapid lateral changes being common and being responsible for local and regional changes in the thickness of the formation. The Carbondale Formation varies from less than 150 feet thick in western and northeastern Illinois to more than 400 feet near its outcrop belt in southern Illinois. The Carbondale sandstones occur in elongate, channel facies up to about 100 feet thick and also in thinner sheets. Most of the sandstones are subgraywackes and are more argillaceous than the older Pennsylvania sandstones. The most abundant type of rock in the Carbondale Formation are the gray shales. Some of these shales contain only marine fossils while others contain only plant fossils. Most of the thicker gray shales probably are either delta front or prodelta deposits. Siderific nodules and bands are common, especially in the lower parts of the shales. The widespread limestones are gray to dark gray, argillaceous, and normally fossiliferous. Associated with the marine limestones are extensive black fissile shales that are generally less than 2 feet thick and contain marine fossils. Occurring in the bottom part of the widespread underclays are light gray limestones, commonly without fossils. The four principal

economic coals of Illinois are contained in the Carbondale Formation. These are, in order of importance as reserves, the Herrin (No. 6), Springfield-Harrisburg (No. 5), the Colchester (No. 2), and the Danville (No. 7) Coal Members. The Carbondale Formation in Illinois correlates with the upper portion of the Carbondale Formation and approximately the lower 50 feet of the Sturgil Formation in western Kentucky, where the boundaries are the base of the Kentucky No. 6 coal and the top of the Kentucky No. 11 coal. Essentially, this same interval is called the Carbondale Formation in Indiana.

The primary site that was visited was a long pit running generally east and west in which the No. 5 and No. 6 coals were being removed. The unconsolidated material at the site was bowl shaped in the direction of the pit with a much larger depth of unconsolidated material toward the center of the pit. On the other hand, the rock formations were arch shaped with a larger amount of rock at each end of the pit. This caused the No. 6 coal to be discontinuous in the center and generally not mineable there. The total overburden in the center consisted of approximately 20 feet of unconsolidated material and 20 feet of rock. The ends of the pit consisted of approximately 40 feet of unconsolidated material and 60 feet of rock. The parting (interburden) between the No. 5 and 6 coals was approximately 20 feet thick.

Most of the strata observed during the site visit indicated that a large part of the material was limestone or carbonaceous shale and no acid problems could be expected. This was somewhat verified by analyzing samples of overburden and an interburden sample collected during the site visit. The results of this testing are listed in Table C-9 and show that the potential acidity of all materials analyzed was zero. This table does show that the material has the potential for adding alkalinity to the water as indicated by the neutralization potential results.

In order to obtain an indication of the suitability of the overburden and interburden materials for revegetation, certain chemical and physical analyses were made on the overburden and interburden. The results of this testing are shown in Table C-10. They indicated that all materials were very high in pH. The gray shale and the shaley limestone had a fair amount of organic matter while all the others were lower in organic matter content. All were low in total nitrogen, with the gray shale being the highest. All materials were extremely low in available phosphorus and high in available potassium. The conclusion from this testing is that only the gray shale has any potential as a revegetation agent. The properties of the shale that were analyzed were similar to the C horizon material except that the available potassium content was much higher in the shale. The weathering characteristics of the shale were not investigated; however, poor weathering could be anticipated.

Table C-9. Results of Potential Acidity and Neutralization Potential Testing of Overburden Samples

| <u>MATERIAL</u> | <u>DEPTH BELOW SURFACE (FT)</u> | <u>POTENTIAL ACIDITY (TONS/ACRE)</u> | <u>NEUTRALIZATION POTENTIAL (TONS/ACRE)</u> |
|---------------------------|---------------------------------|--------------------------------------|---|
| Red (weathered) Limestone | 15-16 | 0 | 3.4 |
| Light Gray Limestone | 16-20 | 0 | 0 |
| Dark Gray Shale | 20-35 | 0 | 1.5 |
| Shaley Limestone | 30-50 | 0 | 19.2 |
| Gray Limestone | (Interburden) | 0 | 1.0 |

Table C-10. Results of Overburden and Interburden Testing

| <u>MATERIAL</u> | <u>pH</u> | <u>ORGANIC MATTER (%)</u> | <u>TOTAL NITROGEN (LBS./ACRE)</u> | <u>AVAILABLE PHOSPHORUS (LBS./ACRE)</u> | <u>AVAILABLE POTASSIUM (LBS./ACRE)</u> |
|------------------------------|-----------|---------------------------|-----------------------------------|---|--|
| Red (weathered) Limestone | 9.0 | 0.0 | 97 | 0.4 | 213 |
| Light Gray Limestone | 8.6 | 0.6 | 111 | 0.0 | 254 |
| Dark Gray Shale | 8.0 | 2.2 | 312 | 0.2 | 659 |
| Shaley Limestone | 8.5 | 1.4 | 77 | 0.8 | 219 |
| Gray Limestone (Interburden) | 9.7 | 0.9 | 197 | 0.0 | 646 |

2.3 Groundwater

The occurrence of groundwater in the area has been discussed by Pryor (1956). He states that the most important aquifers are deposits of sand and gravel, sandstone, limestone, chert, and dolomite. Sand and gravel are water yielding because of their relative high porosity and permeability and occur in amounts varying from several inches to over 100 feet thick. The sandstone strata are generally fine-grained and cemented and have relatively low permeabilities. Therefore, they tend not to be good aquifers. Limestone, dolomite, and chert generally have low porosities and permeabilities and yield water only from interconnected cracks and crevices. The success of a well drilled into these rocks depends on encountering one of these crevices. Since these cracks and crevices occur on an irregular basis, their presence at any particular location is difficult to predict.

In the area of the visited mine site, the glacial deposits of sand and gravel tend to be thin with bedrock outcropping in many places. Thin water yielding sand and gravel deposits are restricted to deeper parts of the Beaucoup Creek Valley and the Little Muddy River Valley and are suitable only for domestic water supplies. In nearby areas, ground water is obtained from shallow Pennsylvanian sandstones at depths less than 100 feet. In the area of concern, these permeable Pennsylvanian sandstones are said to occur at depths about 400-500 feet below the ground surface.

The groundwater flow in the shallow sand, gravel, and clay water table aquifer in the area seems to be to the southeast. This is based on a piezometric map of the area. This aquifer generally yields only a small amount of water and the area is said to be a groundwater shortage area. The quality of the water from this shallow aquifer is generally not good, with high salinity reportedly being a problem. Most of the communities in the area have impoundments as their water supply sources. A few communities in the area have wells in the deep sandstone aquifer.

2.4 Surface Water

Surface water in the area of the site visited is principally runoff from the agricultural and forest lands that boundary the creeks. During wet weather periods of the year, the creeks would be expected to flow substantially. The quality during these periods would be expected to be poor with high suspended solids concentrations and probably high concentrations of fertilizers and pesticides from the agricultural lands. The land slope in the area is flat and the streams could be expected to flow slowly in a meandering pattern.

During the site visit, a water sample was taken in this creek above the site and one below the site. These samples were analyzed for several parameters. The results of this testing are shown in the section of this report dealing with water quality during mining.

3. Description of the Area During Mining

3.1 Soils

The State Reclamation Law requires land, which was being used for crops prior to mining, to be reclaimed using 39 inches of what is called texture material and 9 inches of topsoil. This is to allow a deep root zone for the land after mining. The texture material is supposed to be a mixture of the B and C horizons which contains essentially no rocks in the upper 2 feet. The texture material and topsoil must be stockpiled as was the case at the site visited. For areas not in crops prior to mining, the overburden must be graded and 6 inches of topsoil must be placed on top. This type of reclamation requires only stockpiling of topsoil.

At the visited mine site, topsoil was being removed with scrapers and taken to stockpiles. The topsoil was being removed approximately 12 months or 1,000 feet in advance of the overburden removal. The topsoil was being left in the stockpiles for 12-24 months prior to being replaced. During this period, the stockpiles were sometimes mulched and always seeded with rye grass. No chemical sprays were used to stabilize the stockpiles, and in many cases wind and water erosion were a problem. The stockpiles were generally located 2,000-3,000 feet from the areas where the topsoil was removed.

Regrading of the spoil material was sometimes accomplished as much as 2 years after mining. This regrading, prior to replacing the topsoil and texture material, was accomplished with bulldozers and small draglines. After grading and replacing the texture material and/or topsoil, the area was seeded with a quick growing species to control erosion and provide vegetative mulch. The seasons and quick growing species used were rye at 120 pounds per acre in the spring; sudex at 50 pounds per acre during the summer; and a mixture of rye, wheat, oats, and barley at 120 pounds per acre total during the fall. Permanent species are generally seeded in the spring or fall following grading. In general, seeding is accomplished by aerial methods. For areas to be revegetated as forest land, a series of herbaceous species and trees are planted.

3.2 Geology

As the mining has been practiced, the glacial till material tended to end up near the bottom of the spoil pile and the interburden material tended to end up on top. As indicated earlier, the interburden material, where texture material would not be required, is not as good as the glacial till material for use as the lower root zone during revegetation.

Prior to mining, the major creek of the area was diverted around the site. The material removed from the old creek channel during mining was a very mucky type material. This material was very unstable and was causing stability problems with the spoil piles during mining.

Because of the large amount of water being encountered due to wet weather conditions during the site visit, the interburden highwall was not observed. The overburden highwall was observed. Pyrite was observed in the layer just above the No. 6 coal. Some iron staining and a white aluminum oxide precipitate were present in the overburden shale and limestone.

3.3 Groundwater

During mining, a large amount of water was seeping through the spoil piles either directly from precipitation or indirectly through the old works present at the site. The spring had been an exceptionally wet one and water in the pit was a problem at the time of the site visit. Pumping to remove the water was occurring and should have the effect of lowering the groundwater table in the immediate vicinity. As discussed previously, the groundwater in the area is not generally abundant, and this lowering of the water table probably has little significance. The direction of groundwater flow in the immediate vicinity of the mine would be toward the pit instead of its usual direction. The porosity of the spoil material would be greater than the parent material with an increase in water stored in the area during and after mining.

The quality of the groundwater might be expected to decrease somewhat as the water percolating through the spoil piles comes into contact with a finer particle size of material that could be more amenable to dissolution than the parent rock. The rate of this dissolution would probably be slow for the types of rocks encountered at this site. The quality of the groundwater might also be affected by the practice of dumping gob from the coal preparation plant on the No. 6 coal bench and covering it with interburden. The gob tends to be acid forming; and if this practice is not done judiciously, groundwater problems could result. In one instance, what appeared to be acid water was standing in an area close to where the gob was being dumped. Acid problems were

reported in the ponds below the gob piles and in the water seeping through the overburden from the old works. However, sufficient alkaline overburden appeared to be present to neutralize this acid water.

3.4 Surface Water

Surface runoff and pumpage from the pit were passed through sediment basins. Sediment basins were also used to collect sediment below areas being reclaimed. Dugout basins and impoundment type basins were used about equally at the site. The design criteria for the ponds was based on providing the detention time (approximately 10 hours) to settle particles larger than 0.01 mm. This means that the silt gets trapped and clay size particles pass through the ponds. This detention time was based on normal pit pumpage plus the average flow from a 10-year, 24-hour storm event. In practice, the basin size was said to work out to be approximately 500 square feet per acre of drainage area. Short circuiting in the sediment ponds was a problem, and straw bales were sometimes used as energy dissipaters at the inlets to the ponds.

Surface drainage control was practiced during mining. Above high wall diversion ditches, levees, and stream diversion ditches were being used. The design criteria for these flow diversion methods incorporated passing the peak flow from the 100-year, 6-hour storm event for permanent measures and the 10-year, 24-hours storm event for temporary measures.

To get an idea of the quality of surface runoff from the visited mine site, 4 water samples were collected. Two samples were taken from the major creek that had been diverted around the site. One sample was taken just above the site and the other was taken just below the site. A third sample was taken from the pit pumpage just as it left the pipe and just above the sediment pond. A fourth was taken from the outflow pipe of the sediment pond receiving the pit pumpage. The results of chemical analysis of these samples are shown in Table C-11.

Although analysis was not made for all parameters in the state water quality standards, those that were analyzed give an indication as to the possible effects of mining on the aquatic environment. The period of the site visit was during an exceptionally wet spring when stream flows were high. The impact of the high stream flows are shown in water sample analyses by the fact that in most cases, the values of the parameters analyzed in the samples from the pit and the sediment pond effluent were considerably higher than those in the creek. The conclusion is that the discharge from the mine was being significantly diluted by the large stream flows. Based upon the results in Table C-11, the only parameters for which the

Table C-11. Results of the Analysis of Water Samples Collected

| <u>Parameter & Units</u> | <u>Stream Above Site</u> | <u>Active Mine Pit</u> | <u>Sediment Pond Effluent</u> | <u>Stream Below Site</u> |
|---|--------------------------|------------------------|-------------------------------|--------------------------|
| Total Aluminum (mg/l as Al) | 3.6 | 4.2 | 0.3 | 5.7 |
| Total Boron (mg/l as B) | 0.09 | 0.47 | 0.39 | 0.21 |
| Total Calcium (mg/l as Ca) | 180 | 51 | 171 | 60 |
| Total Iron (mg/l as Fe) | 2.3 | 2.1 | 0.4 | 3.4 |
| Total Potassium (mg/l as P) | 7.1 | 20 | 17 | 7.3 |
| Total Magnesium (mg/l as Mg) | 36 | 97 | 96 | 33 |
| Total Manganese (mg/l as Mn) | 0.5 | 0.8 | 0.7 | 0.4 |
| Total Sodium (mg/l as Na) | 55 | 1330 | 1380 | 88 |
| Total Lead (mg/l as Pb) | 0.03 | 0.07 | 0.10 | 0.03 |
| Total Selenium (mg/l as Se) | < 0.3 | < 0.3 | < 0.3 | < 0.3 |
| Total Zinc (mg/l as Zn) | 0.50 | 0.03 | 0.03 | 0.04 |
| Hydroxide (mg/l as OH) | 0.0068 | 0.011 | 0.017 | 0.0054 |
| Bicarbonate (mg/l as HCO ₃) | 126 | 472 | 395 | 122 |
| Carbonate (mg/l as CO ₃) | 0.23 | 1.37 | 1.83 | 0.18 |
| Chloride (mg/l as Cl) | 25 | 649 | 625 | 31 |
| Sulfate (mg/l as SO ₄) | 354 | 2090 | 2120 | 358 |
| Nitrate (mg/l as N) | 0.7 | 4.7 | 5.9 | 1.0 |
| Total Phosphorus (mg/l as P) | 14.7 | 11.3 | 2.5 | 15.8 |
| pH (standard units) | 7.6 | 7.8 | 8.0 | 7.5 |
| Alkalinity (mg/l as CaCO ₃) | 106 | 387 | 326 | 100 |
| Acidity (mg/l as CaCO ₃) | 8 | 15 | 12 | 10 |
| Total Hardness (mg/l as CaCO ₃) | 600 | 530 | 820 | 290 |
| Total Suspended Solids (mg/l) | 39 | 93 | 11 | 81 |
| Total Dissolved Solids (mg/l) | 679 | 3970 | 3930 | 717 |
| Specific Conductance (umhos/cm) | 960 | 5220 | 5130 | 1010 |
| Sodium Adsorption Ratio (meq/meq) | 0.98 | 25.2 | 20.9 | 2.26 |

water quality standards might be exceeded during lower stream flows are chloride, sulfate, and total dissolved solids. The total suspended solids results show that, under the conditions during the site visit, the sediment pond was doing a good job of removing the sediment in the water and the total suspended solids concentrations in the stream were greater than in the sediment pond discharge.

The results in Table C-11 show that the water in the creek would probably be marginally satisfactory for crop irrigation. However, the concentration of dissolved constituents in the water from the pit make it prohibitive for irrigation. The sulfate, chloride, total dissolved solids, specific conductance and sodium adsorption ratio all exceed the values recommended for irrigation water. The concentrations of phosphorus in the creek waters are relatively high and probably reflect the agricultural practices in the area draining to the creek.

4. Description of the Area After Mining

4.1 Vegetative Growth Potential

The practice of adding 4 feet of texture and topsoil as required by the State Law should result in a deep root zone for crop growth. If this practice is properly carried out, the resulting land after mining should develop crop yields similar to those before mining. The practice of putting 6 inches of topsoil on graded overburden for forest and grassland may not return the land to the pre-mining potential unless the unconsolidated glacial till is returned to the surface so that it is the base for the topsoil. If the consolidated overburden and interburden with poor weathering characteristics is used as the base for the topsoil, a very shallow root zone may result. This shallow root zone may be satisfactory for tree growth, but it is still not as desirable for total revegetation as would be the case where the topsoil was laid on the unconsolidated glacial till material. In addition, the water sample analyses indicated that the consolidated overburden contained large quantities of sodium, calcium, and magnesium which can be dissolved in the water as the overburden is pulverized during mining. If this material is near the surface, high salt concentrations could be transported to the root zone by capillary action, thus causing an impairment of growth for sensitive species. Therefore, altering the mining method or dragline operation, where possible, to attain a layer of unconsolidated glacial material below the topsoil would be desirable.

4.2 Groundwater Patterns and Quality

During mining, the ground water table in the local area may be lowered as water tends to flow to the mining pit due to pumpage

from the pit. The quality of this pumpage has been shown to be very high in dissolved minerals. As mining ends and the final pit is left open as part of the reclamation plan, the level of water in the pit should approach that of the local groundwater table. When this occurs, groundwater flow should return to its approximate pre-mining pattern.

The quality of the groundwater will probably be decreased for a long period of time due to the dissolving of the pulverized material. Although the quality of water in the pit may be suitable for wildlife enhancement, it may not be suitable for agricultural irrigation, at least initially.

4.3 Surface Water Quality and Quantity

With the land returned to the approximate original contour and vegetation re-established, the surface runoff pattern should be similar to that before mining. The infiltration rate will probably be similar to that before mining if care has been taken to insure that excessive compaction of the material has not occurred. The quality of surface water runoff after revegetation is completed should be much better than the current pit pumpage. However, as previously indicated, the groundwater recharge to local streams may show an increase in total dissolved solids concentration, reflecting the minerals that have been dissolved as a result of mining.

5. Effects of Mining and Reclamation on Bonding Costs

A conversation with the permits coordinator of the State Department of Mines and Minerals indicated that the individual field inspector tries to calculate actual reclamation cost based on cubic yards of material to be moved during reclamation. The bond cost is then set at this amount. The cost includes the cost of grading the overburden peaks, removing and stockpiling topsoil (and texture material where applicable) and returning the topsoil (and texture) to the land from the stockpiles. Some cost would be allocated for final grading, scarifying, and revegetation, but this cost tends to be minimal in comparison to the cost of moving the overburden and topsoil material. The permit coordinator indicated, for new permits at an existing operation, an allowance may be made to decrease the bonding cost if the mine operator had demonstrated that the dragline is being operated so that the spoil peaks are closer together, etc., so a smaller amount of material would need to be moved during reclamation.

This allowance would be the primary allowance to be made to decrease the cost of bonding based on mining method. This method of calculating the bond amount offers a great potential for reducing the bonding cost to reflect the mining method. For example,

a mining method to put the texture material on top of the overburden without stockpiling could reduce the bonding costs by an amount equal to the cost of moving that amount of texture material. Similar allowances for other mining methods could be made.

6. Effectiveness of Reclamation Practices

As indicated earlier, the reclamation practices being used at the visited mine site are generally effective in preventing deterioration of water resources and restoring disturbed water resources. One exception would be that an excessive amount of time was lapsing between the time topsoil was stockpiled and the time it was removed from the stockpiles. This practice may allow deterioration in the quality of the topsoil which could affect the quickness with which a good stand of erosion preventing vegetation is established. Also, a large amount of spoil was being left ungraded for a long period of time. The long exposure of this spoil to the weather could increase the breaking down of the spoil and dissolution of minerals by the water passing into the spoil. Therefore, keeping grading and revegetation more current might help improve the quality of the groundwater. If grading and revegetation were being kept more current, some topsoil and texture might be hauled directly from the area where it is removed to the areas being reclaimed, thus cutting down on the total amount of material to be removed twice. This might also allow topsoil and texture material that is being stockpiled to be located closer to the area to be reclaimed, cutting down on the haulage distance.

7. Time for Final Reclamation

A good indication of the amount of time spent per acre of land could not be obtained during the site visit. Information indicated that approximately 11 months should elapse between the completion of mining and the completion of grading and seeding. In practice, significantly more time was elapsing than this. If improvement in mining methods could be employed which would result in less required reclamation effort, a savings might result. In effect, a trade-off might be required between the cost of mining and the cost of reclamation.

8. Potential for Improving the Quality of Reclamation Through Selective Interburden and Overburden Material Replacement

In general, there exists a good potential for using the mining methodology to improve the quality of reclamation through

selective interburden and overburden material replacement. At sites similar to that visited, where there is an abundance of unconsolidated material which may at least partially find its way to the top of the spoil piles, the potential may not be as great as in other areas where the large amount of unconsolidated material is not present. Using selective material replacement could have the following advantages:

- unconsolidated material could be placed on top of the spoil piles, insuring a more workable material for grading;
- in areas where texture material is required, it could be placed directly on top of the spoil, thus eliminating the need for stockpiling the texture material;
- unconsolidated material placed on top would insure a deeper and better root zone than just normal graded overburden in areas where texture material was not required.

APPENDIX D
FIELD SURVEY MINE EVALUATION
MINE NO. 2
(CENTRAL REGION)

APPENDIX D
FIELD SURVEY MINE EVALUATION
MINE NO. 2
(Central Region)

INTRODUCTION

This Appendix consists of two parts. The first part contains a quantitative engineering evaluation of the mine operation. The second part contains a qualitative evaluation of reclamation practices, effects of mining on surrounding water resources and potential for improving the existing interburden handling practices.

Estimation of production rates for the observed stripping systems are made using graphical and analytical procedures. Costs are estimated based on purchase of new stripping equipment.

I. ENGINEERING EVALUATION OF THE MINE OPERATION

1. Production Rate Calculations, 400 H.P. Dozer, Dragline
7.5 Cu. Yd., 10 Cu. Yd. Front End Loader

1.1 Input Variables

- W = 80 feet: pit width
- D₁ = 30 feet: depth of dragline bench (stripped by dozers)
- D₂ = 45 feet: height of overburden dragline bench
- D₃ = 5 feet: interburden thickness
- S₁ = 4 feet: upper coal seam thickness
- S₂ = 5.5 feet: lower coal seam thickness

W_K = 10 feet: width of keyway at the bottom
 L = 100 feet: length of block
 ϕ = 70 degrees: highwall angle
 θ = 37 degrees: natural angle of repose
 α = 50 degrees: angle of oversteepened spoil
 δ = 0.30: spoil swell factor
 R_d = 106 feet: dragline dumping radius
 R_{ef} = 88 feet: dragline effective radius
 H_d = 56 feet: dragline dumping height
 b = 7.5 cubic yards: bucket capacity

1.2 Production Calculation for the Top Overburden Lift, 400 HP Dozer With U-Blade (See Figure D-1.)

1.2.1 Pit Pile Up to the Level of the Dragline Bench, H_1 , ft.

- Swelled top lift area ①, ②, ③: $30 \times 80 \times 1.3 = 3,120, \text{ ft.}^2$
- Swelled interburden thickness: $5 \times 1.3 = 6.5, \text{ ft.}$
- Area of pit pile up to level of dragline bench ① :
 bottom width = $(59.5 - 6.5) (1/\tan 37 - 1/\tan 70) = 51, \text{ ft.}$
 area: $(59.5 - 6.5) \times 51.0/2 = 1,351, \text{ ft.}^2$
- Dozer bench due to top lift; We' , ft.
 area = $3,120 - 1,351.5 = 1,768, \text{ ft.}^2$
 width of bench: $(59.5 - 6.5) We' = 1,768, \text{ ft.}^2$
 $We' = 33. \text{ ft.}$
- Corresponding top lift ① level ④, H_1 , ft.
 $1,351 = 80 \times 1.3 \times (30 - H_1)$
 $H_1 = 17, \text{ ft.}$

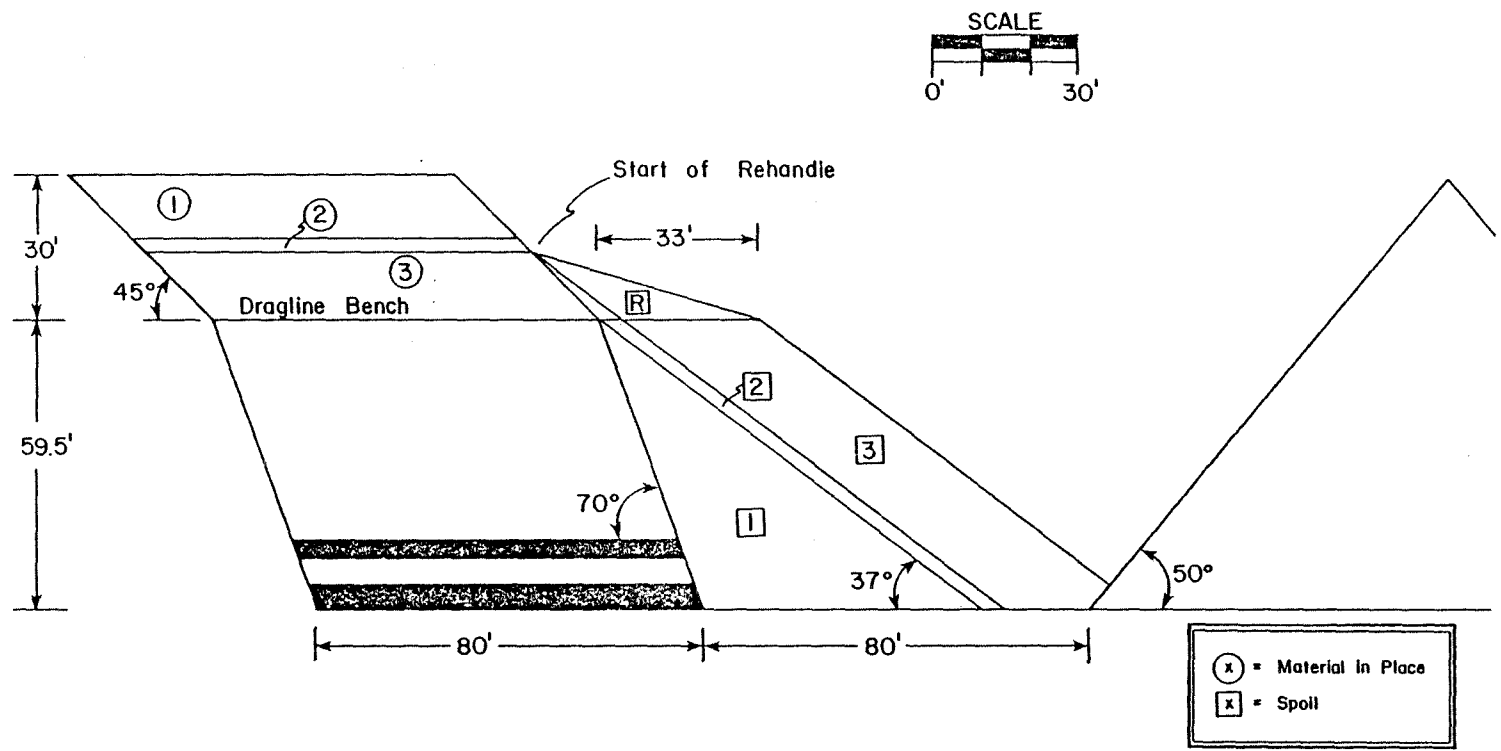


Figure D-1. Cross-Section of Top Lift Removed by Dozers

1.2.2 Dozing Down to the Start of Dozer Rehandle, H_2 , ft.

Equating a section of the top lift (2) with the area of the material (2) dozed down yields:

$$(30-H_2) 80 \times 1.3 = H_2 (1/\tan 37 - 1/\tan 45) \times 53 \\ + H_2^2 (1/\tan 37 - 1/\tan 45)/2 + 1,351$$

$$H_2 = 14. \text{ ft.}$$

1.2.3 Area Rehandle of Dozer, A_r , ft^2

$$A_r = 33.4 \times 14.3/2 = 239, \text{ ft.}^2$$

Table D-1 shows the results of the production calculation for 400 HP dozer with U-Blade working on 100 feet block

1.3 Production Calculations Dragline, 7.5 Cu. Yd.

1.3.1 Height of Final Spoil Pile, H_S , ft. (See Fig. F-2)

$$H_S = (50 + 30) \times 1.3 + 80 \tan 37/4 = 119, \text{ ft.}$$

$$\text{Dragline dumping height needed: } 119 - 59.5 = 59.5, \text{ ft.}$$

1.3.2 Extended Bench Required for Dragline Reach, W_e , ft.

Dragline dump radius: 106 ft.

Dragline width over tracks: 20 ft.

Safety distance: 5 ft.

Dragline effective reach: $R_{ed} = 106 - (20/2 + 5) = 91, \text{ ft.}$

Width of the extended bench: W_e , ft.

$$W_e = \left\{ (D_1 + D_2 + I)(1 + f) + W \tan \theta / 4 - a(\tan \alpha - \tan \theta) / q \right. \\ \left. - \tan \theta (R_{ed} - (D_2 + I + S_1 + S_2) / \tan \phi) \right\} / [2 \tan \alpha / q], \text{ ft.}$$

where

$$a = (D_2 + S_1 + S_2 + I)(1/\tan \theta - 1/\tan \phi)$$

$$q = (1 + \tan \alpha / \tan \theta)$$

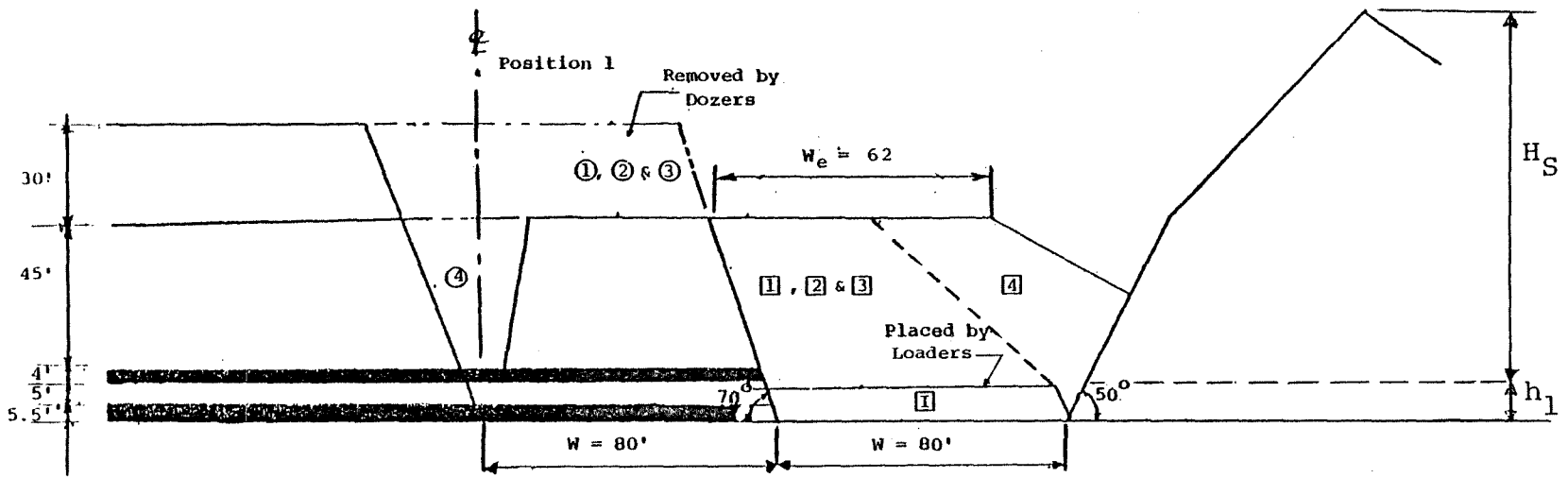


Figure D-2. Section of Pit After Extended Bench Has Been Built and Keyway Dug Out

Table D-1. Production Calculation - 400 HP Dozer
(U-Blade, 100 ft. block)

| Item | Dig Component | | |
|---|---------------|---------|----------|
| | ① & ② | ③ | Total |
| Overburden thickness ft. | 15.7 | 14.3 | 30.0 |
| Bank volume , lcy. | 6,047.4 | 5,508.1 | 11,555.5 |
| Rehandle volume, lcy. | 0.0 | 884.5 | 884.5 |
| Total volume, lcy. | 6,047.4 | 6,392.6 | 12,440.0 |
| Average dozing distance ft. | 80.0 | 96.7 | 88.8 |
| Production rate, lcy. | 780. | 670. | 719 |
| Dozing time required, hrs. | 7.8 | 9.5 | 17.3 |
| Production rate, bcy | 513.81 | | |
| Scheduled hrs. required @ 1.4 per operating hour | 24.2 | | |
| Production per scheduled hour, bcy. | 367.3 | | |
| Monthly production @ 660 sched. | 242,424 | | |
| Annual Production, bcy | 2,909,000 | | |

$$a = 59.5)1/\tan37 - 1/\tan70) = 57.3$$

$$q = 2.582$$

$$We = 80 \times 1.3 + 80(\tan37/4 - 57.3)\tan50 - \tan37)/2.582 \\ - \tan 37(91 - 59.5/\tan700 / (2\tan50/2.582), \text{ ft.}$$

$$We = 62, \text{ ft. (of this, 33.4 ft. are extended by the dozer)}$$

1.3.3 Area Required to Complete the Bench, A_4 , ft.²

(Calculation based on same graphical measurements.)

$$A_4 = (61.9 - 33.4)(42 + (61.9 - 33.4)12/2) = 1368, \text{ ft.}^2$$

Swelled area of minimum keyway: A_k , ft.²

- bottom width, W_k : 10, ft.
- top width: $10 + 45(2/\tan70) = 42.8$, ft.
- $A_k = 13 (42.8 + 10)45/2 = 1,188.0 \times 1.3 = 1,544$, ft.

1.3.4 Area of Dragline Rehandle, A_r , ft.²

$$A_r = 59.5 \times 75/2 = 2,231, \text{ ft.}^2$$

1.3.5 Compute Percent Rehandle, p, percent

$$p = \frac{A_r}{A_T} \times 100 \text{ p,}$$

$$p = \frac{2,231.5}{4,000} \times 100 = 55.8, \text{ percent}$$

Results from the computation of dragline production are summarized in Table D-2.

1.4 Production Rate Calculations for Front End Loader, 10 cu. yd. Bucket

1.4.1 Load Time, T_L , Min.

Time to load one bucket

$$T_L = 0.5 \text{ min.}$$

Add maneuver time, 0.3 min.

Total load time

$$T_L = 0.5 + 0.3 = 0.8 \text{ min.}$$

1.4.2 Travel Time, T_t , min.

Estimated travel time for 200 feet average distance is:

$$T_t = 0.3 \text{ min.}$$

1.4.3 Dump Time, T_d , min.

Estimated dump time is:

$$T_d = 0.2 \text{ min.}$$

1.4.4 Total Cycle Time, T_c , min.

$$T_c = T_L + 2T_t + T_d, \text{ min.}$$

$$T_c = 0.8 + 2 \times 0.3 + 0.2 = 1.6, \text{ min.}$$

1.4.5 Compute Loader Production Per Scheduled Hour, Q_H , bcy

$$Q_H = \frac{60 \times 10 \times \text{Fill Factor} \times \text{Production Factor}}{T_c}, \text{ bcy}$$

$$Q_H = \frac{60 \times 10 \times 0.8 \times 0.7}{1.6} = 210, \text{ bcy}$$

1.4.6 Compute Monthly Production, Q_M , bcy

(100 percent dedication to stripping)

$$Q_M = Q_H \times H_M, \text{ bcy}$$

$$H_M = 660 \text{ hrs: scheduled working hours per month}$$

$$Q_M = 210 \times 660 = 138,600, \text{ bcy}$$

1.4.7 Compute Annual Production, Q_Y , bcy

$$Q_Y = 12 \times Q_M, \text{ bcy}$$

$$Q_Y = 12 \times 138,600 = 1,663,200, \text{ bcy}$$

Table D-2. Production Rate Calculation
7.5 Cu. Yd. Bucket

| Dig Component | Swing Angle, Degrees | Swing Cycles per Hour | Bucket Carrying Capacity, bcy | Volume of Material, bcy | Operating Time, hrs. |
|--|----------------------|-----------------------|-------------------------------|-------------------------|----------------------|
| Keyway to extend bench | 90 | 63 | 5.2 | 5742 | 17.5 |
| Rest of bank to spoil | 90 | 63 | 5.2 | 9073 | 27.7 |
| Rehandle to spoil | 90 | 63 | 6.7 | 8264 | 19.6 |
| Operating time to complete one block, hrs. | | | | | 64.8 |
| Scheduled time for one block, hrs. @ 1.4 per operating hour | | | | | 90.7 |
| Production per scheduled hour, bcy | | | | | 163.3 |
| Monthly production @ 660 sched. hrs/month, bcy | | | | | 108,000 |
| Annual production, bcy | | | | | 1,296,000 |

2. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables D-3 and D-4 show detailed estimated hourly ownership and operating costs for dragline and dozer. The total estimated hourly ownership and operating costs for the dragline with dozer as a utility machine is \$133.73, \$54.51 for the dozer, and \$68.87 for the front-end loader.

3. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by dragline and dozer as a utility machine is \$0.83. Cost per bank cubic yard stripped by dozer is \$0.50 and front-end loader is \$3.84.

Table D-3. Estimated Hourly Ownership and Operating Costs
 Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|----------------------------------|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$300,000 |
| Extras, Freight, & Erection | - |
| Total Price | 300,000 |
| Salvage Value @ 20% | 60,000 |
| Depreciated Amount | 240,000 |
| Life of Machine | 4 years |
| | @ 8030 hrs/yr |
| Depreciation Cost | 7.47 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>4.48</u> |
| Total Ownership Costs | 11.95 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./oper. hour. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | |
| | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/hr. | <u>14.00</u> |
| Total Operating Costs | <u>42.56</u> |
| Total O&O Costs | 54.51 |

Table D-4. Estimated Hourly Ownership and Operating Costs
 Dragline: 7.5 Cu. Yd. Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$ |
| Extras, Freight & Erection | |
| Total Price | 805,000 |
| Salvage Value @ 20% | 161,000 |
| Depreciated Amount | 644,000 |
| Life of Machine | 15 years |
| | @ 8030 hrs/yr |
| Depreciation Cost | 5.34 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>12.02</u> |
| Total Ownership Costs | 17.36 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 25 gal./oper. hr. @ 85¢/gal. | 15.18 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 18.68 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | <u>13.00</u> |
| Total Operating Costs | <u>61.86</u> |
| Total O&O Costs | 79.22 |

Table D-5. Estimated Hourly Ownership and Operating Costs
 Front End Loader: 10 Cu. Yd. Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|----------------------------------|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$450,000 |
| Extras, Freight, & Erection | - |
| Total Price | 450,000 |
| Salvage Value @ 20% | 90,000 |
| Depreciated Amount | 360,000 |
| Life of Machine | 5 years |
| | @ 8030 hrs/yr |
| Depreciation Cost | 8.97 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>6.72</u> |
| Total Ownership Costs | 15.69 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 25 gal./oper. hr. @ 85¢/gal. | 15.18 |
| 4. Maintenance and Supplies | |
| | 21.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/hr. | <u>14.00</u> |
| Total Operating Costs | <u>53.18</u> |
| Total O&O Costs | 68.87 |

II. RECLAMATION AND ENVIRONMENTAL ASSESSMENT (by John Tapp)

1. Introduction

Three periods in the mining cycle of the operation are described in this part. The first is the pre-mining period. Available reports and information on new mining permits on adjacent tracts were used to reconstruct the general characteristics of the site.

The second period of the mining operation described is the during-mining period, including the reclamation that was in progress. Most of the information collected during the visit to the site related to this phase of the mining operation. The third phase described is the post-mining period. This period begins when reclamation has just been completed for the entire site and extends into the future. Since no good data were available for this period, the description is a projection based on the information available for the pre-mining and during-mining periods.

In each of the three periods, the main focus is placed on the soils, geology, groundwater, and surface water.

2. Description of the Area Prior to Mining

2.1 Soils

A conversation with Soil Conservation Service personnel doing soil surveys in the area indicated that the soil units occurring there were Zanesville, Sadler, and Wellston on upland areas and Belknap in river bottom areas. Prior to mining the land use was primarily forestland with some pasture; however, some crops are grown in the area. A description of each soil series was given by SCS (1977) for an adjacent county; these descriptions should be relatively accurate for this site. Observations at the site showed the A horizon to be 8-12 inches thick. The B horizon was approximately 3 feet thick and the C horizon extended for several feet depending on the degree of weathering that had occurred in the underlying sandstone. The results of the chemical and physical analyses of the A, B, and C horizons are shown in Table D-6.

Table D-6. Results of Chemical and Physical Testing
Of The A,B, and C Horizons

| Parameter | A | B | C |
|----------------------------------|-----------|-----------------|-------------|
| pH | 4.8 | 5.3 | 5.9 |
| pH (SMP Buffer) | 6.3 | 6.7 | 7.2 |
| Potential Acidity (tons/acre) | 0.3 | 0.9 | 1.1 |
| Organic Matter (%) | 1.9 | 0.2 | 0.1 |
| Total Nitrogen (lbs./acre) | 1804 | 312 | 159 |
| Available Nitrogen (lbs/acre) | 5.0 | 0.8 | 0.4 |
| Available Phosphorus (lbs./acre) | 25 | 3 | 7 |
| Available Potassium (lbs/acre) | 325 | 174 | 109 |
| Texture (Soil Binder Material) | Silt Loam | Sandy Clay Loam | Silt Loam |
| Plant Available Water (%) | 18.5 | 16.0 | 9.2 (Sandy) |

The results show that the A horizon was acidic, a commonly found characteristic of silt loam soils. The B and C horizons were progressively less acidic. The potential acidities of all three horizons were low as would be expected. The organic matter content was considerably higher in the A horizon as would be expected. Total nitrogen content can usually be correlated with organic matter. This was the case for the above samples with the A horizon having a much higher total nitrogen content. The B horizon had a total nitrogen content approximately twice that of the C horizon. Available nitrogen contents for all 3 soil horizons closely paralleled the total nitrogen contents. The available phosphorus and available potassium contents of the A horizon were considerably higher than the B or C horizons, indicating that the soil may have been fertilized in the past. The texture class analysis showed the A horizon to be a silt loam, the B horizon to be a sandy clay loam and the C horizon to be a silt loam. The B horizon typically contains sand lenses. Sampling one of these sand lenses at the site probably caused the sandy content of the B and C horizons. Plant available water content was slightly higher for the A horizon than the B horizon. The C horizon had a much lower plant available water content due to the lower percentage of clay found in that material.

2.2 Geology

The rock formations that lie below the unconsolidated material at this site are the Lisman Formation and the Carbondale Formation and have been described by Kehn (1968). Both formations are of

Pennsylvanian age. The Lisman Formation contains rock from between the Alluvium along Pond River and the base of the Providence Limestone Member. The Carbondale Formation extends from the base of the Providence Limestone Member to the base of the No. 6 coal bed. The Lisman Formation includes the No. 14 and No. 12 coal beds while the Carbondale Formation includes the No. 11, No. 9, No. 7 and No. 6 coal beds. In the area of the mine site visited the Lisman formation varies from approximately 230 to 290 feet in thickness, while the Carbondale Formation varies from approximately 300-360 feet in thickness. Both the Carbondale and Lisman Formations are composed of primarily shale, sandstone, and limestone.

The upper section of the Lisman Formation contains sandstone, shale, limestone and coal. The sandstone is white to yellowish-brown, fine to coarse grained, friable where weathered, usually thick bedded except where interbedded with shale, and moderately crossbedded. The shale is clay-like, light-gray to red in color, and contains a few small limonite nodules. The base of this section is the Madisonville Limestone Member. This limestone is dark-grey and mottled red, yellow, gray and brown; dense to finely crystalline; and sparsely fossiliferous with brachiopods, crinoids, corals, and pelecypods. It lies in beds $\frac{1}{2}$ to 4 feet thick generally separated by 1 to 5 feet thick clay shale beds. The coal beds in this part of the Lisman Formation are thin and lenticular.

The section of the Lisman Formation which extends to the top of the No. 12 coal bed is comprised mainly of sandstone, siltstone, shale, limestone, coal and underclay. The sandstone is white to yellowish-brown, fine-to medium-grained, locally friable where weathered, moderately crossbedded in the middle part of the unit, and usually thick-bedded except where interbedded with light to dark-gray shale. The siltstone is light to dark gray; weathers yellowish-brown; and is micaceous, especially along bedding surfaces. Part of the shale is clay-like and is light to dark-gray in color. Part of the shale is carbonaceous with a dark-gray to gray-black color. This shale contains scattered limonite and siderite nodules as much as 2 inches in diameter. The limestone is light to medium-gray and is dense to finely crystalline. The No. 14 coal bed is generally thin or absent but is locally thick enough to be mined. The underclay is light to dark gray.

The bottom section of the Lisman Formation contains coal, limestone, shale, and underclay. The No. 12 coal bed is generally thick and usually mined with the No. 11 coal bed. The No. 11 coal bed contains a thin medium-gray to blue-gray pyritic claystone parting about 3 to 4 inches thick approximately 1 foot above the base that is referred to as "blue band" by miners and drillers. The interval between the No. 12 and No. 11 coals is generally about 10-20 feet thick. This interval contains the Providence Limestone Member. This limestone is light to dark-gray, finely crystalline to dense, thin to very thick-bedded, and locally fossiliferous. Where this limestone is locally absent the interval is taken up by carbonaceous shale. The underclay is light to dark gray. The base of the Lisman Formation is the Providence Limestone Member; thus, the No. 11 coal is actual the top of the underlying Carbondale Formation.

Table D-7. Results of Potential Acidity and Neutralization Potential Analyses of Overburden Samples

| MATERIAL | Depth Below Surface (ft.) | Potential Acidity (tons/acre) | Neutralization Potential (tons/acre) |
|------------------------|---------------------------|-------------------------------|--------------------------------------|
| Sandstone | 4-30 | 1.1 | 1.0 |
| Gray Shale | 30-50 | 9.0 | 12.2 |
| Limey Shale | 50-70 | 1.5 | 4.3 |
| Gray Shale | 70-75 | 1.2 | 9.9 |
| Red Sandstone | * | 0.7 | 1.3 |
| Iron-Stained Sandstone | ** | 1.5 | 0.8 |
| Gray Shale | ** | 0.6 | 36.6 |

*Found randomly

**From bench of No. 12 highwall

The section of the Carbondale Formation that is of interest is a sandstone that lies directly below the No. 11 coal. This sandstone is light to medium-gray, weathers brown to yellowish-brown, fine to medium-grained, thin to very thick bedded, and locally crossbedded. It generally is well cemented but locally weathers to a friable sandstone.

In order to characterize the acid or alkalinity forming nature of the strata in the area visited, samples were taken and analyzed for potential acidity and neutralization potential. The majority of these samples were taken from the highwall above the No. 14 coal. Mining had been suspended at the two seam operation (No. 12 and No. 11) due to unfavorable coal market conditions and the pit had filled with water making access to the highwall very difficult. Samples of an iron-stained sandstone and gray shale representative of material in the highwall above the No. 12 coal were taken and analyzed. However, the results indicate that this material had probably weathered to a point where it was not representative of unweathered material. No interburden samples were taken because of the water in the pit. The results of these analyses are shown in Table D-7. These results indicated that of the strata sampled only one gray shale formation had a significant acidity. That formation was the upper gray shale formation and had a potential acidity of 9.0 tons lime equivalent per acre. This formation also had the second highest neutralization potential of 12.2 tons calcium carbonate equivalent per acre. The highest neutralization potential was 36 tons per acre and was exhibited by the shale collected on the bench above the No. 12 coal bed.

Core drill logs were taken by the operator and corresponding set of potential acidity analyses performed. The results were taken in the area of the two seam mining operation and show significant potential acidities in an upper gray shale formation and in the gray shale formations immediately above and below the coal. One coal seam did not show in this highwall and the shale below the coal could be the interburden shale between the coal seams. These analyses do show that materials with significant acid forming potentials were present at the site and probably in both the overburden and interburden.

Additional analyses were conducted on the overburden samples to get an indication of their suitability for use in revegetation. The results of these analyses are shown in Table D-8. All materials were medium to low in organic matter. Total nitrogen content was medium in the shales and low to very low in the sandstones. These results indicated that a mixture of the sandstone and shale with the acid forming material removed through segregation would be a suitable medium for revegetation. However, this mixture of material would probably not be as good as the A horizon soil. The weathering characteristics of the rocks were not investigated, but they would probably be fair to good.

Table D-8. Results of Overburden Testing

| MATERIAL | pH | ORGANIC MATTER % | TOTAL NITROGEN (LBS/ACRE) | AVAILABLE PHOSPHORUS (LBS./ACRE) | AVAILABLE POTASSIUM (LBS./ACRE) |
|--|-----|------------------------|---------------------------------|--|---------------------------------------|
| Brown Sandstone | 5.9 | 0.1 | 159 | 7 | 109 |
| Gray Shale | 6.9 | 1.8 | 1268 | 2 | 562 |
| Light Gray Sandy Shale | 7.2 | 0.3 | 660 | 4 | 380 |
| Gray Shale Over Coal | 7.8 | 1.2 | 972 | 4 | 517 |
| Red Sandstone | 5.8 | 0.1 | 238 | 7 | 76 |
| Brown Sandstone With Iron Streaks-#12 pit | 5.5 | 0.3 | 452 | 7 | 106 |
| Gray Shale-#12 pit | 7.6 | 1.6 | 1064 | 4 | 372 |

2.3 Groundwater

The occurrence of groundwater in this area has been described by Duvaul and Maxwell (1962); Davis, Plebuch, and Whitman (1974); and Wilson and Van Couvering (1965). The most important aquifers in this area are the shallow sand and gravel formations along the Green and Pond Rivers, intermediate sandstone aquifers in the 100-500 deep range and deep sandstone aquifers, which occurs around 500 feet and 1000 feet below the ground surface. The sand and gravel deposits in the alluviums along the Green and Pond Rivers vary from 0-100 feet thick and may yield significant quantities of water.

The intermediate sandstone aquifers may yield quantities sufficient for domestic use but are not considered major aquifers. These aquifers may be within any part of Pennsylvanian rocks that are recharged at their outcrop and contain fresh water for varying distances down the regional dip. In the area of the mine site visited the aquifer tends to occur as the Anvil Rock Sandstone member which, when it is present, lies just above the No. 12 or No. 11 coal beds.

The deep sandstone aquifer around 500 feet below the ground surface is the one from which most groundwater used in the area is withdrawn. The aquifer was deposited as sand in a river channel and as a delta and barrier bars in part of a Pennsylvanian stream distributary system. This aquifer in the area of the mine site is composed of fine to medium grained sandstone. The aquifer's ability to transmit and store water is low. Aquifer tests and a computer simulation showed the transmissivity to range from 800 - 1200 gallons per day per foot with a storage coefficient ranging from 0.00004 to 0.0009. The low storage capacity and transmissivity, along with lithologic changes and faults acting as hydrologic boundaries, have caused excessive drawdowns of the water levels, even though pumpage is low. The aquifer's northern and southern boundaries are faults. Laterally the sandstone changes abruptly to shale which forms the eastern and western boundaries of the aquifer. The aquifer has no outcrop and probably does not have a source of surface recharge. This has caused continually declining water levels in continuously pumped wells penetrating the aquifer.

The deep sandstone aquifer which lies approximately 1000 feet below the ground surface is a channel sandstone aquifer. This aquifer was deposited as sandstone in valleys that had been cut into Mississippian rocks. The thickness of the sandstone is variable; however, that portion containing the water is somewhat more constant in thickness. The aquifer generally consists of medium and fine quartz with occasional pockets of large pebble size quartz. The lithology is similar to that of the other deep aquifer; therefore, by analogy the transmissivity has been estimated as 1,000 gallons per day per foot. The storage coefficient is thought to range from 0.0006 to 0.0009. The water in the aquifer grades

into brackish and saline water to the northeast. The aquifer is isolated hydraulically from the ground surface and is not thought to receive fresh water recharge. There has been essentially no development of this aquifer and development would probably move the fresh-water salt-water interface toward the area of development.

In general, the area of the mine site is not abundant in groundwater. Most municipalities in the area rely on impoundments, although a few do have wells in the upper deep sandstone aquifer. Water districts piping water from the municipalities to the rural areas are common. The quality of the groundwater from all aquifers in the area could only be described as fair to poor. Although strongly influenced by the nearby faults, the general regional dip in the area is westerly at approximately 50 feet per mile, therefore the shallow groundwater should flow generally westward. Since there is no apparent surface recharge for the intermediate and deep aquifers, their flow would only be toward pumped wells.

2.4 Surface Water

Surface water in the area of the mine site would be primarily runoff from forested areas and seepage from reclaimed mine areas. The creeks in the area are relatively flat and sediment washoff from other mining areas has caused excessive siltation of the creeks. In addition, runoff from orphan mine spoil in the area is generally highly acidic and contributes significantly to the degradation of water quality in the area.

During the site visit water samples were taken in this stream above and below the site. The samples were analyzed for several chemical parameters. The results of this testing and testing of water samples from the site itself are shown in the section of this site visit report dealing with surface water during mining.

3. Description of the Area During Mining

3.1 Soils

Prior to the Federal Surface Mining Act of 1977 all that was required at the site was the grading of the spoil material to approximately the original contour and revegetating with grasses, legumes, and trees. The Federal law now requires that the top six inches of soil material stockpiled and replaced on the graded overburden prior to revegetation. Since the land use at the site prior to mining had been forest, no prime farmland restoration was required. The site had been under a topsoil variance so that no topsoil had been stockpiled. The general procedure envisioned by this operator would be to remove the top 6 to 8 inches of soil material approximately one-half to one month or 100 feet in advance of stripping. The soil material would be removed with pans

and dozers and hauled 500-600 feet to stockpiles. No cover crop or soil amendments would be used on the stockpiles and water erosion would tend to be a problem. Chemical sprays are not envisioned for use to stabilize the stockpiles. The topsoil would remain in the stockpiles only one month before it would be replaced. The operator did not have any data on the quality of the topsoil.

Grading of the spoil material was kept current within 3 spoil ridges so that the average spoil age at the time of grading was 6 months. This grading was accomplished primarily with dozers. The grading was to return the land to the approximate original contour. After grading, the spoil material was seeded with quick cover grasses for erosion control and legumes. Eventually the site would be planted with pine and locust trees. Seeding was generally done by drilling. Generally Ky 31 fescue was the primary grass used. According to the State surface mining regulations, seeding must be done in the spring or fall.

3.2 Geology

At the mine site the surface soil material was approximately 4 feet thick. Beneath the soil was a layer of iron stained and weathered sandstone approximately 30-40 feet thick. Beneath the sandstone and lying just above the No. 12 coal was a shale layer 20-30 feet thick. Water that appeared to be acidic was seeping from the base of the sandstone over the No. 12 coal into the pit. The pit was full of water as the site no longer was being mined. Reportedly the shale interburden also tended to be acid forming.

The overburden that was removed ended on top of the spoil material, and was extremely high in acid forming material as acid water was ponded in spots in the overburden. The sediment ponds below the unreclaimed spoil had low pH's, another indication of the presence of the acid material.

Because the pit was filled with water, the interburden highwall was not observed during the site visit. The overburden highwall showed large iron stained streaks in the sandstone. No other stainings or precipitates were observed, except for some nodules in the sandstone. The spoil materials were apparently relatively stable as no stability problems were observed or reported by the operator. The operator reportedly attempts to identify the acid forming material with the potential acidity analyses and segregates the acid forming material with the operation of the dragline. Obviously very little success had been achieved at this site. The spoil was apparently being left exposed for a lengthy period of time and this was allowing the oxidation of the pyritic material and consequently causing the acid drainage.

3.3 Groundwater

During the site visit, water was observed seeping from the overburden into the mine pit. The pit probably had 20-30 feet of water in it and had apparently reached equilibrium such that runoff and seepage into the pit equalled seepage from the pit. Seepage from the pit would be through the spoil nest to the pit and into the lower strata, or surface as recharge to the sediment ponds. A considerable amount of precipitation that fell on the spoil was seeping to the sediment ponds.

Groundwater in the area is not generally abundant and during mining the pumping from the pit probably does little to lower the regional groundwater table. During pumping the flow of groundwater would be toward the pit instead of the general regional direction. The porosity of the spoil material would probably be greater than that of the parent material and the spoil would tend to store more water.

The quality of the groundwater would be expected to decrease as the acid formed on the surface is leached downward. As this acid water comes into contact with calcareous materials, it tends to be neutralized. The neutralization process results in cations being dissolved in the water with a significant increase in total dissolved solids concentration of the water. This is the process that seems to be occurring in the pit filled with water as no acid problems were observed, but the dissolved solids, sulfate, and calcium contents of the water were high. During active mining when the water running into the pit is pumped out frequently, the neutralization might not occur and the pumpage could be acid. The acid water in the sediment ponds also probably recharges the groundwater resulting in relatively poor water quality.

3.4 Surface Water

Surface runoff and pumpage from the pit were passed through two sediment ponds arranged in series. These sediment ponds also collected runoff from areas being reclaimed. Generally, impoundment type basins were used. Short circuiting in the pond appeared to be a problem. The spillway of the upper pond had washed out and the water was flowing through a ditch into the lower pond. A good portion of the storage capacity in the upper pond could not be used because of the eroded spillway. The outlet from the lower spillway was a dirt channel through the embankment. Both ponds were approximately one acre in surface area but probably very shallow due to the sediment that had been trapped in the pond. Iron staining was evident in the ditch between the two ponds. Sediment ponds were the primary methods of runoff and sediment control at the site. Above highwall diversion ditches, levees and stream channel diversions were not being used.

To obtain an indication of the quality of surface water at the site, 4 water samples were collected. Two samples were taken from the major stream draining the area. One of these was taken above the site and the other below the site. Due to access problems, these two sampling points were not immediately above and below the site and the results include effects other than just the visited mine site. Mining was also occurring above the upper sampling site. The third sample was taken from water standing in the mine pit and the fourth was taken in the outflow from the lower sediment pond. The results of these samples analyses are shown in Table D-9.

The pH, iron and manganese exceeded the EPA effluent limitations. The iron concentrations in the stream above and below the site were also high. The aluminum concentrations in the stream above and below the mine site were high, but the aluminum concentrations from the mine site were low. The calcium concentration was very high in the mine pit water due to the dissolution of the limestone interburden through the neutralization of the acid material that was being formed. The calcium concentration was also relatively high in the sediment pond discharge. There was a small net calcium concentration increase from above to below the site. The magnesium concentrations were similar to the calcium except that the stream magnesium concentration tripled from above to below the site. The magnesium was probably being dissolved through neutralization of acid water similar to the calcium. Since calcium and magnesium are the major components of hardness in water, hardness could be expected to parallel the calcium and magnesium results. The data showed that parallel to exist. The sodium concentration was low in the mine pit when compared to calcium and magnesium. This caused a low sodium absorption ratio.

The bicarbonate concentration decreased by about one half from above to below the site as a result of acid material discharged or seeping into the stream between the two sampling points. The sulfate concentration was very high in the water in the mine pit and high in the sediment pond discharge. The sulfate concentration in the stream tripled between the two sampling points. The high sulfate concentration in the mine pit confirms that acid forming material (pyrite) is present and is being oxidized to sulfate and then neutralized by the calcareous material. The high sulfate in the sediment pond discharge was indicative of the sulfuric acid causing the depressed pH in the discharge. The specific conductance and total dissolved solids concentrations were high in the mine pit and in the sediment pond discharge and approximately doubled between the upper and lower stream sampling points. This is reflective of the salts being dissolved in the water as explained earlier for calcium, magnesium, and sulfate.

Table D-9. Results of the Analysis of Water Samples

| PARAMETER & UNITS | LOCATION | | | |
|---|-------------------------|-------------|------------------------------|-------------------------|
| | STREAM ABOVE SITE | MINE PIT | SEDIMENT POND EFFLUENT | STREAM BELOW SITE |
| Total Aluminum (mg/l as Al) | 3.6 | 0.3 | 0.3 | 2.1 |
| Total Boron (mg/l as B) | 0.00 | 0.06 | 0.00 | 0.01 |
| Total Calcium (mg/l as Ca) | 23.9 | 233 | 80 | 25.8 |
| Total Iron (mg/l as Fe) | 2.4 | 0.5 | 6.2 | 1.8 |
| Total Potassium (mg/l as K) | 2.1 | 5.0 | 2.3 | 2.3 |
| Total Magnesium (mg/l as Mg) | 3.2 | 60 | 39 | 10.1 |
| Total Manganese (mg/l as Mn) | 0.1 | 2.5 | 8.3 | 0.7 |
| Total Sodium (mg/l as Na) | 0.1 | 7 | 0.1 | 0.1 |
| Total Lead (mg/l as Pb) | <0.03 | 0.03 | 0.03 | <0.03 |
| Total Selenium (mg/l as Se) | <0.3 | <0.3 | <0.3 | <0.3 |
| Total Zinc (mg/l as Zn) | 0.01 | 0.02 | 0.12 | 0.05 |
| Hydroxide (mg/l as OH) | 0.0017 | 0.0021 | 0.00 | 0.005 |
| Bicarbonate (mg/l as HCO ₃) | 65 | 94 | 0.00 | 34 |
| Carbonate (mg/l as CO ₃) | 0.03 | 0.05 | 0.00 | 0.00 |
| Chloride (mg/l as Cl) | 6.3 | 4.5 | 3.5 | 6.8 |
| Sulfate (mg/l as SO ₄) | 25 | 762 | 4.2 | 75 |
| Nitrate (mg/l as N) | 0.14 | 1.11 | 0.02 | 0.11 |
| Total Phosphorus (mg/l as P) | 0.4 | 0.9 | 0.2 | 0.3 |
| pH (standard units) | 7.0 | 7.1 | 4.3 | 6.5 |
| Alkalinity (mg/l as CaCO ₃) | 53 | 77 | 0 | 28 |
| Acidity (mg/l as CaCO ₃) | 7 | 12 | 12 | 8 |
| Total Hardness (mg/l as CaCO ₃) | 73 | 830 | 360 | 106 |
| Total Suspended Solids (mg/l) | 31 | 7 | 10 | 9 |
| Total Dissolved Solids (mg/l) | 126 | 1251 | 683 | 209 |
| Specific Conductance (mhos/cm) | 173 | 1450 | 815 | 254 |
| Sodium Adsorption Ratio (meq/meq) | 0.005 | 0.11 | 0.002 | 0.004 |

4. Description of the Area After Mining

4.1 Vegetative Growth Potential

The practice of using the mixed sandstone and shale overburden as the revegetation medium will result in a shallow root zone with a corresponding loss in productivity of the land. Although erosion preventing grasses and trees may do well, many years will pass before the reclaimed land could support the same vegetation it could have supported prior to mining. Also future problems could result with the acid material that was present in the overburden. Stockpiling and replacing six inches of topsoil on the land could help. However, a rather shallow root zone would still exist. If the area were considered prime farmland and the pre-mining profile were reconstructed, a deeper root zone would result, with correspondingly better post mining land productivity.

The high salt contents found in the pit water are indicative of the material that may be dissolveable in water at the site. If this water reaches the root zone by capillary action, an impairment of growth of sensitive species could result. A deeper root zone would tend to alleviate this problem. Therefore, altering the mining method to insure that acid forming material is buried and that the more unconsolidated material ends up on the surface would be desirable at this site.

4.2 Groundwater Patterns and Quality

During active mining the groundwater table in the local area will probably be lowered because of pumpage from the pit. Indications are that this pumpage may be acidic if the water that contains acid materials does not have a chance to come in contact with the natural limestone. When it does come in contact with the limestone, it may be neutralized but the result will be a dissolution of minerals in the water. As mining and reclamation is completed, the groundwater table should approach its natural pre-mining level and the groundwater flow pattern should return approximately to its premining state.

The quality of the groundwater may be degraded for a period of time due to the dissolving of minerals from the pulverized rock. If the premining land use called for the final pit to remain the water quality might be suitable for wildlife enhancement but it probably would not be suitable for agricultural irrigation, at least until the water with the high mineral content is flushed out.

4.3 Surface Water Quality and Quantity

When the land is reclaimed to the approximate original contour and revegetation is established similar to that occurring before mining, surface runoff patterns should approximate those existing before mining. The rate of infiltration will probably be higher after mining because of the large mass of disturbed material. However, the disturbed area will tend to hold the water longer. The net result could be reduced runoff peaks and increased base flow during low flow periods. The quality of runoff after mining should be better than the acid conditions that were found in the sediment pond discharge. The runoff quality should also be better than the pit pumpage that occurs during active mining. Runoff quality may be degraded for a period of time if dissolution of the salts in the overburden and interburden material continue. The local streams may show a resulting increase in the total dissolved solids content due to groundwater recharge.

5. Effects of Mining and Reclamation On Bonding Costs

A total of eight items are included in the bond computation format and points are assigned to each. They are (1) special considerations, such as locations of streams, cemeteries, residences, etc; (2) geologic history, which is mainly a reflection of a potential acidic or stability problem; (3) the average slope of the site; (4) the type of operation, i.e., area, contour, auger, multiple seam; (5) the size of operation, broken down by categories; (6) the number of silt structures proposed for the permit; (7) the percent of the permit area in prime farmland; and (8) the necessity for borrow areas for backfilling or fill areas for spoil disposal. The points assigned to each of the above categories and corresponding subcategories are totaled. The points multiplied by 30 gives the total bond amount in dollars per acre.

6. Effectiveness of Reclamation Practices

Little reclamation had been accomplished on the last spoil ridge which was probably causing the acid water discharge. On other spoils grading and revegetation had been accomplished. The revegetation was mostly successful with some bare spots remaining, probably from durable overburden or acid forming material. Grading and successfully revegetating the last spoil outslope while the pit is inactive could probably greatly improve the acid condition in the sediment pond.

The practice of stockpiling and replacing topsoil could also improve the success of the overall revegetation effort. This would eliminate the bare spots caused by durable overburden and acid forming material. It would also provide a suitable medium for plant growth. However, replacing the topsoil would probably require mulching due to the higher erodability of the soil material.

Another practice which should be more fully utilized, until other more permanent reclamation practices solve the problem, is the treatment of the acid water. Technology exists to easily solve the acid problem, although the cost of chemicals can make this an expensive reclamation practice. The water treatment should be used only as a temporary measure.

7. Time for Final Reclamation

The operator indicated that approximately six months lapses between mining and grading prior to revegetation. The time to establish good vegetation on the area could be reduced by better segregation of the acid material, by the stockpiling of topsoil, or by changing the mining method to insure that unconsolidated material ends up on the surface.

8. Potential For Improving the Quality of Reclamation Through Selective Interburden and Overburden Material Replacement

There exists an excellent potential at this site for improving the quality of reclamation through selective overburden and interburden handling and replacement. The site contained about four feet of soil material and several additional feet of highly weathered sandstone material. Selectively removing and replacing this material so that it ends up on top of the consolidated overburden could provide a good medium for revegetation. Doing this could also help alleviate the acid mine drainage problem at the site.

Selective overburden and interburden handling could also be used to segregate and bury the overburden and interburden strata that are found to be potentially acid producing. If this material can be buried during active mining so that it does not have time to oxidize, acid mine drainage can be greatly minimized.

APPENDIX E
FIELD SURVEY MINE EVALUATION
MINE NO. 3
(WESTERN REGION)

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FIELD SURVEY MINE EVALUATION
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INTRODUCTION

This Appendix consists of two parts.. The first part contains a quantitative engineering evaluation of the mine operation. The second part contains a qualitative evaluation of reclamation practices, effects of mining on surrounding water resources and potential for improving the existing interburden practices.

Estimation of production rates for the observed stripping systems are made using graphical and analytical procedures. Costs are estimated based on purchase of new stripping equipment.

I. ENGINEERING EVALUATION OF THE MINE OPERATION

1. Production Rate Calculations - Dragline
75 Cubic Yards, 325 Foot Boom

Two Seams Horseshoe Method

1.1 Input Variables

W = 180 feet: pit width
D = 128 feet: total overburden and interburden depth
D₁ = 35 feet: depth of overburden of upper stray seam
D₂ = 30 feet: depth of upper main seam interburden
D₃ = 4 feet: interburden thickness of lower stray seam
D₄ = 60 feet: interburden thickness of lower main seam
S₁ = 360 inches: thickness of upper main seam
S₂ = 240 inches: thickness of lower main seam
φ = 60 degrees: highwall angle
θ = 38 degrees: natural angle of repose
α = 50 degrees: angle of oversteepened spoil
δ = 30 percent: average spoil factor
L = 150 feet: length of block
R_d = 282 feet: dragline dump radius
R_{ef} = 232 feet: dragline effective radius
H_d = 139 feet: dragline dump height

$b = 75$ cu. yd.: dragline bucket capacity
 $q = 0.83$: dragline utilization
 $p = 0.85$: dragline availability
 $w_b = 10$ feet: width of bucket

1.2 Output Variables

H_S = height of final spoil pile, feet
 H_{OB} = height of spoil pile from upper stray seam and upper main seam
 V_i = dig volumes of each dragline pass, bcy
 S_i = swing cycles per hour for each pass
 C_i = bucket carrying capacity for each pass, bcy
 T_i = time required to dig material of each pass, hours
 T = time to complete one block, hours
 T_S = scheduled time to dig one block, hours
 P_H = dragline production rate perscheduled hour, bcy
 P_M = dragline monthly production, bcy
 P_Y = dragline annual production, bcy

1.3 Height of Final Spoil Pile, H_S , Feet

Swelled overburden cross section: $D \times W \times (1+\delta) = 128 \times 180 \times 1.3$
 $= 29,952, \text{ ft.}^2$

$H_S = D \times 1.3 + W \tan \theta/4, \text{ ft.}$

$H_S = 128 \times 1.3 + 180 \tan 38/4 = 201.6, \text{ ft.}$

1.4 Spoil Pile Height from Upper Stray Seam and Upper Main Seam Overburden, H_{OB} , Feet

(See Figure E-1.)

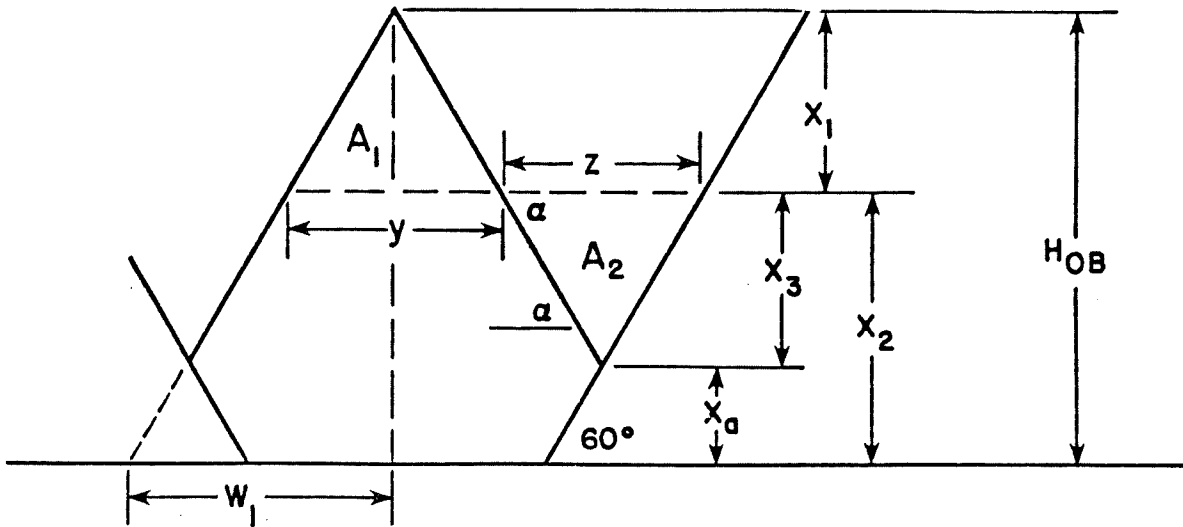


Figure E-1. Spoil Pile from Seam 1 and Seam 2 Overburden

Let A_{12} denote the swelled band above the upper main seam:

$$W_1^2 (\tan 38^\circ - a) + W_1 \times 180a - (180^2 \times a/4 + A) = 0$$

$$a = 1/(1/\tan 60 + 1/\tan 38^\circ) = 0.538$$

$$A_{12} = (35 + 30) \times 180 \times 1.3 = 15,210, \text{ ft.}^2$$

$$W_1 = 147.5, \text{ ft.}$$

$$H_{OB} = 147.5 \tan 38^\circ = 115, \text{ ft.}$$

1.5 Preparation of the Spoil Bench

(See Figure E-2.) It is assumed that the height of the bench will be the same as that of the spoil pile after passes 1 and 2.

Check for dump height:

Required: $201.6 - 115.2 = 86.4 \text{ ft.}$

Available: 148 ft.

Width of the spoil bench: $W_s, \text{ ft.}$

Width of dragline over both shoes: 83 ft.

Safety: 20 ft. in each side

Required bench width = $83 + 2 \times 20 = 123 \text{ ft.}$

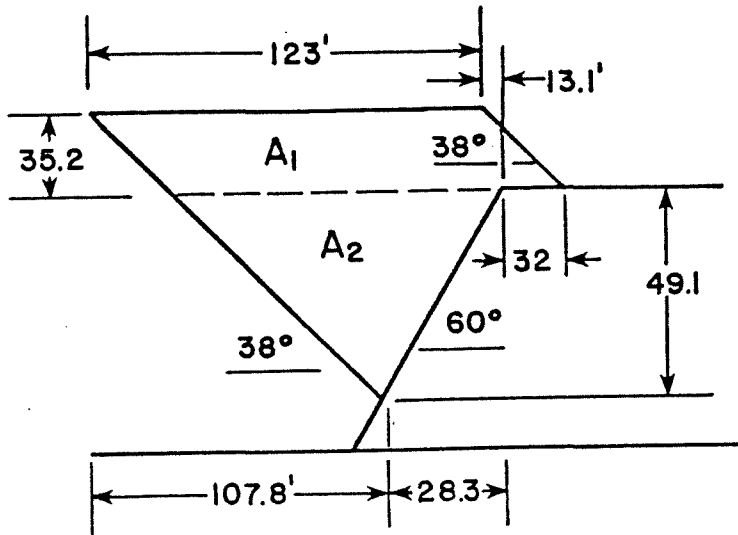


Figure E-2. Spoil Bench Geometry

Area required to complete the bench:

$$A_1 = 35.2 \times 123 = 4,329.6$$

$$A_2 = (123 - 32) \times 49.1/2 = 2,234.0$$

$$\text{Total cross section } A_1 + A_2 = 6,563.6, \text{ ft.}^2$$

Amount of seam 4 interburden required to complete the spoil bench:

$$6,563.6/1.3 = 5,048.9, \text{ ft.}^2$$

1.6 Area of Rehandle, A_r , Feet²

(See Figure E-3.)

$$A_1 = 35.2(100 + 124)/2 = 3,942.4$$

$$A_2 = 92 \times 80/2 = 3,680.9$$

$$\text{Total cross section } A_r = 7,622.4, \text{ ft.}^2$$

Rehandle, as percent of bank yards removed: 39.6%

Results from the computation of dragline production are summarized in Table E-1.

Table E-1. Production Rate Calculation: Marion 8200,
75 Cubic Yard Bucket (150 ft. Block)

| Dig Component | Swing Angle | Swings Per Hour | Bank Cubic Yards Per Bucket | Component bcy. | Operating Hours |
|--|-------------------|-----------------|-----------------------------|----------------|-----------------|
| Seam 1 Overburden to Spoil, from Pass 1 Position | 90° | 56 | 52 | 35,000 | 12.2 |
| Seam 2 Overburden to Spoil, from Pass 2 Position | 90° | 56 | 47 | 30,000 | 11.7 |
| Seam 3 Interburden to Spoil | Taken by Scrapers | | | 3,000 | |
| Part of Seam 4 Interburden to Spoil Bench, from Spoil Bench Position | 90° | 51 | 47 | 28,049 | 11.7 |
| Remaining Seam 4 Interburden to Spoil, from Spoil Bench | 135° | 48 | 52 | 31,951 | 12.8 |
| Rehandle to Spoil, from Spoil Bench Position | 110° | 53 | 73.5 | 49,456 | 12.8 |
| Operating Time for One Block, hrs. | 61.2 | | | | |
| Scheduled Time for One Block, hrs. | 85.7 | | | | |
| Production Per Scheduled Hour, bcy. | 1,458 | | | | |
| Monthly Production Rate, bcy. | 962,700 | | | | |
| Annual Production Rate, bcy. | 11,552,000 | | | | |

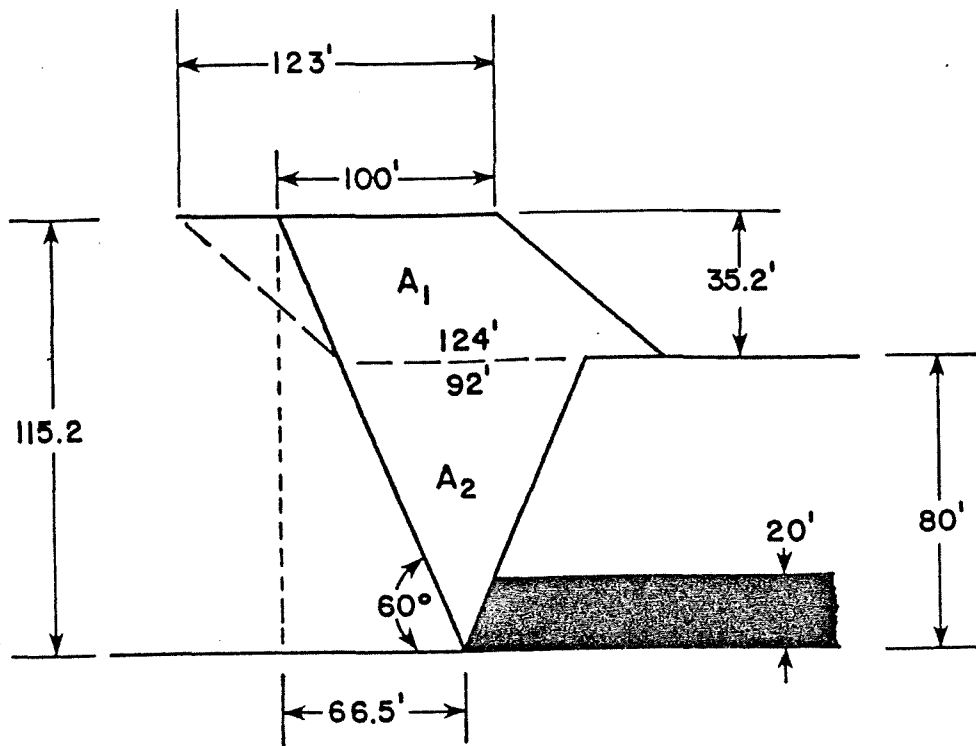


Figure E-3. Area of Rehandle Geometry

2. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables E-2 and E-3 show detailed estimated hourly ownership and operating costs for dragline and dozer. The total estimated hourly ownership and operating costs for the dragline and dozer are \$675.26.

3. Estimate Stripping Cost Per Bank Cubic Yard

The estimated stripping cost per bank cubic yard removed by dragline with a 400 HP dozer used as a utility machine is \$0.47.

Table E-2. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | <u>313.82</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | <u>306.93</u> |
| <u>Total Ownership and Operating Costs</u> | <u>620.75</u> |

Table E-3. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$300,000 |
| Extras, Freight & Erection | -- |
| Total Price | 300,000 |
| Salvage Value @ 20% | 60,000 |
| Depreciated Amount | 240,000 |
| Life of Machine | 4 years @ 8,030 hrs/yr |
| Depreciation Cost | 7.47 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.48 |
| Total Ownership Costs | <u>11.95</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal/hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/hr. | 14.00 |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | <u>54.51</u> |

II. RECLAMATION AND ENVIRONMENTAL ASSESSMENT (by R.Hodder and E. P. Briggs)

1. General Mine Information

The mine is located in high rolling hill country. The area is primarily in rangeland forage with patches of timber ponderosa pine on ridge tops and north slopes. Ranching and the grazing of cattle is the primary use of land in this mining district. Logging of the timber in the area is an industry of minor consequence. Grain and hay production is an important part of some ranching operations, and cultivated fields are a conspicuous component of the countryside.

2. Reclamation Related Practices and Equipment

2.1. Topsoil

Topsoil is variable in both quality and depth, it being best in quantity and quality in swales and in cultivated field areas. In the hilly portions of the mine area itself, topsoil depths vary from 2 inches to 6 inches, a reasonable average being about 3 inches. Unconsolidated soil substitute materials vary from 0 feet to 5 feet, averaging about 34 inches deep. These materials are essential as buffer material to be placed between topsoil and spoil below.

Productivity of the soil within the mine site area must be considered low. Forage production of 1,400 to 3,000 lbs/acre is considered average here.

Because of the terrain, pit configuration, mining plan and season, it is usually necessary to remove topsoil 3 to 6 months in advance of mining. Topsoil is removed and stockpiled in two distinct lifts for separate reapplication on recontoured spoils, if it cannot be directly applied. These operations are accomplished with scrapers, hauling materials a distance averaging 1,500 feet. Topsoil is usually applied directly on recontoured spoils but is sometimes stockpiled.

2.2. Topsoil Stockpiles

Wind erosion is not considered an important problem in the area. However, wind erosion is relatively constant, insidious, and does more damage than is usually attributed to it.

Water erosion concerning stockpiled topsoil is a problem. Because of the nature of violent summer storms, this type of erosion is apparent. Both accumulation of sediments and the formation of rills and gullies testify to the seriousness of this situation.

Cover crops are used at this mine as the most common means of protecting stockpiled topsoil from erosion and from weed infestation. Chemical sprays are not used to stabilize stockpiles.

The average age of stockpiled topsoil at the time of use is 24 months. This term is influenced by the area from which topsoil is removed and by the pit width and by the nature and amount of rehandle necessary to remove the lower seams of coal.

2.3. Segregation and Burial of Materials

Recognized undesirable spoil materials are segregated and buried. These are identified by core drilling and analyses or overburden by soil survey and analysis of topsoil, by texture and by percolation tests.

Perhaps the most toxic and the largest quantity of materials that require segregation and burial concern the sodic interburden. This sodic shaley medium when left close to the surface reduces infiltration and percolation of precipitation and thus directly affects the revegetation effort.

No acid materials appear in amounts that are significant. Large rocks are not frequent, but when encountered, they are buried. Spoil stability is not a problem because the clayey materials are buried and slopes are reduced to acceptable and stable gradients. Segregation and burial are accomplished with the dragline and with scrapers.

2.4. Surface Water

The Yellowstone River is considered the nearest perennial waterway to the mine site, Sarpy Creek is sometimes perennial or nearly so through a good portion of the year. Sarpy Creek and the East Fork of Sarpy Creek are streams located on or near the mine site. Their major flow is during spring, from March to July, caused by snowmelt and spring rain runoff. Turbidity of these streams prior to mining ranged between 130 to 172.

2.5. Surface Drainage Control During Mining

Diversion ditches and levees are not used above the high-wall because they are not needed in this terrain and characteristic runoff pattern. Stream diversion ditches are not needed nor used. Pumps are used to drain excess water from the pit when seepage is excessive or when accumulated runoff is troublesome.

2.6. Surface Water After Mining

The drainage system within the mine site is damaged by mining, but in this high country, this is relatively inconsequential. In the recontouring process, the approximate original drainage system is restored. Settling ponds are recharged by runoff and by groundwater pumped from the pit.

2.7. Ground Water Before Mining

Mining is generally below the premining watertable which in part of the mine area is unusually high. Water does seep from the highwall but usually the flow is relatively minor. On hot, windy days this flow may be evaporated and seepage may be practically undetectable. This seepage does not affect mining significantly.

Depth to groundwater at nearby wells varies from 10 to 60 feet.

2.8. Ground Water After Mining

During mining, the aquifer is removed. After spoils replacement and recontouring, it is expected that accumulation of ground water may appear as a saturation zone above the impermeable clay or shale layer. It is not known at this time how long a period of time this may take if it ever occurs, but monitoring programs have been initiated and will continue indefinitely in order to gain information on ground-water quality and quantity. There is no evidence to date that mining has polluted the ground water.

2.9. Ground Water Supply During Mining

Local wells have been deepened by the mining company even though it has not been assured that mining activity has been the cause of change in ground water supplies. One instance involved the deepening of a well approximately 5 miles away.

Water has not been trucked to local farmers and ranchers and this practice is not expected to be initiated.

2.10. Erosion and Sediment Control

Sediment ponds are used to collect and settle all water leaving the mine area. Prompt grading and seeding of outslope is standard practice in order to minimize erosion. Terracing of outslopes is not used as a soil erosion prevention treatment and is not recommended.

2.11. Incline Reclamation

Inclines and ramps extend through spoils to the pit bottom. These inclines are at specified gradients and migrate with the pit. In this way, inclines are filled in as mining proceeds, thus eliminating depressions and impoundments. However, they usually lend to the design of the drainage system. They will be reclaimed after they have fulfilled their purpose.

2.12. Rough Grading (Backfilling)

Grading and recontouring of spoils is kept current within two spoil ridges from the active pit. Because of multiseam mining, rehandle requirements and length of pit, the average age of spoil pile at time of grading is approximately 6 months. Acceptable gradients of recontoured spoils are: minimum 0%, average 10%, maximum acceptable 20%. Equipment assigned to the recontouring job consist of five bulldozers and eight graders.

2.13. Spoil Amendments

The reclamation process at this mine does not include backblading or dragging. It does, however, include ripping with a subsoiler to relieve compaction and increase percolation and root penetration. It also includes discing to help prepare a seedbed. Fertilizer is used on the topsoil, and straw mulch is applied after seeding. Lime is not used because the soils are alkaline.

2.14. Revegetation

Two types of seeding are typically used at this mine site. They are broadcast seeding and drill seeding. Aerial seeding is not done. Although there is definite state pressure to promote hydromulching, hydroseeding is not used even though the same equipment would be used to accomplish either job.

Trees and bushes are conspicuous on portions of the replanted spoils and a good variety of species are represented. Ponderosa pine is the most obvious because of its size. Test growing plots are included in the revegetation planning process and they include various soil depth studies as well as species trials.

2.15. Final Cut

State law does not allow the final cut to contain a water impoundment. The highwall must be eliminated by filling in with spoils to a gradient of 3:1 or less.

2.16. Topography to Which Land is Reclaimed

The area disturbed by mining averages 50 acres annually. The topography of the disturbed land at the mine site after recontouring is approximately similar to the original contour, thus the surface drainage system is similar also.

The mine is located in hilly country, so concerted effort has been made to meld and blend the recontoured spoils with the unmined area so that intercepted ridges and other features of the landscape are carried into and across the disturbed area.

Steepness of the undisturbed country is much more exaggerated than the recontoured spoils, for state regulations stipulate that no recontoured gradients within the mine site can exceed 5:1. Highwalls, benches and terraces are not acceptable, nor are undrained depressions or pond area.

Although present regulations allow a gradient of 3:1 to be used in eliminating a highwall by fill, it has been found impractical to try to stabilize a slope of this gradient. At this mine, maximum gradients are limited to 5:1 or less.

2.17. Land Use

The land surface, prior to mining, was used primarily for grazing of livestock, for growing of crops and in a minor way for timber production. Use after mining will be limited primarily to grazing by livestock and as unused woodland.

APPENDIX F

FIELD SURVEY MINE EVALUATION

MINE NO. 4

(WESTERN REGION)

APPENDIX F
FIELD SURVEY MINE EVALUATION
MINE NO. 4
(Western Region)

INTRODUCTION

This Appendix consists of two parts. The first part contains a quantitative engineering evaluation of interburden handling practices. The second part contains a qualitative evaluation of reclamation practices, effects of mining on surrounding water resources and potential for improving the existing interburden practices.

Estimation of production rates for the observed stripping systems are made using graphical and analytical procedures. Costs are estimated based on the purchase of new stripping equipment.

I. ENGINEERING EVALUATION OF THE MINE OPERATION

1. Production Rate Estimate, Dragline, 75 Cubic Yards,
310 Foot Boom

1.1. Input Variables

- W = 150 feet: pit width
- D_1 = 60 feet: overburden depth
- D_2 = 20 feet: interburden thickness of second coal seam
- $D_{3,4,5}$ = 15 feet: interburden thickness of 3rd, 4th, and 5th coal seams
- S_i = 5 feet: thickness of coal seams 1 through 5
- L = 150 feet: length of block
- ϕ = 75 degrees: highwall angle
- θ = 37 degrees: natural angle of repose

α = 55 degrees: angle of oversteepened spoil
 δ = 0.30: spoil swell factor
 R_d = 282 feet: dragline dump radius
 R_{ef} = 242 feet: dragline effective radius
 H_d = 139 feet: dragline dump height
 b = 75 cubic yards: bucket capacity
 W_b = 11 feet: width of bucket
 p = 0.85: dragline availability
 q = 0.83: dragline utilization

1.2. Output Variables

H_S = Height of spoil bench above the pit floor, feet
 h_1 = Height of material for filling the spoil bench, feet
 H'_S = Spoil pile height from the overburden pass, feet.
 h_3 = Height of spoil pile toe from the overburden pass, feet
 R_{nr} = Dragline radius required for no rehandle, feet
 R_I = Dragline reach required for the interburden pass, feet
 H_{SF} = Height of spoil pile from the interburden pass, feet
 V_i = Dig volumes of each dragline pass, bcy.
 V_{ROB} = Volume of rehandle material, overburden pass 3, bcy.
 V_{RI} = Volume of rehandle material, spoil side pass 4 and 5, bcy.
 S_i = Swing cycles per hour for each pass of one block
 C_i = Bucket carrying capacity for each component, bcy.

- T_i = Time required to dig material of each pass of one block, hours
 T = Time to complete one block, hours
 T_S = Scheduled time to dig one block, hours
 P_H = Dragline production rate per scheduled hour, bcy.
 P_M = Dragline monthly production, bcy.
 P_Y = Dragline annual production, bcy.

1.3. Step-by-Step Procedure for Computing Dragline Production Rate

1.3.1. Compute Spoil Bench Height After Leveling, H_S , Feet

(See Figure F-1.)

$$H_S = \left(-b \pm \sqrt{b^2 - 4ac} \right) / 2a, \text{ ft.}$$

where

$$a = (1/2 \tan \theta - 1/2 \tan \alpha) - (1/\tan \theta - 1/\tan \alpha)^2 \times \tan \phi \tan \theta / 2 (\tan \theta + \tan \phi)$$

$$b = W + 2W_b - 3W_b (1/\tan \theta - 1/\tan \alpha) \tan \phi \tan \theta \div 2(\tan \theta + \tan \phi)$$

$$c = -W(D_1 + D_2 + D_3)(1 + \delta) - 9W_b^2 \tan \phi \tan \theta \div 8(\tan \theta + \tan \phi)$$

$$a = (1/0.7536 \times 2 - 1/2 \times 1.428) - (1/0.7536 - 1/1.428)^2 \times 3.732 \times 0.7536 / 2(0.7136 + 3.732) = 0.192$$

$$b = 150 + 2 \times 11 - 3 \times 11(1/0.7536 - 1/1.428) \times 3.732 \times 0.7536 / 2(0.7536 + 3.732) = 165.6$$

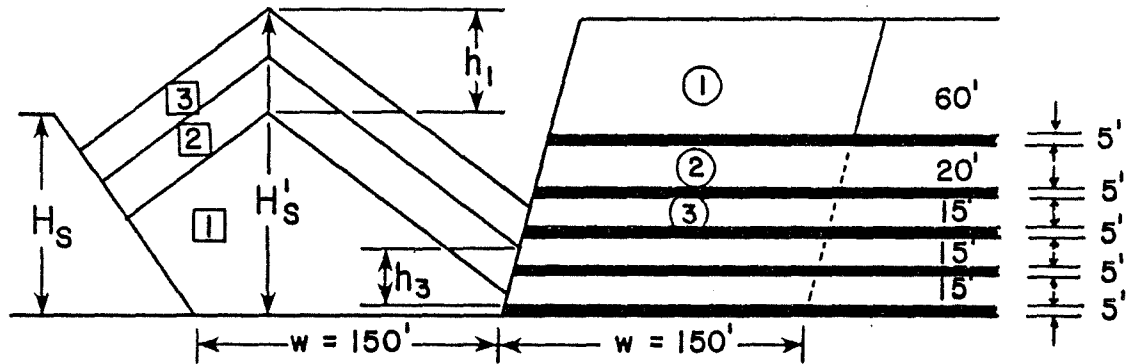


Figure F-1. Spoil Pile from the Highwall Pass 1, 2 and 3

$$\begin{aligned}
 c &= -150(60 + 20 + 15)(1 + 0.3) - 9 \times 11^2 \times 3.732 \\
 &\quad \times 0.7536 \div 8(0.7536 + 3.732) \\
 &= -18,610.36
 \end{aligned}$$

$$\begin{aligned}
 H_S &= \left(-165.6 \pm \sqrt{165.6^2 + 4 \times 0.192 \times 18,610.36} \right) \\
 &\quad \div (2 \times 0.192) \\
 &= 100, \text{ ft.}
 \end{aligned}$$

1.3.2. Compute Spoil Pile Height Before Leveling the Bench, H'_S , Feet (See Figure F-1)

$$H'_S = H_S + h_1, \text{ ft.}$$

where

$$h_1 = \left(-b \pm \sqrt{b^2 - 4ac} \right) / 2a, \text{ ft.}$$

$$a = 2/\tan^2 \theta (1/\tan \theta + 1/\tan \alpha) - 1/\tan \theta$$

$$b = -2(W + 2W_b)/\tan \theta (1/\tan \theta + 1/\tan \alpha)$$

$$c = (W + 2W_b)^2 / 2(1/\tan \theta + 1/\tan \alpha)$$

$$a = 2/0.7536^2 (1/0.7536 + 1/1.428) - 1/0.7536$$

$$= 0.42$$

$$b = -2(150 + 2 \times 11)/0.7536(1/0.7536 + 1/1.428)$$

$$= -226.3$$

$$c = (150 + 2 \times 11)^2/2(1/0.7536 + 1/1.428)$$

$$= 7305$$

$$h_1 = 226.3 \pm \sqrt{-226.3^2 - 4 \times 0.42 \times 7305/2 \times 0.42}$$

$$= 34.5, \text{ ft.}$$

$$H'_S = 34.5 + 100.6 = 135, \text{ ft.}$$

1.3.3. Compute Height of Spoil Pile Toe, h_3 , Feet

$$h_3 = [H'_S(1/\tan \theta - 1/\tan \alpha) + 3W_b/2] \tan \phi \tan \theta$$

$$\div (\tan \theta + \tan \phi), \text{ ft.}$$

$$h_3 = [100(1/0.7536 - 1/1.428) + 3 \times 11/2]$$

$$\times 3.732 \times 0.7536/(0.7536 + 3.732)$$

$$= 49, \text{ ft.}$$

1.3.4. Check Toe Height for Rehandle, Highwall Pass 1, 2 and 3

49 ft. > 40 ft. - rehandle is required.

Rehandle section is:

$$A_{ROB} = [h_3 - (D_3 + D_4 + S_3 + S_4)]^2 \sin \phi^2 \sin[180 - (\alpha + \phi)]$$

$$\times \sin(\theta + \phi)/2 \sin(\alpha - \theta), \text{ ft.}^2$$

$$A_{ROB} = [49 - (15 + 15 + 5 + 5)]^2 \times 0.9659^2 \times 0.76607$$

$$\div 2 \times 0.309 = 93.7, \text{ ft.}^2$$

1.3.5. Check for Dragline Reach for No Rehandle for the Highwall Pass 1, 2 and 3, R_{nr} , Feet

$$R_{ef} > R_{nr}$$

$$R_{nr} = (H'_S - h_3)/\tan \theta + (D_1 + D_2 + D_3 + D_4 + D_5 - h_3)$$

$$\div \tan \phi, \text{ ft.}$$

$$R_{nr} = (135 - 49)/0.7536 + (60 + 60 + 15 + 15 + 15 - 49)/3.732$$

$$= 93.7, \text{ ft.}$$

93.7 < 242 - no extended bench is necessary.

1.3.6. Check for Dragline Reach on the Spoil Pass 4 and 5, R_I , Feet

$$R_I = (H_S - h_3)/\tan \theta + W - h_3/\tan \phi, \text{ ft.}$$

$$R_I = (100 - 49)/0.7536 + 150 - 49/3.732 = 204, \text{ ft.}$$

$$R_I < R_{ef}$$

204' < 242' - dragline has sufficient reach.

1.3.7. Compute Spoil Pile Height for the Spoil Pass 4 and 5, H_{SF} , Feet

$$H_{SF} = A_{sp}/W + W \tan \theta/4, \text{ ft.}$$

where

$$A_{sp} = A_I + A_{rI}, \text{ ft.}^2$$

$$A_I = \text{section of interburden, 4th and 5th coal seams, ft.}^2$$

$$A_I = (D_3 + D_4)W(1 + \delta), \text{ ft.}^2$$

$$A_I = (15 + 15)150(1 + 0.3) = 5850, \text{ ft.}^2$$

$$A_{rI} = \text{rehandle section, ft.}^2$$

$$A_{rI} = A_2 + A_3 - A_4, \text{ ft.}^2$$

$$A_{rI} = 2W_b H_S + H_S^2 [1/2(\tan \theta) - 1/2(\tan \alpha)]$$

$$- [H_S(1/\tan \theta - 1/\tan \alpha) + 3W_b/2]h_3/2, \text{ ft.}^2$$

$$A_{rI} = 2 \times 11 \times 100 + 100^2(1/2 \times 0.7536 - 1/2 \times 1.428)$$

$$- [100(1/0.7536 - 1/1.428) + 3 \times 11/2] \times 49/2$$

$$= 4,178, \text{ ft.}^2$$

$$A_{sp} = 5850 + 4178 = 10,028, \text{ ft.}^2$$

$$H_{SF} = 10028/150 + 150 \times 0.7536/4 = 95.11, \text{ ft.}$$

$$H_{SF} < H_d$$

95.11' < 139' - dragline has sufficient dump height.

1.3.8. Compute Dig Volumes of Each Dragline Pass, V_i , bcy

- Overburden pass 1, V_1 , bcy.

$$V_1 = D_1 \times W \times L/27, \text{ bcy.}$$

$$V_1 = 60 \times 150 \times 150/27 = 50,000, \text{ bcy.}$$

- Interburden pass 2, V_2 , bcy.

$$V_2 = D_2 \times W \times L, \text{ bcy.}$$

$$V_2 = 20 \times 150 \times 150/27 = 16,650, \text{ bcy.}$$

- Interburden pass 3, 4 and 5, $V_{3,4,5}$, bcy.

$$V_{3,4,5} = D_{3,4,5} \times W \times L/27, \text{ bcy.}$$

$$V_{3,4,5} = 15 \times 150 \times 150/27 = 12,500, \text{ bcy.}$$

- Rehandle highwall side, V_{ROB} , bcy.

$$V_{ROB} = [A_{ROB}/(1 + \delta)] \times L/27, \text{ bcy.}$$

$$V_{ROB} = 93.7/1.3 \times 150/27 = 400, \text{ bcy.}$$

- Rehandle spoil side, V_{rI} , bcy.

$$V_{rI} = A_{rI}/(1 + \delta) \times L/27, \text{ bcy.}$$

$$V_{rI} = 4178/1.3 \times 150/27 = 17,855, \text{ bcy.}$$

1.3.9. Compute Percent Rehandle, p, Percent

$$p = \frac{V_{ROB} + V_{rI}}{V_T} \times 100, \text{ percent}$$

where

$$V_T = V_1 + V_2 + V_3 + V_4 + V_5, \text{ bcy.}$$

$$V_T = 50,000 + 16,650 + 3 \times 12,500 = 104,150 \text{ bcy.}$$

$$p = \frac{400 + 17,855}{104,150} \times 100 = 17.5\%$$

1.3.10. Estimate Average Swing Angles for Each Dragline Pass

Average swing angles for each dragline pass are shown in Table F-1.

Table F-1. Average Swing Angles for Each Dragline Pass

| Pass No. | Spoil Dump Position | Average Swing Angle (Degrees) |
|----------------------------------|-------------------------|-------------------------------|
| (1) Overburden Highwall Side | Spoil | 90 |
| (2) Interburden Highwall Side | Building Spoil Bench | 90 |
| (3) Interburden Highwall Side | Building Spoil Bench | 90 |
| (4) Interburden Spoil Side | Main Spoil Pile | 180 |
| (4) Rehandle Spoil Side | Main Spoil Pile | 120 |
| (5) Interburden Spoil Side | Main Spoil Pile | 180 |
| (5) Rehandle Spoil Side | Main Spoil Pile | 120 |

1.3.11. Compute Swings Per Hour for Each Dragline Pass,
 S_i

Swings per hour are computed for each pass from the graph shown in Figure F-2. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging overburden and interburden material. Rehandle is estimated to have 20 second dig time and 4 second delay in one cycle. Swing cycles per hour for each pass are shown in Table F-2.

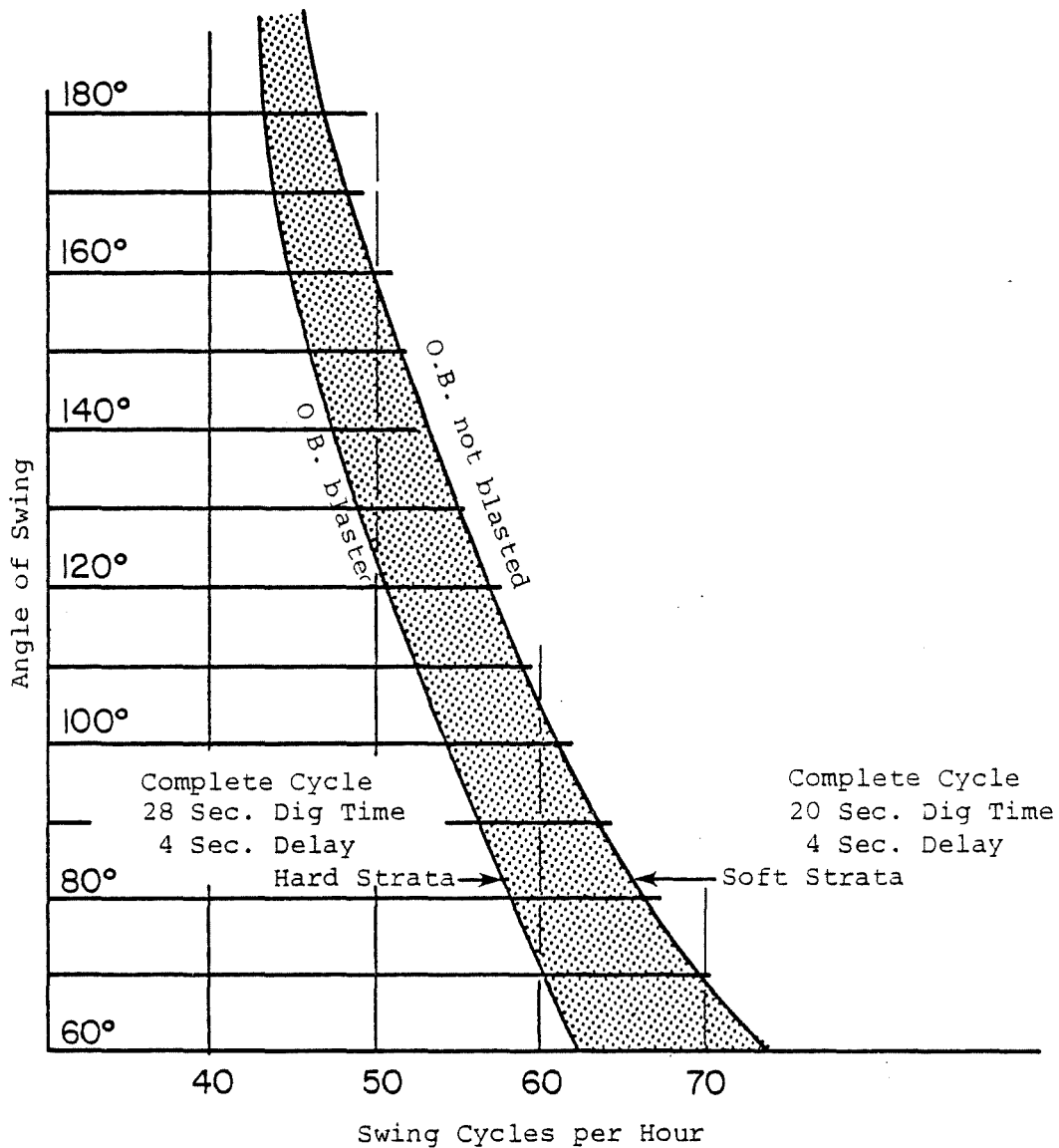


Figure D-2. Swing Cycles per Hour vs. Angle of Swing

Table F-2. Production Rate Calculation for Dragline, 75 Cu. Yd.
Multiple Coal Seam Horseshoe Method

| Output Variable | Dig Component for One Block | | | | | |
|---|-----------------------------|--------|--------|--------|--------|------------------------|
| | Pass 1 | Pass 2 | Pass 3 | Pass 4 | Pass 5 | Rehandle Pass 4 & 5 |
| Volume of Material, bcy. | 50,000 | 16,667 | 13,020 | 12,500 | 12,500 | 17,855 |
| Digging Time, seconds | 32 | 32 | 32 | 32 | 32 | 24 |
| Average Swing Angle, degrees | 90 | 90 | 90 | 180 | 180 | 120 |
| Swing Cycles Per Hour | 56 | 56 | 56 | 43 | 43 | 58 |
| Bucket Fill Factor, percent | 80 | 85 | 85 | 80 | 80 | 90 |
| Bucket Carrying Capacity | 46 | 49 | 49 | 46 | 46 | 68 |
| Required Digging Time, hours | 19.4 | 6.1 | 4.7 | 6.3 | 6.3 | 4.5 |
| Total Time to Complete One Block, hours | 47.3 | | | | | |
| Scheduled Time for One Block, hours | 67 | | | | | |
| Production Per Scheduled Hour, bcy. | 1,555 | | | | | |
| Monthly Production, bcy. | 1,026,000 | | | | | |
| Annual Production, bcy. | 12,312,000 | | | | | |

1.3.12. Compute Bucket Carrying Capacity for Each Pass, C_i , bcy

$$C_i = b \times f_i / (1 + \delta), \text{ bcy.}$$

where, for this mine,

$$f_1 = 0.8 \text{ sandstone}$$

$$f_2 = 0.85 \text{ shale}$$

$$f_3 = 0.9 \text{ rehandle}$$

1.3.13. Compute Time Required to Dig Material at Each Pass, T_i , Hours

$$T_i = V_i / (C_i S_i), \text{ hrs.}$$

1.3.14. Compute Total Time to Complete One Block, T , Hours

$$T = \sum T_i, \text{ hrs.}$$

1.3.15. Compute Scheduled Time to Dig One Block, T_S , Hours

$$T_S = T / pq, \text{ hrs.}$$

1.3.16. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_M / T_S, \text{ bcy.}$$

1.3.17. Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy/month}$$

$$H_M = 660 \text{ hrs.: scheduled working hours per month}$$

1.3.18. Compute Annual Production, P_Y , bcy

$$P_Y = 12 \times P_M, \text{ bcy.}$$

Production rate computations are shown in Table F-2.

2. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables F-3 and F-4 show detailed estimated hourly ownership and operating costs for dragline and dozer. The total estimated hourly ownership and operating costs for the dragline and dozer used as a utility machine are \$675.26.

3. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by the dragline with a dozer used as utility machine is \$0.44.

II. RECLAMATION AND ENVIRONMENTAL ASSESSMENT (by R.Hodder)

1. General Mine Information

The mine is a multiseam operation involving the recovery of only the five uppermost seams in a sequence of 130 seams that are identified in the basin. These series of coal deposits are included in the Medicine Bow Formation. This formation is considered unusual because it is an example of complete geological inversion of strata.

Approximately 350 acres per year are disturbed by the mining activity, but only 160 acres are disturbed by excavation.

The general terrain at the mine site is characterized by a series of parallel reef-like extrusions with minor drainages in between. Coal outcrops are an integral part of the reefs and dip at a decided angle toward the north. The general aspect of the terrain is undulating.

2. Reclamation-Related Practices and Equipment

2.1. Topsoil

Dark quality topsoil varies in depths from none to an average of 8 inches but may be as deep as 18 inches. The site productivity is low with most production appearing in the deeper swales where soils have accumulated to the depths mentioned. Typically, topsoil is removed approximately two months before mining by scrapers. The average haul distance is about 1,000 feet where it is stockpiled and protected. This process segregates the high value topsoil resource from the unconsolidated sandstone material below.

Table F-3. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | <u>313.82</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | <u>306.93</u> |
| <u>Total Ownership and Operating Costs</u> | <u>620.75</u> |

Table F-4. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$300,000 |
| Extras, Freight & Erection | -- |
| Total Price | 300,000 |
| Salvage Value @ 20% | 60,000 |
| Depreciated Amount | 240,000 |
| Life of Machine | 4 years @ 8,030 hrs/yr |
| Depreciation Cost | 7.47 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.48 |
| Total Ownership Costs | <u>11.95</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal/hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/hr. | 14.00 |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | <u>54.51</u> |

2.2. Topsoil Stockpiles

It has recently become necessary to stockpile topsoil for considerable periods of time because of stipulations in the new federal law that requires drastic time-consuming changes in the recontouring and shaping of soils.

Wind erosion is a factor in topsoiling when the materials are being transported, but when piled and stored, wind erosion is reduced to a negligible state because of compaction caused by scrapers in the stockpiling process. Water erosion is not a problem either because of the limited precipitation, compaction and protection.

Cover crops are planted on topsoil stockpiles to protect them from wind, rain and invasion of weeds. The native seeding mixture listed is the usual seeded species. No chemical stabilizing sprays or systems are used on topsoil stockpiles. Seeded vegetation on stockpiles apparently is effective in protecting the topsoil for two years or better, the storage term necessary to prepare spoils for topsoiling following excavation under the new system.

Unconsolidated material beneath the undisturbed topsoil is usually of sandstone origin and thus forms acceptable subsoil material to place between spoils and topsoil. These materials are available in quantities sufficient to supply an effective buffer layer of from 2 to 3 feet in thickness of non-inhibitory yet permeable growth medium.

2.3. Spoil Segregation and Burial

The spoils are typically high in carbonaceous shale content, but according to available analyses, no inhibitory or toxic materials are present in quantities that would suggest the need for special handling with but few exceptions.

Smutty black coal types, partings, and cleanings are segregated and buried. Also, excessive rocks and occasionally debris are also buried. The shaley overburden when moved and mixed is not considered inhibitory or impermeable, yet it forms a stable and secure base. Most segregation and burial is accomplished by the use of draglines and dozers.

2.4. Surface Water

Surface water becomes a problem occasionally because of the nature of the precipitation pattern and the violent storm character. Much of the 8 to 10 inches of annual precipitation comes as winter snow and is lost to sublimation by the persistent

strong winds typical of the area. The remainder usually occurs as spring rains and causes occasional drainage and erosion difficulties.

2.5. Surface Drainage

The nature of the terrain and the lack of precipitation in the area eliminates the need for highwall drainage ditches, levees, and stream diversion ditches. These are not necessary and are not used.

Because of the general 1 to 2 percent slope toward the pit area, the road and ramp system tends to funnel the surface runoff into the pit, but otherwise it creates few problems. It does tend to augment the accumulation of groundwater seepage into the 1570 Pit from which water is pumped for dust control use.

2.6. Surface Water -- After Mining

In a post-mining sense, similar surface runoff patterns persist without major runoff and erosion. This is probably due to the very gradual gradient constructed into the recontoured spoils and the elimination of the swales and low areas that tended to accumulate water previous to mining.

All runoff from the recontoured spoils is directed through two settling ponds, the major one being tied into the area drainage system. Except in February and March, the settling basins are recharged primarily by pumping groundwater seepage from the pit and by occasional summer thunderstorms.

2.7. Ground Water -- Before Mining

As mentioned previously, reliable sources of groundwater are not available above 350 feet from the undisturbed surface. The quality of this water is questionable for drinking purposes because of its mineral content.

Mining in the area does not extend down to the major aquifer system, although the pit may intercept seeps and wet spots as is noticeable along the highwall. The accumulating seepage plus surface runoff is used for dust control purposes or may be pumped into a settling pond.

2.8. Ground Water -- After Mining

It is assumed that mining activity above the major aquifer has not affected nor deteriorated the integrity of the ground water system. No indications are apparent that deterioration has occurred and no analysis data are available presently to suggest this, but monitoring continues.

2.9. Ground Water Supply During Mining

Local wells are scarce in the vicinity of the mine, the closest recognized one being at the mine office. There has been no need by the company to deepen wells in the area because wells are evidently unaffected by the mining. This has also made it unnecessary for the company to haul water to local farmers and ranchers. They are too distant from the mining activity to be affected.

2.10. Erosion and Sediment Control

Sediment control is accomplished by holding all runoff from the mine area in settling ponds.

Prompt grading and seeding of slopes was a procedure used previously to help control erosion, as was the accompanying surface treatment of terracing the outslope. But now that recent federal legislation has affected the gradients and delayed timing of recontouring considerably, these treatments have not recently been done. With the call for much reduced gradients, the need for these treatments has been largely eliminated under ordinary circumstances.

2.11. Inclines and Ramps

Inclines are constructed through the spoil as well as in undisturbed areas. They will be reclaimed after excavation, filled in and leveled so that no depressions will remain.

2.12. Rough Grading (Backfilling)

Previous to the passage of the federal law, grading was kept current within one spoils ridge from the pit. Now the requirements of the new regulations have so defined the recontouring and the elimination of old ridges, that the excessive grading to a nearly flat gradient has been practiced at the mine.

Most of the rough grading work is done with crawler tractors and occasionally front end loaders. There are eight dozers assigned to reclamation recontouring.

2.13. Spoil Amendments

Backblading is done with dozers as a part of the recontouring process. Dragging is not done, but discing is in order to prepare a good seedbed. Fertilizer is not applied at the site. Waste straw is used as a mulch material to help control erosion and increase infiltration and establishment of a stable grass cover.

2.14. Revegetation

The goal of revegetation is to gain surface stability and produce a forage cover suitable for grazing of livestock.

Drilling seed into the seedbed is standard practice at this mine. Hydroseeding or mulching is almost never used. Trees have been planted in the past but tree planting is not a standard procedure.

The present standard seeding mixture contains predominantly native species (six out of eight), and this mixture was determined partially by field-plot testing.

2.15. Final Cut

The final cut will not be an impoundment, as this would be impractical because of lack of water. The highwall in the final cut will be eliminated by backfilling to a gradient of 6:1 or less at tremendous expense.

2.16. Topography to Which Land is Reclaimed

Approximately 160 acres are disturbed by excavation each year. This acreage is returned to approximate original contour. Even though the land aspect was rough and rolling, the post-mining aspect will be flat with few if any interruptions. No highwall or bench will be evident and no pond areas will be left.

The maximum slopes allowable would be 6:1 gradient where box cut spoils would be eliminated. Conditions at this time do not suggest that such gradients will occur. The general strike of topographic features after mining will be 2 degrees or less, or flat.

2.17. Land Use

After mining, the land use is intended to be grazing by livestock just as it was previous to disturbance.

2.18. Costs of Reclamation

Costs of reclamation are variable between pits, but average out presently at \$4,800 to \$6,000 per acre. Grading in the 280 BE shovel pit is slightly more expensive than in the dragline pits, other costs being about the same.

The costs of recontouring and grading are highest varying between \$3,000 and \$4,250 per acre. Stripping, stockpiling and topsoiling average \$1,200 per acre. Mulching and seeding average \$600 per acre.

3. Recommendations

3.1. Recontouring

The intent of the federal regulations should be investigated by the company. Present interpretation of regulations under the unique land forms and spacing of pits encountered definitely negate the intent of recontouring to approximate original contour. Creating practically level terrain in dry rough undulating country does not improve the agricultural potential of the land and does not lend to aesthetic continuity. The greatest land use potential will remain as low productivity grazing lands. This use would benefit tremendously by the creation of an undulating surface rather than an exaggerated flat one.

3.2. Topsoil Stockpile

The topsoiling process seems inefficient in several respects. Topsoil must now be stockpiled for long periods of time (2 to 2-1/2 years) because of the extended term of surface disturbance between topsoil salvage operations and recontouring to create flat terrain.

With more realistic interpretation of the regulations and their intent, a much more efficient operation could be developed that would:

- Shorten the topsoil storage term.
- Eliminate stockpiling in some cases.

- Create a far more usable land form for the grazing purpose intended.
- Create a more aesthetically appealing terrain.
- Offer protection from wind to stock, vegetation and soil surface.

3.3. Terracing

Erosion control treatments at the mines seem to rely on the terracing method of reducing erosion on extensive slopes. This treatment is exacting, requiring surveying to maintain absolute level and thus is expensive. Furthermore, it is not very effective in retaining runoff on hillsides. Creating basins on the approximate contour is much cheaper and more effective. This treatment can be continuous from the top of a slope to the bottom so that excessively large quantities of rain or snow cannot accumulate to gully out the hillside.

APPENDIX G

DOZER PRODUCTIVITY CALCULATION
WHEN STRIPPING THIN INTERBURDEN

APPENDIX G

DOZER PRODUCTIVITY CALCULATION WHEN STRIPPING THIN INTERBURDEN

(By Attilio DiMarco)

1. Introduction

A step-by-step procedure for calculating the production rate for a dozer stripping thin interburden is presented in this Appendix. It is based on a mathematical model developed during this study, and it involves splitting the dozer lift into several horizontal strata or "digging components" and then computing the average dozer push distances and rehandle volumes for each component.

After calculating the push distances and volumes (bank and rehandle) for each dig component, the dozer production rate is calculated using manufacturers' performance curves and factors relating volume moved per hour to push distance, grade, operator efficiency, blade type, and material type.

2. Method of Operation

The method of stripping is one whereby the dozer(s) strip in blocks within a parallel pit. After one block of coal is exposed and hauled out, the dozer(s) proceed to push the next block of material into the mined-out block. In doing this, some of the dozer pushing is uphill. Figure G-1 shows a cross section taken parallel to the length of the pit.

It is assumed that every dozer push starts at the beginning of the block and that the load is pushed until the edge of the highwall is reached or it becomes necessary to push uphill. Rehandle begins when the spoil pile, as it raises up the highwall, reaches the level where the dozing is taking place. From then on every push is longer than the previous one, as a wider and wider "extended bench" is built. Uphill pushing is assumed to start as soon as the extended bench reaches the point where it intersects the new spoil pile (see Figure G-2), whereas in actual practice, uphill pushing would probably not begin until the bench is extended all the way across to the previous spoil pile. The present approach is adopted because it results in a somewhat simpler analytical formulation, although it can be shown that the estimated dozing time difference between the two practices is very small. In the adopted approach, as the dozing level gets lower (closer to the coal), every push will have a level segment followed by an uphill segment.

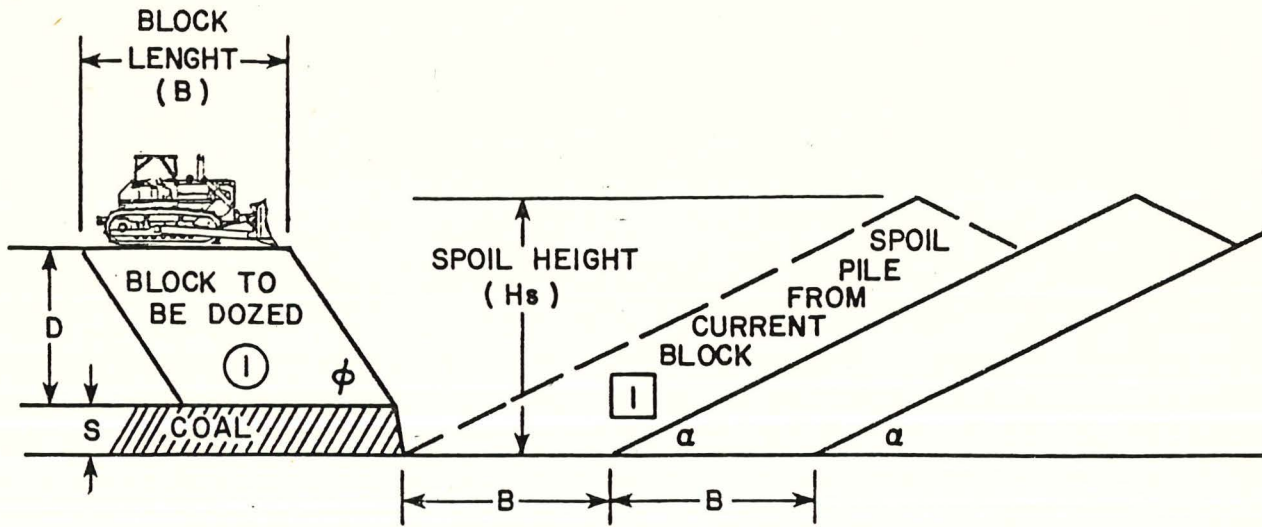


Figure G-1. Cross-Section Parallel to the Length of the Pit

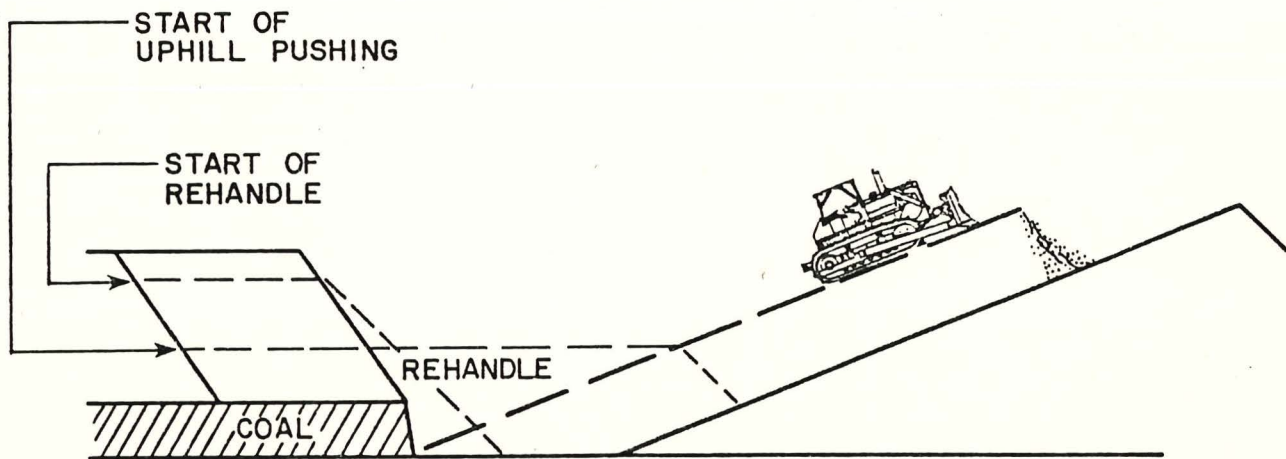


Figure G-2. Cross-Section Showing Uphill Pushing

3. Procedure for Computing the Pit Geometry

Consider as an example a case with 20 ft. of interburden and 4 ft. of coal. Other inputs are as follows:

$B = 50$ ft.: block length $D = 20'$

$f = .30$: swell factor $s = 4'$

$\tan \alpha = .20$, percent: spoil pile slope

$\phi = 45^\circ$: highwall angle

$\theta = 37^\circ$: angle of repose

$W = 100$ ft.: pit width

3.1 Dozing Level for Start of Rehandle

The first quantity of interest is the level at which the dozer begins rehandling material (H_1 in Figure G-3). Rehandle begins when the dozed pile reaches ¹ the level of dozing.

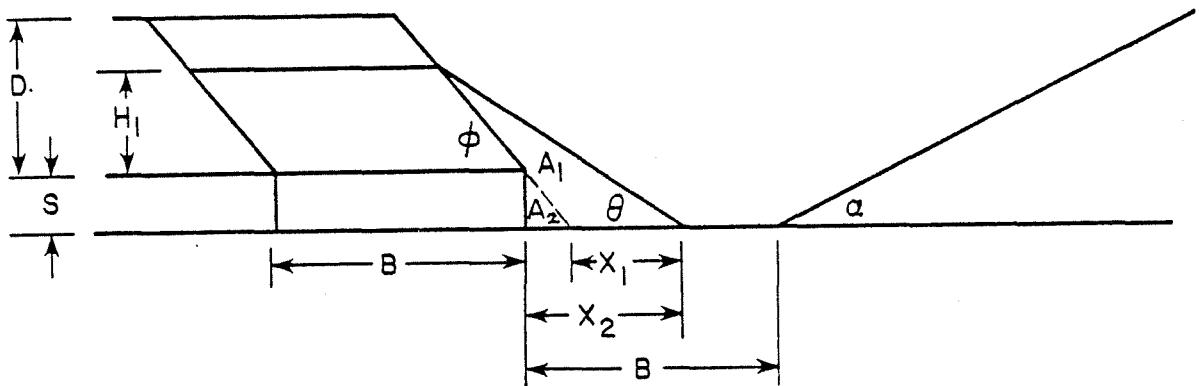


Figure G-3. Pit Geometry at the Start of Rehandle

It results that:

$$X_1 = (H_1 + S) (1/\tan \theta - 1/\tan \phi), \text{ ft.}$$

$$A_1 = (H_1 + S)^2 (1/\tan \theta - 1/\tan \phi) / 2 + S^2 / (2 \tan \phi), \text{ ft.}^2$$

Equating the swelled bank above H_1 , $(D - H_1)B(1 + f)$, with A_1 and solving for H_1 yields

$$H_1^2 a + H_1 [2aS + B(1 + f)] + S^2 a + S^2 / (2 \tan \phi) - DB(1 + f) = 0$$

where $a = (1/\tan\theta - 1/\tan\phi)/2$.

Solving the above quadratic equation yields H_1 . For the example we have

$$a = .1635$$

$$H_1^2 \times .1635 + H_1 \times 66.308 - 1,289.384 = 0$$

or

$$H_1 = (-66.308 \pm 72.388)/.327, \text{ ft.}$$

The "+" solution yields $H_1 = 18.59$ ft. The corresponding value of X_1 is 7.39, and since this is less than $B (=50)$, the value of H_1 is valid. If X_2 had been greater than B , the calculation of H_1 would be somewhat different, and this case is not treated here because it is a rare practical situation.

3.2 Dozing Level for Start of Uphill Pushing

As dozing progresses below H_1 an "extended bench" will be built. It will become wider and wider until at some point it will intersect the new spoil pile. This situation is shown in Figure G-4. We want to determine the value of H_2 such that the swelled bank area above it equals that of area $A_1 + A_2 + A_3$. Now, it results that

$$A_1 = H_2^2(1/\tan\phi + 1/\tan\alpha)/2 + H_2S/\tan\alpha, \text{ ft.}^2$$

$$X_2 = (H_2 + S)/\tan\alpha, \text{ ft.}$$

$$h = B/(1/\tan\theta + 1/\tan\alpha), \text{ ft.}$$

$$A_3 = S^2/(2\tan\alpha), \text{ ft.}^2$$

$$A_2 = By - B^2/[2(1/\tan\theta + 1/\tan\alpha)], \text{ ft.}^2$$

When the extended bench first intersects the new spoil pile,
 $y = H_2 + S$.

To solve for H_2 the swelled bank above, $(D - H_2)B(1 + f)$, is equated with $A_1 + A_2 + A_3$. This yields the quadratic equation

$$H_2^2a + H_2[S/\tan\alpha + B(2 + f)] + S^2/(2\tan\alpha) - b - DB(1 + f) = 0$$

where

$$a_2 = (1/\tan\phi + 1/\tan\alpha)/2$$

$$b = B^2[2(1/\tan\theta + 1/\tan\alpha)]$$

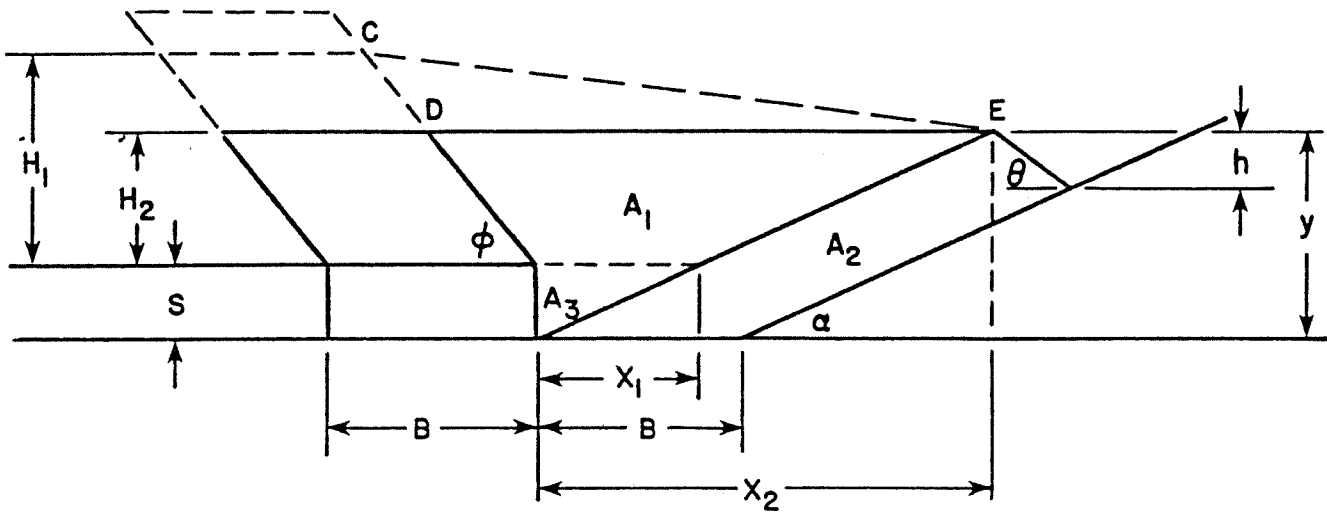


Figure G-4. Dozing Level for Start of Uphill Pushing

For the example we have

$$a_2 = (1/\tan 45 + 1/.2)/2 = 3.0$$

$$b = 50^2/2(1/\tan 37 + 4/.2) = 197.56$$

$$\text{and } 3H_2^2 + 135H_2 - 1,457.56 = 0$$

$$\text{or } H_2 = (-135.0 \pm 188.99)/6$$

$$H_2 = 9, \text{ ft.}$$

The above computational procedure for H_2 is valid only for cases whereby $Y \geq h$, a condition that holds for most practical situations. To test for this condition, we set $Y = h$, calculate $A_1 + A_2 + A_3$ ($h - S$). For the example this test works out as follows:

$$h = 50/(1/\tan 37 + 1/.2) = 7.9$$

$$h - S = 3.9$$

$$A_1 = 3.9^2(1/\tan 45 + 1/.2)/2 + 3.9 \times 4/.2 = 123.63, \text{ ft.}^2$$

$$A_3 = 4^2/(2 \times .2) = 40, \text{ ft.}^2$$

$$A_2 = 50 \times 3.9/2 = 97.5, \text{ ft.}^2$$

$$A_1 + A_2 + A_3 = 261, \text{ ft.}^2$$

Swelled bank area above $h - S$ is

$$(20 - 3.9)50 \times 1.3 = 1,046, \text{ ft.}^2$$

Since this value exceeds 261, the value of H_2 calculated by the procedure given above is correct.

3.3 Height of the Final Spoil Pile, H_S , ft.

This is the value of y for which the area A_2 equals the total swelled bank area. For the example

$$20 \times 50 \times 1.3 = 50y - 197.56,$$

$$\text{or } H_S = 30, \text{ ft.}$$

The final spoil pile will thus extend $30 - (20 + 4) = 6$ ft. above the top of the original bank.

4. Dozer Production Calculations

From the start of dozing a new block down to the point where rehandle beings (H_1), every push is assumed to be equal to the block length. (see¹Figure G-4). For component 1, we thus have

Dozing distance: 50 ft.

Swelled bank area: $(20 - 18.59) \times 50 \times 1.3 = 91.65, \text{ ft.}^2$

Rehandle amount: 0

From H_1 on every push includes some rehandle. When H_2 is reached the push distance is

$$D_{z_2} = B + H_2/\tan\phi + (H_2 + S)/\tan\alpha, \text{ ft.}$$

which for the example works out to 129 ft. The amount of rehandle incurred in going from H_1 to H_2 is approximated as the area of the triangle CDE shown in ¹ Figure G-4*, this area being equal to

$$(H_1 - H_2)(D_{z_2} - B)/2.$$

Therefore, for component 2, we have

Average dozing distance: $(50 + 129)/2 = 89.5, \text{ ft.}$

Swelled bank area: $(18.59 - 9) \times 50 \times 1.3 = 623.4, \text{ ft.}^2$

Rehandle area: $(18.59 - 9.0) \times 79/2 = 378.8, \text{ ft.}^2$

After H_2 is reached, every push will include some uphill pushing. The largest uphill push occurs for the last dozing cycle, when H is zero. At this point the uphill distance is $(H_s - S)/\sin\alpha$, which for the example works out to 132.3 ft. The level push is $B + S/\tan\alpha = 50 + 20 = 70 \text{ ft.}$, for a total push of 202.3 ft.

The rehandle area for component 3 is equal to area A_1 , which amounts to $9^2(1/\tan 45 + 1/.2)/2 + 9 \times 4/.2 = 423$. So for component 3, we have

Average dozing distance: $(129 + 202.3)/2 = 165.7, \text{ ft.}$

Swelled bank area: $9 \times 50 \times 1.3 = 585, \text{ ft.}^2$

Rehandle area: 423, ft.^2

* It can be shown that this area overestimates the rehandle because the extended bench width is not a linear function of H_1 as implied by the triangular area. This overestimate is not significant.

Component 4 is comprised of the dozing required to clean up the rehandle in front of the coal, area A_3 of Figure G-4. This area, which is $S^2/(2\tan\alpha)$ and amounts to 40 ft², is assumed pushed a distance of $H_S/\sin\alpha$ or 152.7 ft. for the example.

Since the uphill dozer work involved here does not involve any uphill cutting, performance will not be penalized by the uphill pushing except for the resulting increase in push distances. The production calculations are shown in Table G-1. They are based on the dozer performance curve shown in Figure G-5.

The effect of increased overburden is shown in Table G-2, which shows the dozer productivity results of 25 ft. cover. With respect to the 20 ft. case, it is seen that a 25% increase in overburden depth results in a 35.6% increase in dozer operating time per block. Rehandle becomes a greater proportion of the total effort, changing from 65 to 89%. The production rate per scheduled hour decreases by about 8%, going down from 241 to 223 bcy.

Table G-1. Dozer Production Rate for Stripping Interburden

Interburden thickness: 20 feet

Block length: 50 feet

Pit width: 100 feet

Efficiency factor: 0.653 (U-blade, good working efficiency)

| Output Variable | Dig Component of One Block | | | | One Block |
|--|----------------------------|---------|---------|-------|-----------------|
| | 1 | 2 | 3 | 4 | |
| Avg. dozing distance, ft. | 50 | 89.5 | 165.7 | 152.7 | |
| Production rate, per scheduled hour, lcy | 1,045.0 | 698.7 | 398.3 | 437.5 | |
| Bank volume, lcy | 339.6 | 2,308.9 | 2,166.7 | 0 | 4,815.2 |
| Rehandle volume, lcy | 0 | 1,403.0 | 1,566.7 | 148.1 | 3,117.8 (64.7%) |
| Scheduled time for one block, hrs. | .33 | 5.31 | 9.37 | .34 | 15.35 |
| Production per scheduled hour, bcy | 241 | | | | |

Table G-2. Dozer Production Rate for Stripping Interburden

Interburden thickness: 25 feet

Block Length: 50 feet

Pit Width: 100 feet

Efficiency factor: 0.653 (U-blade, good working efficiency)

| Output Variable | Dig Component of One Block | | | | One Block |
|--|----------------------------|---------|---------|-------|-----------------|
| | 1 | 2 | 3 | 4 | |
| Avg. dozing distance, ft | 50 | 92 | 184.8 | 185.9 | |
| Production rate, per scheduled hour, lcy | 1,045.0 | 692.2 | 359.2 | 359.0 | |
| Bank volume, lcy | 474.3 | 2,975.6 | 2,568.7 | 0 | 6,018.6 |
| Rehandle volume, lcy | 0 | 1,922.7 | 2,055.4 | 148.1 | 4,126.2 (89.1%) |
| Scheduled time for one block, hrs. | .45 | 7.08 | 12.87 | .41 | 20.81 |
| Production per scheduled hour, bcy | 223 | | | | |

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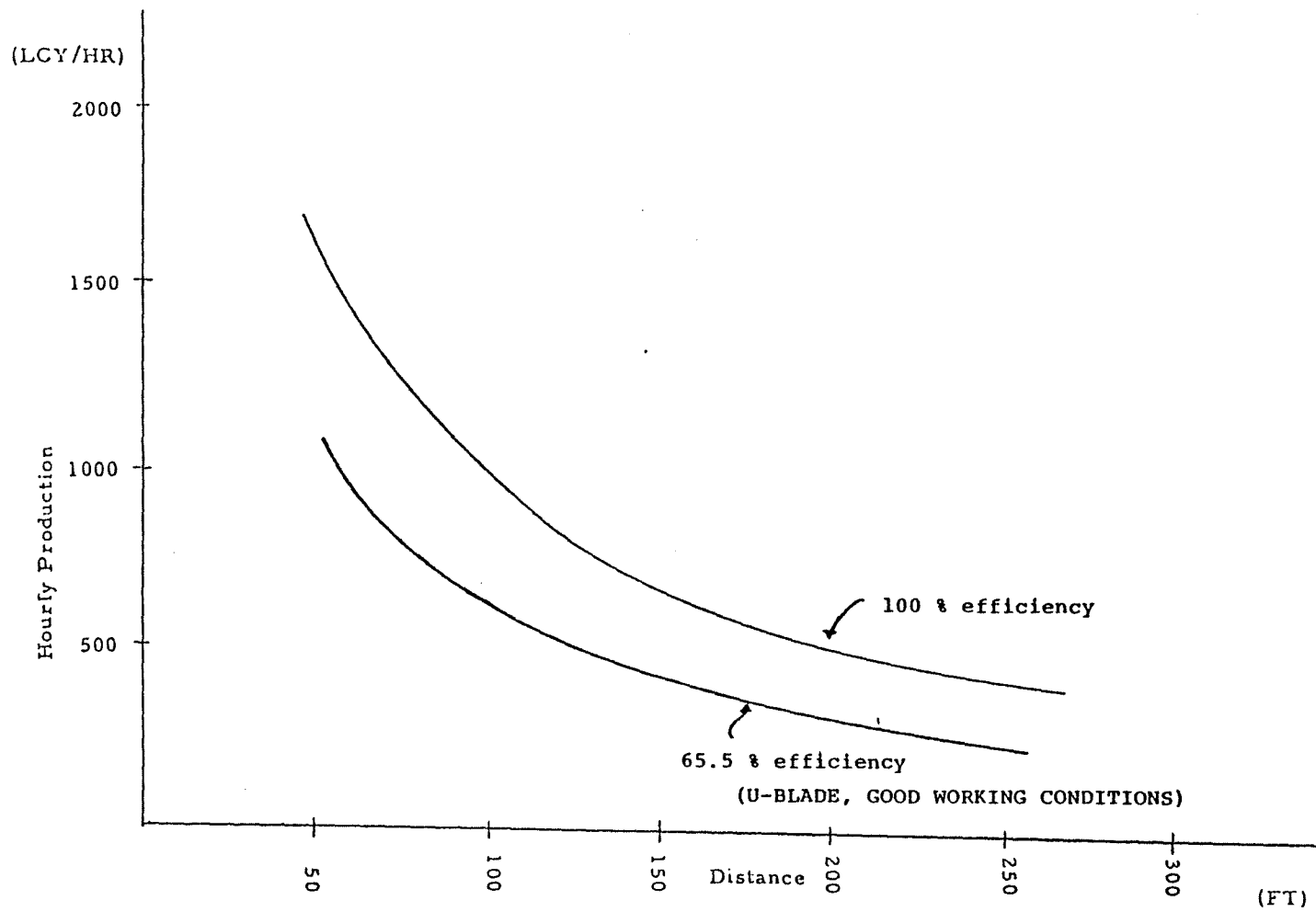


Figure G-5. Production Curve for a 400 H.P. Dozer*

* Obtained from Komatsu Specifications and Application Handbook, Edition 1, pp. 3-70.

APPENDIX H
CENTRAL REGION SITE
SINGLE DRAGLINE HORSESHOE METHOD

APPENDIX H
CENTRAL REGION SITE
SINGLE DRAGLINE HORSESHOE METHOD

1. Introduction

This Appendix contains production and stripping cost estimates of the base case - Single Dragline Horseshoe Method - developed for the Central Region.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

| | | | |
|----------------|---|-------------|--|
| W | = | 150 feet: | pit width |
| D ₁ | = | 70 feet: | total overburden depth |
| D ₂ | = | 20 feet: | interburden thickness shale |
| D ₃ | = | 50 feet: | height of dragline bench |
| D ₄ | = | 20 feet: | thickness of overburden clay strata |
| D ₅ | = | 30 feet: | thickness of overburden sandstone strata |
| D ₆ | = | 20 feet: | thickness of overburden shale strata |
| S ₁ | = | 72 inches: | thickness of upper coal seam |
| S ₂ | = | 48 inches: | thickness of lower coal seam |
| φ | = | 70 degrees: | highwall angle |
| θ | = | 37 degrees: | natural angle of repose |
| α | = | 50 degrees: | angle of oversteepened spoil |
| δ | = | 30 percent: | average spoil factor |
| L | = | 150 feet: | length of block |

- R_d = 282 feet: dragline dump radius
- R_{ef} = 232 feet: dragline effective radius
- H_d = 139 feet: dragline dump height
- b = 75 cubic yards: dragline bucket capacity
- w_b = 10 feet: width of bucket
- q = 0.83: dragline utilization
- p = 0.85: dragline availability

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

- H_S = Height of spoil bench above the pit floor, feet
- h_1 = Height of material for filling the spoil bench, feet
- H'_S = Total spoil pile height from the overburden pass, feet
- h_3 = Height of the spoil pile toe from the overburden pass, feet
- R_{nr} = Dragline radius required for no rehandle, feet
- R_I = Dragline reach required for the interburden pass, feet
- H_{SF} = Height of spoil pile from the interburden pass, feet
- V_i = Dig volumes of each dragline pass, bcy
- V_{rOB} = Volume of rehandle material, overburden pass, bcy
- V_{rI} = Volume of rehandle material, interburden pass, bcy
- S_i = Swing cycles per hour for each pass
- C_i = Bucket carrying capacity for each pass, bcy

- T_i = Time required to dig material of each pass, hours
 T = Time to complete one block, hours
 T_S = Scheduled time to dig one block, hours
 P_H = Dragline production rate per scheduled hour, bcy
 P_M = Dragline monthly production, bcy
 P_Y = Dragline annual production, bcy

4. Procedure for Estimating Production Rate for the Single Dragline Horseshoe Method

A procedure for estimating the production rate for a single dragline using the horseshoe method is given below. Figures H-1 and H-2 illustrate the operating procedure of the base case.

4.1. Compute Spoil Bench Height After Leveling, H_S , Feet

$$H_S = (-b \pm \sqrt{b^2 - 4ac})/2a, \text{ ft.}$$

where

$$a = (1/2 \tan \theta - 1/2 \tan \alpha) - (1/\tan \theta - 1/\tan \alpha)^2 \times \tan \phi \tan \theta / 2(\tan \theta + \tan \phi)$$

$$b = W + 2w_b - 3w_b (1/\tan \theta - 1/\tan \alpha) \tan \phi \tan \theta \div 2(\tan \theta + \tan \phi)$$

$$c = -W(D_1 + D_2)(1 + \delta) - 9w_b^2 \tan \phi \tan \theta \div 8(\tan \theta + \tan \phi)$$

$$a = (1/2 \times 0.7536 - 1/2 \times 1.1918) - (1/0.7536 - 1/1.1918)^2 \times 2.7475 \times 0.7536 / 2(0.7536 + 2.7475) = 0.17$$

$$b = 150 + 2 \times 10 - 3 \times 10 (1/0.7536 - 1/1.1918) \times 2.7475 \times 0.7536 / 2(0.7536 + 2.7475) = 166.87$$

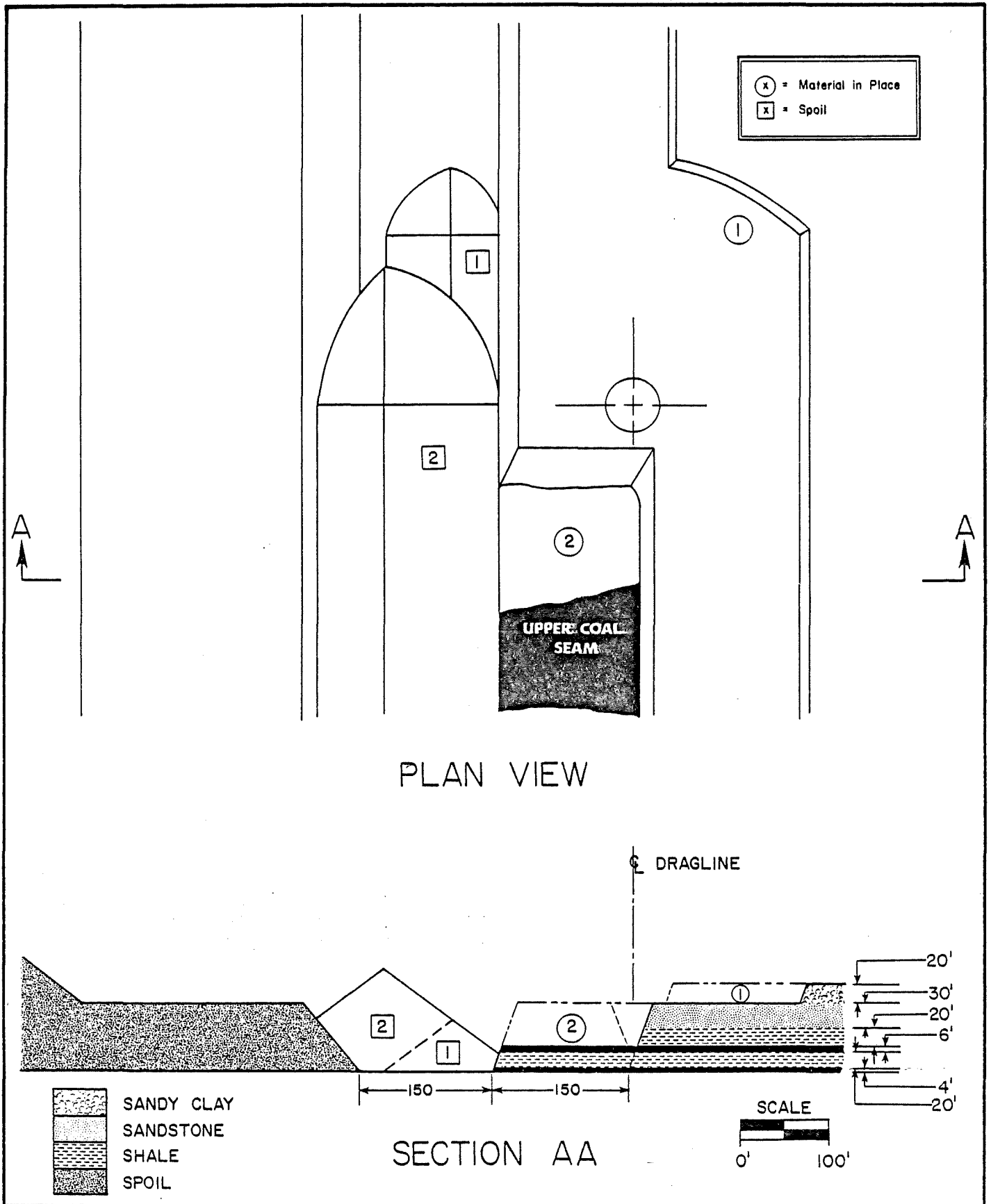


Figure H-1. Dragline Removing Overburden:
Single Dragline Horseshoe Method

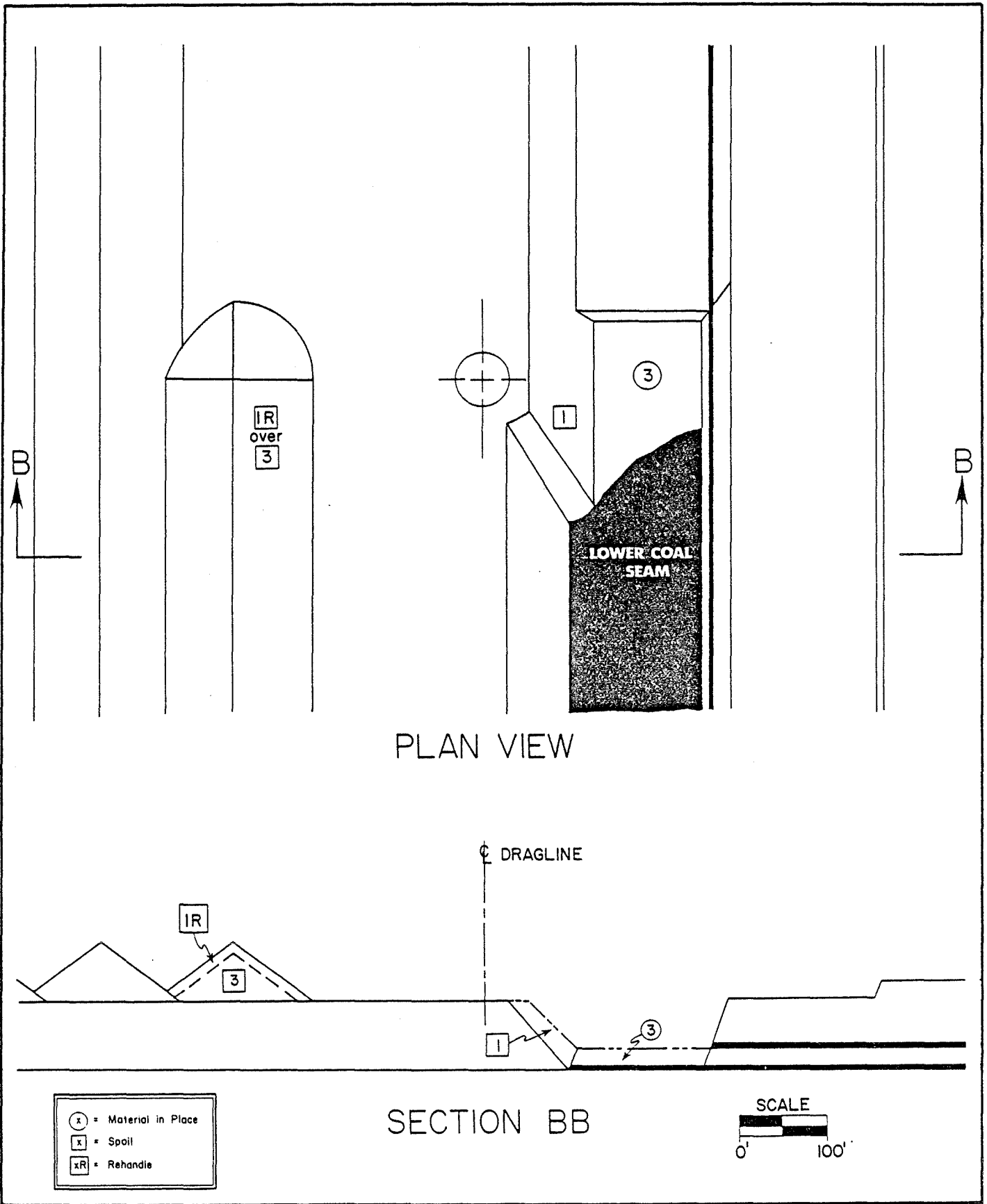


Figure H-2. Dragline Removing Interburden:
Single Dragline Horseshoe Method

$$c = -150(20 + 50)(1 + 0.30) - 9 \times 10^2 \times 2.7475 \times 0.7536$$

$$\div 8 (0.7536 + 2.7475) = -13,716$$

$$H_S = (-166.87 \pm \sqrt{166.87^2 - 4 \times 0.17 \times -13,716}) / 2 \times 0.17$$

$$= 76 \text{ ft.}$$

4.2. Compute Spoil Pile Height Before Leveling the Bench, H'_S , Feet

$$H'_S = H_S + h_1, \text{ ft.}$$

where

$$h_1 = (-b \pm \sqrt{b^2 - 4ac}) / 2a, \text{ ft.}$$

$$a = 2/\tan^2 \theta (1/\tan \theta + 1/\tan \alpha) - 1/\tan \theta$$

$$b = -2(W + 2w_b)/\tan \theta (1/\tan \theta + 1/\tan \alpha)$$

$$c = (W + 2w_b)^2 / 2(1/\tan \theta + 1/\tan \alpha)$$

$$a = 2/0.7536^2 (1/0.7536 + 1/1.1918) - 1/0.7536$$

$$= 0.31$$

$$b = -2(150 + 2 \times 10)/0.7536(1/0.7536 + 1/1.1918) = -183$$

$$c = (150 + 2 \times 10)^2 / 2(1/0.7536 + 1/1.1918) = 6,671$$

$$h_1 = 183 \pm \sqrt{-183^2 - 4 \times 0.31 \times 6,671} / 2 \times 0.31 = 39, \text{ ft.}$$

$$H'_S = 39 + 76 = 115 \text{ ft.}$$

4.3. Compute Height of Spoil Pile Toe, h_3 , Feet

$$h_3 = [H_S(1/\tan \theta - 1/\tan \alpha) + 3w_b/2] \tan \phi \tan \theta$$

$$\div (\tan \theta + \tan \phi), \text{ ft.}$$

$$h_3 = [76(1/0.7536 - 1/1.1918) + 3 \times 10/2]$$

$$\times 2.7475 \times 0.7536 / (0.7536 + 2.7475) = 30, \text{ ft.}$$

4.4. Check Toe Height for Rehandle, Overburden Pass

$$30' = 30' - \text{no rehandle is required}$$

4.5. Check for Dragline Reach for No Rehandle for the Overburden Pass, R_{nr} , Feet

$$R_{nr} = (H'_S - h_3) / \tan \theta + (D_1 + D_2 + D_3 - h) \div \tan \phi$$

$$R_{nr} = (115 - 30) / 0.7536 + (20 + 50 + 20 - 30) / 2.7475 = 132 \text{ ft.}$$

$$R_{nr} < R_{ef}$$

132' < 232', no extended bench necessary

4.6. Check for Dragline Reach on the Interburden Pass, R_I , Feet

$$R_I = (H_S - h_3) / \tan \theta + W - h_3 / \tan \phi, \text{ ft.}$$

$$R_I = (77 - 30) / 0.7536 + 150 - 30 / 2.7475 = 184 \text{ ft.}$$

$$R_I < R_{ef}$$

184' < 232', dragline has sufficient reach

4.7. Compute Spoil Pile Height for the Interburden Pass, H_{SF} , Feet

$$H_{SF} = A_{SP} / W + W \tan \theta / 4, \text{ ft.}$$

where

$$A_{SP} = A_I + A_{rI}, \text{ ft.}^2$$

$$A_I = \text{section of interburden, ft.}^2$$

$$A_I = D_3 W (1 + \delta), \text{ ft.}^2$$

$$A_I = 20 \times 150 (1 + 0.30) = 3,900, \text{ ft.}^2$$

$$A_{rI} = \text{rehandle section, ft.}^2$$

$$A_{rI} = A_2 + A_3 - A_4, \text{ ft.}^2$$

$$A_{rI} = 2w_b H_S + H_S^2 [1/2(\tan \theta) - 1/2(\tan \alpha)] - [H_S(1/\tan \theta - 1/\tan \alpha) + 3w_b/2] h_3/2, \text{ ft.}^2$$

$$A_{rI} = 2 \times 10 \times 76 + 76^2 (1/2 \times 0.7536 - 1/2 \times 1.1918) - [76(1/0.7536 - 1/1.1918) + 3 \times 10/2] \times 30/2 = 2,171, \text{ ft.}^2$$

$$A_{SP} = 3900 + 2171 = 6,071, \text{ ft.}^2$$

$$H_{SF} = 6071/150 + 150 \times 0.7536/4 = 63, \text{ ft.}$$

4.8. Check for Dragline Dump Height

$$H_{SF} < H_d$$

63' < 139' , dragline has sufficient dump height.

4.9. Compute Dig Volumes of Each Dragline Pass, V_i , bcy

- Overburden pass, $V_{1,2}$, bcy.

$$V_1 = D_1 \times W \times L/27, \text{ bcy}$$

$$V_1 = 20 \times 150 \times 150/27 = 16,667, \text{ bcy}$$

$$V_2 = D_2 \times W \times L/27, \text{ bcy}$$

$$V_2 = 50 \times 150 \times 150/27 = 41,667, \text{ bcy}$$

- Interburden pass, V_2 , bcy.

$$V_2 = D_3 \times W \times L/27, \text{ bcy}$$

$$V_2 = 20 \times 150 \times 150/27 = 16,667, \text{ bcy}$$

- Dragline rehandle, V_{rI} , bcy.

$$V_{rI} = [A_{rI}/(1 + \delta)] \times L/27, \text{ bcy}$$

$$V_{rI} = [2171/(1 + 0.30)] \times 150/27 = 9,277, \text{ bcy}$$

4.10. Compute Percent Rehandle, p, Percent

$$p = \frac{V_{rI}}{V_T} \times 100, \text{ percent}$$

where

$$V_T = V_1 + V_2 + V_3, \text{ bcy}$$

$$V_T = 16,667 + 41,667 + 16,667 = 75,000, \text{ bcy}$$

$$p = \frac{9,277}{75,000} \times 100 = 12.4\%$$

4.11. Estimate Average Swing Angles for Each Dragline Pass

The values of the average swing angles computed graphically for the horseshoe method are shown in Table H-1.

4.12. Compute Swings Per Hour for Each Dragline Pass, S_i

Swings per hour are computed for each pass from the graph shown in Figure H-3. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging overburden and interburden material. Rehandle is estimated to have 20 second dig time and 4 second delay in one cycle. Swing cycles per hour for each pass are shown in Table H-1.

4.13. Compute Bucket Carrying Capacity for Each Pass, C_i , bcy

$$C_i = bf_i / (1 + \delta) , \text{ bcy}$$

where, for this case

$$f_1 = 0.8 \text{ sandy clay}$$

$$f_2 = 0.80 \text{ shale and sandstone}$$

$$f_3 = 0.5 \text{ interburden shale}$$

$$f_4 = 0.95 \text{ rehandle}$$

4.14. Compute Time Required to Dig Material at Each Pass, T_i , Hours

$$T_i = V_i / (C_i S_i) , \text{ hrs.}$$

4.15. Compute Total Time to Complete One Block, T, Hours

$$T = \sum T_i , \text{ hrs.}$$

4.16. Compute Scheduled Time to Complete One Block, T_S , Hours

$$T_S = T / pq , \text{ hrs.}$$

4.17. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_M / T_S , \text{ bcy}$$

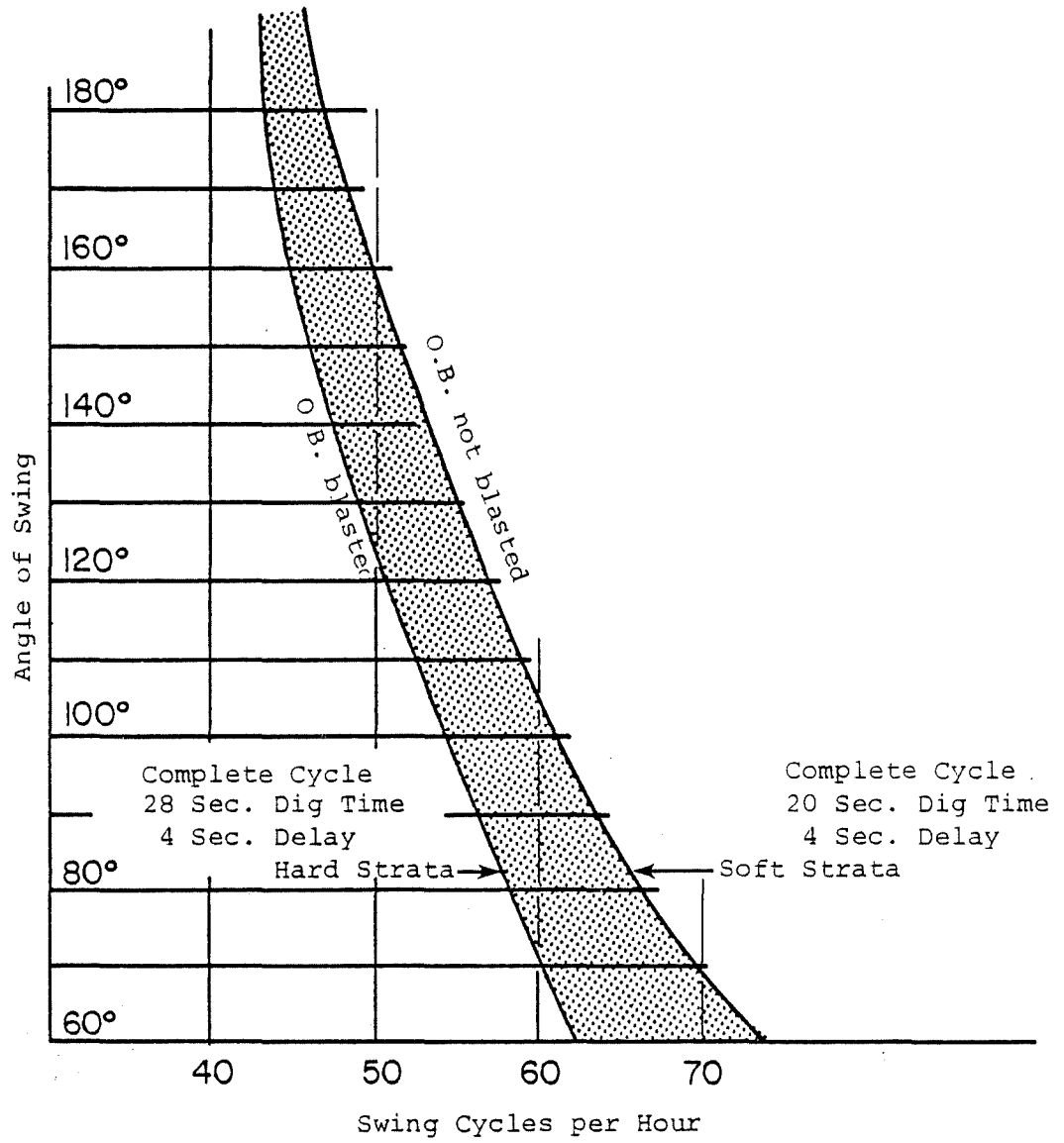


Figure H-3. Swing Cycles per Hour vs. Angle of Swing

4.18. Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy.}$$

where

$$H_M = 660 \text{ hrs.: scheduled hours per month.}$$

4.19. Compute Dragline Annual Production, P_Y , bcy

$$P_Y = 12 \times P_M, \text{ bcy.}$$

Table H-1 summarizes the results of the production computation for the base case - single dragline horseshoe method.

5. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables H-2 and H-3 show detailed estimated hourly ownership and operating costs for dragline and dozer.

The total estimated hourly ownership and operating costs for the dragline and dozer are \$679.90.

6. Estimate Stripping Cost Per Bank Cubic Yard

Estimated stripping costs per bank cubic yard removed by dragline with 400 H.P. dozer used as a utility machine is \$0.50.

Table H-1. Production Rate Calculation for Base Case -
Single Dragline Horseshoe Method

| Output Variable | Dig Component for One Block | | | |
|--|-----------------------------|--------------------------|-----------------------|--------------------|
| | Overburden Side Bench | Overburden Solid Bank | Interburden Pass 2 | Rehandle Pass 2 |
| Volume of Material, bcy. | 16,667 | 41,667 | 16,667 | 9,277 |
| Average Swing Angle, degrees | 90 | 90 | 180 | 120 |
| Swing Cycles Per Hour | 56 | 56 | 43 | 58 |
| Bucket Carrying Capacity, bcy. | 46 | 46 | 29 | 55 |
| Required Digging Time, hours | 6.47 | 16.2 | 13.4 | 2.9 |
| Total Time to Complete One Block, hours | 38.97 | | | |
| Scheduled Time for One Block, hours | 54.55 | | | |
| Production Per Scheduled Hour, bcy. | 1,374 | | | |
| Monthly Production, bcy. | 907,400 | | | |
| Annual Production, bcy. | 10,889,000 | | | |

Table H-2. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | 313.82 |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | 306.93 |
| <u>Total Ownership and Operating Costs</u> | 620.75 |

Table H-3. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.45 |
| Total Ownership Costs | <u>16.59</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | 14.00 |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | <u>59.15</u> |

APPENDIX J
CENTRAL REGION SITE
SINGLE DRAGLINE MINING CONCEPT
ELEVATED BENCH METHOD

APPENDIX J
CENTRAL REGION SITE
SINGLE DRAGLINE MINING CONCEPT
ELEVATED BENCH METHOD

1. Introduction

This Appendix contains production and stripping cost estimates for the single dragline mining concept - elevated bench method - developed for the Central Region.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

| | | | |
|----------------|---|-------------|--|
| W | = | 200 feet: | pit width |
| D ₁ | = | 70 feet: | total overburden depth |
| D ₂ | = | 20 feet: | interburden thickness |
| D ₃ | = | 50 feet: | height of dragline bench |
| D ₄ | = | 20 feet: | thickness of overburden clay strata |
| D ₅ | = | 30 feet: | thickness of overburden sandstone strata |
| D ₆ | = | 20 feet: | thickness of overburden shale strata |
| S ₁ | = | 72 inches: | thickness of upper coal seam |
| S ₂ | = | 48 inches: | thickness of lower coal seam |
| φ | = | 70 degrees: | highwall angle |
| θ | = | 37 degrees: | natural angle of repose |
| α | = | 50 degrees: | angle of oversteepened spoil |
| δ | = | 30 percent: | average spoil factor |
| L | = | 150 feet: | length of block |

R_d = 282 feet: dragline dump radius
 R_{ef} = 232 feet: dragline effective radius
 H_d = 139 feet: dragline dump height
 b = 75 cubic yards: dragline bucket capacity
 q = 0.83: dragline utilization
 p = 0.85: dragline availability

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

H_S = height of spoil pile above the pit floor, feet
 V_i = volume of dig component i , bcy.
 V_r = volume of rehandle material, bcy.
 S_i = swings per hour
 C_i = bucket carrying capacity, bcy.
 T_i = time to dig component i , hrs.
 T = operating time to dig one block, hrs.
 T_S = scheduled time to dig one block, hrs.
 P_H = dragline production rate, bcy.
 P_M = dragline monthly production, bcy.
 P_Y = dragline annual production, bcy.

4. Procedure for Estimating Production Rate for the Single Dragline Mining Concept - Elevated Bench Method

A procedure for estimating the production rate for a single dragline using the elevated bench method is given below. Figure J-1 illustrates the operating procedure for this mining concept.

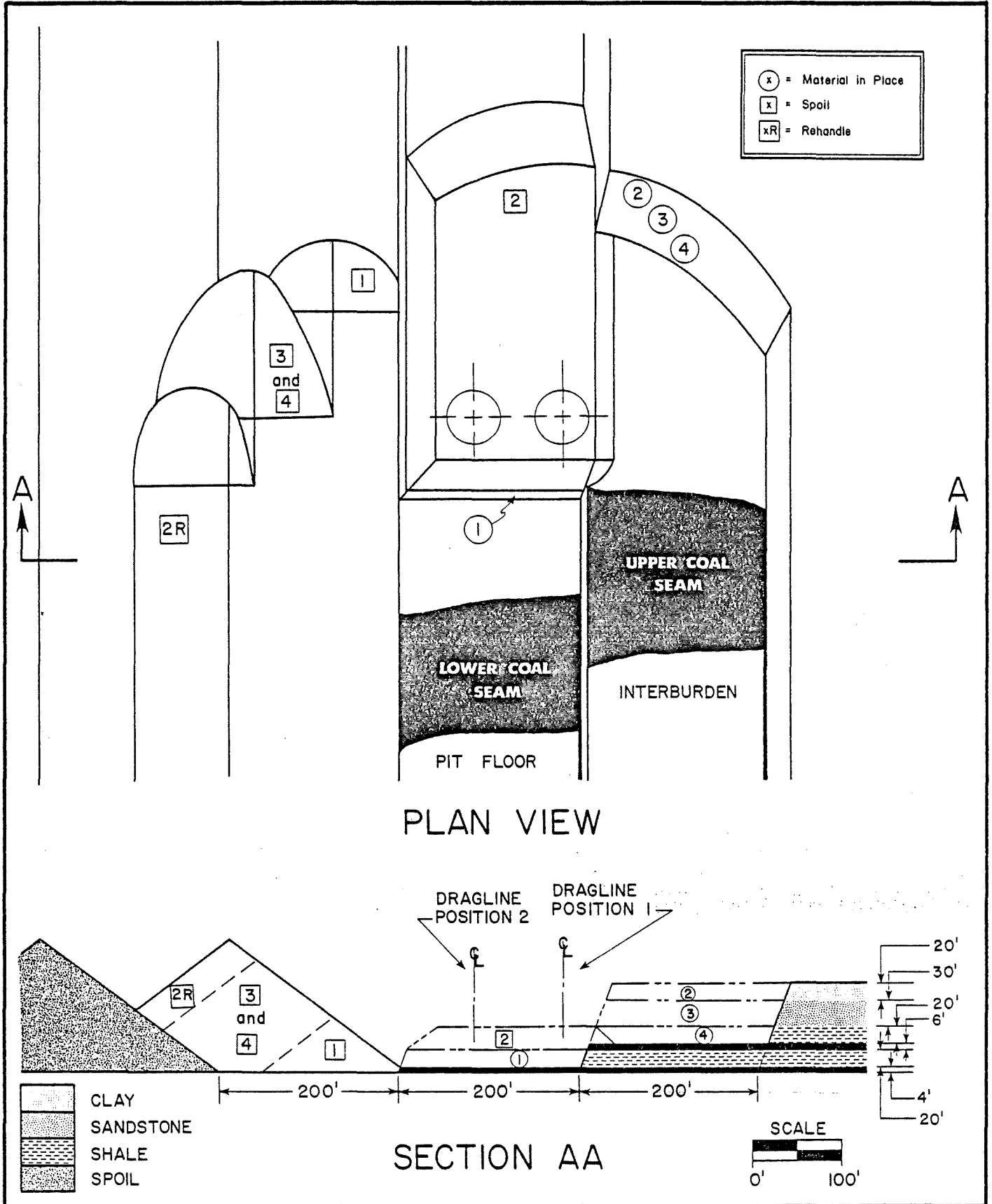


Figure J-1. Single Dragline Mining Concept - Elevated Bench Method Operating Procedure

4.1. Compute Final Spoil Pile Height, H_S , Feet

$$H_S = (D_1 + D_2)(1 + \delta) + W \tan \theta/4, \text{ ft.}$$

$$H_S = (70 + 20)(1 + 0.30) + 200 \tan 37^\circ/4 = 155 \text{ ft.}$$

4.2. Check for Dragline Dump Height, H_d , Feet

$$H_d \geq H_S - D_3$$

139' > 104' -- dragline has sufficient dump height.

4.3. Compute Dig Volumes of Each Component, V_i , bcy

- Volume of interburden, V_I , bcy.

$$V_I = D_2 \times W \times L/27, \text{ bcy.}$$

$$V_I = 20 \times 200 \times 150/27 = 22,222, \text{ bcy.}$$

- Volume of overburden clay material, V_2 , bcy.

$$V_2 = D_4 \times W \times L/27, \text{ bcy.}$$

$$V_2 = 20 \times 200 \times 150/27 = 22,222, \text{ bcy.}$$

- Volume of shale and sandstone overburden material, V_3 , bcy.

$$V_3 = (D_5 + D_6) \times W \times L/27, \text{ bcy.}$$

$$V_3 = (20 + 30) \times 200 \times 150/27 = 55,555, \text{ bcy.}$$

- Volume of rehandle material, V_r , bcy.

$$V_r = V_2 = 22,222, \text{ bcy.}$$

4.4. Compute Percent Rehandle, p , Percent

$$p = \frac{V_r \times 100}{(V_1 + V_2 + V_3)}, \text{ percent}$$

$$p = \frac{22,222}{100,000} \times 100 = 29\%$$

4.5. Estimate Average Swing Angles for Each Dig Component

Average swing angles for each dig component are estimated graphically and are shown in Table J-1.

4.6. Compute Swings Per Hour for Each Dig Component, S_i

Swings per hour are computed for each dig component from the graph shown in Figure J-2. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging overburden and interburden material. Rehandle is estimated to have 20 second dig time and 4 second delay in one cycle. Swing cycles per hour for each dig component are shown in Table J-1.

4.7. Compute Bucket Carrying Capacity for Each Dig Component, C_i , bcy

$$C_i = bf_i / (1 + \delta) , \text{ bcy.}$$

where, for this case

$$f_1 = 1.0 \text{ clay}$$

$$f_2 = 0.8 \text{ sandstone}$$

$$f_3 = 0.85 \text{ shale}$$

$$f_4 = 0.95 \text{ rehandle}$$

4.8. Compute Time Required to Dig Material for Each Dig Component, T_i , Hours

$$T_i = V_i / (C_i S_i) , \text{ hrs.}$$

4.9. Compute Total Time to Complete One Block, T , Hours

$$T = \sum T_i , \text{ hrs.}$$

4.10. Compute Scheduled Time to Dig One Block, T_S , Hours

$$T_S = T / pq , \text{ hrs.}$$

4.11. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_M / T_S , \text{ bcy.}$$

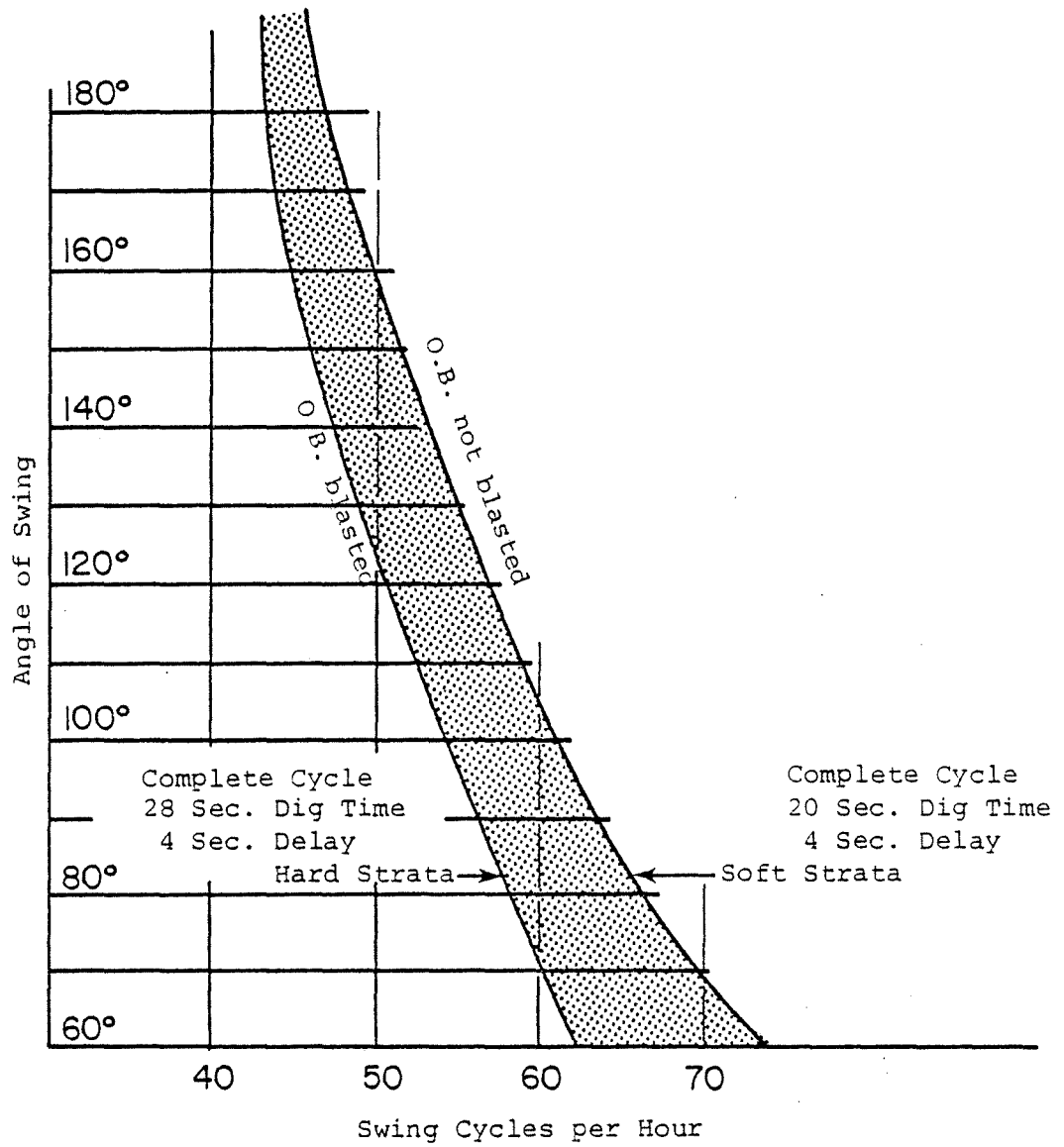


Figure J-2. Swing Cycles per Hour vs. Angle of Swing

4.12. Compute Dragline Monthly Production, bcy

$$P_M = P_H \times H_M , \text{ bcy.}$$

$$H_M = 660 \text{ hrs.: scheduled hours per month}$$

4.13. Compute Dragline Annual Production, P_y, bcy

$$P_Y = P_M \times 12 , \text{ bcy.}$$

Table J-1 summarizes the results of the computations for the single dragline mining concept - elevated bench method.

5. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables J-2 and J-3 show detailed estimated hourly ownership and operating costs for dragline and dozer.

The total estimated hourly ownership and operating costs for the dragline and dozer used as a utility machine are \$679.90.

6. Estimate Stripping Cost Per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by dragline with a 400 HP dozer used as a utility machine is \$0.43.

Table J-1. Production Rate Calculation for Single Dragline
Mining Concept - Elevated Bench Method

| Output Variable | Dig Component for One Block | | | |
|---|-----------------------------|---------------------|---------------------------|----------|
| | Interburden | Side Bench, Clay | Side Bench, Snd. & Sh. | Rehandle |
| Volume of Material, bcy. | 22,222 | 22,222 | 55,555 | 22,222 |
| Average Swing Angle, degrees | 95 | 90 | 150 | 110 |
| Swing Cycles Per Hour | 55 | 63 | 50 | 59 |
| Bucket Carrying Capacity, bcy. | 49 | 57 | 48 | 55 |
| Required Digging Time, hours | 8.25 | 6.19 | 23.14 | 6.88 |
| Total Time to Complete One Block, hours | 44.43 | | | |
| Scheduled Time for One Block, hours | 62.2 | | | |
| Production Per Scheduled Hour, bcy. | 1,607 | | | |
| Monthly Production, bcy. | 1,061,000 | | | |
| Annual Production, bcy. | 12,732,000 | | | |

Table J-2. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | <u>313.82</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | <u>306.93</u> |
| <u>Total Ownership and Operating Costs</u> | <u>620.75</u> |

Table J-3. Estimated Hourly Ownership and Operating Costs
 Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>4.45</u> |
| Total Ownership Costs | 16.59 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | <u>14.00</u> |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | 59.15 |

APPENDIX K

CENTRAL REGION SITE
TANDEM MINING CONCEPT
DRAGLINE/DOZER SYSTEM

APPENDIX K
CENTRAL REGION SITE
TANDEM MINING CONCEPT
DRAGLINE/DOZER SYSTEM

1. Introduction

This Appendix contains production and stripping cost estimates of the tandem mining concept - Dragline/Dozer System - developed for the Central Region.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

- W = 180 feet: pit width
- D_1 = 70 feet: total overburden depth
- D_2 = 20 feet: interburden thickness
- D_3 = 70 feet: depth of dragline bench
- D_4 = 20 feet: thickness of overburden clay strata
- D_5 = 30 feet: thickness of overburden sandstone strata
- D_6 = 20 feet: thickness of overburden shale strata
- S_1 = 72 inches: thickness of upper coal seam
- S_2 = 48 inches: thickness of lower coal seam
- ϕ_1 = 70 degrees: highwall angle overburden
- ϕ_2 = 45 degrees: highwall angle interburden
- θ = 37 degrees: natural angle of repose
- α = 55 degrees: angle of oversteepened spoil
- δ = 30 percent: average spoil factor

L = 100 feet: length of block
 R_d = 282 feet: dragline dump radius
 R_{ef} = 239 feet: dragline effective radius
 H_d = 139 feet: dragline dump height
 b = 75 cubic yards: dragline bucket capacity
 q = 0.83: dragline utilization
 p = 0.85: dragline availability

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

H_1 = Height of overburden spoil pile, feet
 H_2 = Average height of interburden spoil pile, feet
 H_S = Height of final spoil pile, feet
 R_{nr} = Dragline reach for no rehandle from highwall, feet
 V_i = Volume of dig component i , bcy.
 V_r = Volume of rehandle material, bcy.
 S_i = Swings per hour
 C_i = Bucket carrying capacity, bcy.
 T_i = Time to dig component i , hours
 T = Operating time to dig one block, hours
 T_S = Elapsed time to dig one block, hours
 P_H = Dragline production rate, bcy.
 P_M = Dragline monthly production, bcy.
 P_Y = Dragline annual production, bcy.

- H_{1Z} = Dozing level for start of rehandle, feet
 H_{2Z} = Dozing level for start of uphill pushing, feet
 H_{SI} = Height of final interburden spoil pile, feet
 D_{Zi} = Average dozing distance, feet
 A_{Zi} = Area of dig component i, feet²
 R_{Zi} = Rehandle area i, feet²

4. Procedure for Estimating Production Rate for the Dragline/Dozer Tandem System

A procedure for estimating the production rate for a dragline and dozer working in tandem is given below. Figures K-1 and K-2 illustrate the operating procedure for the tandem system.

4.1. Estimate Dragline Production Rate

4.1.1. Height of Overburden Spoil Pile, H_1 , Feet

$$D_1W(1 + \delta) = H_1W - W^2 \tan \alpha/4$$

or

$$H_1 = D_1(1 + \delta) + W \tan \alpha/4, \text{ ft.}$$

$$H_1 = 70 \times 1.3 + 180 \tan 37^\circ/4 = 125, \text{ ft.}$$

4.1.2. Average Height of Interburden Spoil Pile, H_2 , Feet

$$\begin{aligned} \text{Spoil area} &= H_2W + H_2(H_2/\tan \phi_1 + H_2/\tan \alpha)/2 \\ &= H_2W + H_2^2(1/\tan \phi_1 + 1/\tan \alpha)/2 \end{aligned}$$

$$\text{Inplace interburden: } D_2W(1 + \delta)$$

Solving for H_2 :

$$H_2 = \left(-W \pm \sqrt{W^2 + 4aD_2W(1 + \delta)} \right) / 2a, \text{ ft.}$$

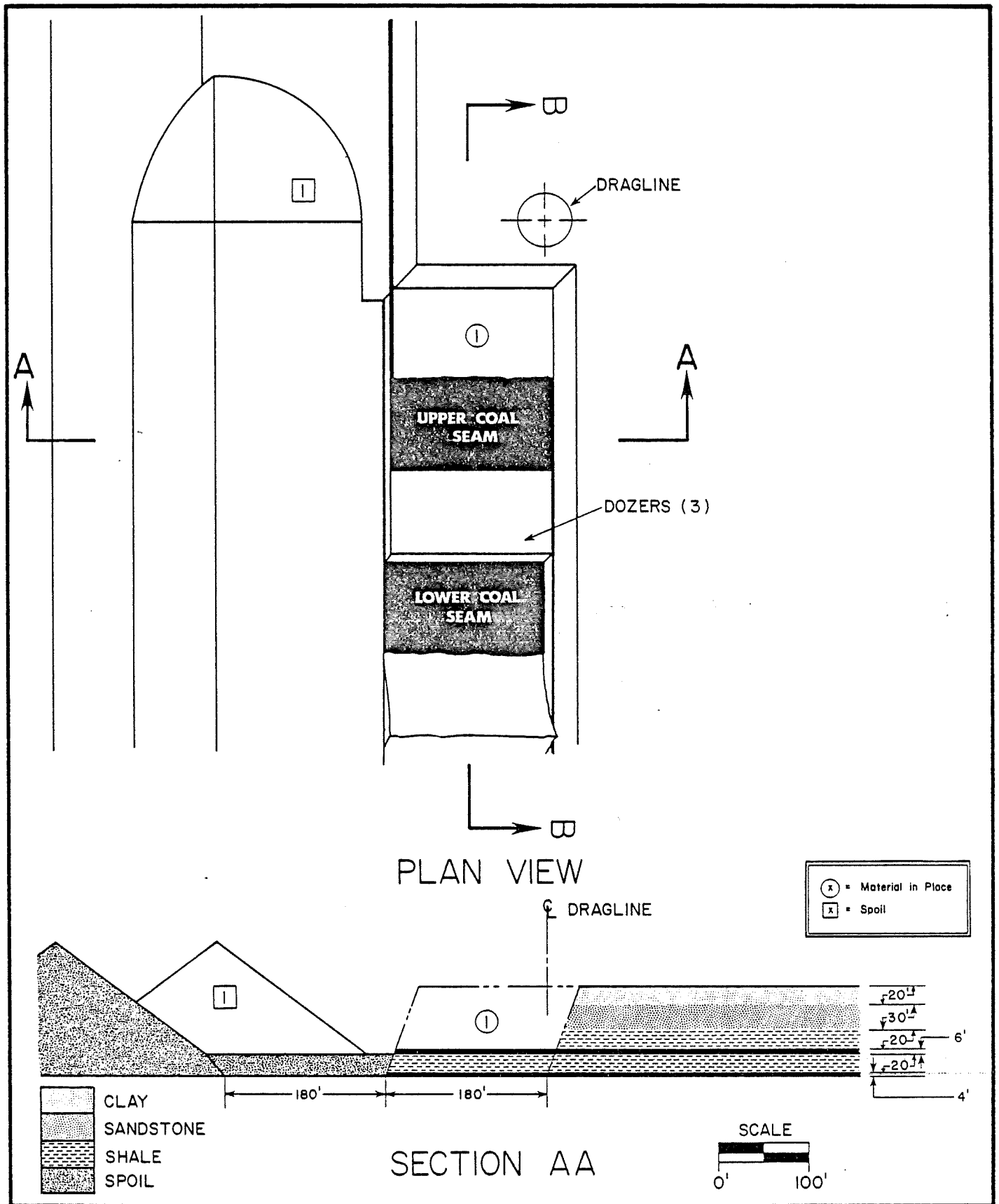


Figure K-1. Dragline Removes the Overburden in Dragline/Dozer Tandem System

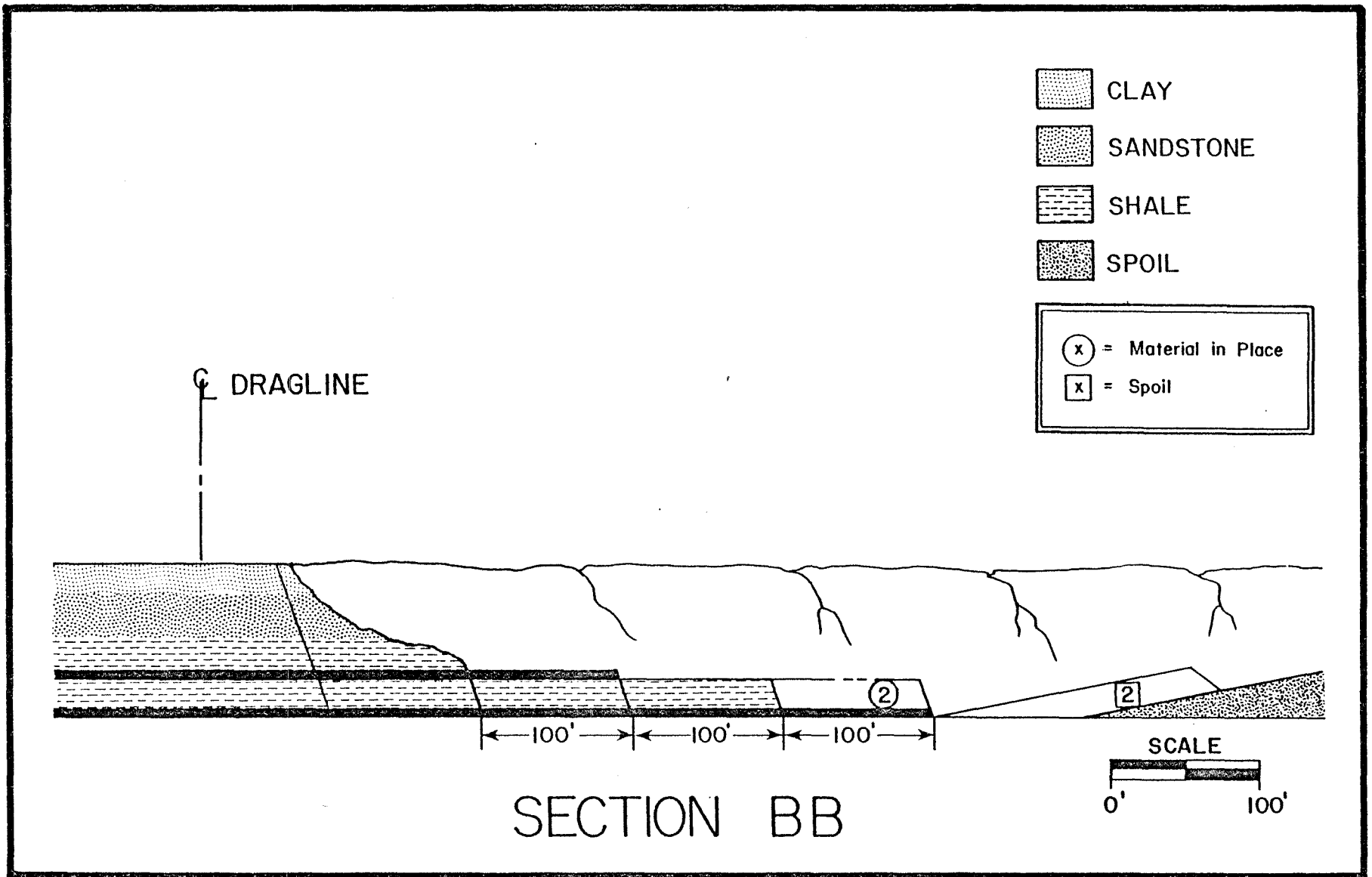


Figure K-2. Dozer Removes the Interburden in Dragline/Dozer Tandem System (Pushback)

where

$$a = (1/\tan \theta + 1/\tan \alpha)/2 = 0.0952$$

$$H_2 = (-180 \pm 224.11)/1.9 = 23, \text{ ft.}$$

Final height of spoil pile is:

$$H_S = H_1 + H_2 = 125 + 23 = 148, \text{ ft.}$$

4.1.3 Dragline Reach for No Rehandle from the Highwall, R, Feet

$$\begin{aligned} R_{nr} &= (70 + 6 + 20 + 4)/\tan 70^\circ + 23.16/\tan 55^\circ \\ &\quad + 124.91/\tan 37^\circ \\ &= 240, \text{ ft.} \end{aligned}$$

No extended bench needed because $240 \approx 239$, effective dump radius.

4.1.4. Compute Volume of Dig Components, V_i , bcy

- Volume of bank overburden material, V_1 , bcy.

$$V_1 = D_1 \times W \times L/27, \text{ bcy.}$$

$$V_1 = 70 \times 180 \times 100/27 = 46,667, \text{ bcy.}$$

- Volume of rehandle, V_{ROB} , bcy.

$$V_{ROB} = A_{ROB} \times 100/27 \times 1.3, \text{ bcy.}$$

where

$$A_{ROB} = H_2^2 a, \text{ ft.}^2$$

$$A_{ROB} = 23.16^2 \times 0.0952 = 510, \text{ ft.}^2$$

$$V_{ROB} = 510 \times 100/27 \times 1.3 = 1,453, \text{ bcy.}$$

4.1.5. Compute Percent Rehandle, p, Percent

$$p = \frac{V_{ROB} \times 100}{V_1}, \text{ percent}$$

$$p = \frac{1,453 \times 100}{46,000} = 3\%$$

4.1.6. Estimate Average Swing Angles for Each Dig Component

Average swing angles for each dig component are estimated graphically and are shown in Table K-1.

4.1.7. Compute Swings Per Hour for Each Dig Component, S_i

Swings per hour are computed for each dig component from the graph shown in Figure K-3. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging overburden and interburden material. Rehandle is estimated to have 20 second dig time and 4 second delay in one cycle. Swing cycles per hour for each pass are shown in Table K-1.

4.1.8. Compute Bucket Carrying Capacity for Each Dig Component, C_i , bcy

$$C_i = bf_i / (1 + \delta), \text{ bcy.}$$

where; for this concept

$$f = 1.0 \text{ clay}$$

$$f_1 = 0.8 \text{ sandstone}$$

$$f_2 = 0.7 \text{ shale}$$

$$f_3 = 0.85 \text{ rehandle}$$

4.1.9. Compute Time Required to Dig Each Dig Component, T_i , Hours

$$T_i = V_i / (C_i S_i), \text{ hrs.}$$

4.1.10. Compute Total Time to Complete One Block, T , Hours

$$T = \sum T_i, \text{ hrs.}$$

4.1.11. Compute Scheduled Time to Dig One Block, T_S , Hours

$$T_S = T / pq, \text{ hrs.}$$

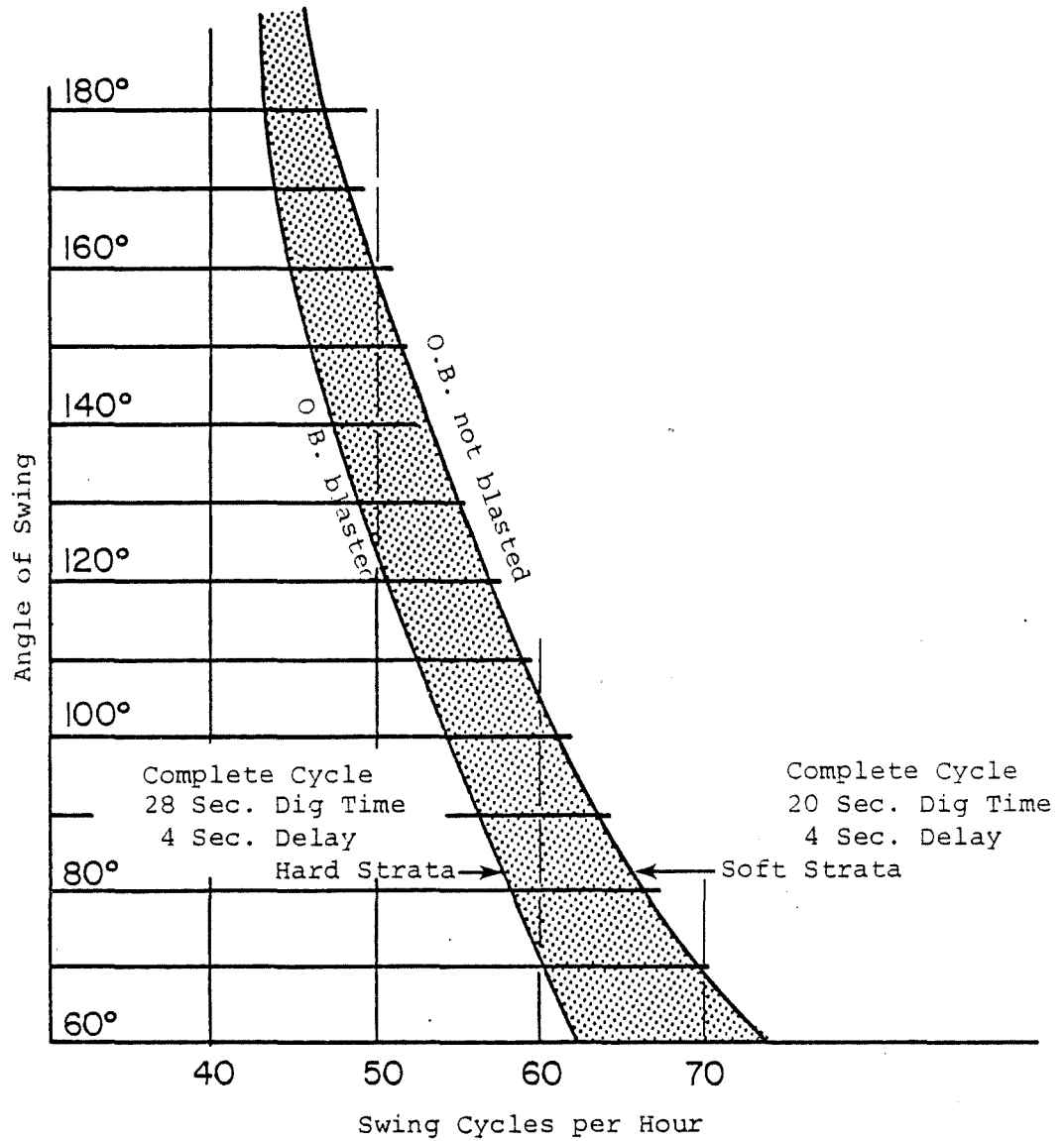


Figure K-3. Swing Cycles per Hour vs. Angle of Swing

4.1.12. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_1/T_S, \text{ bcy.}$$

4.1.13. Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy.}$$

$$H_M = 660 \text{ hrs: scheduled hours per month}$$

4.1.14. Compute Dragline Annual Production, P_Y , bcy

$$P_Y = P_M \times 12, \text{ bcy.}$$

Results of the production computations for the dragline are summarized in Table K-1.

4.2. Dozer Production Rate Estimate

5.2.1. Compute Dozing Level for Start of Rehandle, H_{1Z} , Feet (See Figure K-4)

$$H_1^2 a_1 + H_1(2a_1 S_2 + L(1 + \delta)) + S_2^2 a_1 + S_2^2 / (2 \tan \phi_2) - D_2 L(1 + \delta) = 0$$

where

$$a_1 = (1/\tan \theta - 1/\tan \phi_2) / 2$$

4.2.2. Compute Dozing Level for Start of Uphill Pushing, H_{2Z} , Feet (See Figure K-5)

$$H_{2Z}^2 a_2 + H_{2Z}(S_2/\tan \alpha + L(2 + \delta)) + S_2^2 / (2/\tan \alpha) - b - D_2 L(1 + \delta) = 0$$

where

$$a_2 = (1/\tan \phi_2 + 1/\tan \alpha) / 2$$

$$b = L^2 / 2(1/\tan \theta + 1/\tan \alpha)$$

Check for validity of H_{2Z}

$$A_1 + A_2 + A_3 \leq (D_2 - h - S_2)L \times (1 + \delta)$$

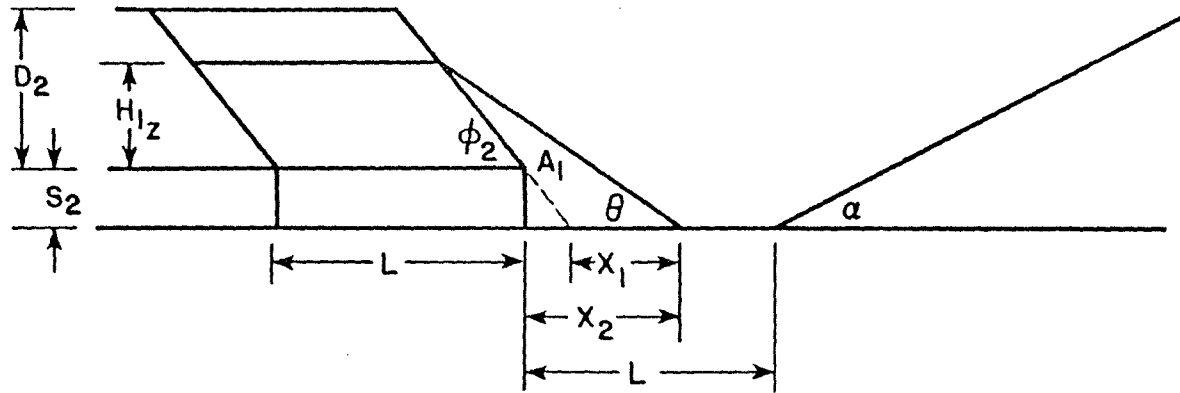


Figure K-4. Dozing level for start of rehandle

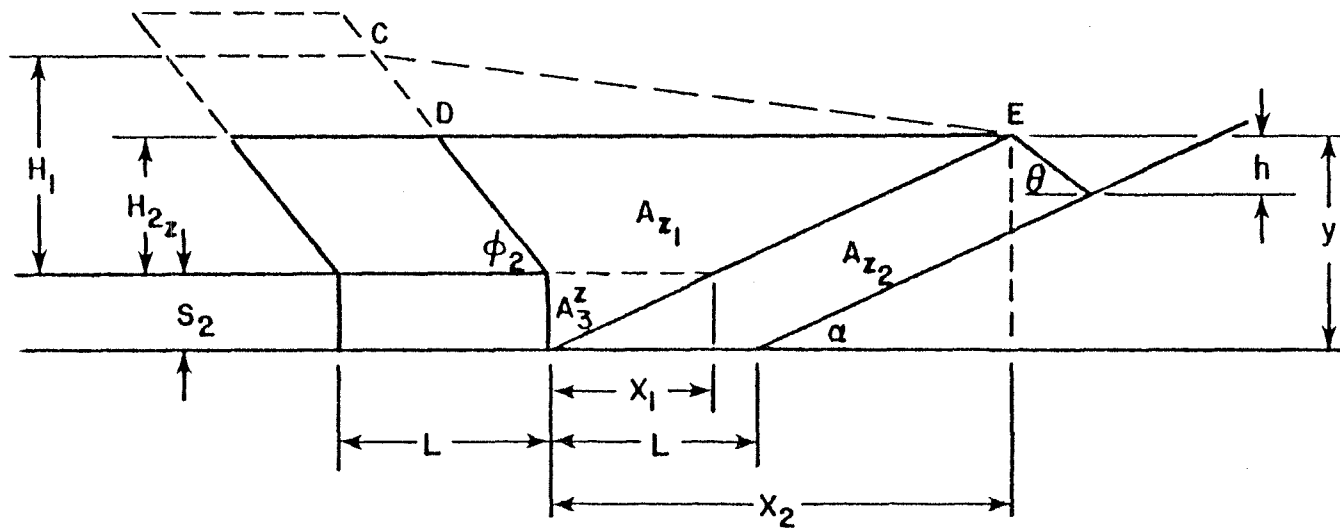


Figure K-5. Dozing level for start of uphill pushing

Table K-1. Dragline Production Rate Calculation for Dragline/Dozer Tandem System

| Output Variable | Dig Component for One Block | |
|---|-----------------------------|----------|
| | Overburden | Rehandle |
| Volume of Material, bcy. | 46,667 | 1,453 |
| Average Swing Angle, degrees | 90 | 57 |
| Swing Cycles Per Hour | 56 | 75 |
| Bucket Carrying Capacity, bcy. | 51 | 55 |
| Required Digging Time, hours | 16.34 | 0.35 |
| Total Time to Complete One Block, hours | 16.69 | |
| Scheduled Time for One Block, hours | 23.37 | |
| Production Per Scheduled Hour, bcy. | 1,997 | |
| Monthly Production Rate, bcy. | 1,318,000 | |
| Annual Production Rate, bcy. | 15,816,000 | |

where

$$A_1 = (h - S_2)^2 (1/\tan \phi_2 + 1/\tan \alpha) / 2 + (h - S_2) S_2 / \tan \alpha, \text{ feet}^2$$

$$A_2 = L \times (h - S_2) - L^2 / (2(1/\tan \theta + 1/\tan \alpha)), \text{ feet}^2$$

$$A_3 = S_2^2 / (2 \tan \alpha), \text{ feet}^2$$

$$h = L / (1/\tan \theta + 1/\tan \alpha), \text{ feet}$$

4.2.3. Compute Height of the Final Interburden Spoil Pile, H_{S_I} , Feet

$$H_{S_I} = D_2 \times (1 + \delta) + L/2(1/\tan \theta + 1/\tan \alpha), \text{ feet}$$

Check for extension of final spoil pile above interburden level, h_e , feet.

$$h_e = H_{S_I} - (D_2 + S_2), \text{ feet}$$

4.2.4. Component 1 - Average Dozing Distance and Area Computations (See Figure K-5)

- Average dozing push distance, D_{Z_1} , feet

$$D_{Z_1} = L, \text{ feet}$$

- Swelled bank area, A_{Z_1} , feet²

$$A_{Z_1} = (D_2 - H_{1Z}) \times L \times (1 + \delta), \text{ feet}^2$$

- Rehandle area, R_{Z_1} , feet²

$$R_{Z_1} = 0, \text{ feet}$$

4.2.5. Component 2 - Average Dozing Distance and Area Computations (See Figure K-5)

- Average dozing push distance, D_{Z_2} , feet

$$D_{Z_2} = \left(D_{Z_{2MIN}} + D_{Z_{2MAX}} \right) / 2, \text{ feet}$$

where

$$D_{Z_{2MIN}} = D_{Z_1} = L, \text{ feet}$$

$$D_{Z_{2MAX}} = L + H_{2Z} / \tan \phi_2 + (H_{2Z} + S_2) / \tan \alpha, \text{ feet}$$

- Swelled bank area, A_{Z_2} , feet²

$$A_{Z_2} = (H_{1Z} - H_{2Z}) \times L \times (1 + \delta), \text{ feet}^2$$

- Rehandle area, R_{Z_2} , feet²

$$R_{Z_2} = (H_{1Z} - H_{2Z}) \times \left(D_{Z_{2MAX}} - L/2 \right), \text{ feet}^2$$

4.2.6. Component 3 - Average Dozing Distance and Area Computations (See Figure K-5)

- Average dozing push distance, D_{Z_3} , feet

$$D_{Z_3} = \left(D_{Z_{3MIN}} + D_{Z_{3MAX}} \right) / 2, \text{ feet}$$

where

$$D_{Z_{3MIN}} = D_{Z_{2MAX}}, \text{ feet}$$

$$D_{Z_{3MAX}} = L + S_2 / \tan \alpha + \left(H_{S_I} - S_2 \right) / \sin \alpha, \text{ feet}$$

- Swelled bank area, A_{Z_3} , feet²

$$A_{Z_3} = H_{2Z} \times L \times (1 + \delta), \text{ feet}^2$$

- Rehandle area, R_{Z_3} , feet²

$$R_{Z_3} = A_1, \text{ feet}^2$$

where

$$A_1 = H_{2Z}^2 (1 / \tan \phi_2 + 1 / \tan \alpha) / 2 + H_{2Z} S_2 / \tan \alpha, \text{ feet}^2$$

4.2.7. Component 4 - Average Dozing Distance and Area Computations

- Average push distance, D_{Z_4} , feet

$$D_{Z_4} = H_{S_I} / \sin \alpha, \text{ feet}$$

- Rehandle area, R_{Z_4} , feet²

$$R_{Z_4} = S_2^2 / (2 \tan \alpha), \text{ feet}^2$$

Dozer production calculations* based on the adjusted dozer hourly figures are shown in Tables K-2 and K-3. Production rate was computed for 50 and 100 foot block length using Adjusted Production Curve for 400 H.P. dozer shown in Figure K-4. Results from 100 foot block length were used to compute total system production rate.

5. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables K-4 and K-5 show detailed estimated hourly ownership and operating costs for dragline and dozer.

The total estimated hourly ownership and operating costs for dragline are \$620.75 and for dozer are \$59.15.

6. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by dragline with dozer used as utility machine is \$0.34. Cost per bank cubic yard when dozer is used as a stripping machine is \$0.31 for 100 foot block. Cost per bank cubic yard for the total system using one dragline, utility dozer and three stripping dozers is \$0.33.

* Note: Since the uphill dozer work involved here does not involve any uphill cutting, performance will not be penalized by the uphill pushing except for the resulting increase in push distances.

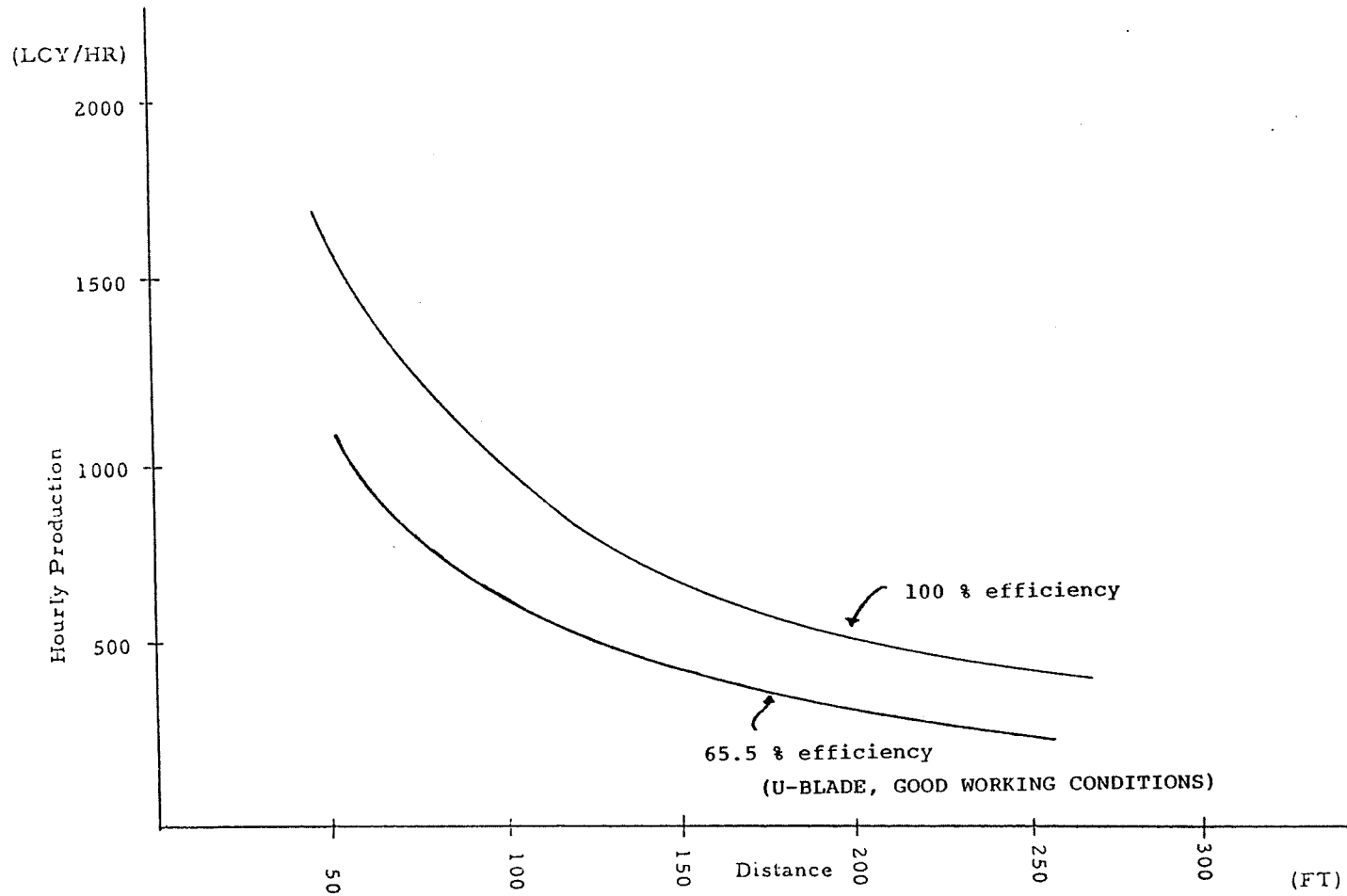


Figure K-4. Production Curve* Adjusted for an Additional 8.2 Seconds to Back-up Highwall to Start New Push

* Based on 400 HP dozer

Table K-2. Dozer Production Rate Calculations for Dragline/Dozer Tandem System

Interburden Thickness: 20 ft.
 Block Length: 50 ft.
 Pit Width: 180 ft.

| Output Variable | Dozed Component | | | | Block |
|--|-----------------|-------|-------|-------|---------------|
| | 1 | 2 | 3 | 4 | |
| Average Dozing Distance, feet | 50 | 89.5 | 165.7 | 152.7 | |
| Production Rate Per Scheduled Hour, bcy. | 1,045 | 699 | 398 | 438 | |
| Bank Volume, lcy. | 611 | 4,156 | 3,900 | 0 | 8,667 |
| Rehandle Volume, lcy. | 0 | 2,525 | 2,820 | 267 | 5,612 (64.7%) |
| Scheduled Time for One Block, hours | 0.58 | 9.56 | 16.87 | 0.61 | 27.62 |
| Production Per Scheduled Hour, bcy. | 241 | | | | |
| Monthly Production Rate, bcy. | 159,000 | | | | |
| Annual Production Rate, bcy. | 1,908,000 | | | | |

Table K-3. Dozer Production Rate Calculations for Dragline/Dozer Tandem System

Interburden Thickness: 20 ft.
 Block Length: 100 ft.
 Pit Width: 180 ft.

| Output Variable | Dozed Component | | | | Block |
|--|-----------------|-------|--------|------|-------------|
| | 1 | 2 | 3 | 4 | |
| Average Dozing Distance, feet | 100 | 145 | 230 | 170 | |
| Production Rate Per Scheduled Hour, bcy. | 655 | 460 | 285 | 397 | |
| Bank Volume, lcy. | 650 | 6,543 | 10,140 | 0 | 17,333 |
| Rehandle Volume, lcy. | 0 | 2,265 | 4,050 | 267 | 6,582 (38%) |
| Scheduled Time for One Block, hours | 0.99 | 19.15 | 49.7 | 0.67 | 70.51 |
| Production Per Scheduled Hour, bcy. | 189 | | | | |
| Monthly Production Rate, bcy. | 125,000 | | | | |
| Annual Production Rate, bcy. | 1,500,000 | | | | |

Table K-4. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | 313.82 |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | 306.93 |
| <u>Total Ownership and Operating Costs</u> | 620.75 |

Table K-5. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.45 |
| Total Ownership Costs | 16.59 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | 14.00 |
| Total Operating Costs | 42.56 |
| <u>Total Ownership and Operating Costs</u> | <u>59.15</u> |

APPENDIX L
CENTRAL REGION SITE
NO DRAGLINE MINING CONCEPT
BWE/CONVEYOR SYSTEM

APPENDIX L
CENTRAL REGION SITE
NO DRAGLINE MINING CONCEPT
BWE/CONVEYOR SYSTEM

1. Introduction

This Appendix contains production and stripping cost estimates of a single mining concept - bucket wheel excavator/conveyor system developed for the Central Region.

2. Input Variables

| | | |
|-----------|---------------|---|
| W | = 120 feet: | pit width |
| D_1 | = 35 feet: | first lift overburden depth, unconsolidated material |
| D_2 | = 35 feet: | second lift overburden depth, unconsolidated material |
| D_3 | = 20 feet: | interburden thickness, soft shale |
| S_1 | = 72 inches: | thickness of upper coal seam |
| S_2 | = 48 inches: | thickness of lower coal seam |
| ϕ | = 55 degrees: | highwall angle |
| θ | = 35 degrees: | natural angle of repose |
| δ | = 30 percent: | average spoil factor |
| R_{d_1} | = 46 feet: | BWE dig radius |
| R_{d_2} | = 82 feet: | BWE dump radius |
| H_{d_1} | = 50 feet: | BWE dig height |
| P_T | = 1570 bcy: | hourly rated output |
| q | = 0.65: | BWE/conveyor system utilization |
| p | = 0.85: | BWE availability |

3. Output Variables

H_S = height of spoil pile above the pit floor, ft.

P_H = BWE production rate per scheduled hour, bcy

P_M = BWE monthly production, bcy

P_Y = BWE annual production, bcy

4. Procedure for Estimating Production Rate for Single Mining Concept - BWE/Conveyor System

A procedure for estimating the production rate for a BWE/Conveyor system is given below. Figure L-1 illustrates the operating procedure of this mining concept.

4.1 Compute Final Spoil Pile Height, H_S , ft.

$$H_S = (D_1 + D_2 + D_3) (1 + \delta), \text{ ft.}$$

$$H_S = (20 + 35 + 35) (1 + 0.3) = 117, \text{ ft.}$$

4.2 Compute BWE Production per Scheduled Hour, P_H , bcy

$$P_H = P_T \times q \times p, \text{ bcy}$$

where P_T = hourly rated output, bcy

4.3 Compute BWE monthly production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy}$$

H_M = 660 hrs.: scheduled working hours per month

4.4 Compute BWE annual production, P_Y , bcy

$$P_Y = P_M \times 12, \text{ bcy}$$

Results from the computation of BWE production rates are summarized in Table L-1.

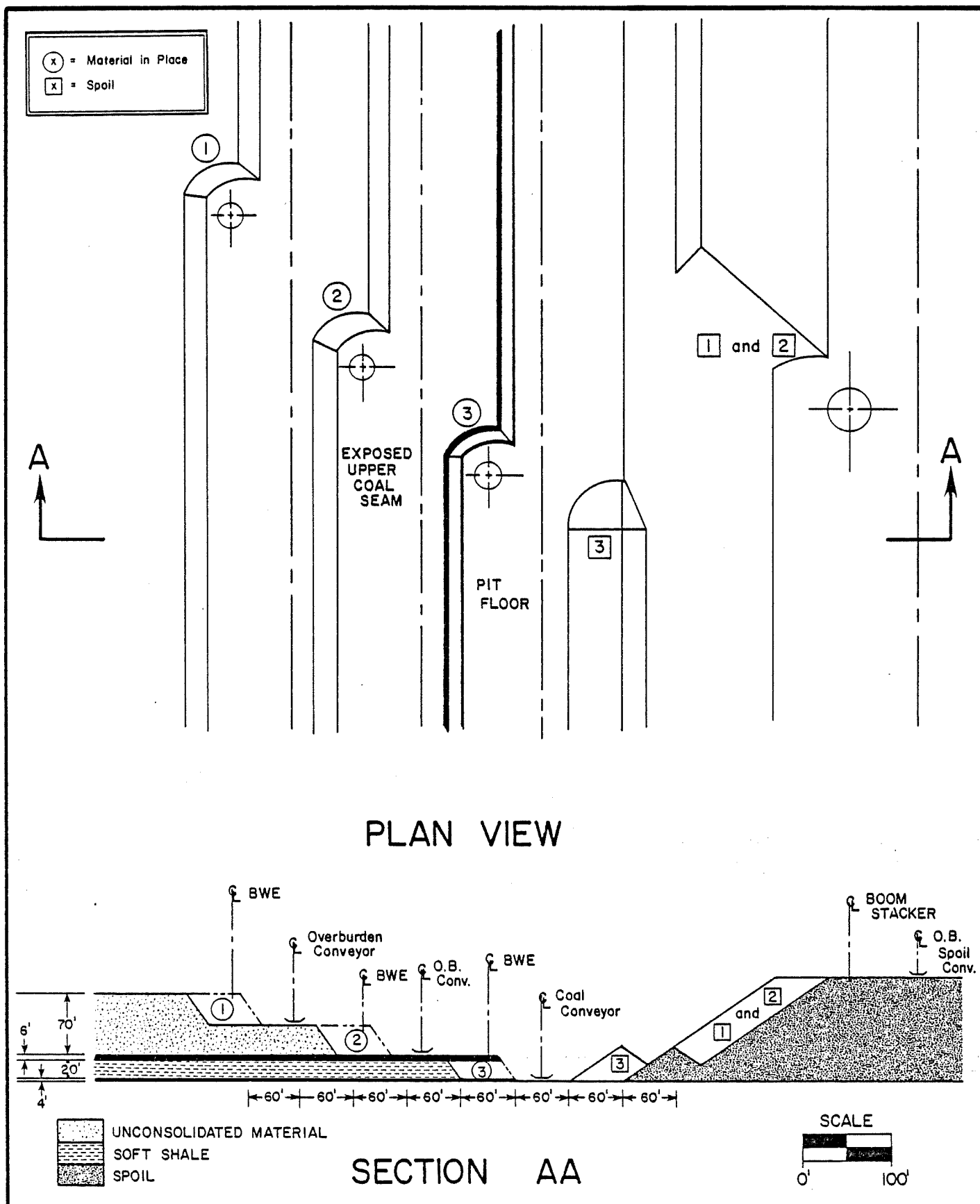


Figure L-1. Operating Procedure for BWE/Conveyor System

Table L-1. BWE Production Rate Calculation

| OUTPUT VARIABLE | OVERBURDEN (2-BWE) | INTERBURDEN (1-BWE) |
|---|-----------------------|------------------------|
| Hourly rated output, bcy | 3,140 | 1,570 |
| Production per scheduled hour, bcy | 1,734 | 578 |
| Production per month, bcy | 1,144,000 | 382,000 |
| Production per year, bcy | 13,728,000 | 4,584,000 |
| Total overburden and interburden production per year, bcy | 18,312,000 | |

5. Estimate Ownership and Operating Costs

Capital and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables L-2 and L-3 show detailed estimated ownership and operating costs for BWE/conveyor system. The total O&O costs for BWE/conveyor system with 2 dozers is \$1,480.59.

6. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by BWE/conveyor system with two (2) dozers used as utility machines is \$0.57.

Table L-2. Estimated Hourly Ownership and Operating Costs: BWE/Conveyor System

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | |
| 3 - BWE | \$20,550,000 |
| 1 - Stacker with Tripper Car | 5,000,000 |
| 1 - Conveyor Gantry Bridge | 550,000 |
| 1 - Belt Wagon | 1,640,000 |
| 2 - Conveyors | <u>5,535,000</u> |
| Total Price | 33,275,000 |
| Salvage Value @ 10% | 3,327,500 |
| Depreciated Amount | 29,947,500 |
| Life of Machine | 20 years @ 8,030 hrs/yr |
| Depreciation Cost | <u>186.47</u> |
| 2. Interest, Taxes and Insurance | 455.82 |
| 20% of Average Investment | |
| Total Ownership Costs | <u>642.29</u> |
| <u>Operating Costs</u> | |
| 3 - BWE | \$327 |
| 1 - Stacker | 85 |
| 1 - Conveyor Gantry Bridge | 30 |
| 1 - Belt Wagon | 48 |
| 2 - Conveyors | 230 |
| Total Operating Costs | <u>720.00</u> |
| <u>Total Ownership and Operating Costs</u> | <u>1,362.29</u> |

Table L-3. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|----------------------------------|--------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | - |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>4.45</u> |
| Total Ownership Costs | 16.59 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | |
| | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/hr. | <u>14.00</u> |
| Total Operating Costs | <u>42.56</u> |
| Total O&O Costs | 59.15 |

APPENDIX M

WESTERN REGION SITE

SINGLE DRAGLINE HORSESHOE METHOD

APPENDIX M
WESTERN REGION SITE
SINGLE DRAGLINE HORSESHOE METHOD

1. Introduction

This Appendix contains production and stripping cost analysis of the Single Dragline Horseshoe Method developed for the Western Region.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

| | | | |
|----------------|---|-------------|--|
| W | = | 200 feet: | pit width |
| D | = | 125 feet: | total overburden and interburden depth |
| D ₁ | = | 65 feet: | total overburden thickness |
| D ₂ | = | 60 feet: | interburden thickness, shale |
| D ₃ | = | 10 feet: | thickness of overburden clay strata |
| D ₄ | = | 20 feet: | thickness of overburden sandstone strata |
| D ₅ | = | 35 feet: | thickness of overburden shale strata |
| S ₁ | = | 360 inches: | thickness of upper coal seam |
| S ₂ | = | 240 inches: | thickness of lower coal seam |
| φ | = | 75 degrees: | highwall angle |
| θ | = | 37 degrees: | natural angle of repose |

α = 55 degrees: angle of oversteepened spoil
 δ = 30 percent: average spoil factor
 L = 150 feet: length of block
 R_d = 282 feet: dragline dump radius
 R_{ef} = 232 feet: dragline effective radius
 H_d = 139 feet: dragline dump height
 b = 75 cubic yards: dragline bucket capacity
 w_b = 10 feet: width of bucket
 q = 0.83: dragline utilization
 p = 0.85: dragline availability

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

H_S = Height of spoil bench above the pit floor, feet
 H_F = Height of the final spoil pile, feet
 h_l = Height of the spoil pile toe from the overburden pass, feet
 H_I = Height of spoil pile from the interburden pass, feet
 V_i = Dig volumes of each dragline pass, bcy.
 V_{rI} = Volume of rehandle material, interburden pass, bcy.
 S_i = Swing cycles per hour for each pass
 C_i = Bucket carrying capacity for each pass, bcy.
 T_i = Time required to dig material of each pass, hours
 T = Time to complete one block, hours
 T_S = Scheduled time to dig one block, hours

P_H = Dragline production rate per scheduled hours,
bcy.

P_M = Dragline monthly production, bcy.

P_Y = Dragline annual production, bcy.

4. Procedure for Estimating Production Rate for the Base Case - Single Dragline Method

A procedure for estimating the production rate for a single dragline using the horseshoe method is given below. Figure M-1, M-2 and M-3 illustrate the operating procedures of the base case.

4.1. Height of Final Spoil Pile, H_F , Feet

$$H_F = D(1 + \delta) + W \tan \theta/4, \text{ ft.}$$

$$H_F = 125(1 + 0.30) + 200 \tan 37^\circ/4 = 200, \text{ ft.}$$

4.2. Height of the Spoil Pile from Overburden Pass, H_S , Feet

(See Figure M-1.)

$$H_S = W_1 \tan \theta, \text{ ft.}$$

where

$$W_1(\tan \theta - a) + W_1 a - (W^2 a/4 + A) = 0$$

$$a = 1/(1/\tan \phi + 1/\tan \theta)$$

$$A = (D_1 + D_2)W(1 + \delta), \text{ ft.}^2$$

$$W_1 = 146.2, \text{ ft.}$$

$$H_S = 146.2 \tan 37^\circ = 110.2, \text{ ft.}$$

$$H_S = 110, \text{ ft.}$$

4.3. Height of the Spoil Pile Toe from the Overburden Pass, h_1 , Feet

$$h_1 = (W_1 - W/2)a, \text{ ft.}$$

$$h_1 = 29, \text{ ft.}$$

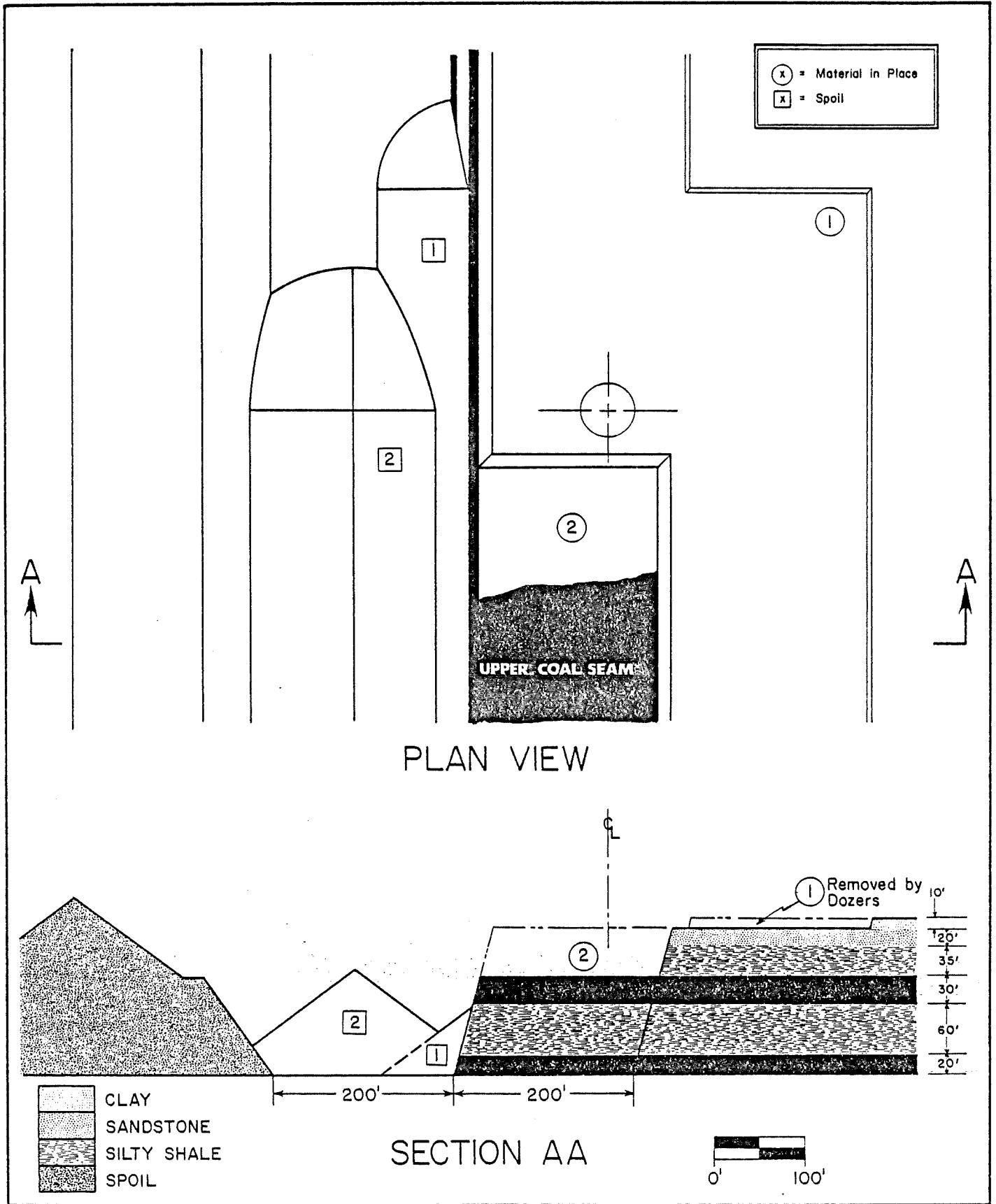


Figure M-1. Dragline Removing Overburden:
 Single Dragline Horseshoe Method

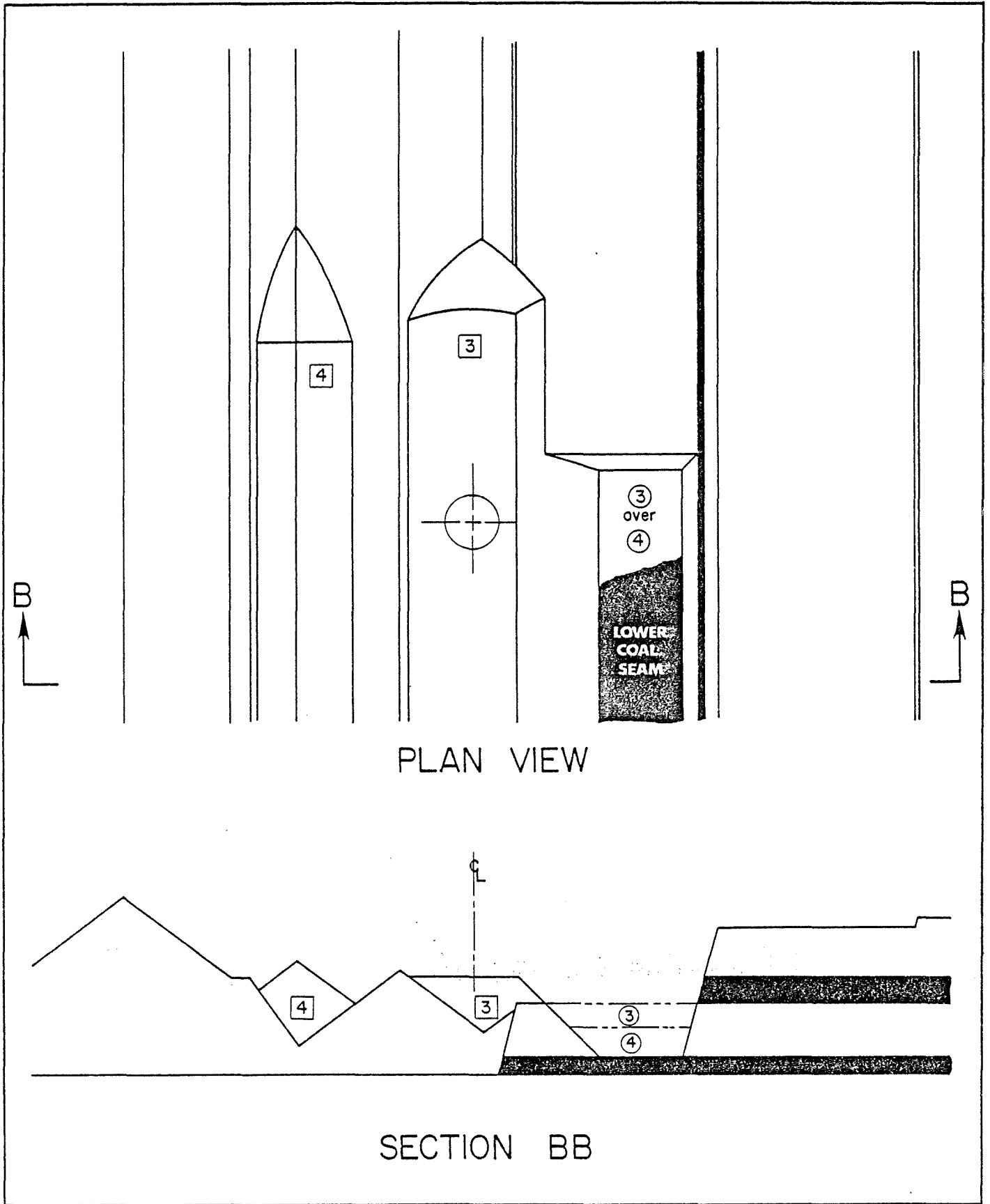


Figure M-2. Dragline Removing Interburden:
Single Dragline Horseshoe Method

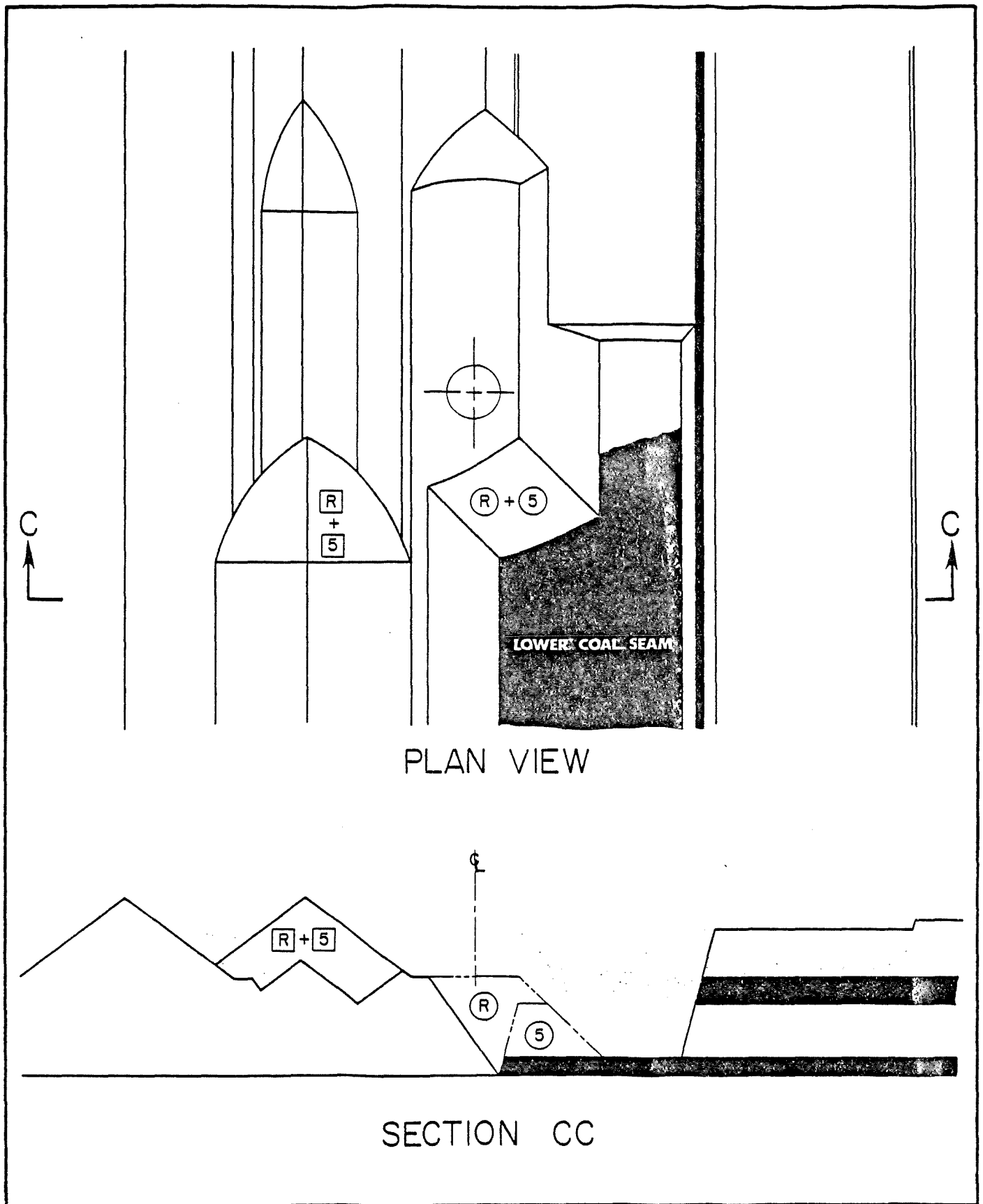


Figure M-3. Dragline Removing Rehandle:
Single Dragline Horseshoe Method

4.4. Preparation of the Spoil Bench, H_B , Feet

Height of the spoil bench to be the same as that of the spoil pile after the overburden pass.

$$H_B = H_S = 110, \text{ ft.}$$

4.5. Check for Required Dump Height on the Interburden Pass, H_I , Feet

$$H_I = H_F - H_S, \text{ ft.}$$

$$H_I = 200 - 110 = 90, \text{ ft.}$$

90' > 139' -- dragline has sufficient dump height.

4.6. Width of the Spoil Bench, W_S , Feet

Width over dragline shoes: 83 ft.
Safety distance: 20 ft. each side.

$$W_S = 83 + 2 \times 20 = 123, \text{ ft.}$$

4.7. Area to Complete the Spoil Bench, A_S , Feet²

$$A_S = A_1 + A_2, \text{ ft.}^2$$

where

$$A_1 = W_S (H_S - D_2 - S_2), \text{ ft.}^2$$

$$A_2 = \left[W_S - (H_S - D_2 - S_2) / \tan \alpha \right] (D_2 + S_2 - h_1) / 2, \text{ ft.}^2$$

$$A_1 = 3,690, \text{ ft.}^2$$

$$A_2 = 2,116, \text{ ft.}^2$$

$$A_S = 5,806, \text{ ft.}^2$$

4.8. Amount of Interburden Required to Complete the Spoil Bench, A_I , Feet²

$$A_I = A'_S / (1 + \delta), \text{ ft.}^2$$

$$A_I = 5,806 / (1 + 0.3) = 4,466, \text{ ft.}^2$$

4.9. Amount of Rehandle on the Interburden Pass, A_R , Feet²

$$A_R = A_1 + A_2, \text{ ft.}^2$$

$$A_1 = 30(100.4 + 119.4) / 2 = 3,297, \text{ ft.}^2$$

$$A_2 = 77 \times 80 / 2 = 3,080, \text{ ft.}^2$$

$$A_R = 6,377, \text{ ft.}^2$$

4.10. Dig Volumes of Each Dragline Pass, V_i , bcy

- Volume of dozed material, V_{DZ} , bcy.

$$V_{DZ} = D_3 \times W \times L / 27, \text{ bcy.}$$

$$V_{DZ} = 10 \times 200 \times 150 / 27 = 11,111, \text{ bcy.}$$

- Volume of overburden moved by dragline, V_1 , bcy.

$$V_1 = (D_4 + D_5) \times W \times L / 27, \text{ bcy.}$$

$$V_1 = 55 \times 200 \times 150 / 27 = 61,111, \text{ bcy.}$$

- Volume of interburden to build the spoil bench, V_2 , bcy.

$$V_2 = A_S \times L / 27, \text{ bcy.}$$

$$V_2 = 4,466 \times 150 / 27 = 24,811, \text{ bcy.}$$

- Volume of remaining interburden, V_3 , bcy.

$$V_3 = V_I - V_2, \text{ bcy.}$$

where

$$V_I = D_2 \times W \times L / 27, \text{ bcy.}$$

$$V_I = 60 \times 200 \times 150 / 27 = 66,667, \text{ bcy.}$$

$$V_3 = 66,667 - 24,811 = 41,856, \text{ bcy.}$$

- Volume of rehandle, V_r , bcy.

$$V_r = A_r \times L/27 \times (1 + \delta), \text{ bcy.}$$

$$V_r = 6,377 \times 150/27 \times 1.3 = 27,252, \text{ bcy.}$$

4.11. Percent Dragline Rehandle, p, Percent

$$p = \frac{V_r \times 100}{V_T}, \text{ percent}$$

where

$$V_T = (D_3 + D_4 + D_5) \times W \times L/27, \text{ bcy.}$$

$$V_T = 115 \times 200 \times 150/27 = 127,778, \text{ bcy.}$$

$$p = \frac{27,252 \times 100}{127,778} = 21\%$$

4.12. Estimate Average Swing Angles for Each Dragline Pass

The values of the average swing angles compared graphically for the horseshoe method are shown in Table M-1.

4.13. Compute Swings Per Hour for Each Dragline Pass, S_i

Swings per hour are computed for each pass from the graph shown in Figure M-4. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging overburden and interburden material. Rehandle is estimated to have 20 second dig time and 4 second delay in one cycle. Swing cycles per hour for each pass are shown in Table M-1.

4.14. Compute Bucket Carrying Capacity for Each Pass, C_i , bcy

$$C_i = bf_i/(1 + \delta), \text{ bcy.}$$

where for this concept

$$f_1 = 0.8 \text{ sandstone}$$

$$f_2 = 0.85 \text{ shale}$$

$$f_3 = 0.50 \text{ interburden shale}$$

$$f_4 = 1.00 \text{ rehandle}$$

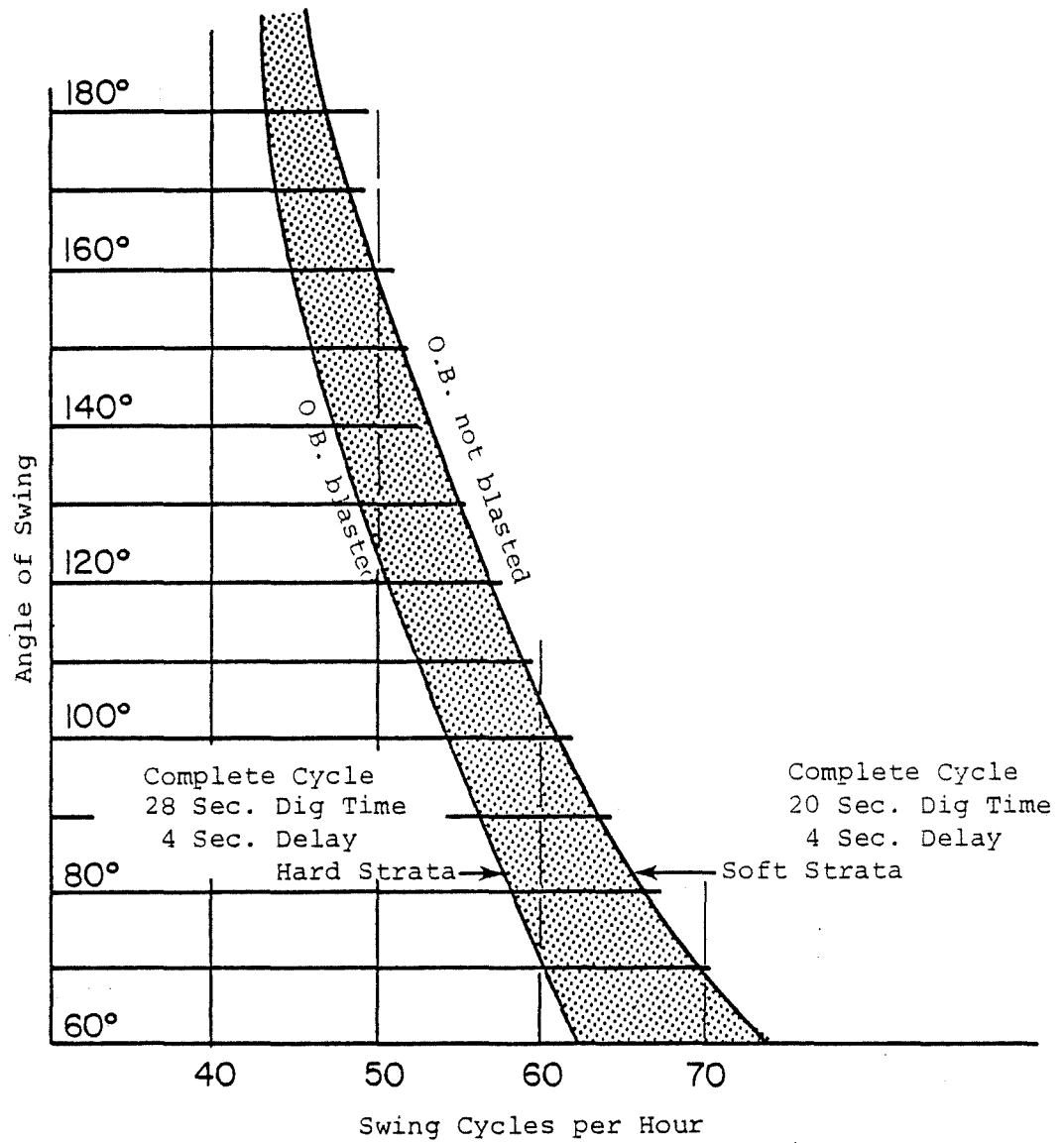


Figure M-4. Swing Cycles per Hour vs. Angle of Swing

4.15. Compute Time Required to Dig Material at Each Pass, T_i , Hours

$$T_i = V_i / (C_i S_i), \text{ hrs.}$$

4.16. Compute Total Time to Complete One Block, T , Hours

$$T = \sum T_i, \text{ hrs.}$$

4.17. Compute Scheduled Time to Dig One Block, T_S , Hours

$$T_S = T/pq, \text{ hrs.}$$

4.18. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_T / T_S, \text{ bcy.}$$

4.19. Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy.}$$

$$H_M = 660 \text{ hrs: scheduled hours per month.}$$

4.20. Compute Dragline Annual Production, P_Y , bcy

$$P_Y = 12 \times P_M, \text{ bcy.}$$

Results from the computation of dragline production rate are summarized in Table M-1.

5. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables M-2 and M-3 show detailed estimated hourly ownership and operating costs for dragline and dozer.

The total estimated hourly ownership and operating costs for the dragline and dozer are \$679.90.

6. Estimate Stripping Costs per Bank Cubic Yard

Estimated cost per bank cubic yard removed by dragline with 400 H.P. dozer used as utility machine is \$0.45.

Table M-1. Production Rate Calculations for
Single Dragline Horseshoe Method

| Output Variable | Dig Component for One Block | | | |
|--|-----------------------------|----------------------------------|--------------------------------|---------------------------|
| | Overburden | Interburden to Spoil Bench | Interburden to Main Pile | Rehandle & Interburden |
| Volume of Material, bcy. | 61,111 | 24,812 | 19,250 | 49,857 |
| Average Swing Angle, degrees | 90 | 90 | 135 | 110 |
| Swing Cycles Per Hour | 56 | 56 | 48 | 52 |
| Bucket Carrying Capacity, bcy. | 46 | 36 | 29 | 58 |
| Required Digging Time, hrs. | 23.7 | 12.3 | 13.8 | 16.2 |
| Total Time to Complete One Block, hours | 66.0 | | | |
| Scheduled Time for One Block, hours | 92.5 | | | |
| Production Per Scheduled Hour, bcy. | 1,500 | | | |
| Monthly Production, bcy. | 990,000 | | | |
| Annual Production, bcy. | 11,880,000 | | | |

Table M-2. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | 313.82 |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | 306.93 |
| <u>Total Ownership and Operating Costs</u> | <u>620.75</u> |

Table M-3. Estimated Hourly Ownership and Operating Costs
 Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.45 |
| Total Ownership Costs | <u>16.59</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | 14.00 |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | <u>59.15</u> |

APPENDIX N
WESTERN REGION SITE
SINGLE DRAGLINE MINING CONCEPT
ELEVATED BENCH METHOD

APPENDIX N
WESTERN REGION SITE
SINGLE DRAGLINE MINING CONCEPT
ELEVATED BENCH METHOD

1. Introduction

This Appendix contains production and stripping cost estimates of the Single Dragline Mining Concept - Elevated Bench Method - developed for the Western Region.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

| | | | |
|----------------|---|-------------|--|
| W | = | 180 feet: | pit width |
| D | = | 125 feet: | total overburden and interburden depth |
| D ₁ | = | 65 feet: | total overburden thickness |
| D ₂ | = | 60 feet: | total interburden thickness, shale |
| D ₃ | = | 10 feet: | thickness of overburden clay strata |
| D ₄ | = | 20 feet: | thickness of overburden sandstone strata |
| D ₅ | = | 35 feet: | thickness of overburden shale strata |
| S ₁ | = | 360 inches: | thickness of upper coal seam |
| S ₂ | = | 240 inches: | thickness of lower coal seam |
| ϕ | = | 75 degrees: | highwall angle |
| θ | = | 37 degrees: | natural angle of repose |
| α | = | 55 degrees: | angle of oversteepened spoil |
| δ | = | 30 percent: | average spoil factor |
| L | = | 150 feet: | length of block |

R_d = 282 feet: dragline dump radius
 R_{ef} = 232 feet: dragline effective radius
 H_d = 139 feet: dragline dump height
 b = 75 cubic yards: dragline bucket capacity
 q = 0.83: dragline utilization
 p = 0.85: dragline availability

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

H_S = Height of spoil pile above the pit floor, feet
 R_{nr} = Dragline radius required for no rehandle, feet
 V_i = Dig volumes of each dragline pass, bcy.
 V_r = Volume of rehandle material, bcy.
 S_i = Swing cycles per hour for each pass, bcy.
 C_i = Bucket carrying capacity for each pass, bcy.
 T_i = Time required to dig material of each pass, hours
 T = Time to complete one block, hours
 T_S = Scheduled time to dig one block, hours
 P_H = Dragline production rate per scheduled hours, bcy.
 P_M = Dragline monthly production, bcy.
 P_Y = Dragline annual production, bcy.

4. Procedure for Estimating Production Rate for the Single Dragline Mining Concept - Elevated Bench Method

A procedure for estimating the production rate for a single dragline using the elevated bench method is given below. Figure N-1 illustrates the operating procedures for this mining concept.

4.1. Compute Final Spoil Pile Height, H_S , Feet

$$H_S = (D_1 + D_2)(1 + f) + W \tan \theta/4, \text{ ft.}$$

$$H_S = (65 + 60)(1 + 0.30) + 180 \tan 37^\circ/4 = 196.41, \text{ ft.}$$

4.2. Check for Dragline Dump Height, H_d , Feet

$$H_d \geq H_S - D_2 - S_1 - S_2$$

139' > 86.41' -- dragline has sufficient dump height.

4.3. Compute Dig Volumes of Each Component, V_i , bcy

- Volume of interburden, V_1 , bcy.

$$V_1 = D_2 \times W \times L/27, \text{ bcy.}$$

$$V_1 = 60 \times 180 \times 150/27 = 60,000, \text{ bcy.}$$

- Volume of overburden material to build the elevated bench, V_2 , bcy.

$$V_2 = (D_3 + 0.75 D_4) \times W \times L/27, \text{ bcy.}$$

$$V_2 = 25 \times 180 \times 150/27 = 25,000, \text{ bcy.}$$

- Volume of overburden material to main spoil pile, V_3 , bcy.

$$V_3 = (0.25 D_4 + D_5) \times W \times L/27, \text{ bcy.}$$

$$V_3 = 40 \times 180 \times 150/27 = 40,000, \text{ bcy.}$$

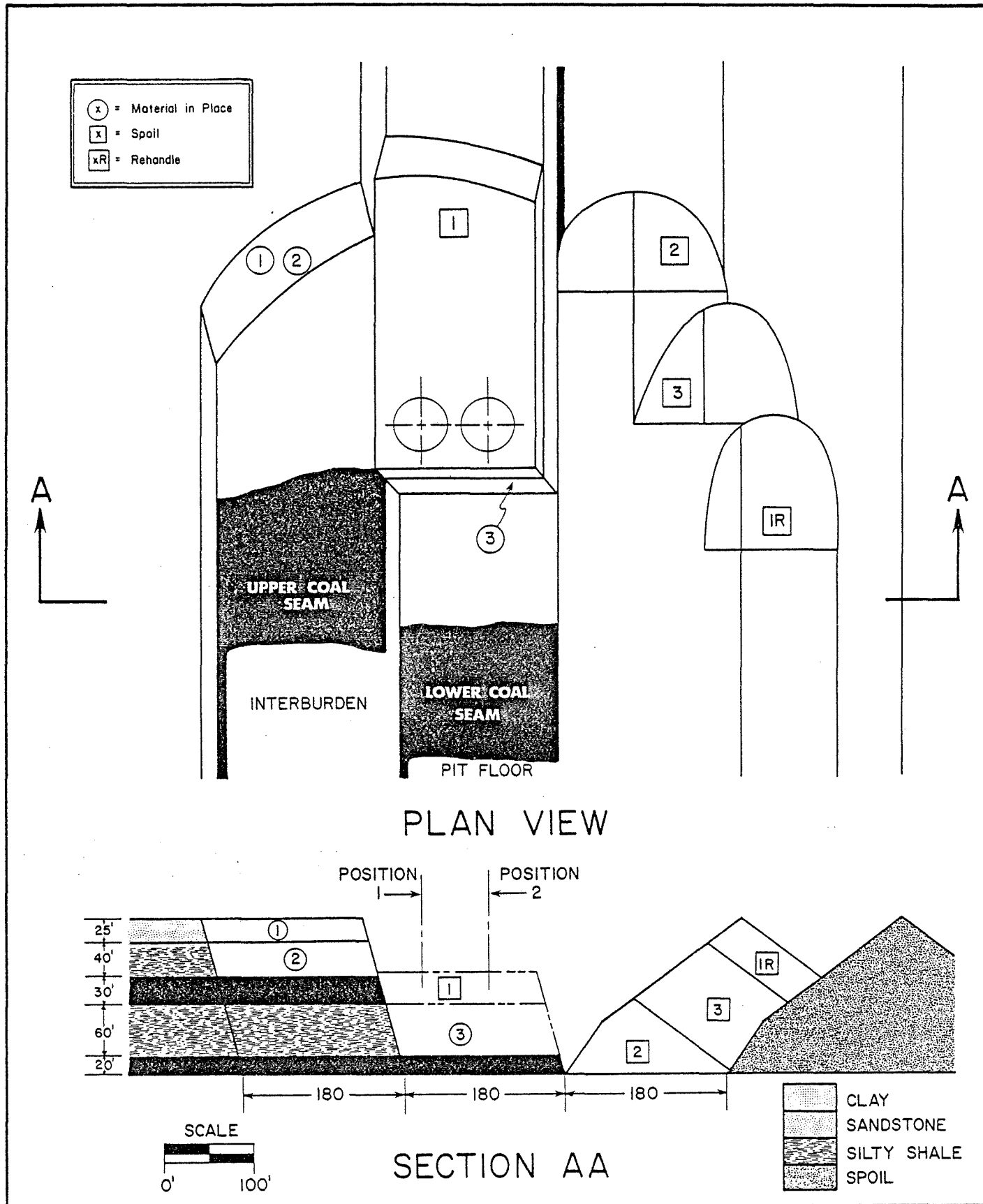


Figure N-1. Single Dragline Mining Concept - Elevated Bench Method - Operating Procedure

- Volume of rehandle material, V_r , bcy.

$$V_r = (D_3 + 0.75 D_4) \times W \times L/27, \text{ bcy.}$$

$$V_r = 25 \times 180 \times 150/27 = 25,000, \text{ bcy.}$$

4.4. Compute Percent Rehandle, p, Percent

$$p = \frac{V_r \times 100}{V_T}, \text{ percent}$$

where

$$V_T = V_1 + V_2 + V_3, \text{ bcy.}$$

$$p = \frac{25,000}{125,000} \times 100 = 20\%$$

4.5. Estimate Average Swing Angles for Each Dig Component

Average swing angles for each dig component are estimated graphically and are shown in Table N-1.

4.6. Compute Swings Per Hour for Each Dig Component, S_i

Swings per hour are computed for each dig component from the graph shown in Figure N-2. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging overburden and interburden material. Rehandle is estimated to have 20 second dig time and 4 second delay in one cycle. Swing cycles per hour for each pass are shown in Table N-1.

4.7. Compute Bucket Carrying Capacity for Each Dig Component, C_i , bcy

$$C_i = bf_i/(1 + \delta), \text{ bcy.}$$

where, for this concept

$$f_1 = 0.8 \text{ clay}$$

$$f_2 = 0.8 \text{ sandstone}$$

$$f_3 = 0.85 \text{ shale}$$

$$f_4 = 0.95 \text{ rehandle}$$

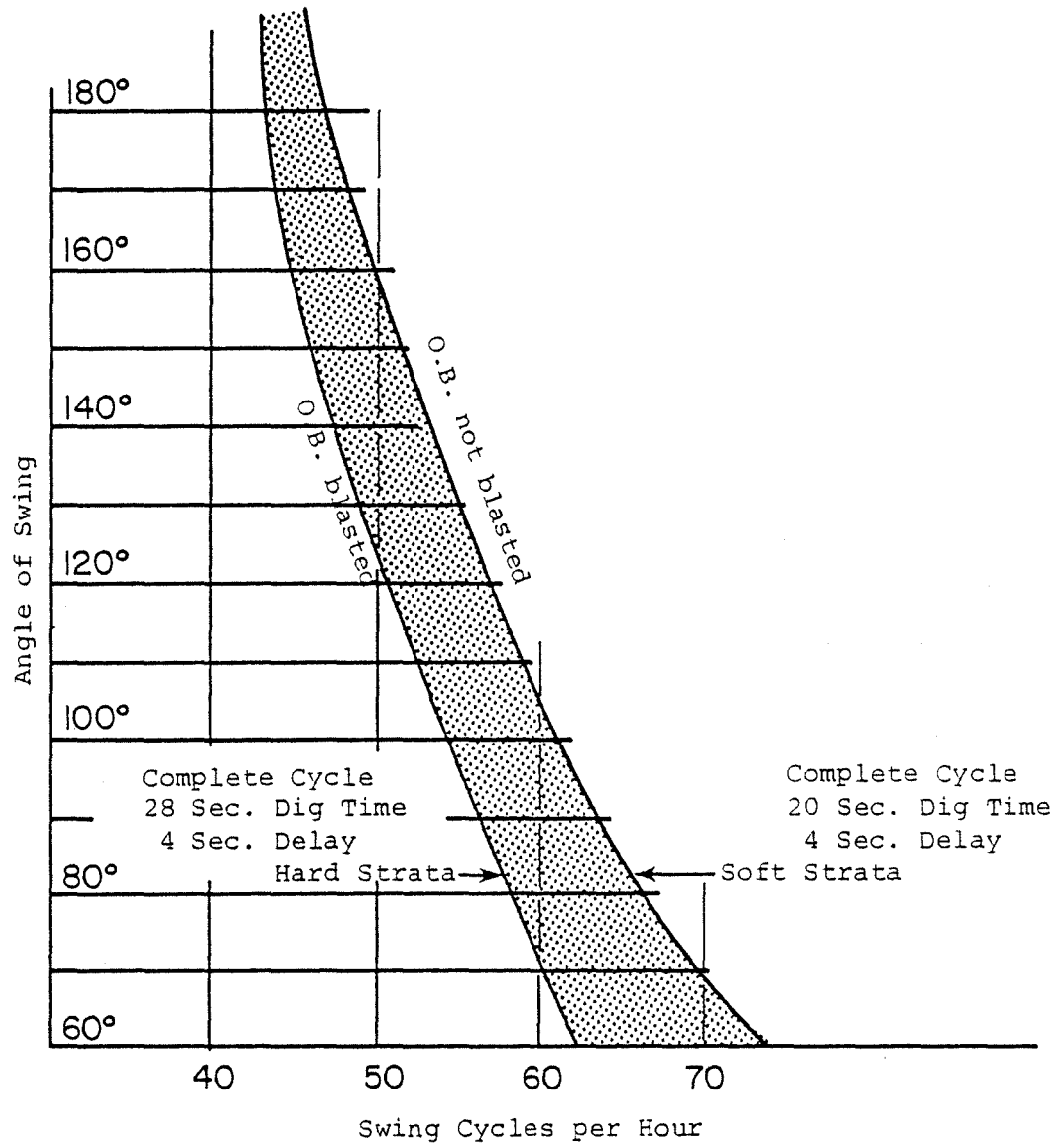


Figure N-2. Swing Cycles per Hour vs. Angle of Swing

- 4.8. Compute Time Required to Dig Each Dig Component, T_i , Hours

$$T_i = V_i / (C_i S_i), \text{ hrs.}$$

- 4.9. Compute Total Time to Complete One Block, T, Hours

$$T = \sum T_i, \text{ hrs.}$$

- 4.10. Compute Scheduled Time to Dig One Block, T_S , Hours

$$T_S = T/pq, \text{ hrs.}$$

- 4.11. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_T / T_S, \text{ bcy.}$$

- 4.12. Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy}$$

where

$$H_M = 660 \text{ hrs: scheduled hours per month.}$$

- 4.13. Compute Dragline Annual Production, P_Y , bcy

$$P_Y = P_M \times 12, \text{ bcy.}$$

Results from the computation of dragline production rate are summarized in Table N-1.

5. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables N-2 and N-3 show detailed estimated hourly ownership and operating costs for dragline and dozer.

The total estimated hourly ownership and operating costs for the dragline and dozer used as a utility machine are \$679.90.

Table N-1. Production Rate Calculations for Single Dragline Mining
Concept - Elevated Bench Method

| Output Variable | Dig Component for One Block | | | |
|---|-----------------------------|------------------|-------------------|----------|
| | Overburden Sandstone | Overburden Shale | Interburden Shale | Rehandle |
| Volume of Material, bcy. | 25,000 | 40,000 | 60,000 | 25,000 |
| Average Swing Angle, degrees | 50 | 80 | 110 | 90 |
| Swing Cycles Per Hour | 68 | 60 | 59 | 64 |
| Bucket Carrying Capacity, bcy. | 46 | 49 | 49 | 55 |
| Required Digging Time, hours | 7.99 | 13.6 | 20.7 | 7.1 |
| Total Time to Complete One Block, hours | 49.39 | | | |
| Scheduled Time for One Block, hours | 69.14 | | | |
| Production Per Scheduled Hour, bcy. | 1,808 | | | |
| Monthly Production, bcy. | 1,193,000 | | | |
| Annual Production, bcy. | 14,319,000 | | | |

6. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by dragline with 400 H.P. dozer used as utility machine is \$0.38.

Table N-2. Estimated Hourly Ownership and Operating Costs
 Dragline: 75 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$16,500,000 |
| Extras, Freight & Erection | 1,500,000 |
| Total Price | 18,000,000 |
| Salvage Value @ 10% | 1,800,000 |
| Depreciated Amount | 16,200,000 |
| Life of Machine | 30 years @ 8,030 hrs/yr |
| Depreciation Cost | 67.25 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 246.57 |
| Total Ownership Costs | <u>313.82</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 66.14 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 186.79 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | <u>306.93</u> |
| <u>Total Ownership and Operating Costs</u> | <u>620.75</u> |

Table N-3. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.45 |
| Total Ownership Costs | <u>16.59</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | <u>14.00</u> |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | 59.15 |

APPENDIX P

WESTERN REGION SITE
TANDEM MINING CONCEPT
SHOVEL/TRUCK/Dragline SYSTEM

APPENDIX P
WESTERN REGION SITE
TANDEM MINING CONCEPT
SHOVEL/TRUCK/DragLINE SYSTEM

1. Introduction

This Appendix contains procedures for computation of production rates and stripping cost estimate of dragline/shovel/truck tandem stripping system. Each step of estimating these parameters is discussed separately.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

| | | | |
|-----------------|---|-------------|--|
| L | = | 4,500 feet: | pit length |
| W | = | 150 feet: | pit width |
| D | = | 125 feet: | total overburden and interburden thickness |
| D ₁ | = | 35 feet: | height of first shovel lift |
| D ₂ | = | 30 feet: | height of second shovel lift |
| D ₃ | = | 60 feet: | height of dragline bench |
| S ₁ | = | 360 inches: | thickness of upper coal seam |
| S ₂ | = | 240 inches: | thickness of lower coal seam |
| L ₁ | = | 150 feet: | block length |
| φ ₁ | = | 75 degrees: | highwall angle |
| θ | = | 37 degrees: | natural angle of repose |
| α | = | 55 degrees: | angle of steepened spoil |
| δ | = | 30 percent: | average spoil swell factor |
| R _{d1} | = | 241 feet: | dragline dump radius |

R_e = 205 feet: dragline effective spoil radius
 H_d = 132 feet: dragline dumping height
 D_d = 150 feet: dragline digging depth
 b_1 = 50 cubic yards: dragline bucket capacity
 b_2 = 18 cubic yards: shovel bucket capacity
 H_L = 35 feet: dumping height maximum
 H_C = 51.25 feet: cutting height maximum
 R_{d2} = 54.75 feet: dumping radius maximum
 R_C = 63 feet: cutting radius maximum
 R_{CL} = 23 feet: clearance radius
 T_T = 85 tons: truck capacity

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

V = Total overburden volume for one block, bcy.
 V_{S_i} = Volumes moved by shovel/truck, bcy.
 V_{D_i} = Volumes of each dragline dig component, bcy.
 H_S = Height of final spoil pile, feet
 H_{DS} = Height of dragline spoil pile, feet
 R_{nr} = Dragline radius required for no rehandle, feet
 S_i = Dragline swing cycles per hour for each dig component
 C_i = Dragline bucket carrying capacity for each dig component, bcy
 T_i = Time required to dig material of each dig dragline component, hours
 T = Time to complete one dragline block, hours
 T_S = Scheduled time to dig one dragline block, hours

- P_H = Dragline production rate per scheduled hours, bcy.
 t_p = Tons per shovel pass, t.
 T_p = Cycle time per shovel pass, min.
 N_p = Number of passes to fill one truck
 T_f = Truck fill time, min.
 T_l = Truck load time, min.
 T_d = Truck dump time, min.
 T_F = Truck fixed time, min.
 T_t = Truck travel time, min.
 T_c = Total truck cycle time, min.
 Q_t = Production per truck scheduled hour, bcy.
 Q_H = Production per shovel scheduled hour, bcy.
 N_t = Number of operating trucks per shovel
 Q_{HT} = Production per hour, bcy.
 Q_M = Production per month, bcy.
 Q_Y = Production per year, bcy.

4. Procedure for Estimating Production Rate for the Shovel/Truck/Dragline Tandem System

A procedure for estimating the production rate for a shovel/truck/dragline tandem system is given below. Figure P-1 illustrates the operating procedure of this mining concept.

4.1. Shovel/Truck Production Rate Estimate

4.1.1. Compute Overburden Volume for Each Lift, V_{S_i} , bcy

$$V_{S_i} = D_i \times W \times L_i / 27, \text{ bcy.}$$

Volume calculations are shown in Table P-1.

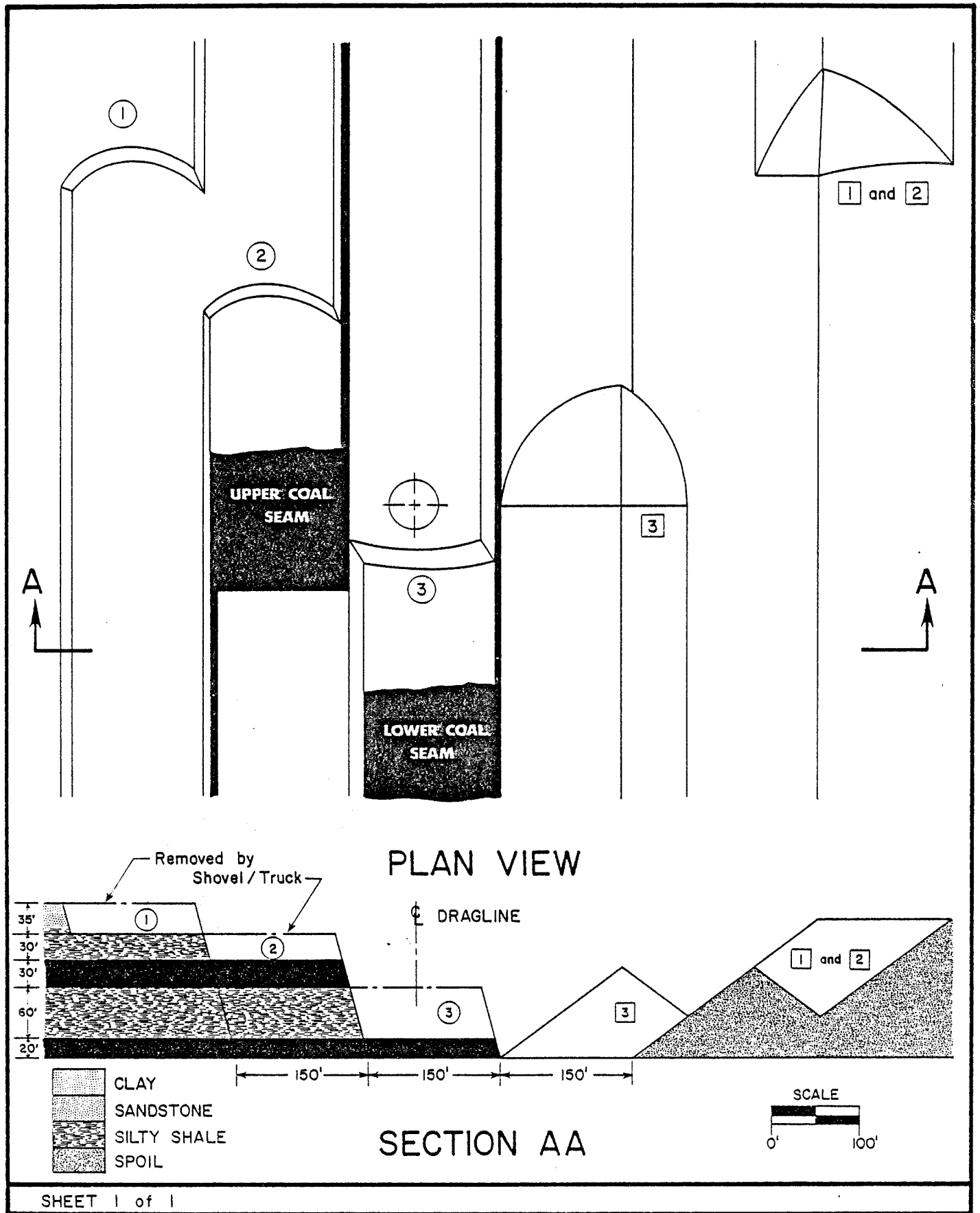


Figure P-1. Typical Cross Section and Plan of Shovel/Truck/Dragline Tandem System

Table P-1. Haul Road Profile

Truck Capacity: 85 tons
 Maximum Allowable Grade: 10 percent
 Pit Length: 4,500 feet

| LIFT NO. | LIFT HEIGHT, FEET | HAUL ROAD SEGMENT | PIT WIDTH, FT. | | |
|----------|-------------------|----------------------------|----------------|----------|----|
| | | | 150 | | |
| | | | LENGTH, FT. | GRADE, % | |
| 1 | 35 | L O A D E D | 1 | 2250 | 0 |
| | | | 2 | 750 | +3 |
| | | | 3 | 2250 | 0 |
| | | E M P T Y | 1 | 2250 | 0 |
| | | | 2 | 750 | -3 |
| | | | 3 | 2250 | 0 |
| 2 | 30 | L O A D E D | 1 | 2250 | 0 |
| | | | 2 | 600 | 0 |
| | | | 3 | 2250 | 0 |
| | | E M P T Y | 1 | 2250 | 0 |
| | | | 2 | 600 | 0 |
| | | | 3 | 2250 | 0 |

4.1.2. Compute Final Spoil Pile Height, H_S , Feet

$$H_S = D \times (1 + \delta), \text{ ft.}$$

$$H_S = 125 \times (1 + 0.3) = 163, \text{ ft.}$$

4.1.3. Haul Road Configuration

The following assumptions have been made when designing the haul roads:

- Average length of the haul road segment on the highwall side is half of the length of the pit.
- Length of the incline is minimum four (4) pit widths.
- Average length of the haul road segment on the spoil side is half of the length of the pit.

Haul road profiles are summarized in Table P-1.

4.1.4. Compute Shovel/Truck Cycle Times

4.1.4.1. Truck Load Time, T_1 , Minutes

- Tons per shovel pass, t_p , t.

$$t_p = \text{bcy.} \times \text{load factor} \times \text{fill factor} \times \text{bank density}$$

| | | |
|----------------------|---|-------|
| bcy. | = | 18 |
| load factor | = | 0.769 |
| fill factor | = | 0.8 |
| bank density, t/bcy. | = | 2.05 |

$$t_p = 18 \times 0.769 \times 0.8 \times 2.05 = 22.7, \text{ t.}$$

- Cycle time per shovel pass, T_p , min.

$$T_p = 0.5 \text{ min.}$$

- Number of passes to fill one truck, N_p

$$N_p = \frac{T_t}{t_p}$$

$$N_p = \frac{85}{22.7} = 3.75 \approx 4 \text{ passes}$$

- Truck fill time, T_f , min.

$$T_f = N_p \times T_p$$

$$T_f = 4 \times 0.5 = 2.0 \text{ min.}$$

Add spot and maneuver time of 1 minute.

- Truck load time, T_l , min.

$$T_l = T_f + T_m, \text{ min.}$$

$$T_l = 2.0 + 1.00 = 3.0 \text{ min.}$$

4.1.4.2. Truck Dump Time, T_d , Minutes

Nominal backup of 0.5 min. Add a spot and maneuver time of 1.0 min.

$$T_d = T_b + T_m, \text{ min.}$$

$$T_d = 0.5 + 1.0 = 1.5 \text{ min.}$$

4.1.4.3. Truck Fixed Time, T_F , Minutes

$$T_F = T_l + T_d, \text{ min.}$$

$$T_F = 3.0 + 1.50 = 4.5 \text{ min.}$$

4.1.4.4. Truck Travel Time, T_t , Minutes

Travel time is computed based on haul road profile and using travel time charts for 85 ton rock truck. Travel cycle time computations are shown in Table P-2.

4.1.4.5. Total Truck Cycle Time, T_C , Minutes

$$T_C = T_F + T_t, \text{ min.}$$

Total cycle time computations are shown in Table P-2.

4.1.4.6. Shovel Production Per Scheduled Hour, Q_H , bcy

$$Q_H = 60 \times T_T \times \text{Production Factor} / T_l \times 2.05, \text{ bcy.}$$

where

$$\text{Production Factor} = 0.714$$

Table P-2. Truck Cycle Time, Minutes

Truck Capacity: 85 tons
 Pit Length: 4,500 ft.
 Pit Width: 150 ft.

| LIFT NO. | LIFT HEIGHT, FEET | HAUL ROAD SEGMENT | | TRAVEL TIME, MIN. | | |
|-------------------------|-------------------|----------------------------|----|----------------------------|---|------|
| 1 | 35 | L O A D E D | 1 | 1.0 | | |
| | | | 2 | 0.55 | | |
| | | | 3 | 1.0 | | |
| | | E M P T Y | 1 | 0.92 | | |
| | | | 2 | 0.45 | | |
| | | | 3 | 0.92 | | |
| | | TOTAL TRAVEL TIME, MIN. | | | | 4.84 |
| | | FIXED TIME, MIN. | | | | 4.5 |
| | | TOTAL CYCLE TIME, MIN. | | | | 9.34 |
| | | 2 | 30 | L O A D E D | 1 | 0.95 |
| 2 | 0.48 | | | | | |
| 3 | 0.95 | | | | | |
| E M P T Y | 1 | | | 0.90 | | |
| | 2 | | | 0.38 | | |
| | 3 | | | 0.90 | | |
| TOTAL TRAVEL TIME, MIN. | | | | 4.56 | | |
| FIXED TIME, MIN. | | | | 4.50 | | |
| TOTAL CYCLE TIME, MIN. | | | | 9.06 | | |

4.1.4.7. Truck Production Per Scheduled Hour, Q_t , bcy

$$Q_t = 60 \times T_T \times \text{Production Factor}/T_C \times 2.05, \text{ bcy.}$$

4.1.4.8. Number of Operating Trucks Per Shovel, N_t

$$N_t = \frac{T_C}{T_1}$$

4.1.4.9. Production Rate Per Scheduled Hour, Per Lift, Q_{HT} , bcy

$$Q_{HT} = Q_t \times N_t, \text{ bcy.}$$

4.1.4.10. Production Rate Per Month, Q_M , bcy

$$Q_M = Q_{HT} \times H_M, \text{ bcy.}$$

where

$$H_M = 660 \text{ hrs.: scheduled hours per month.}$$

4.1.4.11. Annual Rate of Production, Q_Y , bcy

$$Q_Y = 12 \times Q_M, \text{ bcy}$$

Production rate computations are shown in Table P-3. Two shovels and three trucks for each shovel were used in the production and cost calculations.

4.2. Computing Dragline Production Rate

4.2.1. Height of the Dragline Spoil Pile, H_F , Feet

$$H_{DS} = D_3 \times (1 + \delta) + W \tan \theta/4, \text{ ft.}$$

$$H_{DS} = 60 \times (1 + 0.30) + 150 \tan 37^\circ/4 = 106 \text{ ft.}$$

$$\text{Height above the dragline bench} = 106' - 20' - 60' = 26 \text{ ft.}$$

Since the dragline dump height is 132 feet, it is sufficient to dump on top of the spoil.

Table P-3. Shovel/Truck Production Rate Calculation

Truck Capacity: 85 tons
 Shovel Capacity: 18 Cu. Yd.

| Output Variables | Overburden | |
|--|-------------------------------|--------------------------------|
| | First Lift (35 ft. height) | Second Lift (30 ft. height) |
| Volume of Material in One Block, bcy. | 29,167 | 25,000 |
| Shovel Production Per Scheduled Hour, bcy. | 592 | 592 |
| Truck Production Per Scheduled Hour, bcy. | 190 | 196 |
| Number of Operating Trucks | 3 | 3 |
| Production Rate Per Scheduled Hour, bcy. | 570 | 588 |
| Production Rate Per Month, bcy. | 764,280 | |
| Production Rate Per Year, bcy. | 9,171,360 | |

4.2.2. Effective Dragline Reach Required for No Rehandling, R_{nr} , Feet

$$R_{nr} = (D_3 + S_2)/\tan \phi + H_f/\tan \theta$$
$$= 169 \text{ ft.}$$

Since the effective dragline radius (205 feet) exceeds R_{nr} , no extended bench is required.

4.2.3. Compute Dig Volumes of Each Dig Component, V_i

- Compute volume of keyway, V_{D1} , bcy.

$$V_{D1} = A_K \times L/27, \text{ bcy.}$$

where

$$A_K = \text{cross-sectional area of keyway, ft.}^2$$

$$A_K = W_K D_2 + D_2^2 \left((\tan \phi)/2 + (\tan \phi_1)/2 \right), \text{ ft.}^2$$

- Compute volume of bank material, V_{D2} , bcy.

$$V_{D2} = V_{DI} - V_{D1}, \text{ bcy.}$$

where

$$V_{DI} = \text{volume of interburden material}$$

$$V_{DI} = D_3 \times W \times L/27, \text{ bcy.}$$

Dig volumes calculations are shown in Table P-4.

4.2.4. Estimate Average Swing Angles for Each Dig Component

The values of the average swing angles computed graphically for the dragline are shown in Table P-4.

4.2.5. Compute Swings Per Hour for Each Dig Component, S_i

Swings per hour are computed for each dig component from the graph shown in Figure P-2. Complete cycle is estimated to have 28 second dig time and 4 second delay for digging interburden material. Swing cycles per hour for each dig component are shown in Table P-4.

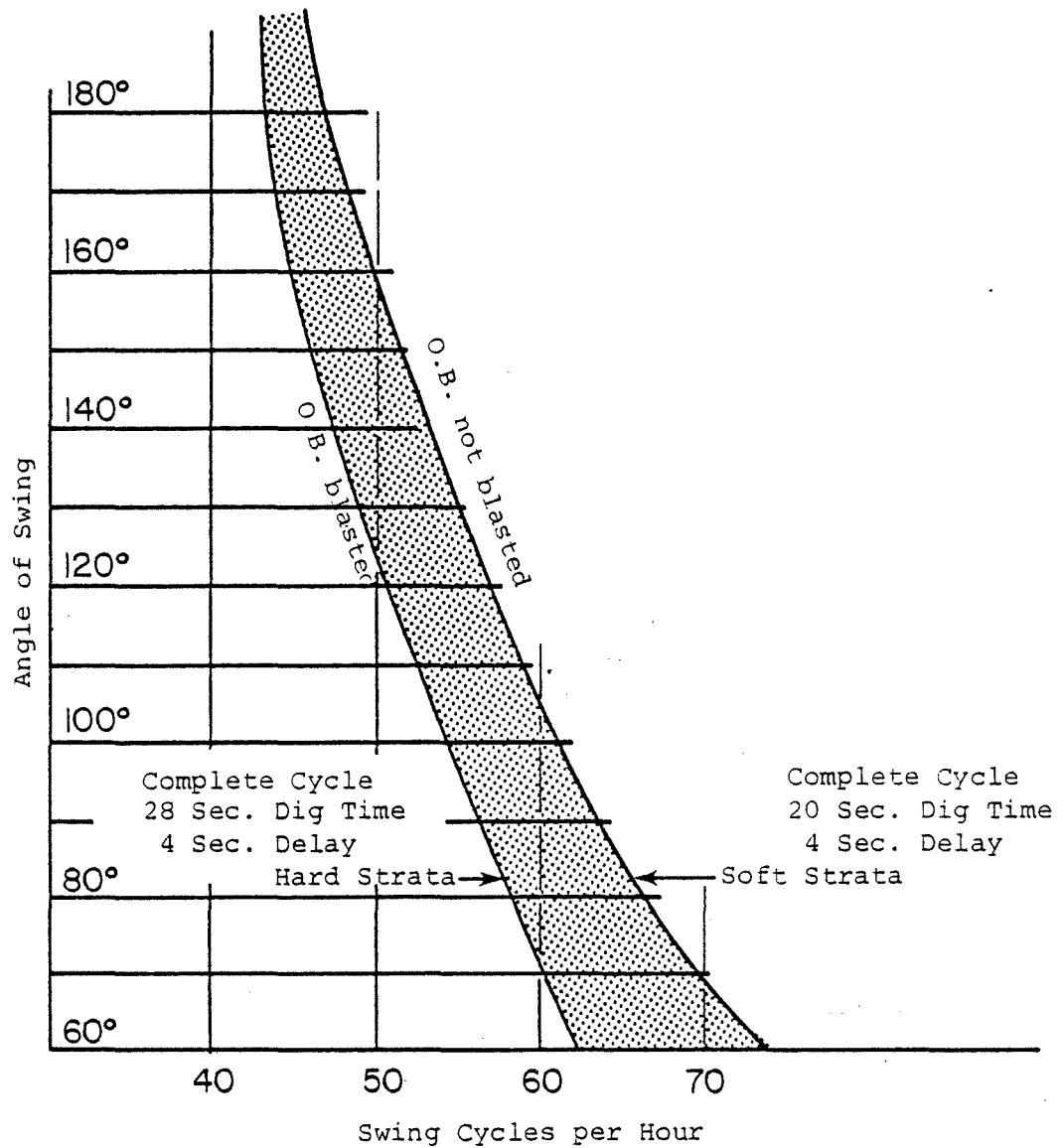


Figure P-2. Swing Cycles per Hour vs. Angle of Swing

Table P-4. Production Rate Calculation for Dragline Stripping the Interburden

| Output Variable | Dig Component for One Block | |
|---|-----------------------------|-------------------------|
| | Keyway | Remaining Bank Material |
| Volume of Material, bcy. | 8,692 | 41,308 |
| Average Swing Angle, degrees | 90 | 90 |
| Swing Cycles Per Hour | 55 | 55 |
| Bucket Carrying Capacity, bcy. | 32 | 33 |
| Required Digging Time, hours | 4.9 | 22.7 |
| Total Time to Complete One Block, hours | 27.7 | |
| Scheduled Time for One Block, hours | 38.7 | |
| Production Per Scheduled Hour, bcy. | 1,286 | |
| Production Rate Per Month, bcy. | 848,760 | |
| Production Rate Per Year, bcy. | 10,185,000 | |

4.2.6. Compute Bucket Carrying Capacity for Each Dig Component, C_i , bcy

$$C_i = b_x f_i / (1 - \delta), \text{ bcy.}$$

where for this concept

$$f_1 = 0.75 \text{ keyway material}$$

$$f_2 = 0.80 \text{ remaining bank material}$$

4.2.7. Compute Time Required to Dig Material at Each Dig Component, T_i , Hours

$$T_i = V_{D_i} / (C_i S_i), \text{ hrs.}$$

4.2.8. Compute Total Time to Complete One Block, T , Hours

$$T = \sum T_i, \text{ hrs.}$$

4.2.9. Compute Scheduled Time to Dig One Block, T_S , Hours

$$T_S = T/pq, \text{ hrs.}$$

where

$$p = 0.85 \text{ dragline availability}$$

$$q = 0.83 \text{ dragline utilization}$$

4.2.10. Compute Dragline Production Rate Per Scheduled Hour, P_H , bcy

$$P_H = V_{DL} / T_S, \text{ bcy.}$$

4.2.11. Compute Dragline Monthly Production, P_M , bcy

$$P_M = P_H \times H_M, \text{ bcy.}$$

$$H_M = 660 \text{ hrs: scheduled working hours per month.}$$

4.2.12. Compute Dragline Annual Production, P_Y , bcy

$$P_Y = P_M \times 12, \text{ bcy.}$$

Results from the computation of dragline production rate are summarized in Table -4.

5. Estimate Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables P-5, P-6, P-7, and P-8 show detailed estimated hourly ownership and operating costs for shovel, truck, dragline and dozer.

The total estimated hourly ownership and operating costs for one shovel working with three trucks and a dozer used as utility machine is \$355.25. Total ownership and operating cost for a dragline with dozer used as a utility machine is \$554.28.

6. Estimate Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by shovel/truck system is \$0.62.

Estimated stripping cost per bank cubic yard of material removed by dragline is \$0.43. Total stripping cost per bank cubic yard for shovel/truck/dragline tandem system is \$0.49.

Table P-5. Estimated Hourly Ownership and Operating Costs
Shovel: 18 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$ 2,300,000 |
| Extras, Freight & Erection | 200,000 |
| Total Price | 2,500,000 |
| Salvage Value @ 10% | 250,000 |
| Depreciated Amount | 2,250,000 |
| Life of Machine | 18 years @ 8,030 hrs/yr |
| Depreciation Cost | 15.57 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 34.25 |
| Total Ownership Costs | <u>49.82</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 15.03 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 44.83 |
| 5. Direct Labor and Fringes | |
| Operator | 14.00 |
| Oiler | 13.00 |
| Total Operating Costs | <u>86.86</u> |
| <u>Total Ownership and Operating Costs</u> | <u>136.68</u> |

Table P-6. Estimated Hourly Ownership and Operating Costs
 Rock Truck: 85 Ton Capacity

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$300,000 |
| Extras, Freight & Erection | -- |
| Total Price | 300,000 |
| Less Original Tires | 14,000 |
| Amount to be Salvaged | 286,000 |
| Salvage Value @ 10% | 28,600 |
| Depreciated Amount | 257,400 |
| Life of Machine | 3 years @ 8,030 hrs/yr. |
| Depreciation Cost | 10.68 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.09 |
| Total Ownership Costs | <u>14.77</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 15 gal/operating hr. @ \$0.85/unit | 9.11 |
| 4. Tire Replacement and Repair | |
| \$14,000 @ 1,500 hrs. + 15% for repair | 10.73 |
| 5. Lubricants, Filters & Grease | |
| Est. at 20% of Fuel Cost | 1.80 |
| 6. Maintenance and Supplies | |
| Est. at 10% of Investment | 3.73 |
| 7. Direct Labor and Fringes | |
| Operator | <u>13.00</u> |
| Total Operating Costs | <u>38.37</u> |
| <u>Total Ownership and Operating Costs</u> | <u>53.14</u> |

Table P-7. Estimated Hourly Ownership and Operating Costs
 Dragline: 50 Cubic Yard Bucket

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$12,000,000 |
| Extras, Freight & Erection | 2,000,000 |
| Total Price | 14,000,000 |
| Salvage Value @ 10% | 1,400,000 |
| Depreciated Amount | 12,600,000 |
| Life of Machine | 18 years @ 8,030 hrs/yr |
| Depreciation Cost | 87.17 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 191.78 |
| Total Ownership Costs | <u>278.95</u> |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 37.65 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 124.53 |
| 5. Direct Labor and Fringes | |
| Operator | 15.00 |
| Oiler | 13.00 |
| Groundman (2) | 26.00 |
| Total Operating Costs | <u>216.18</u> |
| <u>Total Ownership and Operating Costs</u> | <u>495.13</u> |

Table P-8. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.45 |
| Total Ownership Costs | <u>16.59</u> |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | 14.00 |
| Total Operating Costs | <u>42.56</u> |
| <u>Total Ownership and Operating Costs</u> | <u>59.15</u> |

APPENDIX Q

WESTERN REGION SITE
NO DRAGLINE MINING CONCEPT
SHOVEL/TRUCK SYSTEM

APPENDIX Q
WESTERN REGION SITE
NO DRAGLINE MINING
SHOVEL/TRUCK SYSTEM

1. Introduction

This Appendix contains procedures for computation of production rates and stripping cost estimate of the shovel/truck mining concept.

2. Input Variables

Variables describing numerical values that must be input to the concept developed in this Appendix are listed below.

- L = 4,500 feet: pit length
- W = 150 feet: pit width
- D = 125 feet: total overburden and interburden thickness
- D_i = 30-35 feet: height of shovel lift
- S_1 = 360 inches: thickness of upper coal seam
- S_2 = 240 inches: thickness of lower coal seam
- L_1 = 100 feet: block length
- ϕ = 75 degrees: highwall angle
- θ = 37 degrees: natural angle of repose
- δ = 30 percent: average spoil swell factor
- b = 18 cubic yards: bucket capacity
- H_L = 34 feet: dumping height maximum

H_C = 51.25 feet: cutting height maximum
 R_d = 54.75 feet: dumping radius maximum
 R_C = 63 feet: cutting radius maximum
 R_{CL} = 23 feet: clearance radius
 T_T = 85 tons: truck capacity

3. Output Variables

Variables representing numerical values to be calculated using the procedures described in this concept are listed below.

V = Total overburden volume for one block, bcy.
 V_{S_i} = Volumes moved by shovel/truck for each lift, bcy.
 H_S = Height of final spoil pile, feet
 H_{OB} = Height of spoil pile for overburden lifts, feet
 t_p = Tons per shovel pass, ton
 T_p = Cycle time per shovel pass, minutes
 N_p = Number of passes
 T_f = Truck fill time, minutes
 T_l = Truck load time, minutes
 T_d = Truck dump time, minutes
 T_F = Truck fixed time, minutes
 T_t = Truck travel time, minutes
 T_C = Total truck cycle time, minutes
 Q_t = Production per truck per scheduled hour, bcy.
 Q_H = Production per shovel per scheduled hour, bcy.
 N_t = Number of operating trucks per shovel
 Q_{H_T} = Production per scheduled hour per lift, bcy.
 Q_M = Production per month, bcy.
 Q_Y = Production per year, bcy.

4. Procedure for Estimating Production Rate for the Shovel/Truck System

A procedure for estimating the production rate for a shovel/truck system is given below. Figure Q-1 illustrates the operating procedure of this mining concept.

4.1. Compute Volume of Overburden for Each Lift, V_{S_i} , bcy

$$V_{S_i} = D_i \times W \times L_i / 27, \text{ bcy.}$$

Volume calculations are shown in Table Q-1.

4.2. Compute Final Spoil Pile Height, H_S , Feet

$$H_S = D \times (1 + \delta), \text{ ft.}$$

$$H_S = 125 \times (1 + 0.3) = 163, \text{ ft.}$$

4.3. Compute Spoil Pile Height for the Second Overburden Lift, H_{OB} , Feet

$$H_{OB} = D_1 + D_2 \times (1 + \delta), \text{ ft.}$$

$$H_{OB} = 30 + 30 \times (1 + 0.3) = 78, \text{ ft.}$$

4.4. Haul Road Configuration

The following major assumptions have been made when designing the haul roads:

- Average length of the haul road segment on the highwall side is half of the length of the pit.
- Maximum allowable grade of the inclines is 10 degrees.
- Average length of the haul road segment on the spoil side is half of the length of the pit.

Haul road profiles are summarized in Table Q-1.

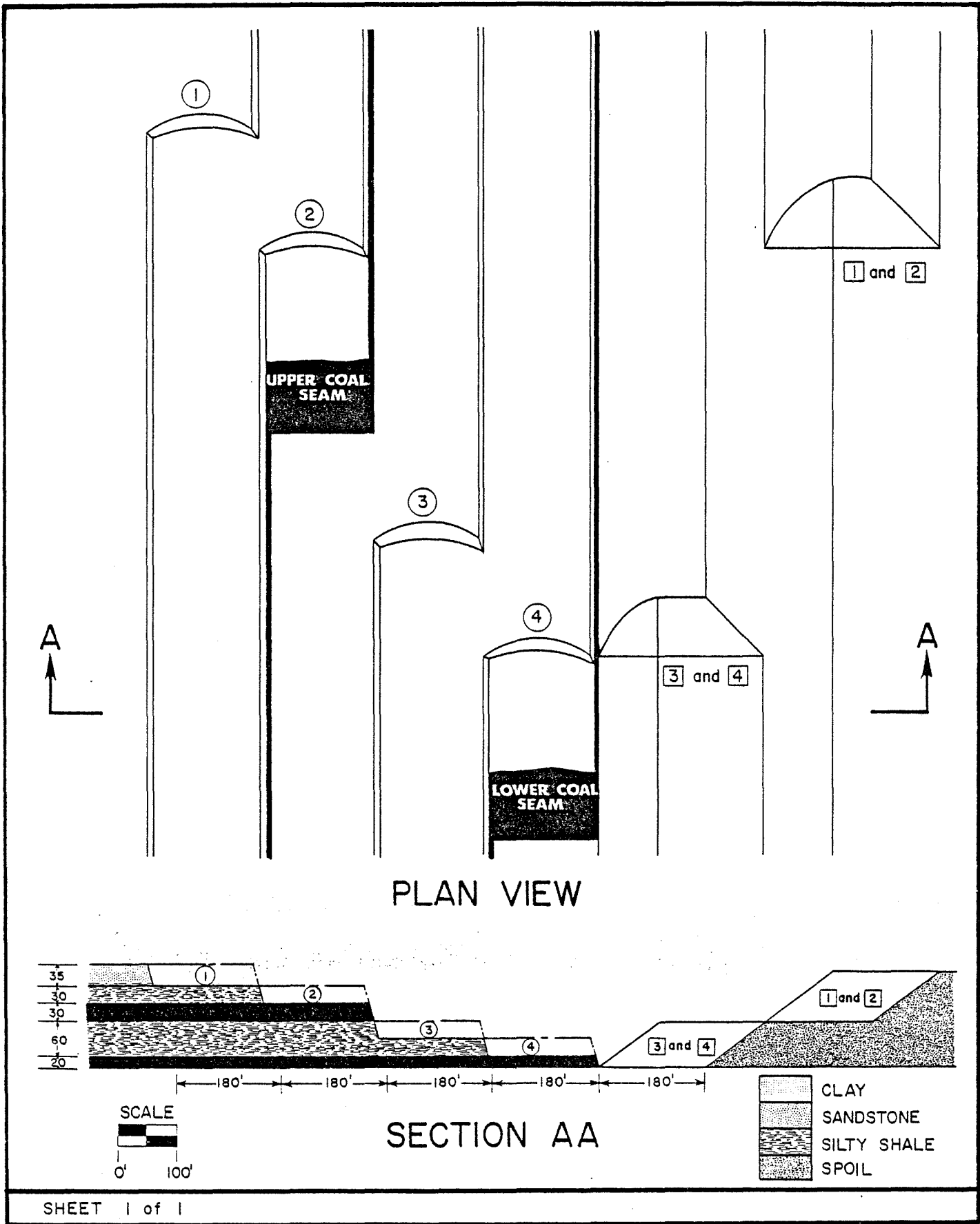


Figure Q-1. Typical Cross-Section and Plan of Shovel/Truck Selective Mining Concept

Table Q-1. Haul Road Profile

Rock Truck: 85 tons
 Maximum Allowable Grade: 10 percent
 Pit Length: 4,500 feet

| LIFT NO. | LIFT HEIGHT, FEET | HAUL ROAD SEGMENT | PIT WIDTH, FT. | | |
|----------|-------------------|----------------------------|----------------|----------|-----|
| | | | 150 | | |
| | | | LENGTH, FT. | GRADE, % | |
| 1 | 35 | L O A D E D | 1 | 2250 | 0 |
| | | | 2 | 230 | +10 |
| | | | 3 | 670 | 0 |
| | | | 4 | 2250 | 0 |
| | | E M P T Y | 1 | 2250 | 0 |
| | | | 2 | 670 | 0 |
| | | | 3 | 230 | -10 |
| | | | 4 | 2250 | 0 |
| 2 | 30 | L O A D E D | 1 | 2250 | 0 |
| | | | 2 | 530 | +10 |
| | | | 3 | 220 | 0 |
| | | | 4 | 2250 | 0 |
| | | E M P T Y | 1 | 2250 | 0 |
| | | | 2 | 220 | 0 |
| | | | 3 | 530 | -10 |
| | | | 4 | 2250 | 0 |
| 3 | 30 | L O A D E D | 1 | 2250 | 0 |
| | | | 2 | 280 | +10 |
| | | | 3 | 170 | 0 |
| | | | 4 | 2250 | 0 |
| | | E M P T Y | 1 | 2250 | 0 |
| | | | 2 | 170 | 0 |
| | | | 3 | 280 | -10 |
| | | | 4 | 2250 | 0 |
| 4 | 30 | L O A D E D | 1 | 2250 | 0 |
| | | | 2 | 580 | +10 |
| | | | 3 | 0 | 0 |
| | | | 4 | 2250 | 0 |
| | | E M P T Y | 1 | 2250 | 0 |
| | | | 2 | 0 | 0 |
| | | | 3 | 580 | -10 |
| | | | 4 | 2250 | 0 |

4.5. Compute Shovel/Truck Cycle Times

4.5.1. Truck Load Time, T_1 , Minutes

- Tons per shovel pass, t_p , tons

$$t_p = \text{bcy} \times \text{load factor} \times \text{fill factor} \times \text{bank density}$$

$$\text{bcy} = 18$$

$$\text{load factor} = 0.769$$

$$\text{fill factor} = 0.8$$

$$\text{bank density, ton/bcy} = 2.05$$

$$t_p = 18 \times 0.769 \times 0.8 \times 2.05 = 22.7, \text{ tons}$$

- Cycle time per shovel pass, T_p , min.

$$T_p = 0.5 \text{ min.}$$

- Number of passes to fill one truck, N_p

$$N_p = \frac{T_t}{t_p}$$

$$N_p = \frac{85}{22.7} = 3.74 \approx 4 \text{ passes}$$

- Truck fill time, T_f , min.

$$T_f = N_p \times T_p, \text{ min.}$$

$$T_f = 4 \times 0.5 = 2.0, \text{ min.}$$

Add spot and maneuver time of 1 minute.

- Total load time, T_1 , min.

$$T_1 = T_f + T_m, \text{ min.}$$

$$T_1 = 2.0 + 1.00 = 3.0 \text{ min.}$$

4.5.2. Truck Dump Time, T_d , Minutes

Nominal backup of 0.5 minutes. Add a spot and maneuver time of 1.0 minutes.

$$T_d = T_b + T_m, \text{ min.}$$

$$T_d = 0.5 + 1.0 = 1.5 \text{ min.}$$

4.5.3. Truck Fixed Time, T_F , Minutes

$$T_F = T_1 + T_d, \text{ min.}$$

$$T_F = 3.0 + 1.50 = 4.5 \text{ min.}$$

4.5.4. Truck Travel Time, T_t , Minutes

Travel time is computed based on haul road profile and using travel time charts for 85 ton rock truck. Travel cycle time computations are shown in Table Q-2.

4.5.5. Total Cycle Time, T_C , Minutes

$$T_C = T_F + T_t, \text{ min.}$$

Total cycle time computations are shown in Table Q-2.

4.5.6. Shovel Production Per Scheduled Hour, Q_H , bcy

$$Q_H = 60 \times T_T \times \text{Production Factor} / T_1 \times 2.05, \text{ bcy}$$

where

$$\text{Production Factor} = 0.714$$

4.5.7. Truck Production Per Scheduled Hour, Q_t , bcy

$$Q_t = 60 \times T_T \times \text{Production Factor} / T_C \times 2.05, \text{ bcy.}$$

4.5.8. Number of Operating Trucks Per Shovel, N_t

$$N_t = \frac{T_C}{T_1}$$

Table Q-2. Truck Cycle Time, Minutes

Truck: 85 tons
 Pit Length: 4,500 ft.
 Pit Width: 150 ft.

| LIFT NO. | LIFT HEIGHT, FEET | HAUL ROAD SEGMENT | TRAVEL TIME, MINUTES |
|-------------------------|-------------------|-------------------|----------------------|
| 1 | 35 | L | |
| | | O | 1 1.0 |
| | | A | 2 0.37 |
| | | D | 3 0.47 |
| | | E | 4 1.0 |
| | | D | |
| | | E | 1 0.92 |
| | | M | 2 0.38 |
| | | P | 3 0.23 |
| | | T | 4 0.92 |
| | | Y | |
| TOTAL TRAVEL TIME, MIN. | | | 5.29 |
| FIXED TIME, MIN. | | | 4.50 |
| TOTAL CYCLE TIME, MIN. | | | 9.79 |
| 2 | 30 | L | |
| | | O | 1 1.0 |
| | | A | 2 0.74 |
| | | D | 3 0.22 |
| | | E | 4 1.0 |
| | | D | |
| | | E | 1 0.92 |
| | | M | 2 0.20 |
| | | P | 3 0.43 |
| | | T | 4 0.92 |
| | | Y | |
| TOTAL TRAVEL TIME, MIN. | | | 5.43 |
| FIXED TIME, MIN. | | | 4.50 |
| TOTAL CYCLE TIME, MIN. | | | 9.93 |
| 3 | 30 | L | |
| | | O | 1 1.0 |
| | | A | 2 0.42 |
| | | D | 3 0.15 |
| | | E | 4 1.0 |
| | | D | |
| | | E | 1 0.92 |
| | | M | 2 0.15 |
| | | P | 3 0.28 |
| | | T | 4 0.92 |
| | | Y | |
| TOTAL TRAVEL TIME, MIN. | | | 4.84 |
| FIXED TIME, MIN. | | | 4.50 |
| TOTAL CYCLE TIME, MIN. | | | 9.34 |
| 4 | 30 | L | |
| | | O | 1 1.0 |
| | | A | 2 0.77 |
| | | D | 3 0.0 |
| | | E | 4 1.0 |
| | | D | |
| | | E | 1 0.92 |
| | | M | 2 0.0 |
| | | P | 3 0.46 |
| | | T | 4 0.92 |
| | | Y | |
| TOTAL TRAVEL TIME, MIN. | | | 5.07 |
| FIXED TIME, MIN. | | | 4.50 |
| TOTAL CYCLE TIME, MIN. | | | 9.57 |

4.5.9. Production Rate Per Scheduled Hour Per Lift,
 Q_{HT} , bcy

$$Q_{HT} = Q_t \times N_t, \text{ bcy.}$$

4.5.10. Production Rate Per Month, Q_M , bcy

$$Q_M = Q_{HT} \times H_M, \text{ bcy.}$$

where

$$H_M = 660 \text{ hrs.: scheduled hours per month.}$$

4.5.11. Production Rate per Year, Q_Y , bcy

$$Q_Y = 12 \times Q_M, \text{ bcy.}$$

Production rate computations are shown in Table Q-3. Four shovels and four trucks each shovel were used in the production and cost calculations.

5. Estimated Hourly Ownership and Operating Costs

Hourly ownership and operating costs are estimated based on 365 working days per year, 22 scheduled hours per day, and 7 days per week. Tables Q-4, Q-5 and Q-6 show detailed estimated hourly ownership and operating costs for shovel, truck and dozer.

The total estimated hourly ownership and operating costs for one shovel working with four trucks and dozer used as a utility machine is \$408.39.

6. Estimated Stripping Cost per Bank Cubic Yard

Estimated stripping cost per bank cubic yard removed by the shovel/truck system is \$0.69.

Table Q-3. Production Rate Estimate
Shovel/Truck System

Shovel: 18 Cubic Yard Dipper Capacity
Truck: 85 Ton Capacity

| Output Variable | Overburden | | Interburden | |
|--|--------------------|--------------------|--------------------|--------------------|
| | Lift 1 (35 ft.) | Lift 2 (30 ft.) | Lift 3 (30 ft.) | Lift 4 (30 ft.) |
| Volume of One Block, bcy. | 19,444 | 16,666 | 16,666 | 16,666 |
| Shovel Production Per Scheduled Hour, bcy. | 592 | 592 | 592 | 592 |
| Truck Production Per Scheduled Hour, bcy. | 181 | 179 | 190 | 186 |
| Number of Operating Trucks Per Shovel | 4 | 4 | 4 | 4 |
| Production Rate Per Month, tons | 390,720 | 390,720 | 390,720 | 390,720 |
| Total Monthly Production, bcy. | 1,562,880 | | | |
| Annual Production, bcy. | 18,755,000 | | | |

Table Q-4. Estimated Hourly Ownership and Operating Costs
Shovel: 18 Cubic Yard Dipper

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$ 2,300,000 |
| Extras, Freight & Erection | 200,000 |
| Total Price | 2,500,000 |
| Salvage Value @ 10% | 250,000 |
| Depreciated Amount | 2,250,000 |
| Life of Machine | 18 years @ 8,030 hrs/yr |
| Depreciation Cost | 15.57 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 34.25 |
| Total Ownership Costs | 49.82 |
| <u>Operating Costs</u> | |
| 3. Power Cost | |
| 35% of Peak Power @ 2.51¢/kwh | 15.03 |
| 4. Maintenance and Supplies | |
| 250,000 yd ³ /year/yd of bucket @ 8¢/yd | 44.83 |
| 5. Direct Labor and Fringes | |
| Operator | 14.00 |
| Oiler | 13.00 |
| Total Operating Costs | 86.86 |
| <u>Total Ownership and Operating Costs</u> | 136.68 |

Table Q-5. Estimated Hourly Ownership and Operating Costs
 Rock Truck: 85 Ton Capacity

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|----------------------------|
| 1. Depreciation | |
| Purchase Price | \$300,000 |
| Extras, Freight & Erection | -- |
| Total Price | 300,000 |
| Less Original Tires | 14,000 |
| Amount to be Salvaged | 286,000 |
| Salvage Value @ 10% | 28,600 |
| Depreciated Amount | 257,400 |
| Life of Machine | 3 years @ 8,030 hrs/yr. |
| Depreciation Cost | 10.68 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | <u>4.09</u> |
| Total Ownership Costs | 14.77 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 15 gal/operating hr. @ \$0.85/unit | 9.11 |
| 4. Tire Replacement and Repair | |
| \$14,000 @ 1,500 hrs. + 15% for repair | 10.73 |
| 5. Lubricants, Filters & Grease | |
| Est. at 20% of Fuel Cost | 1.80 |
| 6. Maintenance and Supplies | |
| Est. at 10% of Investment | 3.73 |
| 7. Direct Labor and Fringes | |
| Operator | <u>13.00</u> |
| Total Operating Costs | <u>38.37</u> |
| <u>Total Ownership and Operating Costs</u> | <u>53.14</u> |

Table Q-6. Estimated Hourly Ownership and Operating Costs
Dozer: 400 HP

| <u>Ownership Costs</u> | <u>\$/Scheduled Hour</u> |
|--|---------------------------|
| 1. Depreciation | |
| Purchase Price | \$325,000 |
| Extras, Freight & Erection | -- |
| Total Price | 325,000 |
| Salvage Value @ 10% | 32,500 |
| Depreciated Amount | 292,500 |
| Life of Machine | 3 years @ 8,030 hrs/yr |
| Depreciation Cost | 12.14 |
| 2. Interest, Taxes and Insurance | |
| 20% of Average Investment | 4.45 |
| Total Ownership Costs | 16.59 |
| <u>Operating Costs</u> | |
| 3. Fuel Cost | |
| 17.4 gal./operating hr. @ 85¢/gal. | 10.56 |
| 4. Maintenance and Supplies | 18.00 |
| 5. Direct Labor and Fringes | |
| 1 Operator @ \$14.00/scheduled hr. | 14.00 |
| Total Operating Costs | 42.56 |
| <u>Total Ownership and Operating Costs</u> | 59.15 |

APPENDIX R
OPERATING COST ANALYSIS
METHODOLOGY

APPENDIX R

OPERATING COST ANALYSIS METHODOLOGY

(by Robert Lappi)

1. Introduction

In order to analyze operating costs for eight separate mining cases, it is necessary to develop a systematic method to assure consistency and relative accuracy between the cases. The methodology described in this Appendix will permit calculation of costs associated with alternative concepts from two base cases (one for the Central coal region and one for the Western coal region).

2. Cost Classification

For purposes of this analysis operating costs are classed as functional, non-functional, fixed, and variable. Provision must be made in the methodology to enable calculation of functional/variable costs, functional/fixed costs, non-functional/variable costs, and non-functional fixed costs.

3. Functional Costs

Functional costs are costs which are incurred to directly affect the accomplishment of a process in the mining operation. For purposes of this analysis functional costs are: site preparation, drilling and blasting, stripping, coal loading, coal hauling, coal processing, rough grading, and final reclamation.

3.1 Site Preparation

Site preparation includes clearing and grubbing, topsoil removal, road building, and road maintenance.

3.2 Drilling and Blasting

Drilling and blasting includes drill benching, drilling, and blasting.

3.3 Stripping

Stripping includes all costs associated with removing overburden. Specifically included are all costs of the prime mover and required support equipment.

3.4 Coal Loading

Coal loading costs include coal drilling and blasting (if required), coal loading, and pit maintenance.

3.5 Coal Hauling

Coal hauling costs are restricted to movement of coal from pit to row coal stockpile either for direct shipment or for further processing.

3.6 Coal Processing

Coal processing costs include all costs incurred from raw coal stockpile to placement in railroad cars. Washing or screening, if required, are coal processing costs.

3.7 Rough Grading

Rough grading costs are those required to place overburden spoil in a condition that it is ready for top soil, seeding and fertilizing. These costs are excluded from final reclamation costs primarily because sum of stripping and rough grading costs is a useful indication of the cost-effectiveness of various overburden handling techniques.

3.8 Final Reclamation

Final reclamation includes topsoil replacement, seeding, and fertilizing.

4.0 Non-Functional Costs

Non-functional costs are costs which do not directly impact on production when incurred. Examples are overhead, engineering, prospecting, and severance taxes.

5. Variable Costs

Variable costs are costs which are changed directly by incremental production. Examples are direct labor, fuel and power, and operation supplies.

6. Fixed Costs

Fixed costs are costs which are not changed directly by incremental production. Fixed is a relative term; it can be effectively argued that there are no fixed costs in the long run. Nevertheless, the concept of fixed costs is useful analytically. In this context, examples of fixed costs are depreciation, overhead, and engineering.

7. Functional/Variable Costs

Functional/variable costs are costs which are incurred to directly affect a production process and which vary directly with incremental production. An example is drilling and blasting explosive costs. Functional/variable costs generally account for the major amount of total operating costs in mining operations.

8. Functional/Fixed Costs

Functional/fixed costs are costs which are incurred to directly affect a production process but which do not vary directly with incremental production. Depreciation on a drill rig is an example of a functional/fixed cost. For purposes of this analysis, equipment depreciation is the only recognized functional/fixed cost.

9. Non-Functional/Variable Costs

Non-functional/variable costs are costs which do not directly affect production but which vary directly with incremental production. State and federal severance taxes are non-functional/variable costs.

10. Non-Functional/Fixed Costs

Non-functional/fixed costs are costs which do not directly affect production and do not vary with incremental production. Prospecting and land taxes are examples of non-functional/fixed costs.

11. General Analytical Approach

The objective of the methodology described in this paper is to enable calculation of operating costs for a number of alternative mining methods employed on a given tract of coal from the available cost estimates of a base case. Cost estimates for the base case must be provided in the classes discussed above: functional/variable, functional/fixed, non-functional variable, and non-functional fixed.

The alternative mining methods to be costed may employ different equipment as the prime overburden mover, may place spoil in different configurations, and will result in varying production. The first two effects above will be estimated by direct, detailed build-up of stripping and rough grading costs. The balance of the costs will be calculated using new production volumes from the base case costs.

12. Functional/Variable Cost Estimates

The functional areas of site preparation, drilling and blasting, coal loading coal hauling, coal processing, and final reclamation are substantially unchanged for different overburden mining methods except for differing production volumes when considering a given coal tract. Furthermore, although the appropriate unit production volume for site preparation is acres mined and that for drilling and blasting is bank cubic yards, coal production tonnage is an acceptable proxy because a given coal tract is common to all concepts being analyzed from the same base case. Therefore, acres, bank cubic yards, and coal tons all change in the same proportion on the given tract.

For each of the unchanged functional areas above, the functional variable costs for each alternative mining method can be estimated by multiplying the appropriate cost per coal ton for the base case by the new coal tonnage.

13. Functional/Fixed Cost Estimates

Equipment depreciation is the only functional/fixed cost recognized in this analysis. The base case will have a depreciation table. The revised equipment depreciation cost for the alternative methods is obtained by adding and deleting equipment as required.

14. Non-Functional/Variable Costs

Non-functional/variable costs are obtained for each alternative method by multiplying the base case costs per ton by the alternative method tonnage.

15. Non-Functional/Fixed Costs

Non-functional fixed costs are obtained for each alternative directly from the base case.

APPENDIX S
FINANCIAL ANALYSIS METHODOLOGY

APPENDIX S

FINANCIAL ANALYSIS METHODOLOGY

(by Robert Lappi)

1. Introduction

This Appendix describes the financial profitability measures, and the methodology used to financially analyze alternative mining methods. Alternative methods and the horseshoe method are mutually exclusive for each mine site. That is, a given tract can only be mined once. Accordingly, this paper deals with the analysis of mutually exclusive investments.

Two financial parameters are utilized as financial measures of the value of alternative mining concepts relative to the base case for the Central and Western coal regions. The primary measure is the net present worth (NPW). A second parameter which is calculated when appropriate is the internal rate of return (IRR).

2. Net Present Worth

The NPW of an investment is the present value of the cash inflows less the present value of the cash outflows. Of the generally known investment criteria, NPW has become recognized as the theoretically soundest measure of profitability (Ref. 1 & 2). Implicit in the use of NPW, is selection of an appropriate discount rate for discounting future cash flows. For a firm utilizing NPW as an investment criterion, the appropriate discount rate is the firm's cost of capital. In actual practice, and because the appropriate cost of capital is difficult to quantify, firms tend to select a hurdle or cut-off rate as the discount rate below which investments are considered undesirable. For all NPW calculations in this project, the hurdle rate is 15%.

3. Internal Rate of Return

IRR is the discount rate that results in a NPW of zero for a stream of cash flows. It is a commonly used profitability measure, but relative to NPW has severe limitations (Ref. 3). In particular, mutually exclusive projects that require different initial outlays or have different economic lives are difficult to compare using IRR. In addition, IRR is meaningless for cases with no initial outlay. Consequently, we shall use IRR with discretion in the financial analyses of interburden handling methods and only when meaningful.

4. Investment Cash Flows

Annual investment cash flow is the after-tax net cash paid out or taken in by the firm as a result of an investment. It is equal to cash margin minus income taxes minus investment. A stream of cash flows for a hypothetical coal mining investment are shown in Table S-1.

Table S-1. Calculation of Cash Flows

| Year | Investment | Revenue | Cash Costs | Depreciation | Depletion | Income Taxes | Cash Flow |
|------|------------|---------|------------|--------------|-----------|--------------|-----------|
| 0 | 1,000 | — | — | — | — | — | (1,000) |
| 1 | — | 1,000 | 500 | 200 | 100 | 100 | 400 |
| 2 | — | 1,000 | 500 | 200 | 100 | 100 | 400 |
| 3 | — | 1,000 | 500 | 200 | 100 | 100 | 400 |
| 4 | — | 1,000 | 500 | 200 | 100 | 100 | 400 |
| 5 | — | 1,000 | 500 | 200 | 100 | 100 | 400 |

In the table above, investment, revenue, and cash costs are all given. DEPRECIATION is obtained by subtracting the salvage value from the investment and dividing the result by the useful life. In this case, $(\$1000 - 0) \div 5 = \200 . DEPLETION (for coal) is 10% of revenue, or \$100. Only tax depreciation and depletion are relevant to cash flow analysis. Companies can and do report both depreciation and depletion to stockholders on a different basis than that used for tax purposes, but such "book" costs are excluded from cash flow analysis. INCOME TAXES are revenue minus cash costs minus depreciation minus depletion, the result multiplied by the income tax rate. For convenience, the tax rate here is assumed to be 50% and includes federal and state income taxes. In Table 1, $(\$1000 - \$500 - \$200 - \$100) \times .50 = \$100$. CASH FLOW is revenue minus cash costs minus income taxes minus investment. For year 0, cash flow is $0 - 0 - 0 - \$1,000 = -\$1,000$. For years 1 - 5, cash flow is $\$1,000 - \$500 - \$100 - 0 = \400 .

5. Calculation of Net Present Worth and Incremental Rate of Return

NPW (at 15%) can be calculated from the stream of cash flows in Table S-1. The calculation is shown in Table S-2.

Analysis based on IRR is not as straightforward. Investment 1 certainly has the higher IRR, 28.6% versus 18.7%. However, both investments are above the cut-off rate of 15%. The question arises is 28.6% IRR on a \$1,000 investment superior to 18.7% IRR on a \$2,000 investment. The answer can be obtained by analysis of the incremental cash flow. In this case, IRR for the additional \$1,000 investment required for Investment 2 is 7.9% compared to the cut-off rate of 15%. Therefore, the same conclusion indicated by NPW is drawn: reject 2 and accept 1.

IRR is more difficult to interpret than NPW. In addition, there are cases where IRR can not be calculated. Consider the case of the incremental cash flow in Table S-4. If the \$1,000 required investment was zero, or even an investment saving, no calculation of IRR is possible.

Both measures of profitability, NPW and IRR, are used in the analysis of alternative mining methods. NPW is the primary measure because of its theoretical superiority, ease of interpretation, and general applicability.

References

1. H.B. Bierman & S. Smidt, "The Capital Budgeting Decision: Economic Analysis and Financing of Investment Projects" (New York: The Macmillan Company, 1966).
2. J.L. Bower, "Managing the Resource Allocation Process: A Study of Corporate Planning and Investment" (Boston: Division of Research, Graduate School of Business Administration, Harvard University, 1970).
3. L. Daniel Maxim & Frank X. Cook, Jr., "Financial Risk Analysis--An AMA Management Briefing" (New York: American Management Association, 1972).

APPENDIX T
COST ITEMIZATION AND
CASH FLOW ANALYSIS

APPENDIX T

COST ITEMIZATION AND CASH FLOW ANALYSIS

1. Introduction

This Appendix contains the tabulated results of the economic evaluation of the selective interburden handling systems analyzed for the two mine sites. The following three sets of tables are included:

- Capital Cost Estimate
- Annual Operating Costs Estimate
- Cash Flow Analysis

Table T-1. Single Dragline Horseshoe Method - Central Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 1 | Dozer 400 H.P. | Site Preparation | 325 | 25 | 5 | 60 |
| 1 | Grader 250 H.P. | Site Prep. & Reclamation | 200 | 20 | 5 | 36 |
| 1 | Scraper 30 C.Y. | Site Prep. & Reclamation | 350 | 50 | 6 | 50 |
| 1 | Blast Drill 12½ in. | Drilling | 900 | 99 | 3 | 267 |
| 1 | Dozer 400 H.P. | Drilling and Blasting | 325 | 25 | 3 | 100 |
| 1 | Dragline 75 C.Y. | Stripping | 16,000 | 160 | 33 | 480 |
| 1 | Dozer 400 H.P. | Stripping | 325 | 25 | 3 | 100 |
| 1 | Shovel 18 C.Y. | Coal Loading | 2,000 | 20 | 33 | 60 |
| 1 | Loader 12 C.Y. | Coal Loading | 530 | 50 | 6 | 80 |
| 3 | Trucks 150 T. | Coal Hauling | 1,920 | 240 | 6 | 280 |
| 1 | Washer 400 TPH | Coal Processing | 5,000 | 50 | 33 | 150 |
| 1 | Dozer 400 H.P. | Rough Grading | 325 | 25 | 6 | 50 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 6 | Trucks: Support & Pick Up | Miscellaneous | 75 | 15 | 3 | 20 |
| 1 | Cable Reeler | Electrical | 65 | 5 | 10 | 6 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 600 | 39 | 33 | 17 |
| Set | Pit Electrics | Electrical | 600 | 39 | 33 | 17 |
| Set | R.R. Lead & Yard Tracks | Rail Transport | 1,000 | 76 | 33 | 28 |
| Set | Buildings | Miscellaneous | 750 | 24 | 33 | 22 |
| Set | Roads | Miscellaneous | 150 | -- | 30 | 5 |
| Set | Tools | Miscellaneous | 1,000 | 250 | 30 | 25 |
| | Start-Up & Contingency | Miscellaneous | 5,300 | -- | 30 | 177 |
| Totals | | | 38,000 | 1,257 | | 2,054 |

Table T-2. Single Dragline - Elevated Bench Method - Central Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 1 | Dozer 400 H.P. | Site Preparation | 325 | 25 | 5 | 60 |
| 1 | Grader 250 H.P. | Site Prep. & Reclamation | 200 | 20 | 5 | 36 |
| 1 | Scraper 30 C.Y. | Site Prep. & Reclamation | 350 | 50 | 5 | 60 |
| 1 | Blast Drill 12½ in. | Drilling | 900 | 99 | 3 | 267 |
| 1 | Dozer 400 H.P. | Drilling and Blasting | 325 | 25 | 3 | 100 |
| 1 | Dragline 75 C.Y. | Stripping | 16,000 | 2,560 | 28 | 480 |
| 1 | Dozer 400 H.P. | Stripping | 325 | 25 | 3 | 100 |
| 1 | Shovel 18 C.Y. | Coal Loading | 2,000 | 320 | 28 | 60 |
| 1 | Loader 12 C.Y. | Coal Loading | 530 | 50 | 5 | 96 |
| 3 | Trucks 150 T. | Coal Hauling | 1,920 | 240 | 5 | 336 |
| 1 | Washer 400 TPH | Coal Processing | 5,000 | 212 | 28 | 171 |
| 1 | Dozer 400 H.P. | Rough Grading | 325 | 25 | 5 | 60 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 6 | Trucks: Supply & Pick-Up | Miscellaneous | 75 | 15 | 3 | 20 |
| 1 | Cable Reeler | Electrical | 65 | 5 | 10 | 6 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 600 | 40 | 28 | 20 |
| Set | Pit Electrics | Electrical | 600 | 40 | 28 | 20 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 1,000 | 104 | 28 | 32 |
| Set | Buildings | Miscellaneous | 750 | 22 | 28 | 26 |
| Set | Roads | Miscellaneous | 150 | 10 | 28 | 5 |
| Set | Tools | Miscellaneous | 1,000 | 244 | 28 | 27 |
| | Start-Up & Contingency | Miscellaneous | 5,300 | -- | 28 | 190 |
| Totals | | | 38,000 | 4,151 | | 2,196 |

Table T-3. Dragline/Dozer Tandem System - Central Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 1 | Dozer 400 H.P. | Site Preparation | 325 | 25 | 5 | 60 |
| 1 | Dozer 400 H.P. | Site Preparation | 325 | 25 | 6 | 50 |
| 1 | Grader 250 H.P. | Site Prep. & Reclamation | 200 | 20 | 5 | 36 |
| 1 | Grader 250 H.P. | Site Prep. & Reclamation | 200 | 20 | 6 | 30 |
| 2 | Scraper 30 C.Y. | Site Prep. & Reclamation | 700 | 100 | 6 | 100 |
| 2 | Blast Drill 12½ in. | Drilling | 1,800 | 198 | 3 | 534 |
| 2 | Dozer 400 H.P. | Drilling and Blasting | 650 | 50 | 3 | 200 |
| 1 | Dragline 75 C.Y. | Overburden Stripping | 16,000 | 7,360 | 18 | 480 |
| 4 | Dozer 400 H.P. | Interburden Stripping | 1,300 | 100 | 3 | 400 |
| 1 | Shovel 18 C.Y. | Coal Loading | 2,000 | 920 | 18 | 60 |
| 2 | Loader 15 C.Y. | Coal Loading | 550 | 58 | 6 | 82 |
| 5 | Trucks 150 T. | Coal Hauling | 3,200 | 398 | 6 | 467 |
| 1 | Washer 400 TPH | Coal Processing | 5,000 | 212 | 18 | 266 |
| 2 | Dozer 400 H.P. | Rough Grading | 650 | 50 | 6 | 100 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 6 | Trucks: Supply & Pick-Up | Miscellaneous | 75 | 15 | 3 | 20 |
| 1 | Cable Reeler | Electrical | 65 | 5 | 10 | 6 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 600 | 30 | 18 | 32 |
| Set | Pit Electrics | Electrical | 600 | 30 | 18 | 32 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 1,000 | 100 | 18 | 50 |
| Set | Buildings | Miscellaneous | 750 | 30 | 18 | 40 |
| Set | Roads | Miscellaneous | 150 | -- | 18 | 8 |
| Set | Tools | Miscellaneous | 1,000 | 244 | 18 | 42 |
| | Start-Up & Contingency | Miscellaneous | 5,600 | -- | 18 | 311 |
| Totals | | | 43,000 | 10,010 | | 3,430 |

Table T-4. Bucket Wheel Excavator/Conveyor System - Central Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 2 | Dozer 400 H.P. | Site Preparation | 650 | 62 | 6 | 98 |
| 2 | Grader 250 H.P. | Site Prep. & Reclamation | 400 | 43 | 7 | 51 |
| 1 | Scraper 30 C.Y. | Site Prep. & Reclamation | 350 | 49 | 7 | 43 |
| 2 | BWE 25 Ft. Wheel | Overburden Stripping | 13,700 | 1,380 | 20 | 616 |
| 1 | BWE 25 Ft. Wheel | Intbdn. & Coal Removal | 6,850 | 690 | 20 | 308 |
| 1 | Stacker with Tripper Car | Overburden Spoil | 5,000 | 500 | 20 | 225 |
| 1 | Conveyor Gantry Bridge | Overburden Haulage | 550 | 50 | 20 | 25 |
| 1 | Belt Wagon | Interburden Haulage | 1,640 | 160 | 20 | 74 |
| 1 | Conveyor 48 in., 4500 Ft. | Overburden Haulage | 2,475 | 235 | 20 | 112 |
| 1 | Conveyor 60 in., 5100 Ft. | Overburden Haulage | 3,060 | | 20 | 135 |
| 2 | Dozers 400 H.P. | Stripping | 650 | 65 | 3 | 195 |
| 1 | Conveyor 48 in., 14,500 Ft. | Coal Haulage | 7,540 | 740 | 20 | 340 |
| 1 | Washer 400 TPH | Coal Processing | 5,000 | 60 | 20 | 247 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 6 | Trucks: Supply & Pick-Up | Miscellaneous | 75 | 15 | 3 | 20 |
| 3 | Cable Reelers | Electrical | 190 | 20 | 10 | 17 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 1,000 | 100 | 20 | 45 |
| Set | Pit Electrics | Electrical | 1,500 | 100 | 20 | 70 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 1,000 | 100 | 20 | 45 |
| Set | Buildings | Miscellaneous | 750 | 30 | 20 | 36 |
| Set | Roads | Miscellaneous | 150 | -- | 20 | 8 |
| Set | Tools | Miscellaneous | 1,000 | 260 | 20 | 37 |
| | Start-Up & Contingency | Miscellaneous | 7,210 | -- | 20 | 360 |
| Totals | | | 61,000 | 5,039 | | 3,131 |

Table T-5. Single Dragline Horseshoe Method - Western Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 1 | Dozer 400 H.P. | Site Preparation | 325 | 25 | 5 | 60 |
| 1 | Grader 250 H.P. | Site Prep. & Reclamation | 200 | 20 | 5 | 36 |
| 1 | Scraper 30 C.Y. | Site Prep. & Reclamation | 350 | 50 | 6 | 50 |
| 1 | Blast Drill 12½ in. | Drilling | 900 | 99 | 3 | 267 |
| 1 | Dozer 400 H.P. | Drilling and Blasting | 325 | 25 | 3 | 100 |
| 1 | Dragline 75 C.Y. | Stripping | 16,000 | 1,600 | 30 | 480 |
| 1 | Dozer 400 H.P. | Stripping | 325 | 25 | 3 | 100 |
| 1 | Shovel 22 C.Y. | Coal Loading | 2,500 | 250 | 30 | 75 |
| 1 | F.E.L. 15 C.Y. | Coal Loading | 550 | 50 | 5 | 100 |
| 6 | Trucks 150 T. | Coal Hauling | 3,840 | 480 | 5 | 672 |
| 1 | Dry Plant 1400 TPH | Coal Processing | 5,000 | 200 | 30 | 160 |
| 1 | Dozer 400 H.P. | Rough Grading | 325 | 25 | 6 | 50 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 9 | Trucks: Supply & Pick-Up | Miscellaneous | 100 | 16 | 3 | 28 |
| 1 | Cable Reeler | Electrical | 65 | 5 | 10 | 6 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 600 | 30 | 30 | 19 |
| Set | Pit Electrics | Electrical | 600 | 30 | 30 | 19 |
| Set | Buildings | Miscellaneous | 700 | 10 | 30 | 23 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 800 | 50 | 30 | 25 |
| Set | Roads | Miscellaneous | 150 | -- | 30 | 5 |
| Set | Tools | Miscellaneous | 1,000 | 250 | 30 | 25 |
| | Start-Up & Contingency | Miscellaneous | 5,585 | -- | 30 | 186 |
| Totals | | | 40,500 | 3,260 | | 2,510 |

Table T-6. Single Dragline - Elevated Bench Method - Western Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 1 | Dozer 400 H.P. | Site Preparation | 325 | 25 | 4 | 75 |
| 1 | Grader 250 H.P. | Site Prep. & Reclamation | 200 | 20 | 4 | 45 |
| 1 | Scraper 30 C.Y. | Site Prep. & Reclamation | 350 | 50 | 5 | 60 |
| 1 | Blast Drill 12½ in. | Drilling | 900 | 99 | 3 | 267 |
| 1 | Dozer 400 H.P. | Drilling and Blasting | 325 | 25 | 3 | 100 |
| 1 | Dragline 75 C.Y. | Stripping | 16,000 | 4,000 | 25 | 480 |
| 1 | Dozer 400 H.P. | Stripping | 325 | 25 | 3 | 100 |
| 1 | Shovel 22 C.Y. | Coal Loading | 2,500 | 250 | 25 | 90 |
| 1 | F.E.L. 15 C.Y. | Coal Loading | 550 | 50 | 4 | 125 |
| 6 | Trucks 150 T. | Coal Hauling | 3,840 | 480 | 4 | 840 |
| 1 | Dry Plant 1400 TPH | Coal Processing | 5,000 | 200 | 25 | 192 |
| 1 | Dozer 400 H.P. | Rough Grading | 325 | 25 | 5 | 60 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 9 | Trucks: Supply & Pick-Up | Miscellaneous | 100 | 16 | 3 | 28 |
| 1 | Cable Reeler | Electrical | 65 | 5 | 10 | 6 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 600 | 25 | 25 | 23 |
| Set | Pit Electrics | Electrical | 600 | 25 | 25 | 23 |
| Set | Buildings | Miscellaneous | 700 | 25 | 25 | 27 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 800 | 50 | 25 | 30 |
| Set | Roads | Miscellaneous | 150 | -- | 25 | 6 |
| Set | Tools | Miscellaneous | 1,000 | 250 | 25 | 30 |
| | Start-Up & Contingency | Miscellaneous | 5,585 | -- | 25 | 223 |
| Totals | | | 40,500 | 5,665 | | 2,854 |

Table T-7. Shovel/Truck/Dragline Tandem System - Western Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 2 | Dozers 400 H.P. | Site Preparation | 650 | 50 | 6 | 100 |
| 2 | Graders 250 H.P. | Site Prep. & Reclamation | 400 | 36 | 7 | 52 |
| 2 | Scrapers 30 C.Y. | Site Prep. & Reclamation | 700 | 98 | 7 | 86 |
| 2 | Blast Drill 12½ in. | Drilling | 1,800 | 200 | 4 | 400 |
| 2 | Dozers 400 H.P. | Drilling and Blasting | 650 | 50 | 4 | 150 |
| 1 | Dragline 50 C.Y. | Interburden Stripping | 13,000 | 6,920 | 19 | 320 |
| 2 | Shovels 18 C.Y. | Overburden Stripping | 4,000 | 580 | 19 | 180 |
| 6 | Rock Trucks 85 T. | Overburden Stripping | 2,460 | 246 | 3 | 738 |
| 2 | Dozers 400 H.P. | Overburden Stripping | 650 | 50 | 3 | 200 |
| 2 | Shovels 22 C.Y. | Coal Loading | 5,000 | 1,580 | 19 | 180 |
| 2 | F.E.L. 15 C.Y. | Coal Loading | 1,100 | 98 | 6 | 167 |
| 12 | Trucks 150 T. | Coal Hauling | 7,680 | 960 | 6 | 1,120 |
| 1 | Dry Plant 1400 TPH | Coal Processing | 6,000 | 262 | 19 | 302 |
| 1 | Dozer 400 H.P. | Rough Grading | 325 | 25 | 4 | 75 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 9 | Trucks: Supply & Pick-Up | Miscellaneous | 100 | 16 | 3 | 28 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| 1 | Cable Reeler | Electrical | 65 | 5 | 10 | 6 |
| Set | Power Line & Main Transformer | Electrical | 600 | 30 | 19 | 30 |
| Set | Pit Electrics | Electrical | 900 | 45 | 19 | 45 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 800 | 40 | 19 | 40 |
| Set | Buildings | Miscellaneous | 700 | 35 | 19 | 35 |
| Set | Roads | Miscellaneous | 150 | -- | 19 | 8 |
| Set | Tools | Miscellaneous | 1,000 | 240 | 19 | 40 |
| | Start-Up & Contingency | Miscellaneous | 7,010 | -- | 19 | 369 |
| Totals | | | 56,000 | 11,586 | | 4,695 |

Table T-8. Shovel/Truck System - Western Region
Capital Cost Estimate

| Quantity | Description | Usage | Capital Cost (Thous. \$) | Salvage Value (Thous. \$) | Depreciation Schedule (Years) | Yearly Charge (Thous. \$) |
|----------|-------------------------------|--------------------------|-----------------------------|------------------------------|----------------------------------|------------------------------|
| 2 | Dozers 400 H.P. | Site Preparation | 650 | 50 | 6 | 100 |
| 2 | Graders 250 H.P. | Site Prep. & Reclamation | 400 | 40 | 6 | 60 |
| 2 | Scrapers 30 C.Y. | Site Prep. & Reclamation | 700 | 98 | 7 | 86 |
| 2 | Blast Drills 12½ in. | Drilling | 1,800 | 200 | 4 | 400 |
| 2 | Dozers 400 H.P. | Drilling and Blasting | 650 | 50 | 4 | 150 |
| 4 | Shovels 18 C.Y. | Stripping | 8,000 | 1,160 | 19 | 360 |
| 16 | Rock Trucks | Stripping | 6,560 | 656 | 3 | 1,968 |
| 3 | Dozers 400 H.P. | Stripping | 975 | 75 | 3 | 300 |
| 2 | Shovels 22 C.Y. | Coal Loading | 5,000 | 250 | 19 | 250 |
| 2 | F.E.L. 15 C.Y. | Coal Loading | 1,100 | 98 | 6 | 167 |
| 12 | Trucks 150 T. | Coal Hauling | 7,680 | 960 | 6 | 1,120 |
| 1 | Dry Plant 1400 TPH | Coal Processing | 6,000 | 300 | 19 | 300 |
| 1 | Dozer 400 H.P. | Rough Grading | 325 | 25 | 4 | 75 |
| 1 | Truck | Road Wetting | 75 | 5 | 10 | 7 |
| 1 | Truck | Crane | 75 | 5 | 10 | 7 |
| 9 | Trucks: Supply & Pick-Up | Miscellaneous | 100 | 16 | 3 | 28 |
| 1 | Drill | Prospecting | 110 | 10 | 10 | 10 |
| Set | Power Line & Main Transformer | Electrical | 600 | 30 | 19 | 30 |
| Set | Pit Electrics | Electrical | 900 | 45 | 19 | 45 |
| Set | R.R. Yard & Lead Tracks | Rail Transport | 800 | 40 | 19 | 40 |
| Set | Buildings | Miscellaneous | 700 | 35 | 19 | 35 |
| Set | Roads | Miscellaneous | 150 | -- | 19 | 8 |
| Set | Tools | Miscellaneous | 1,000 | 240 | 19 | 40 |
| | Start-Up & Contingency | Miscellaneous | 6,650 | -- | 19 | 350 |
| Totals | | | 51,000 | 4,388 | | 5,936 |

Table T-9. Single Dragline Horseshoe Method
Central Region
Annual Operating Costs Estimate
Coal Production 908,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|---|---|--|--|--|--|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 1 Dragline, 4 C.Y. (leased) | 4,800 | 36.00 | 173 | | |
| 1 Dozer, 400 H.P. | 4,800 | 42.56 | 204 | | |
| 1 Motor Grader, 250 H.P. | 3,800 | 21.25 | 81 | | |
| 1 Elevating Scraper, 30 C.Y. | 2,000 | 31.72 | 63 | | |
| Subtotal, Site Preparation | | | 521 | 0.05 | 0.57 |
| <u>Drilling and Blasting</u> | | | | | |
| 1 Blasthole Drill, 12½ inc. | 8,030 | 53.10 | 426 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Explosives, ANFO | | | 1,090 | | |
| Subtotal, Drilling & Blasting | | | 1,858 | 0.17 | 2.05 |
| <u>Stripping</u> | | | | | |
| 1 Dragline, 75 C.Y. | 8,030 | 306.93 | 2,465 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Subtotal, Stripping | | | 2,807 | 0.26 | 3.09 |
| <u>Coal Loading</u> | | | | | |
| 1 Shovel, 18 C.Y. | 4,000 | 70.67 | 283 | | |
| 1 Loader, 12 C.Y. | 4,000 | 51.00 | 204 | | |
| Explosives, ANFO | | | 25 | | |
| Subtotal, Coal Loading | | | 512 | 0.05 | 0.56 |
| <u>Coal Hauling</u> | | | | | |
| 3 Coal Haulers, 150 T. | 12,000 | 33.50 | 402 | 0.04 | 0.44 |
| <u>Coal Processing</u> | | | | | |
| 1 Washer, 400 TPH | 4,000 | 91.00 | 363 | 0.03 | 0.40 |
| <u>Rough Grading</u> | | | | | |
| 1 Dozer, 400 H.P. | 4,000 | 42.56 | 170 | 0.02 | 0.19 |
| <u>Final Reclamation</u> | | | | | |
| 1 Scraper, 30 C.Y. | 2,000 | 31.72 | 63 | | |
| 1 Grader, 250 H.P. | 1,000 | 21.25 | 21 | | |
| 1 Hydroseeder | 1,000 | 35.00 | 35 | | |
| Subtotal, Final Reclamation | | | 119 | 0.01 | 0.13 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 6,752 | 0.62 | 7.44 |
| NON-FUNCTIONAL VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 354 | | |
| UMWA Pension Fund | | | 1,406 | | |
| Insurance | | | 136 | | |
| Royalties & Lease Rent | | | 907 | | |
| Subtotal, Non-Functional/Variable Costs | | | 2,803 | 0.26 | 3.09 |
| NON-FUNCTIONAL FIXED COSTS | | | | | |
| General & Administrative | | | 905 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 500 | | |
| Miscellaneous | | | 273 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 1,973 | 0.18 | 2.17 |
| TOTAL COST | | | 11,528 | 1.06 | 12.69 |

Table T-10. Single Dragline - Elevated Bench Method
 Central Region
 Annual Operating Costs Estimate
 Coal Production 1,061,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 1 Dragline, 4 C.Y. | 5,200 | 36.00 | 187 | | |
| 1 Dozer, 400 H.P. | 5,200 | 42.56 | 221 | | |
| 1 Motor Grader, 250 H.P. | 4,160 | 21.25 | 88 | | |
| 1 Elevating Scraper, 30 C.Y. | 2,200 | 31.72 | 70 | | |
| Subtotal, Site Preparation | | | 566 | 0.05 | 0.54 |
| <u>Drilling and Blasting</u> | | | | | |
| 1 Blasting Drill, 12½ in. | 8,030 | 53.10 | 426 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Explosives | | | 1,061 | | |
| Subtotal, Drilling & Blasting | | | 1,829 | 0.14 | 1.72 |
| <u>Stripping</u> | | | | | |
| 1 Dragline, 75 C.Y. | 8,030 | 306.93 | 2,465 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Subtotal, Stripping | | | 2,807 | 0.22 | 2.64 |
| <u>Coal Loading</u> | | | | | |
| 1 Shovel, 18 C.Y. | 4,700 | 70.67 | 332 | | |
| 1 F.E.L., 12 C.Y. | 4,700 | 51.00 | 240 | | |
| Explosives | | | 21 | | |
| Subtotal, Coal Loading | | | 593 | 0.05 | 0.56 |
| <u>Coal Hauling</u> | | | | | |
| 3 Coal Haulers, 150 T. | 14,000 | 33.50 | 469 | | 0.44 |
| <u>Coal Processing</u> | | | | | |
| 1 Washer, 400 TPH | 4,700 | 91.00 | 428 | | 0.40 |
| <u>Rough Grading</u> | | | | | |
| 1 Dozer, 400 H.P. | 4,700 | 42.56 | 200 | | 0.19 |
| <u>Final Reclamation</u> | | | | | |
| 1 Scraper, 30 C.Y. | 2,200 | 31.72 | 70 | | |
| 1 Grader, 250 H.P. | 1,040 | 21.25 | 22 | | |
| 1 Hydroseeder | 1,290 | 35.00 | 42 | | |
| Subtotal, Final Reclamation | | | 134 | 0.01 | 0.12 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 7,026 | 0.55 | 6.62 |
| NON-FUNCTIONAL/VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 414 | | |
| UMWA Pension Fund | | | 1,645 | | |
| Insurance | | | 159 | | |
| Royalties & Lease Rent | | | 1,049 | | |
| Subtotal, Non-Functional/Variable Costs | | | 3,267 | 0.26 | 3.09 |
| NON-FUNCTIONAL/FIXED COSTS | | | | | |
| General & Administrative | | | 950 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 500 | | |
| Miscellaneous | | | 272 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 1,972 | 0.15 | 1.86 |
| TOTAL COST | | | 12,265 | 0.95 | 11.57 |

Table T-11. Dragline/Dozer Tandem System
Central Region
Annual Operating Costs Estimate
Coal Production 1,692,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 2 Draglines, 4 C.Y. | 7,600 | 36.00 | 274 | | |
| 2 Dozers, 400 H.P. | 7,600 | 42.56 | 323 | | |
| 1 Motor Grader, 250 H.P. | 6,300 | 21.25 | 134 | | |
| 1 Elevating Scraper, 30 C.Y. | 3,300 | 31.72 | <u>105</u> | | |
| Subtotal, Site Preparation | | | 836 | 0.04 | 0.49 |
| <u>Drilling and Blasting</u> | | | | | |
| 2 Blasting Drills, 124 in. | 13,200 | 53.10 | 701 | | |
| 2 Dozers, 400 H.P. | 13,200 | 42.56 | 562 | | |
| Explosives | | | <u>1,692</u> | | |
| Subtotal, Drilling & Blasting | | | 2,955 | 0.15 | 1.75 |
| <u>Stripping</u> | | | | | |
| 1 Dragline, 75 C.Y. | 8,030 | 306.93 | 2,465 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| 3 Dozers, 400 H.P. | 24,090 | 42.56 | <u>1,025</u> | | |
| Subtotal, Stripping | | | 3,832 | 0.19 | 2.26 |
| <u>Coal Loading</u> | | | | | |
| 1 Shovel, 18 C.Y. | 4,000 | 70.67 | 283 | | |
| 2 F.E.L., 15 C.Y. | 8,000 | 51.00 | 408 | | |
| Explosives | | | <u>33</u> | | |
| Subtotal, Coal Loading | | | 724 | 0.04 | 0.43 |
| <u>Coal Hauling</u> | | | | | |
| 5 Coal Haulers, 150 T. | 20,000 | 33.50 | 670 | 0.03 | 0.40 |
| <u>Coal Processing</u> | | | | | |
| 1 Washer, 400 TPH | 6,600 | 91.00 | 600 | 0.03 | 0.36 |
| <u>Rough Grading</u> | | | | | |
| 2 Dozers, 400 H.P. | 6,600 | 42.56 | 281 | 0.01 | 0.17 |
| <u>Final Reclamation</u> | | | | | |
| 1 Scraper, 30 C.Y. | 3,300 | 31.72 | 105 | | |
| 1 Motor Scraper, 250 H.P. | 1,600 | 21.25 | 34 | | |
| 1 Hydroseeder | 1,600 | 35.00 | <u>56</u> | | |
| Subtotal, Final Reclamation | | | <u>195</u> | <u>0.01</u> | <u>0.11</u> |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 10,093 | 0.50 | 5.96 |
| NON-FUNCTIONAL VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 660 | | |
| UMWA Pension Fund | | | 2,623 | | |
| Insurance | | | 254 | | |
| Royalties & Lease Rent | | | <u>1,690</u> | | |
| Subtotal, Non-Functional/Variable Costs | | | 5,227 | 0.26 | 3.09 |
| NON-FUNCTIONAL FIXED COSTS | | | | | |
| General & Administrative | | | 950 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 500 | | |
| Miscellaneous | | | <u>272</u> | | |
| Subtotal, Non-Functional/Fixed Costs | | | <u>1,972</u> | <u>0.10</u> | <u>1.16</u> |
| TOTAL COST | | | 17,292 | 0.86 | 10.22 |

Table T-12. BWE/Conveyor System
 Central Region
 Annual Operating Costs Estimate
 Coal Production 1,526,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 2 Draglines, 4 C.Y. (leased) | 8,070 | 36.00 | 291 | | |
| 2 Dozers, 400 H.P. | 8,070 | 42.56 | 343 | | |
| 2 Motor Graders, 250 H.P. | 6,400 | 21.25 | 136 | | |
| 1 Elevating Scraper, 30 C.Y. | 3,360 | 31.72 | 107 | | |
| Subtotal, Site Preparation | | | 877 | 0.05 | 0.57 |
| <u>Stripping</u> | | | | | |
| 2 BWE, 25 ft. Wheel | 16,060 | 109.00 | 1,751 | | |
| 1 BWE, 25 ft. Wheel | 5,350 | 109.00 | 583 | | |
| 1 Stacker with Tripper Car | 8,030 | 85.00 | 683 | | |
| 1 Conveyor Gantry Bridge | 8,030 | 30.00 | 241 | | |
| 1 Belt Wagon | 8,030 | 48.00 | 385 | | |
| Conveyors, 48 in. | 8,030 | 115.00 | 923 | | |
| Conveyors, 60 in. | 8,030 | 115.00 | 923 | | |
| 2 Dozers, 400 H.P. | 16,060 | 42.56 | 683 | | |
| Subtotal, Stripping | | | 6,172 | 0.34 | 4.04 |
| <u>Coal Loading</u> | | | | | |
| 1 BWE, 25 ft. Wheel | 2,680 | 100.00 | 268 | | |
| <u>Coal Hauling</u> | | | | | |
| 1 Conveyor, 48 in. | 2,680 | 110.00 | 295 | | |
| Subtotal, Coal Hauling | | | 563 | 0.03 | 0.37 |
| <u>Coal Processing</u> | | | | | |
| 1 Washer, 400 TPH | 6,720 | 91.00 | 611 | 0.03 | 0.04 |
| <u>Final Reclamation</u> | | | | | |
| 1 Scraper, 30 C.Y. | 3,360 | 31.72 | 107 | | |
| 1 Grader, 250 H.P. | 1,680 | 21.25 | 36 | | |
| 1 Hydroseeder | 1,680 | 35.00 | 59 | | |
| Subtotal, Final Reclamation | | | 202 | 0.01 | 0.13 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 8,425 | 0.46 | 5.52 |
| NON-FUNCTIONAL VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 595 | | |
| UMWA Pension Fund | | | 2,365 | | |
| Insurance | | | 229 | | |
| Royalties & Lease Rent | | | 1,526 | | |
| Subtotal, Non-Functional Variable Costs | | | 4,715 | 0.26 | 3.09 |
| NON-FUNCTIONAL FIXED COSTS | | | | | |
| General & Administrative | | | 905 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 500 | | |
| Miscellaneous | | | 273 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 1,973 | 0.11 | 1.29 |
| TOTAL COST | | | 15,113 | 0.82 | 9.90 |

Table T-13. Single Dragline Horseshoe Method
Western Region
Annual Operating Costs Estimate
Coal Production 4,280,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 1 Dragline, 3-4 C.Y. | 4,800 | 36.00 | 173 | | |
| 1 Dozer, 400 H.P. | 4,800 | 42.56 | 204 | | |
| 1 Motor Grader, 250 H.P. | 3,800 | 21.25 | 81 | | |
| 1 Elevating Scraper | 2,000 | 31.72 | 63 | | |
| Subtotal, Site Preparation | | | 521 | 0.04 | 0.12 |
| <u>Drilling and Blasting</u> | | | | | |
| 1 Blasthole Drill, 12½ in. | 8,030 | 53.10 | 426 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Explosives | | | 1,340 | | |
| Subtotal, Drilling & Blasting | | | 2,108 | 0.18 | 0.49 |
| <u>Stripping</u> | | | | | |
| 1 Dragline, 75 C.Y. | 8,030 | 306.93 | 2,465 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Subtotal, Stripping | | | 2,807 | 0.23 | 0.65 |
| <u>Coal Loading</u> | | | | | |
| 1 Shovel, 22 C.Y. | 4,800 | 70.67 | 339 | | |
| 1 F.E.L., 15 C.Y. | 4,800 | 57.25 | 275 | | |
| Explosives | | | 282 | | |
| Subtotal, Coal Loading | | | 896 | 0.07 | 0.20 |
| <u>Coal Hauling</u> | | | | | |
| 6 Coal Haulers, 150 T. | 28,800 | 33.50 | 965 | 0.08 | 0.23 |
| <u>Coal Processing</u> | | | | | |
| 1 Preparation Plant, 1400 TPH | 4,800 | 166.74 | 800 | 0.07 | 0.19 |
| <u>Rough Grading</u> | | | | | |
| 1 Dozer, 400 H.P. | 4,000 | 42.56 | 170 | 0.01 | 0.04 |
| <u>Final Reclamation</u> | | | | | |
| 1 Scraper, 30 C.Y. | 2,000 | 31.72 | 63 | | |
| 1 Grader, 250 H.P. | 1,000 | 21.25 | 21 | | |
| 1 Hydroseeder | 1,000 | 35.00 | 35 | | |
| Subtotal, Final Reclamation | | | 119 | 0.01 | 0.03 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 8,386 | 0.70 | 1.96 |
| NON-FUNCTIONAL/VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 7,062 | | |
| UMWA Pension Fund | | | 6,634 | | |
| Insurance | | | 85 | | |
| Royalties & Lease Rent | | | 1,449 | | |
| Subtotal, Non-Functional/Variable Costs | | | 15,230 | 1.28 | 3.56 |
| NON-FUNCTIONAL/FIXED COSTS | | | | | |
| General & Administrative | | | 1,883 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 1,626 | | |
| Miscellaneous | | | 257 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 4,016 | 0.34 | 0.94 |
| TOTAL COST | | | 27,632 | 2.32 | 6.46 |

Table T-14. Single Dragline Elevated Bench Method
 Western Region
 Annual Operating Costs Estimate
 Coal Production 5,155,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 1 Dragline, 3 C.Y. | 5,800 | 36.00 | 209 | | |
| 1 Dozer, 400 H.P. | 5,800 | 42.56 | 247 | | |
| 1 Motor Grader, 250 H.P. | 4,600 | 21.25 | 98 | | |
| 1 Elevating Scraper, 30 C.Y. | 2,400 | 31.72 | 76 | | |
| Subtotal, Site Preparation | | | 630 | 0.04 | 0.12 |
| <u>Drilling and Blasting</u> | | | | | |
| 1 Blasthole Drill, 12½ in. | 8,030 | 53.10 | 426 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 372 | | |
| Explosives | | | 1,614 | | |
| Subtotal, Drilling & Blasting | | | 2,412 | 0.17 | 0.47 |
| <u>Stripping</u> | | | | | |
| 1 Dragline, 75 C.Y. | 8,030 | 306.93 | 2,465 | | |
| 1 Dozer, 400 H.P. | 8,030 | 42.56 | 342 | | |
| Subtotal, Stripping | | | 2,807 | 0.20 | 0.54 |
| <u>Coal Loading</u> | | | | | |
| 1 Shovel, 22 C.Y. | 5,800 | 70.67 | 410 | | |
| 1 F.E.L., 15 C.Y. | 5,800 | 57.25 | 332 | | |
| Explosives | | | 340 | | |
| Subtotal, Coal Loading | | | 1,082 | 0.07 | 0.19 |
| <u>Coal Hauling</u> | | | | | |
| 6 Coal Haulers, 150 T. | 34,800 | 33.50 | 1,166 | 0.08 | 0.23 |
| <u>Coal Processing</u> | | | | | |
| 1 Preparation Plant, 1400 TPH | 5,800 | 166.74 | 967 | 0.07 | 0.19 |
| <u>Rough Grading</u> | | | | | |
| 1 Dozer, 400 H.P. | 4,800 | 42.56 | 204 | 0.01 | 0.04 |
| <u>Final Reclamation</u> | | | | | |
| 1 Scraper, 30 C.Y. | 2,400 | 31.72 | 76 | | |
| 1 Grader, 250 H.P. | 1,200 | 21.25 | 26 | | |
| 1 Hydroseeder | 1,200 | 35.00 | 42 | | |
| Subtotal, Final Reclamation | | | 144 | 0.01 | 0.03 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 9,412 | 0.66 | 1.82 |
| NON-FUNCTIONAL/VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 8,506 | | |
| UMWA Pension Fund | | | 7,990 | | |
| Insurance | | | 103 | | |
| Royalties & Lease Rent | | | 1,836 | | |
| Subtotal, Non-Functional/Variable Costs | | | 18,435 | 1.29 | 3.57 |
| NON-FUNCTIONAL/FIXED COSTS | | | | | |
| General & Administrative | | | 1,883 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 1,626 | | |
| Miscellaneous | | | 257 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 4,016 | 0.28 | 0.78 |
| TOTAL COST | | | 31,863 | 2.22 | 6.17 |

Table T-15. Shovel/Truck/Dragline Tandem System
Western Region
Annual Operating Costs Estimate
Coal Production 6,968,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COSTS | | | | | |
| <u>Site Preparation</u> | | | | | |
| 2 Draglines, 3 C.Y. | 7,800 | 36.00 | 281 | | |
| 2 Dozers, 400 H.P. | 7,800 | 42.56 | 332 | | |
| 2 Motor Graders, 250 H.P. | 5,450 | 21.25 | 116 | | |
| 1 Elevating Scraper, 30 C.Y. | 3,240 | 31.72 | 103 | | |
| Subtotal, Site Preparation | | | 832 | 0.04 | 0.12 |
| <u>Drilling & Blasting</u> | | | | | |
| 2 Blasthole Drills, 12½ in. | 1,300 | 53.10 | 690 | | |
| 2 Dozers, 400 H.P. | 1,300 | 42.56 | 553 | | |
| Explosives | | | 2,181 | | |
| Subtotal, Drilling & Blasting | | | 3,424 | 0.18 | 0.49 |
| <u>Stripping</u> | | | | | |
| 1 Dragline, 50 C.Y. | 8,030 | 216.18 | 1,736 | | |
| 2 Shovels, 18 C.Y. | 16,060 | 70.67 | 1,135 | | |
| 6 Trucks, 85 T. | 48,180 | 45.87 | 2,210 | | |
| 2 Dozers, 400 H.P. | 16,060 | 42.56 | 684 | | |
| Subtotal, Stripping | | | 5,765 | 0.30 | 0.83 |
| <u>Coal Loading</u> | | | | | |
| 2 Shovels, 22 C.Y. | 7,800 | 70.67 | 551 | | |
| 2 F.E.L., 15 C.Y. | 7,800 | 57.25 | 447 | | |
| Explosives | | | 460 | | |
| Subtotal, Coal Loading | | | 1,458 | 0.07 | 0.21 |
| <u>Coal Hauling</u> | | | | | |
| Coal Haulers, 150 T. | 46,700 | 33.50 | 1,564 | | 0.22 |
| <u>Coal Processing</u> | | | | | |
| 1 Preparation Plant, 1400 TPH | 7,800 | 166.74 | 1,301 | | 0.19 |
| <u>Rough Grading</u> | | | | | |
| 1 Dozer, 400 H.P. | 6,500 | 42.56 | 277 | | 0.04 |
| <u>Final Reclamation</u> | | | | | |
| 1 Elevating Scraper, 30 C.Y. | 3,240 | 31.72 | 103 | | |
| 1 Motor Grader, 250 H.P. | 1,600 | 21.25 | 34 | | |
| 1 Hydroseeder | 1,600 | 35.00 | 56 | | |
| Subtotal, Final Reclamation | | | 193 | 0.01 | 0.03 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 14,814 | 0.76 | 2.13 |
| NON-FUNCTIONAL/VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 11,497 | | |
| UMWA Pension Fund | | | 10,800 | | |
| Insurance | | | 139 | | |
| Royalties & Lease Rent | | | 2,439 | | |
| Subtotal, Non-Functional/Variable Costs | | | 24,875 | 1.28 | 3.57 |
| NON-FUNCTIONAL/FIXED COSTS | | | | | |
| General & Administrative | | | 1,883 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 1,626 | | |
| Miscellaneous | | | 257 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 4,016 | 0.21 | 0.58 |
| TOTAL COST | | | 43,705 | 2.26 | 6.27 |

Table T-16. Shovel/Truck System
Western Region
Annual Operating Costs Estimate
Coal Production 6,752,000 T/Year

| Cost Item | Scheduled Hours Per Year (Hours) | Cash Operating Costs Per Scheduled Hour (\$) | Cash Operating Cost Per Year (Thous. \$) | Cash Cost Per Bank Cubic Yard (\$) | Cash Cost Per Ton of Coal (\$) |
|--|----------------------------------|--|--|------------------------------------|--------------------------------|
| FUNCTIONAL/VARIABLE COST | | | | | |
| <u>Site Preparation</u> | | | | | |
| 2 Draglines, 3 C.Y. | 7,600 | 36.00 | 274 | | |
| 2 Dozers, 400 H.P. | 7,600 | 42.56 | 323 | | |
| 2 Motor Graders, 250 H.P. | 6,000 | 21.25 | 128 | | |
| 1 Elevating Scraper, 30 C.Y. | 3,200 | 31.72 | 102 | | |
| Subtotal, Site Preparation | | | 827 | 0.04 | 0.12 |
| <u>Drilling and Blasting</u> | | | | | |
| 2 Blasthole Drills, 12½ in. | 12,700 | 53.10 | 674 | | |
| 2 Dozers, 400 H.P. | 12,700 | 42.56 | 541 | | |
| Explosives | | | 2,113 | | |
| Subtotal, Drilling & Blasting | | | 3,328 | 0.18 | 0.49 |
| <u>Stripping</u> | | | | | |
| 4 Shovels, 18 C.Y. | 32,120 | 70.67 | 2,270 | | |
| 16 Haul Trucks, 85 T. | 128,480 | 45.87 | 5,893 | | |
| 3 Dozers, 400 H.P. | 24,090 | 42.56 | 1,025 | | |
| Subtotal, Stripping | | | 9,188 | 0.49 | 1.36 |
| <u>Coal Loading</u> | | | | | |
| 2 Shovels, 22 C.Y. | 7,600 | 70.67 | 537 | | |
| 2 F.E.L., 15 C.Y. | 7,600 | 57.25 | 435 | | |
| Explosives | | | 445 | | |
| Subtotal, Coal Loading | | | 1,417 | 0.07 | 0.21 |
| <u>Coal Hauling</u> | | | | | |
| 12 Coal Haulers, 150 T. | 45,500 | 33.50 | 1,524 | 0.08 | 0.23 |
| <u>Coal Processing</u> | | | | | |
| 1 Preparation Plant, 1400 TPH | 7,600 | 166.74 | 1,267 | 0.07 | 0.19 |
| <u>Rough Grading</u> | | | | | |
| 1 Dozer, 400 H.P. | 6,320 | 42.56 | 269 | 0.01 | 0.04 |
| <u>Final Reclamation</u> | | | | | |
| 1 Elevating Scraper, 30 C.Y. | 3,200 | 31.72 | 102 | | 0.01 |
| 1 Motor Grader, 250 H.P. | 1,600 | 21.25 | 34 | | 0.01 |
| 1 Hydroseeder | 1,600 | 35.00 | 56 | | 0.01 |
| Subtotal, Final Reclamation | | | 192 | 0.01 | 0.03 |
| SUBTOTAL, FUNCTIONAL/VARIABLE COSTS | | | 18,012 | 0.96 | 2.67 |
| NON-FUNCTIONAL VARIABLE COSTS | | | | | |
| Coal Severance Taxes (State & Federal) | | | 11,141 | | |
| UMWA Pension Fund | | | 10,466 | | |
| Insurance | | | 135 | | |
| Royalties & Lease Rent | | | 2,393 | | |
| Subtotal, Non-Functional/Variable Costs | | | 24,135 | 1.28 | 3.57 |
| NON-FUNCTIONAL FIXED COSTS | | | | | |
| General & Administrative | | | 1,883 | | |
| Engineering Staff, Prospecting & Land Management | | | 250 | | |
| Taxes, Property & Personal Property | | | 1,626 | | |
| Miscellaneous | | | 257 | | |
| Subtotal, Non-Functional/Fixed Costs | | | 4,016 | 0.21 | 0.59 |
| TOTAL COST | | | 46,163 | 2.46 | 6.83 |

Table T-17. Single Dragline Horseshoe Method
 Central Region
 Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS \$) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|-----------------------------------|------------------------|---------------------------|---------------------------------|------------------------------|-----------------------------------|--------------------------|
| 0 | 38,000 | 0 | 0 | 0 | 0 | 0 | (38,000) |
| 1 | 0 | 20,430 | 11,528 | 2,054 | 2,043 | 2,403 | 6,499 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 3 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 4 | 1,461 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,038 |
| 5 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 6 | 480 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,019 |
| 7 | 4,221 | ↑ | ↑ | ↑ | ↑ | ↑ | 2,278 |
| 8 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 9 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 10 | 1,461 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,038 |
| 11 | 780 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,719 |
| 12 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 13 | 4,221 | ↑ | ↑ | ↑ | ↑ | ↑ | 2,278 |
| 14 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 15 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 16 | 1,941 | ↑ | ↑ | ↑ | ↑ | ↑ | 4,558 |
| 17 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 18 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 19 | 4,221 | ↑ | ↑ | ↑ | ↑ | ↑ | 2,278 |
| 20 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 21 | 780 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,719 |
| 22 | 1,461 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,038 |
| 23 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 24 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,499 |
| 25 | 4,221 | ↑ | ↑ | ↑ | ↑ | ↑ | 2,278 |
| 26 | 480 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,019 |
| 27 | 0 | ↑ | ↑ | ↓ | ↓ | ↓ | 6,499 |
| 28 | 1,461 | ↑ | ↑ | ↓ | ↓ | ↓ | 5,038 |
| 29 | 0 | ↑ | ↑ | 2,054 | ↓ | 2,403 | 6,499 |
| 30 | 0 | ↑ | ↑ | 2,044 | ↓ | 2,408 | 6,494 |
| 31 | 5,001 | ↑ | ↑ | 1,847 | ↓ | 2,506 | 1,395 |
| 32 | 0 | ↓ | ↓ | 1,847 | ↓ | 2,506 | 6,396 |
| 33 | 0 | 20,430 | 11,528 | 1,847 | 2,043 | 2,506 | 6,396 |
| 34 | (3,039) | 810 | 457 | | 81 | 136 | 3,256 |
| TOTALS | 67,151 | 675,000 | 380,881 | 67,151 | 67,500 | 79,749 | 147,219 |

Table T-18. Single Dragline - Elevated Bench Method
 Central Region
 Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) | | | |
|--------|--------------------------------------|---------------------------|---------------------------|------------------------------------|---------------------------------|---|-----------------------------|-------|-------|-------|
| 0 | 38,000 | 0 | 0 | 0 | 0 | 0 | (38,000) | | | |
| 1 | 0 | 23,872 | 12,265 | 2,196 | 2,387 | 3,512 | 8,095 | | | |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 8,095 | | | |
| 3 | 0 | | | | | | 8,095 | | | |
| 4 | 1,461 | | | | | | 6,634 | | | |
| 5 | 0 | | | | | | 8,095 | | | |
| 6 | 3,240 | | | | | | 4,855 | | | |
| 7 | 1,461 | | | | | | 6,634 | | | |
| 8 | 0 | | | | | | 8,095 | | | |
| 9 | 0 | | | | | | 8,095 | | | |
| 10 | 1,461 | | | | | | 6,634 | | | |
| 11 | 3,540 | | | | | | 4,555 | | | |
| 12 | 0 | | | | | | 8,095 | | | |
| 13 | 1,461 | | | | | | 6,634 | | | |
| 14 | 0 | | | | | | 8,095 | | | |
| 15 | 0 | | | | | | 8,095 | | | |
| 16 | 4,701 | | | | | | 3,394 | | | |
| 17 | 0 | | | | | | 8,095 | | | |
| 18 | 0 | | | | | | 8,095 | | | |
| 19 | 1,461 | | | | | | 6,634 | | | |
| 20 | 0 | | | | | | 8,095 | | | |
| 21 | 3,540 | | | | | | 4,555 | | | |
| 22 | 1,461 | 6,634 | | | | | | | | |
| 23 | 0 | 8,095 | | | | | | | | |
| 24 | 0 | 8,095 | | | | | | | | |
| 25 | 1,461 | 6,634 | | | | | | | | |
| 26 | 3,240 | 4,855 | | | | | | | | |
| 27 | 0 | ↓ | ↓ | 2,196 | ↓ | 3,512 | 8,095 | | | |
| 28 | 0 | | | 23,872 | | 12,265 | 1,689 | 2,387 | 3,766 | 7,841 |
| 29 | (5,507) | | | 6,584 | | 3,383 | 658 | 1,272 | 7,436 | |
| TOTALS | 60,981 | 675,000 | 346,803 | 60,981 | 67,494 | 99,862 | 167,354 | | | |

Table T-19. Dragline/Dozer Tandem System
Central Region
Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS \$) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|--------------------------------------|---------------------------|------------------------------|------------------------------------|---------------------------------|---|-----------------------------|
| 0 | 43,000 | 0 | 0 | 0 | 0 | 0 | (43,000) |
| 1 | 0 | 38,070 | 17,292 | 3,430 | 3,807 | 6,771 | 14,007 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 3 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 4 | 3,462 | ↑ | ↑ | ↑ | ↑ | ↑ | 10,545 |
| 5 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 6 | 480 | ↑ | ↑ | ↑ | ↑ | ↑ | 13,527 |
| 7 | 8,436 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,571 |
| 8 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 9 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 10 | 3,462 | ↑ | ↑ | ↑ | ↑ | ↑ | 10,545 |
| 11 | 780 | ↑ | ↑ | ↑ | ↑ | ↑ | 13,227 |
| 12 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 13 | 8,436 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,571 |
| 14 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 15 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,007 |
| 16 | 3,942 | ↓ | ↓ | ↓ | ↓ | ↓ | 10,065 |
| 17 | 0 | 38,070 | 17,292 | 3,430 | 3,807 | 6,771 | 14,007 |
| 18 | 0 | 27,810 | 12,632 | 3,426 | 2,781 | 4,486 | 10,692 |
| 19 | (10,262) | 0 | 0 | 0 | 0 | 0 | 10,262 |
| TOTALS | 61,736 | 675,000 | 306,596 | 61,736 | 67,500 | 119,593 | 187,075 |

Table T-20. BWE/Conveyor System
Central Region
Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|--------------------------------------|---------------------------|---------------------------|------------------------------------|---------------------------------|---|-----------------------------|
| 0 | 61,000 | 0 | 0 | 0 | 0 | 0 | (61,000) |
| 1 | | 34,335 | 15,113 | 3,131 | 3,433 | 6,329 | 12,893 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 3 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 4 | 645 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,248 |
| 5 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 6 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 7 | 1,233 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,660 |
| 8 | 658 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,235 |
| 9 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 10 | 645 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,248 |
| 11 | 410 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,483 |
| 12 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 13 | 1,233 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,660 |
| 14 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 15 | 658 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,235 |
| 16 | 645 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,248 |
| 17 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,893 |
| 18 | 0 | ↓ | ↓ | 3,131 | ↓ | 6,329 | 12,893 |
| 19 | 645 | 34,335 | 15,113 | 3,031 | 3,433 | 6,379 | 12,198 |
| 20 | 0 | 27,635 | 9,963 | 3,035 | 2,263 | 3,687 | 8,985 |
| 21 | (5,348) | 0 | 0 | 0 | 0 | 0 | 5,348 |
| TOTALS | 62,424 | 675,000 | 297,110 | 62,424 | 67,490 | 123,988 | 191,478 |

Table T-21. Single Dragline Horseshoe Method
Western Region
Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS \$) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|-----------------------------------|------------------------|---------------------------|---------------------------------|------------------------------|-----------------------------------|--------------------------|
| 0 | 40,500 | 0 | 0 | 0 | 0 | 0 | (40,500) |
| 1 | 0 | 38,520 | 27,632 | 2,510 | 3,852 | 2,263 | 8,625 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 8,625 |
| 3 | 0 | | | | | | 8,625 |
| 4 | 1,485 | | | | | | 7,140 |
| 5 | 0 | | | | | | 8,625 |
| 6 | 4,340 | | | | | | 4,285 |
| 7 | 2,085 | | | | | | 6,540 |
| 8 | 0 | | | | | | 8,625 |
| 9 | 0 | | | | | | 8,625 |
| 10 | 1,485 | | | | | | 7,140 |
| 11 | 4,640 | | | | | | 3,985 |
| 12 | 0 | 8,625 | | | | | |
| 13 | 2,085 | 6,540 | | | | | |
| 14 | 0 | 8,625 | | | | | |
| 15 | 0 | 8,625 | | | | | |
| 16 | 5,825 | 2,800 | | | | | |
| 17 | 0 | 8,625 | | | | | |
| 18 | 0 | 8,625 | | | | | |
| 19 | 2,085 | 6,540 | | | | | |
| 20 | 0 | 8,625 | | | | | |
| 21 | 4,640 | 3,985 | | | | | |
| 22 | 1,485 | 7,140 | | | | | |
| 23 | 0 | 8,625 | | | | | |
| 24 | 0 | 8,625 | | | | | |
| 25 | 2,085 | 6,540 | | | | | |
| 26 | 4,340 | 4,285 | | | | | |
| 27 | 0 | 8,625 | | | | | |
| 28 | 1,485 | 7,140 | | | | | |
| 29 | 0 | ↓ | ↓ | ↓ | ↓ | ↓ | 8,625 |
| 30 | 0 | | | | | | 38,520 |
| 31 | (3,260) | 14,400 | 10,330 | 0 | 1,440 | 1,315 | 6,015 |
| TOTALS | 75,305 | 1,170,000 | 839,290 | 75,305 | 117,000 | 69,203 | 186,202 |

Table T-22. Single Dragline - Elevated Bench Method
 Western Region
 Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS \$) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES, (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|--------------------------------------|---------------------------|------------------------------|------------------------------------|---------------------------------|--|-----------------------------|
| 0 | 40,500 | 0 | 0 | 0 | 0 | 0 | (40,500) |
| 1 | 0 | 46,395 | 31,863 | 2,854 | 4,639 | 3,520 | 11,012 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 3 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 4 | 1,485 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,527 |
| 5 | 4,340 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,672 |
| 6 | 600 | ↑ | ↑ | ↑ | ↑ | ↑ | 10,412 |
| 7 | 1,485 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,527 |
| 8 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 9 | 4,340 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,672 |
| 10 | 1,485 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,527 |
| 11 | 900 | ↑ | ↑ | ↑ | ↑ | ↑ | 10,112 |
| 12 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 13 | 5,825 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,187 |
| 14 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 15 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 16 | 2,085 | ↑ | ↑ | ↑ | ↑ | ↑ | 8,927 |
| 17 | 4,340 | ↑ | ↑ | ↑ | ↑ | ↑ | 6,672 |
| 18 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 19 | 1,485 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,527 |
| 20 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 21 | 5,240 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,772 |
| 22 | 1,485 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,527 |
| 23 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,012 |
| 24 | 0 | ↓ | ↓ | 2,854 | ↓ | 3,520 | 11,012 |
| 25 | 0 | 46,395 | 31,863 | 1,284 | 4,639 | 4,305 | 10,227 |
| 26 | (5,815) | 10,125 | 6,954 | 0 | 1,012 | 1,080 | 7,906 |
| TOTALS | 69,780 | 1,170,000 | 803,529 | 69,780 | 116,987 | 89,865 | 206,826 |

Table T-23. Shovel/Truck/Dragline Tandem System
Western Region
Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS \$) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50%) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|--------------------------------------|---------------------------|------------------------------|------------------------------------|---------------------------------|---|-----------------------------|
| 0 | 56,000 | 0 | 0 | 0 | 0 | 0 | (56,000) |
| 1 | 0 | 62,712 | 43,705 | 4,695 | 6,271 | 4,021 | 14,986 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 14,986 |
| 3 | 0 | | | | | | 14,986 |
| 4 | 2,898 | | | | | | 12,088 |
| 5 | 2,500 | | | | | | 12,486 |
| 6 | 0 | | | | | | 14,986 |
| 7 | 11,220 | | | | | | 3,766 |
| 8 | 966 | | | | | | 14,020 |
| 9 | 2,500 | | | | | | 12,486 |
| 10 | 2,898 | | | | | | 12,088 |
| 11 | 300 | | | | | | 14,686 |
| 12 | 0 | 14,986 | | | | | |
| 13 | 13,720 | 1,266 | | | | | |
| 14 | 0 | 14,986 | | | | | |
| 15 | 966 | 14,020 | | | | | |
| 16 | 2,898 | 12,088 | | | | | |
| 17 | 2,500 | 12,486 | | | | | |
| 18 | 0 | 62,712 | 43,705 | 4,695 | 6,271 | 4,021 | 14,986 |
| 19 | 0 | 41,184 | 28,701 | 2,339 | 4,118 | 3,013 | 9,470 |
| 20 | (12,517) | 0 | 0 | 0 | 0 | 0 | 12,517 |
| TOTALS | 86,849 | 1,170,000 | 815,391 | 86,849 | 116,996 | 75,391 | 192,369 |

Table T-24. Shovel/Truck System
Western Region
Cash Flow Analysis

| YEAR | CAPITAL INVESTMENT (THOUSANDS \$) | REVENUE (THOUSANDS \$) | CASH COSTS (THOUSANDS \$) | TAX DEPRECIATION (THOUSANDS \$) | TAX DEPLETION (THOUSANDS \$) | INCOME TAXES (50 %) (THOUSANDS \$) | CASH FLOW (THOUSANDS \$) |
|--------|--------------------------------------|---------------------------|------------------------------|------------------------------------|---------------------------------|--|-----------------------------|
| 0 | 51,000 | 0 | 0 | 0 | 0 | 0 | (51,000) |
| 1 | 0 | 60,768 | 46,133 | 5,936 | 4,335 | 2,167 | 12,438 |
| 2 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,438 |
| 3 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,438 |
| 4 | 6,888 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,550 |
| 5 | 2,500 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,938 |
| 6 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,438 |
| 7 | 15,570 | ↑ | ↑ | ↑ | ↑ | ↑ | (3,132) |
| 8 | 602 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,836 |
| 9 | 2,500 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,938 |
| 10 | 6,888 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,550 |
| 11 | 240 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,198 |
| 12 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,438 |
| 13 | 18,070 | ↑ | ↑ | ↑ | ↑ | ↑ | (5,632) |
| 14 | 0 | ↑ | ↑ | ↑ | ↑ | ↑ | 12,438 |
| 15 | 602 | ↑ | ↑ | ↑ | ↑ | ↑ | 11,836 |
| 16 | 6,888 | ↑ | ↑ | ↑ | ↑ | ↑ | 5,550 |
| 17 | 2,500 | ↑ | ↑ | ↑ | ↑ | ↑ | 9,938 |
| 18 | 0 | ↓ | ↓ | 5,936 | 4,335 | 2,167 | 12,438 |
| 19 | 0 | 60,768 | 46,133 | 2,191 | 6,077 | 3,169 | 11,436 |
| 20 | (5,209) | 15,408 | 11,705 | 0 | 1,541 | 1,081 | 7,831 |
| TOTALS | 109,039 | 1,170,000 | 888,802 | 109,039 | 85,648 | 43,256 | 128,903 |