OPPORTUNITIES FOR IMPROVED SURFACE MINE RECLAMATION
IN THE CENTRAL APPALACHIAN COAL REGION

by

Carl E. Zipper

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APPROVED:

D. C. Martens, Co-chairman

W. L. Daniels, Co-chairman

L. A. Shabman

D. B. Taylor

J. A. Burger

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(ABSTRACT)

The Appalachian coal mining region is subject to a number of environmental and economic problems; many are a result of the steeply sloping topography. The extensive surface mining activities in the area appear to offer the opportunity to produce more favorable landforms at minimum marginal costs. Yet, despite this apparent opportunity and the success of research efforts to develop improved mine soil construction and revegetation techniques, the majority of the mining and reclamation activities in the Virginia coal region are carried out using conventional methods: reconstructing steeply sloping mining areas to their approximate original contours.

The purpose of this research was to estimate the costs of coal surface mine reclamation methods designed to prepare mined lands for improved use in areas of steeply sloping topography. During the course of this research, a computer-based mining and reclamation cost estimating system was developed. COSTSUM is a set of seven programs designed to analyze data from active surface mining sites to determine spoil handling and reclamation costs. OPSIM is a surface mining simulator designed to estimate the differences in spoil handling costs among reclamation and postmining landform alternatives.

This cost-estimating system was utilized during an intensive study of mining and reclamation costs at a surface mining site in Wise County, Virginia, where a number of improved reclamation practices were implemented. At this site, a steeply sloping premining topography was transformed to a postmining landform containing an extensive

near-level area covered with deep, uncompacted, potentially productive mine soils. Analysis of daily records of operations revealed that the cost of mining and reclaiming this site was comparable to industry average costs in the area in spite of departure from conventional methods. The results of simulation procedures indicated that the cost of mining so as to produce this landscape was less than than the estimated cost of conventional mining methods. Since the topography of the site is typical of surrounding areas, there are opportunities to produce near-level landforms with deep, productive soils as a byproduct of coal surface mining activities.

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I. INTRODUCTION

Coal is a resource vital to the economic well being of our nation. Land is also a natural resource. The effect of coal surface mining operations upon local land uses has been a problem since the inception of surface mining methods (Caudill, 1962), an inevitable result of the extreme land disturbance associated with those methods. Such problems have been particulary acute in the steeply sloping areas of the central Appalachian region, due to steep surface topography and an abundance of near-surface coal deposits. Coal surface mining has been widespread in eastern Kentucky, southern West Virginia, southwestern Virginia, and eastern Tennessee since the 1950's.

Given the nature of the surface mining process and the central Appalachian terrain, conflicts between local land uses and mining occur in the absence of government regulation. Early researchers had hopes that the problem could be solved through identification of the "optimal" degree of mined land reclamation activities, that level of reclamation which struck an equitable balance between the economic interests of the mining industry and local land users (Brooks, 1966). Laws and regulations could then be written to impose this level of reclamation upon industry, and the "public interest" would be served.

Subsequent research has shown that identification of "optimal" reclamation procedures is so easily achieved (Brock and Brooks, 1968; Howard, 1971; Randall et al., 1978; Kalt, 1983). Indeed, despite the passage of the Surface Mining Control and Reclamation Act of 1977 (SMCRA) after years of study and research, controversy over mined land reclamation regulation continues. Certain aspects of that controversy are centered in the central Appalachian region, due to the unique topography of that region and the unique effects of SMCRA upon mining operations and land use within that topographic setting (Rowe, 1976).

Coal Surface Mining in Steeply Sloping Regions

Coal surface mining is essentially an earth moving process. In its simplest form, the first step is to scalp the mining site, generally with a dozer, removing all earth and vegetation and exposing the horizontally bedded rock below. Then, a blasthole drill bores holes vertically towards the coal seam; these are loaded with explosives and the rock covering the coal ("overburden") is shattered to form "spoil" that can be removed with earth moving equipment. The most economical methods of spoil removal are, generally, to push with a dozer or carry with a front-end loader over a short distance. In haulback mining, the majority of the spoil is loaded into haul trucks by a loader or small shovel; most of these materials are hauled back to cover the previously mined areas. However, the excess spoil produced by the tendency of rock to "swell" when blasted must also be disposed of. Otherwise, the operation becomes "spoil bound". The necessity to remove spoil from the mining pit in spite of insufficient disposal areas causes previously mined spoil to interfere with active operations; hauler routes become narrow and steep, flat areas suitable for maintenance of equipment become scarce and, at worst, the backfilled spoil begins to spill over into the mining pit. Depending upon the depth to the coal, the process of drilling, blasting, and removing overburden may be repeated (multiple "lifts" are removed) sufficient times to expose a "block" of coal, which is also loaded into trucks for removal from the site.

Reclamation generally follows, a process whose purpose is to hide the mining scar, to smooth the spoil bank and cover it with vegetation so as to reduce movement of material off the site by wind and water and/or to prepare the mined land for a productive post-mining land use. If spoil removal is carried out with a reclamation plan in mind, post-coal-removal activities required to reclaim the site are kept to a minimum.

In sloping topography, the least expensive coal to mine generally occurs at the outcrop, where the near-flat lying beds of coal intersect the sloping surface. Here, the volume of overburden that must be removed to mine each ton of coal ("stripping ratio") is minimized. Generally, mining in these areas is initiated by removing the overburden covering the coal near the outcrop. The stripping ratio increases with distance from the outcrop. One or more "cuts" may be taken, working into the mountain, until mining reaches a point where further increases in the stripping ratio, coupled with spoil disposal requirements, prevent further economic recovery. The contour mining method consists of working a coal seam laterally along the side of a mountain or ridge in this fashion. Where multiple seams outcrop with a vertical separation small enough that they can be mined together, the sequence of steps required for contour mining becomes more complex but the principles and purposes remain the same.

Since coal becomes less profitable to mine as stripping ratio increases, the interests of the mining operator may differ from those of the land owner in contract mining situations. The land owner generally receives a percentage of the selling price of the coal without bearing mining costs. Therefore, the land owner's profits are maximized when the maximum quantity of coal that can be recovered from a particular operation is actually mined. The mining operator, however, bears the costs. Cost per ton is minimized, and operator profits are maximized, when only the most profitable coal is mined. Thus, the mining operator will favor maintaining an average stripping ratio below that level which allows that recovery level favored by the land owner.

Local Land Uses in Central Appalachian Areas

The Appalachian region consists of flat lying sandstones, siltstones, and shales, primarily of the Missippian and Pennsylvanian geologic eras (Miller, 1974). It is an ancient landscape which is deeply dissected, and the predominant topography is steep slopes. In the central Appalachian mining region, these slopes generally exceed 20%, and often exceed 20°. The natural soils which form on these highly weathered, steeply sloping landforms are generally thin, acidic, and infertile (Daniels and Amos, 1984).

In this topography, there are limited amounts of land that could be considered "flat", with slopes that favor residential or agricultural development. Such lands include, primarily, the present ridgetops, which are remnants of an ancient plateau, and alluvial areas adjacent to rivers and streams. Historically, land use has been most intense in these areas. The ridgetops have proven to be excellent locations for small-scale agricultural enterprise, in spite of generally limited areal extent. A leading industry of Wise County, Virginia, during the 1930's and 1940's was apple production, primarily based in the highlands (Addington, 1956). However, the terrain caused the primary transportation routes to locate in the lowlands, adjacent to waterways. Access to water and transportation also made these low areas preferred for residential development.

Today, the prevailing settlement pattern is small communities located adjacent to waterways. As a result of the mountainous topography, many are subject to periodic flooding (SWVA 208, 1978). It is expensive to provide the dispersed, rural population with public services, such as water and sewers (CPPDC, 1981; LPDC, 1981). The ridgetop areas are not large enough to support economically-viable modern agriculture, and broad, level areas suitable for industrial and commercial development are scarce. The result is a region where the mining industry is the primary employer. Due to the recent difficulties of the coal mining industry, the 1980's have not been prosperous times for the communities of the Virginia coal region (Table 1).

Effects of Mining upon Land Use

Abundant coal deposits close to the surface have caused the central Appalachian area to be intensely affected by mining activities. In Virginia, for example, an estimated 26,000 hectares were affected by mining between the 1950's and 1975 (SWVA 208, 1978). Since 1975, an additional 18,000 hectares have been disturbed (VDMLR, 1975-85), the estimated area of the Virginia coal region is 481,000 hectares (SWVA 208, 1978). An approximatiion of the proportion of post-1975 disturbance which consisted of previously mined areas is 20%. Thus, approximately 8.5 % of the land surface in the Virginia coal region has been directly disturbed by mining.

Early surface mining was unregulated, and its effect upon local land uses was sometimes severe (Shelton, 1977). The predominant mining method ("shoot and shove") consisted of pushing the blasted overburden with a dozer over the edge of the slope to expose the coal. Land use problems often resulted from these practices, including displacement

Table 1. Selected economic characteristics of the populations of Virginia's coal counties, and state averages.

Area	Per Capita Income[a] (1983)	Incomes below poverty level[b] (1980)	Mining employment[c] (1982)	Unemployment[d] (1985)
	- $/year -	- % -	--- % of workforce	---
Buchanon	8496	19	69	19.7
Dickenson	7972	17	64	20.2
Wise	9558	15	50	15.8
Virginia	12122	11	1.6	5.6

[a]: U.S. Dept. of Commerce, 1985.

[b]: U.S. Dept. of Commerce, 1980.

[c]: U.S. Dept. of Commerce, 1984.

[d]: Virginia Employment Commission.

of residents from the mining area and areas below. The angle of repose of the pushed material could exceed the stable maximum, resulting in landslides. Instability was also favored by the tendency to push from points into hollows, where natural flowing waters would saturate the spoil mass (Mathematica, Inc., 1974). Water quality was often adversely affected when material pushed over the outslope proved difficult to revegetate. This was a frequent occurrence, as "shoot and shove" mining tended to turn the overburden mass "bottom side up", burying surface soils and exposing sometimes phytotoxic materials adjacent to the coal seam.

The immediate effects of these practices included displacement of local land uses. Other effects tended to occur downstream. Water quality in the coal region was severely affected during the period of unregulated mining; abandoned non-reclaimed areas continue to affect water quality (Skelly and Loy, 1983). Where mining-induced siltation caused a decrease in stream channel volumes, flood frequency increased. Also, the mining of a watershed affects the infiltration: runoff ratio, and thus the magnitude of peak flows, although such relationships are not well documented (Curtis, 1979). In addition, the aesthetic impacts of exposed highwalls were judged to be objectionable by many (Shelton, 1977; Mathematica, Inc., 1974; Randall et.al., 1978).

The reasons for these land use conflicts are understandable. Surface mining was a competitive but unregulated industry. The relatively small capital requirements, abundant coal deposits, and resultant ease of entry created a situation that remains today: the potential supply of coal exceeded demand, and mining firm survival was dependent upon cost minimization. The least-cost mining method was "shoot and shove"; the firm engaging in "unnecessary" spoil handling and reclamation practices risked losing market share and/or failure to recover costs.

Much state regulatory activity during the 1960's and early 1970's (Meiners, 1964; Imhoff, 1976). However, ease of entry and mobility of resources continued to hinder regulatory efforts. Mineable coal deposits exist in 11 Appalachian states. Those states imposing stiff reclamation requirements risked loss of mining activity and resultant employment and tax revenues to neighboring states with less stringent regulations.

In the mid-to-late 1970's, there was a movement to impose federal regulation upon coal surface mining activities which culminated with the passage of SMCRA. The many controversial provisions of this law included a revegetation standard requiring the mining firm to establish a plant community capable of sustaining itself upon the reclaimed land for 5 full years without fertilization or re-seeding, unless the land was to be placed in a "higher" use such as agricultural, residential, commercial, or industrial development. In addition, SMCRA requires that mined land be returned to its "approximate original contours" (AOC) after mining, unless it is to be placed in a higher land use.

Since the industry had little experience with the reclamation procedures required by SMCRA, much research has been performed since its passage. Research performed by graduate students, staff, and faculty of the Agronomy Department at Virginia Polytechnic Institute and State University (VPI&SU) has shown that coal surface mining and reclamation activities continue to have a profound effect upon land use in the central Appalachian region, while the potential to have future effects is also great.

The primary research effort in the Agronomy Department has been with mine soil revegetation methods. The results have shown that the rocks of the Virginia coal mining region, in general, have favorable plant growth characteristics (Howard, 1979; Everett, 1981) relative to the more acidic spoils of adjacent states and to the natural soils which form on steep slopes. However, the plant growth properties of the resulting mine soils were often limited by physical properties which could be controlled during material placement (Daniels and Amos, 1981). By selecting strata with the most favorable qualities, placing materials from such strata containing a minimum of >2mm fragments at the surface in a layer of at least 1 meter, minimizing compaction while grading, adding organic ammendments if possible, and managing carefully for nitrogen and phosphorus, mining firms could construct mine soils with productive potentials exceeding many of the natural soils of the area (Daniels and Amos, 1984). This method of mine soil construction is called controlled overburden placement. In addition, research has shown that proper selection and placement of rock material during post-mining landform construction could prepare mined lands to support roads and buildings, as well as associated residential, commercial, and industrial development (Bell, 1982). However, in spite of the development of these reclamation techniques and their potential to provide mined lands capable of improved use, the methods are not being implemented in any widespread fashion. Suggested reasons for this lack of implementation include excessive costs and the predominance of AOC reclamation in steep slope topography, which produces landforms unsuitable for improved use.

On the other hand, the potential for future land use impacts by AOC backfills being constructed in accordance with the provisions of the Law is also great. Bell and Daniels (1984) studied a number of AOC backfills, looking primarily at stability. They found the potential for instability could result from a number of factors, including excessive steepness, improper construction techniques, and seepage of subsurface water into the fills. Since some of the sites studied were typical, in many respects, of other backfills, these results portend potentially widespread stability problems. When AOC backfills do fail, the effects upon local land use are similar to those which formerly resulted from uncontrolled "shoot and shove" mining activities.

Purpose of Research

This research project was developed in response to the observation that conventional reclamation practices had little apparent relationship to the future land use requirements of the Virginia mining region. In spite of the needs generated by unfavorable topography, reconstruction of the original steep slopes after mining was then and remains the conventional procedure; the opportunity to take advantage of the earth moving capabilities of surface mining operations and newly-developed reclamation techniques to produce lands capable of improved use was not being realized in any widespread fashion. Conversations with representitives of mining firms indicated a concern with the potential costs of alternative mining and reclamation practices.

The purpose of this research was to estimate the cost of improved reclamation in southwestern Virginia. In order to accomplish this purpose, the following objectives were established:

1. To study active haulback mining operations so as to determine the costs associated with spoil handling and reclamation activities.

2. To develop computerized methods capable of estimating spoil handling and reclamation costs in steeply sloping topography.

3. To use those methods to estimate the cost of reclaiming surface mined land in southwestern Virginia in a fashion that will favor improved land uses.

4. To determine if cost.was, in fact, the major factor preventing implementation of improved reclamation practices.

The practice chosen for study was the production of a stable post-mining landform capable of agricultral use in an area of steeply sloping pre-mining contours. Such a landform will have an extensive near-level area with soils constructed using controlled overburden placement techniques.

Since the form of "reclamation" required to produce an alternative post-mining landform is, in effect, an alteration of the mining process, the research approach required that the entire mining process be studied. Due to the current requirement to restore the land to its AOC, unless an improved land use is being implemented, the cost of the reclamation practice under study was taken to be the change in overall mining cost resulting from implementation of the improved practices rather than conventional AOC techniques.

II. REVIEW OF THE LITERATURE

Very few precedents were found to guide this study. Although studies of the economics of mining and reclamation are common in the literature of the past 10 years, surprisingly few make use of original, site-measured data. The requirement that usable studies pertain to steep-slope haulback mining situations restricts applicable studies even further.

Nephew and Spore (1976) attempted to determine the effect of "level of reclamation" upon mining cost. In order to accomplish this objective, they modeled 9 mining plans (three postmining landforms, three mining methods) for six stripping ratios in three topographies (15^o, 20^o, and 25^o slopes; 18 m and 27 m highwalls). The mining/reclamation process was assumed to consist of 10 separate unit operations: haul road construction, clearing and scalping, topsoil removal, drilling and blasting, overburden removal, coal loading, coal hauling, backfilling and grading, topsoil replacement, and revegetation. With the aid of consulting mining engineers, they developed a series of simple equations to estimate the cost of each unit operation in each of the 42 "model mines" studied, and then normalized those costs to per-ton bases. Not surprisingly, they found that the restoration of approximate original contours (AOC) was more expensive than either of the two "terraced backfill" landform alternatives studied, and that the degree of cost difference increased with highwall height and land slope. However, both terraced backfills were assumed to have 37^o outslopes with toes on the undisturbed slope below the mining bench, a situation that subsequent research has shown to be frequently unstable (Bell and Daniels, 1985).

Mathematica (1977) and Leckie Smokeless Coal Co. (1982) developed more sophisticated cost estimating methods. Both were based upon extensive site data and an assumed conservation of volume: 100% of the total overburden must be moved in order for coal to be mined. Given the availability of multiple modes of spoil movement, the total overburden quantity may be partitioned among those modes of movement. Then, equations were developed to estimate the rate of spoil movement by each of the available modes, given a distance to a disposal area.

In neither case were the form of these equations published.

The Mathematica model was used to estimate the effect of landform and mining method upon mining costs for two hypothetical mountain top removal operations. The first situation was an 11 hectare virgin mountaintop with a low stripping ratio and a single coal seam. Here, a cross-ridge mining method was found to be inferior to the conventional contour method, which takes the most profitable coal first, primarily due to cash flow considerations. A disadvantage to the conventional contour method was the possibility that extremely high stripping ratios in the final cuts might cause an operator to terminate activities before completing the mining plan. However, the cross ridge method was found to be advantageous for a 35 hectare irregularly shaped ridge with two mineable seams and a low stripping ratio. Improved operating efficiency and flexibility, and increased profitability, resulted from use of the cross-ridge method.

III. MATERIALS AND METHODS

III.A. SUMMARY

Since the effect of reclamation activities upon the overall mining operation was under study, a site-data based computer simulation approach to cost estimation was chosen. Original cost estimation methods were developed and applied to a case study in Wise County, Virginia, at a mining site where landform alteration and controlled overburden placement procedures were being implemented.

The first step of this study was to gain familiarity with surface mining operations. A number of mining sites in Wise County, Virginia were visited in the summer of 1982 for this purpose. During the Summer of 1983, a system was established to record daily operations data at the case study site. These data were supplemented by machinery operation data recorded during 1983, 1984, and 1985 at two other sites in Wise County, where steep slope haulback contour mining operations were being performed.

Computerized data analysis and mining simulation techniques were developed in order to process the site data and to use that data as a basis for estimating the probable cost of operations under an alternative mining plan. COSTSUM is a set of seven programs designed to analyze data from active surface mining sites to determine spoil handling and reclamation costs. OPSIM is a surface mining simulator designed to estimate the differences in spoil handling costs among reclamation and post-mining landform alternatives. These programs were supplemented with use of CPS/PC, a software system for gridding, mapping, and analysis of topographic data produced by Radian Corporation (Radian, 1986). CPS/PC was used to estimate the spoil movement volumes required to produce specific post-mining landforms.

These computer programs were used to estimate costs at the case study site. The actual cost of mining was estimated by analyzing daily operations data using COSTSUM. The cost of the improved reclamation techniques under study (landform alteration and controlled overburden placement) was estimated by using OPSIM to simulate mining and reclamation as performed and the most likely alternative: mining so as to return the land to its AOC using conventional reclamation procedures.

Due to the facts that English units are commonly used within the mining industry, and that the inputs and outputs of the computer programs used in this analysis are defined in English units, the use of SI units and notation will be foregone in the remaining sections of this dissertation.

III.B. COSTSUM

COSTSUM (COST of SUrface Mining) is a system for collection and analysis of coal surface mine data. The system was developed for the specific purpose of analyzing data collected during this study. However, it can be usefully applied at other haulback mining operations, where dozers, loaders, and haulers are the primary equipment used to move spoil. See Zipper and Daniels (1986a) for more detailed discussion of the use and logoc of COSTSUM.

The primary assumption of COSTSUM is that the major costs of coal surface mining are machinery operation and labor. Thus, a per hour operating cost which includes operator wage is assigned to each machine on the site; the cost of using that machine is calculated as the product of total hours of operation and hourly operating cost and the primary component of total mining cost is the sum of the machinery operating costs. Aside from machinery and labor used for machinery operation and other defined tasks, only three other categories of cost are recognized: coal hauling, supplies, and overhead. It is assumed that some portion of the coal produced is hauled from the site by a contract hauling firm at a per-ton rate. The quantities and costs of seeding and blasting supplies are input by the user and totaled by the computer. Seeding supplies are accounted on an area treated, or on a "batches" of seeding mix applied, basis. Blasting supplies may be accounted on per pound of explosive, per hole, and per foot of hole bases. All other costs are lumped into one category: overhead. These are entered as a per-ton-of-coal-produced figure; COSTSUM calculates an initial estimate for the total overhead cost as the product of the per-ton input and the coal produced.

The objective of the operation is assumed to be the mining of coal; thus, no other classes of revenue are recognized. Also, the overall mining site is seen as a series of mining blocks, and profitability is seen as the result of a series of decisions regarding mining of these blocks. Those decisions include whether each block should be mined, and what spoil-handling practices should be used to remove the overburden from each mining block. The objective of the COSTSUM programs is to assign all costs to the mining blocks, and to break those costs down in detail. Thus, the results of the decisions made can be assessed, in terms of their effect upon block profitability.

The assumptions used in assigning costs to mining blocks are straightforward. First, all costs of overburden handling are charged to the block of spoil origin. Thus, all costs of blasting, dozing, carrying, loading, hauling, and reclaiming mined material are charged to the mining block where the spoil originated. Likewise, costs of coal augering, loading, and hauling are charged to the block of coal origin. Finally, since the primary cost of mining coal is moving overburden, the total overhead cost is distributed to the mining blocks on a per undisturbed cubic yard (bank cubic yard; bcy) of overburden basis.

Seven data analysis computer programs form the body of COSTSUM. Five Level 1 programs (HAULER, LOADER, DOZER, DRILL, and OTHER) are designed to analyze the files containing daily records of machine operations. These programs compile totals of the hours spent performing each type of operation subtotaled by location, destination (for overburden movement operations) and other criteria. Other quantities, such as hauler loads, feet of drill hole, and pounds of explosives, are also totaled. Two additional programs are defined as Level 2, since their primary inputs are the outputs of the Level 1 programs. The program MOVE estimates the quantities of material moved from each mining block to each spoil disposal area, and the program COST provides a detailed analysis of cost and profitability.

Level 1 Programs

The purpose of the five Level 1 programs is to total the daily operational data recorded on the mine site. Their methods of operation are similar; each requires an input file composed of a portion of the daily data. The first four programs total the data on operation of a particular type of machine, while the fifth (OTHER) accepts all additional daily data, The outputs of the Level 1 programs consist of machinery operation time totals.

HAULER FORTRAN totals hauler data (hours and loads) by block and lift of spoil origin and by destination. Thus, the output of the program tells the user how much hauling time was spent and how many loads were carried over each haul route on the site. In addition, loads per hour (lph) hauling rates are listed for each hauler route.

LOADER FORTRAN and DOZER FORTRAN accumulate time of operation data by machine, location, and operation type for the loaders and dozers. In the output, the hours spent by each machine performing each type of operation are totaled by source block and lift. In addition, hours moving overburden (loader carry and dump, dozer push) are totaled by machine, source block, source lift, and destination. Hours working dumpsites are totaled by machine and dumpsite location.

DRILL FORTRAN is the simplest and shortest of the seven programs. It totals all data recorded for the drill and blast operation by block and lift. In addition, average hole depths and drilling rates are calculated.

OTHER FORTRAN totals all additional data recorded on a daily basis. Labor hours, labor days, supervisor days, and fuel purchases are added into simple totals. Coal tonnage is totaled by block and mining method (auger or strip). The hours worked by machines other than haulers, loaders, dozers, and drills ("other machines") are totaled by location and by cost category. The labor hours required to operate the auger in each coal block are also totaled. The number of "batches" of seed and fertilizer applied by the the seeder (or the acres treated) are totaled by location.

Level 2 Programs

The primary inputs to MOVE FORTRAN and COST FORTRAN are the outputs of the Level 1 programs. MOVE estimates the quantities of overburden moved from each source location to each destination, while COST provides a detailed cost and profitablility analysis.

MOVE

The purpose of MOVE FORTRAN is to generate a file to direct the distribution of reclamation expenses to the mining blocks on the basis of the relative quantities of material disposed in each reclaimed area originating in each mining block. In the process of generating this information, MOVE performs a series of calculations to estimate the quantities of material moved by each of three modes (hauler, loader carry, dozer push) over each source-to-destination route. The accuracy of the moved quantity estimates performed by program MOVE will vary, depending upon the importance placed by the user upon the resultant cost data and willingness to spend time developing accurate movement rate estimates.

Inputs to MOVE include the undisturbed (bank cubic yard, or bcy) volume of each lift and the swell expected upon disturbance; these quantities are used to calculate the amount of material (in loose cubic yards, or lcy) which must be removed from each block in order to expose the coal. The primary inputs used to calculate moved volume estimates are hours spent hauling, carrying, and pushing material over each transport route (Level 1 output files), and estimates of the rates of material movement. For loaders and haulers, movement rates are estimated as a product of the loads carried or hauled per hour and average load lcy volumes. For the dozer, the rates are estimated directly as a lcy-per-hour quantity. Average hauler and loader loads per hour, and dozer push rates, are estimated by the user for the entire job as default values; the user may override these defaults by estimating the route specific rates, where that information is available.

An initial series of transport quantity estimates is calculated as the product of hours x rate for each transport route. These estimates are refined through use of a correction factor called the "moved:topo ratio", calculated for each lift of each block as the ratio of the total of the initial hours x rate moved quantity estimate to the input lcy volume. The initial transport quantity estimates are adjusted with this factor. Thus, volume is conserved in the final transport estimates. The program output includes a listing of the initial calculated estimates of material quantities moved from each source block and lift to each destination by each of the three modes (push, carry, haul). An additional output file lists the bcy volumes of the mining blocks and lifts, volumes removed from each block by each mode of movement, and estimated proportions of the material disposed in each reclamation area originating in each source; this file is used as an input to program COST.

COST

The purpose of COST FORTRAN is to provide a detailed analysis of the costs of mining and reclamation so as to allow assessment of the effects of those costs upon mining profitability. The method used in programming to meet this purpose is to provide a system whereby all mining expenses are charged to the mining blocks. The expenses are further broken down into 15 cost categories; these are calculated on a gross basis and recalculated on a per bcy basis for spoil-handling operations and on a per-ton basis for coal- handling operations. Thus, the user is able to identify the mining areas where spoil was handled in cost effective fashion and those areas where spoil handling was most costly, and the reasons why.

Costs incurred while removing overburden to expose coal are charged directly to the mining block on the basis of machine operating hours and hourly operating costs. However, two costs which cannot be directly charged are reclamation expenses and overhead. As stated above, reclamation expenses are distributed to mining blocks in amounts proportionate to quantities of reclaimed spoil originating in each block. Overhead costs are also distributed on a per bcy basis because overburden movement is the primary cost of mining coal.

Costs totaled as overhead include operations of "other" machines listed as overhead costs in the input to OTHER FORTRAN. The total cost of operating machinery in any location may be considered as overhead if it is so identified in the COST input file. Costs which can be conveniently handled in this fashion may include construction and maintenance of roads to the site or sediment ponds and other environmental control structures. In addition, the input overhead cost per ton is multiplied by the total tonnage produced and added to the overhead total; that quantity is then distributed to the mining blocks and lifts on a bcy volume basis.

The primary operations considered as reclamation are dozer grading, seeding, excess topsoil (or topsoil substitute) hauling time. The user may specify a percentage of the total hauling cost incurred during any day (or portion of a day) between any source and destination as a reclamation expense, rather than a source block mining expense. Thus, "excess" topsoil hauling costs (costs required for topsoil hauling judged to be over and above the costs required for routine disposal) are identified by the user and appropriately accounted. In addition, if a location is appropriately identified in the COST input file, all operations performed at that location will be considered as reclamation costs. This option has proved useful for handling the costs of constructing excess spoil disposal facilities.

Execution of COST FORTRAN results in the output of a variety of cost data. The job cost summary includes a listing of the input costs, a profit summary for each block, total job costs by cost category, machine cost and operating hours totals, and block and lift cost summaries which include stripping ratios, and per-bcy and per-ton cost totals. The costs for each block are also listed and broken down by lift and by cost category. There are two separate listings for each block, the first containing cost totals and the second containing per-block-bcy overburden-handling costs and per-ton coal-handling costs. A record of the distribution of overhead and reclamation costs is also produced.

III.C. OPSIM

The primary cost of surface coal mining is overburden handling. These materials must be disposed of by constructing and reclaiming landforms which satisfy regulatory requirements and serve the purposes of the land owner. The post-mining landform is a product of the mining plan utilized and directly determines the cost of mining under that plan. In steeply sloping areas, the constraints imposed by topography make it essential to develop an effective spoil handling strategy before initiating mining activities. Thus, the OPSIM simulator was developed for the purpose of facilitating the comparison of overburden handling alternatives (Cakraborty, 1985; Zipper, Chakraborty, Topuz, and Daniels, 1985).

The major assumptions of the OPSIM model are that the primary cost of surface coal mining is overburden removal, transport, and placement, and that, given a volume of coal to be surface mined, the magnitude of this cost is primarily determined by the pre- and post-mining landforms and by the mining strategy used to transform the former to the latter. Major inputs to the model include equipment availablility and operating characteristics, costs experienced by the mining firm, and overburden movement specifications. Operation of the model consists of the simulation of overburden movement and coal removal operations, as specified by the user, while maintaining overburden volume, machine time, and cost accounts. Output includes information on costs, machine scheduling, and overburden movement. The model is designed to simulate haulback mining in situations where off-road haul trucks, front end loaders, and dozers are the machines used to move the majority of the mining spoil.

The model is being developed as a cooperative project by the Department of Agronomy and the Department of Mining and Minerals Engineering at Virginia Polytechnic Institute and State University. See Chakraborty (1985) and Zipper, Chakraborty, Topuz, and Daniels (1985) for more complete discussion of model logic and operation.

The Model

The simulation process carried out by the model is represented by the macro flow chart of Fig. 1. The program keeps an account of time during execution of the mining plan input by the user. That mining plan is composed of a sequence of jobs, or unit operations. The times required to perform the various operations required by the mining plan are translated into costs using the input data.

After reading the input data and initializing internal variables, the program executes via assignment of machines to jobs in mining blocks according to the mining plan. The activity of each machine is simulated by a separate subroutine or subroutine segment. Assignment of a job to a machine results in the main program calling up the appropriate subroutine and initializing the necessary subroutine variables. When the job is completed, the time of operation and other quantities required to update system attributes are recorded and the control is returned to the main program. The program will then attempt to assign a new job to that machine. If no jobs are available, the machine accumulates idle time until the completion of other job(s) by other machine(s) on the site causes a job to become available to the idle machine. Mining is completed when all jobs required by the mining plan are completed. The final step in the simulation program is the calling of a subroutine to calculate and print an overall mining and cost summary.

The primary unit for assignment of jobs to machines is the lift, which is a vertical segment of a mining block. Before work can begin on any lift, overburden removal must be completed for all previous lifts of that block. Jobs may be performed by different machines in two or more blocks simultaneously; however, overburden removal from the top lift of any block must be preceeded by the completion of overburden removal from any prerequisite lifts defined in the input data. If a job is available to a machine in each of two or more blocks, priority is assigned according to the mining block sequence input by the user.

Within each block, jobs are assigned in order of priority (Table 2). For each lift, the first job assigned is drilling and blasting

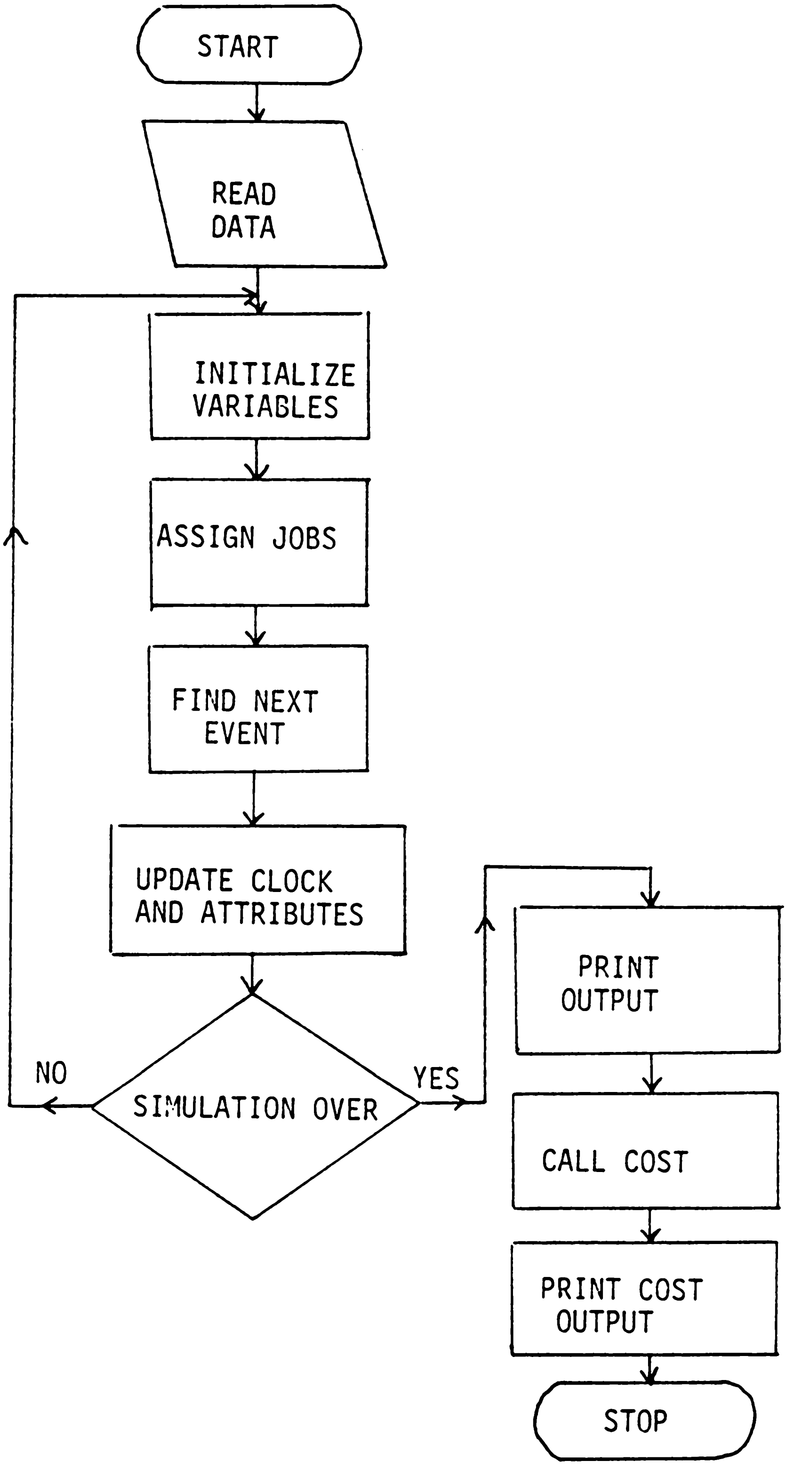


Figure 1. Macro flow chart of the OPSIM simulation program.

Table 2. Surface mining operations modeled by the mining simulator OPSIM, in order of execution.

Order	Machine(s)	Operation
1	Dozer	Prepare for drilling[a]
2	Drill	Drill blastholes
3	Dozer	Push overburden
4	Loader (& Dozer)	Carry overburden (with or without dozer assist)[b]
5	Loader & Hauler(s) (& Dozer)	Load and haul overburden (with or without dozer assist)[b]
6	Coal Loader	Prepare coal for loading[a]
7		Load coal into contract haulers
8	Dozer	Reclamation grading[a]

a: Time requirement input directly by user.
b: Dozer time input as fraction of loader time.

(with or without dozer preparation) if that lift is to be drilled. Next, the three transport operations are assigned in the order: dozer push, loader carry, load and haul. The total of the percentages to be moved by these three operations should equal 100% or distortion of the volume accounts will occur. Any of the lift percentages may equal zero. For example, if topsoil removal is to be included in the modeling of a particular block, the topsoil could be specified as constituting the top lift, of which 0% is to be drilled. The dozer is available for assignment to non-overburden removal operations (such as reclamation grading) only when no overburden removal jobs are available. Coal removal is simulated after all overburden removal jobs are completed for all lifts of the mining block.

The program simulates the operations required by the mining plan until the mining plan is completed. As described above, stripping and filling are simulated in sequence and accounts are maintained for times of operation, idle times, and volumes of material moved from each mining block to each fill cell.

Inputs

Data is input to the program in the form of a data file. This file may be constructed through the use of a program developed for use with the simulator which assigns default values to all variables, excepting those determining the form of the file (i.e. number of mining blocks, etc.) which are input directly by the user. The user then modifies the data file using an interactive full screen editor.

Required inputs are primarily of three types: characteristics of the mining firm, characteristics of the site, and a mining plan, which is a sequence of operations to be used in simulating the mining of the site (Appendix A).

The characteristics of the firm which are required by the simulation program are related to costs and equipment. The required cost data include payroll and wage information, estimates of overhead costs, and hourly owning and operating costs for the equipment whose operations are to be modeled. Such equipment will include drills, dozers, overburden loaders, coal loaders, overburden haul trucks, and coal haul trucks. The simulator allows for only one model of each of the above types of machines; however, it is able to handle multiple units of each. The user specifies how many of each model will be considered to be on the site during simulation.

Additional data is input in order to define the operating characteristics of the equipment. This includes such information as bucket capacities, loader and dozer traveling speeds, rimpull in various gears and maximum downhill speeds for trucks, truck dumping times, drilling rates, and average loader cycle times.

Information to be supplied for the mining site includes descriptions of the pre- and post-mining topographies and of the haulage routes to be utilized during the mining operation.

The pre-mining topography is defined as a series of mining blocks, which are segmented into a user-defined number of lifts. The dimensions of each lift are defined by the user, as is the area and thickness of the underlying coal. Each lift is defined as being composed of specifed percentages of the rock types found at the site; loose (post-blasting) density and swell are calculated for each lift based upon rock composition. Lift dimensions and swell are used to calculate loose volume. Separate bucket-fill factors are also input for the loader and the dozer for each lift to allow consideration of the influence of material properties upon spoil movement efficiencies. In addition, prerequisites may be defined for each block and/or lift i.e. blocks where mining must be completed before the mining of the unit in question can begin.

Post-mining topography is defined as a series of "fill cells," or spoil disposal areas. These are of two major types: backfill cells and excess spoil disposal areas, such as hollow fills. Each fill cell is defined as a specified volume with mining prerequisites. For surface cells, the user may input the number of dozer hours required to prepare the area for seeding. Temporary spoil storage areas may also be defined, if rehandling of spoil will be required.

Spoil transport routes are defined from mining blocks to fill cells. Only those routes that the user feels are likely to be used during execution of the program need be defined; if the program attempts to move material over an undefined route, an error message results. For haulers, each route is defined as a series of segments of specified length and grade which allow the simulator to calculate average spoil movement rates over that route. In addition, the user may specify either or both of two types of speed limits for any road segment, one type being the speed limit as usually defined, i.e. the maximum speed attainable by a vehicle traveling over that road segment. An "entry limit" may also be defined, which is a maximum speed to be attained by a vehicle at the initial point of a particular road segment. These entry limits are used to represent "speed constraint" points, such as hairpin curves. The same function could be performed by defining a short road segment with a normal speed limit in these locations. However, memory limitations made it desireable to restrict the maximum number of road segments per transport route to 5; the entry limit concept allows more precise road definitions within this constraint. If desired, the user may bypass this calculation by inputting haul times directly for individual haul routes.

The user must also provide a mining plan. Mining blocks are stripped in an order defined by the user as the "stripping sequence," from the top lift down. For each lift, the user must specify the percentage of the total volume which is to be drilled and blasted, the percentages which are to be moved by each of the three available modes: dozer push, loader carry, load and haul, and the percentage of the loader-handled material which is to be moved to the loader with a dozer feed operation. In addition, the user must specify a fill cell to serve as the primary destination for material moved by each mode of transport for each lift. A backfilling sequence is also defined, i.e. the order in which the backfill cells are to be filled with spoil. If the simulator is directed to deliver spoil to a backfill cell whose capacity has been reached, the next available cell in the sequence will be used for spoil disposal. If no backfill cells are found to be available or if the attempt to locate an available cell in the sequence brings the simulator to a "fill break point" (a user-defined break in the backfilling sequence), the user-defined alternate dumpsite (generally the closest excess spoil disposal area) will receive the mined spoil. Excess spoil disposal areas are also filled in sequence. Lack of available backfill cells or excess spoil disposal areas results in transport of mined spoil to a temporary storage area. The primary destination for spoil removed from any lift by any transport mode may be defined as a backfill cell, excess spoil disposal area, or temporary storage area. The user also defines a point or points in the stripping sequence when temporarily stored materials (such as stockpiled topsoil) may be rehandled through hauling to a backfill cell.

Simulation Algorithms

The primary operations simulated are drilling, pushing, carrying, and hauling. The drilling, pushing, and carrying algorithms are quite simple. User-input rates and cycle time constants are used in conjunction with lift dimensions, rock properties, and the fraction of the lift to be treated by any particular operation to determine the total time required by the operation. For the drill, the input rate is a foot-per-minute drilling rate, while the constant is the average time to move between drill holes. Although the public release version of OPSIM considers drilling to be a property of rock type, drilling rates were input individually for each lift for the purposes of this study.

Dozer push and loader carry cycle times are determined on the basis of transport distance, traveling rate, and loading time constant inputs. The volume moved per cycle is a simple product of the bucket/blade capacity multiplied by a lift-specific bucket-fill correction factor. The total time required per lift is simply calculated, given the lift volume and the fraction to be pushed or carried.

The hauling time algorithm is more complex. A separate subroutine is called to calculate haul time over a given transport route; a separate calculation is performed for the loaded portion of the haul cycle and for the empty return trip. The volume of material carried is a product of the number of loader buckets specified per hauler load, the loader bucket capacity, and the loader bucket correction factor. Vehicle mass is the sum of the mass carried (volume x average density) and the empty vehicle mass. The haul time calculation subroutine breaks the total haul up into a sequence of time increments of variable length. The initial increment is 0.01 second. Changes in speed over each time increment are calculated based upon the initial speed, the available power of the vehicle at that speed, and resistance to motion resulting from the vehicle mass, the grade, and the condition of the road surface. The final speed and initial speed over the increment are averaged to determine the average speed and time required, the duration of the next increment is calculated based upon the change of speed calculated and a user-input maximum time increment value, and the process is repeated. These calculations are modified, of course, if the situation does not call for acceleration. For example, if the road segment's speed limit is attained, constant speed is assumed until the end of the segment. Approach of a speed limit less than present speed or the end of the final road segement will cause the vehicle to brake at a user-defined deceleration rate. Many of the assumptions used by the Caterpiller Tractor Co. (1984) VEHSIM Hauling Unit Simulator have been incorporated into the OPSIM haul time calculation subroutine, and the OPSIM results are generally within 0.5 % of those calculated by VEHSIM for identical conditions. In addition, the user may also assign a value to an overall haul time correction factor (OPFAC) to adjust calculated haul times in accordance with on-site measurements. This was felt to be necessary because vehicles are seldom operated to perform at their maximum capabilities.

Total hauler cycle time is calculated as the sum of the hauling time, maneuvering and dumping time at the dumpsite, return time, maneuvering time at the loading site, loading time, and waiting time if haulers are running in tandem. The number of haulers over any route is determined by calculating the cost per loose cubic yard for each possible number up to the maximum available and choosing the combination that results in the lowest per-lcy cost.

Output

Three main categories of output data result from execution of the program. These are equipment scheduling data, per-block cost and overburden movement data, and overall cost data.

Equipment scheduling information is generated continuously during execution of the program. As each job is completed, the program records the machine, location, job type, and the starting and elapsed times. This information is available to the user when the program completes execution.

As mining blocks are completed, the program calculates the total and per-cúbic-yard costs associated with each operation required to mine that block of coal. Only operational times are considered in this set of calculations. Other data output at this time include the total yardage treated by each operation, the yardage moved by each mode of transport to each fill cell, and loads per hour (lph) rates for hauled materials.

Completion of simulation results in the output of an overall mining and cost summary. Costs are calculated from accumulated operational and idle times. These costs are, in turn, broken down into components attributable to operation of each machine, wage and salaried payroll, and various overhead costs.

Uses and Limitations of the Simulation Model

A primary limitation of the model is that not all of the operations required for surface coal mining are simulated. Haul road maintenance, drainage and sediment pond construction, and the maintenance of haul roads and dumpsites are but a few of the many operations required on haulback mining sites that are not considered. However, the time schedule output allows the user who wishes to consider the effect of these neglected operations on the overall mining plan to integrate them manually into the simulation results.

There are many additional simplifications inherent in the construction of a model such as this. The difficulties of accurately predicting the variety of conditions which will affect the rate at which overburden is moved and coal is mined should not be understated. For example, the manner in which rock is shot will affect the rate at which it is moved. Also, the effects of weather, equipment failure, and other more or less 'random' events have not been considered, other than by allowing the user to consider such events when specifying input values. The limitation of machine models to one per machine type and the simulator's inability to handle more than one suite of equipment will prevent its application to certain mining sites. The above limitations indicate that it may not be realistic to expect the model to calculate an overall mining cost with exceptional accuracy, for a particular mining situation.

However, the model's strength is its ability to compare the costs that might result from alternative overburden handling plans. With the exception of haul road and dumpsite maintainance, the cost effects of most of the limitations described above remain constant, given a particular mining site. The user has the ability to assign values to most of the parameters which determine spoil movement rates, many on a route-specific basis. Thus, if the user takes the time required to develop accurate estimates of spoil movement parameters, the cost difference that results from comparing alternative mining plans should reflect the actual cost difference. Information on the cost effects of alternative mining strategies should be of use to land owners and mining firms in planning mining strategies and post-mining land uses.

III.D. SITE DATA

The case study site is located near the intersection of Rogers Ridge and Amos Ridge, as designated on the U. S. Geological Survey 7.5 minute Flat Gap and Norton quadrangle maps, approximately centered at $37^{o}59'30"$N latitude and $82^{o}42'00"$W longitude. The pre-mining topography consisted of a series of finger ridges protruding from a central "spine", Amos Ridge (Figs. 2, 3a). Excepting the tops of the fingers, nearly all the land being mined had slopes in excess of 20^{o}. This type of topography is common throughout the central Appalachian region.

The site at Amos Ridge is being mined under an experimental variance from the provisions of SMCRA which deals with the method of constructing a post-mining topography; the variance was obtained with cooperation of the U.S. Office Surface Mining (OSM) and Virginia Division of Mined Land Reclamation (VDMLR). Three hollow fills are being constructed so that their upper surfaces are contiguous with flat areas on the tops of the finger ridges, which are not being returned to their original heights (Fig 4). The first hollow fill (HF1) is complete, excepting approximately one acre of upper surface area which requires topsoiling and revegetating. HF1 was constructed using conventional durable rock hollow fill techniques, as defined in section 816.74 of the Virginia regulations implementing SMCRA: it was end dumped, and a "blanket drain" of large "durable" (will not slake in water) rocks extends beneath the entire body of the fill. The second and third hollow fills (HF2 and HF3) are permitted for construction using experimental techniques. In both, up to 45% non-durable rock may be used. HF2 is being constructed by side dumping to form a durable-rock core "chimney drain" approximately 25 feet thick extending from the base of the fill to its surface along its entire length; HF3 is planned for construction with a 15 foot diameter durable rock underdrain, covered by filter fabric extending beneath the full length of the fill. The objective of producing usable land is being pursued by constructing all three fill outslopes at a 3:1 grade, rather than the conventional 2:1, and by using 3 to 4 feet of uncompacted soil and spoil materials to construct plant growth media

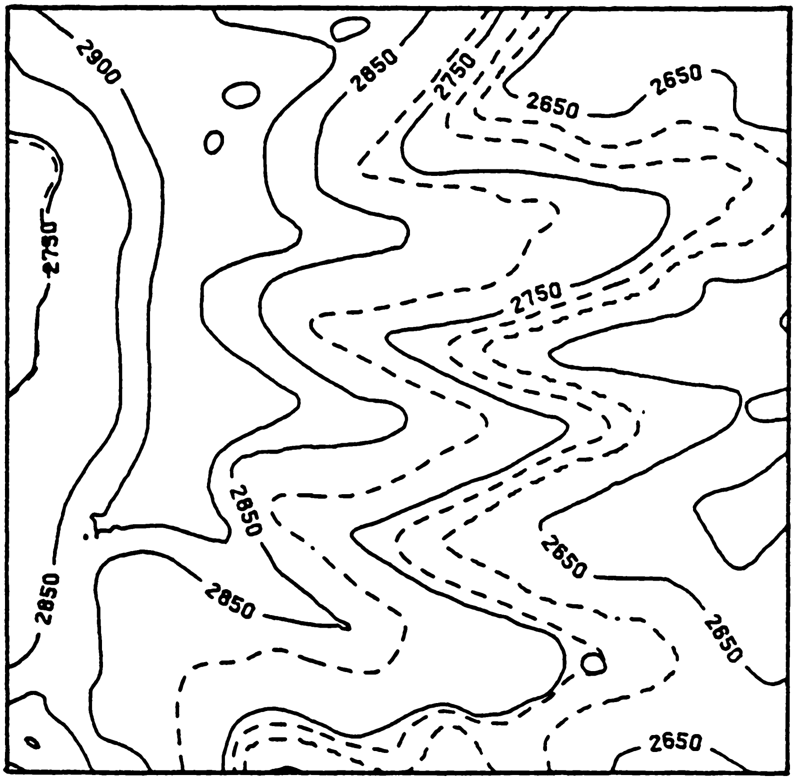
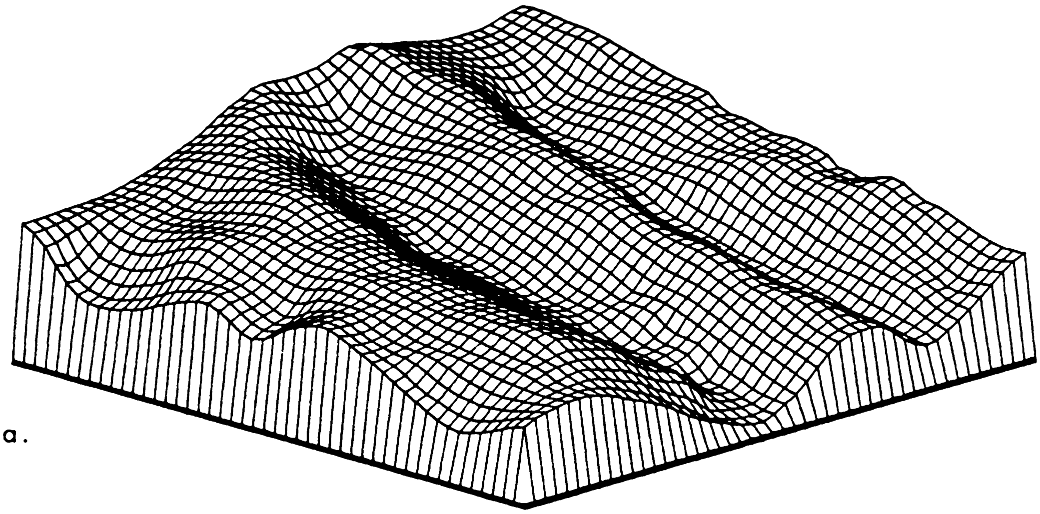
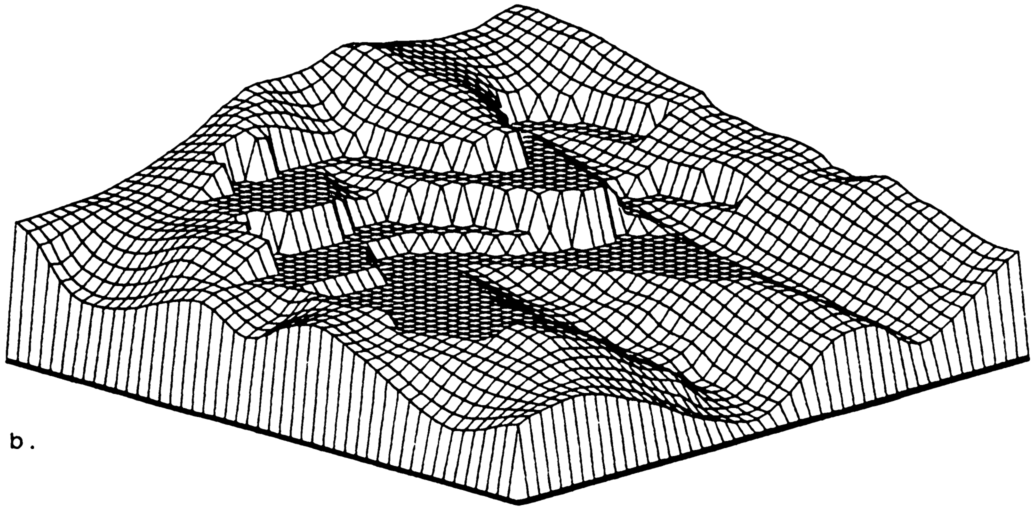


Figure 2. Pre-mining contours at the Amos Ridge case study site. The area represented is 2000 feet square. Dotted lines represent coal seam outcrops. The hollow whose center is approximately 600 feet north of the southern (lower) map border is the location of the first hollow fill (HF1). The center of hollow fill 2 (HF2) is located approximately 400 feet further north, while the third fill will be constructed in the hollow closest to the northern edge of the map.



a.



b.

Figure 3. Isometric representations of the Amos Ridge mining site. The area represented is 1400 feet square; the bottom corners of the isometrics are located near the southeastern corner of Figure 2. (a). The pre-mining landform, showing finger ridges protruding from the central "spine" of Amos Ridge at the western edge of the image. (b). A representation of the portion of the topography mined during the period of study. This would have been the appearance of the site, had all spoil been removed from the area. However, the site never took this appearance since the economics of mining dictate minimization of spoil movement distance.

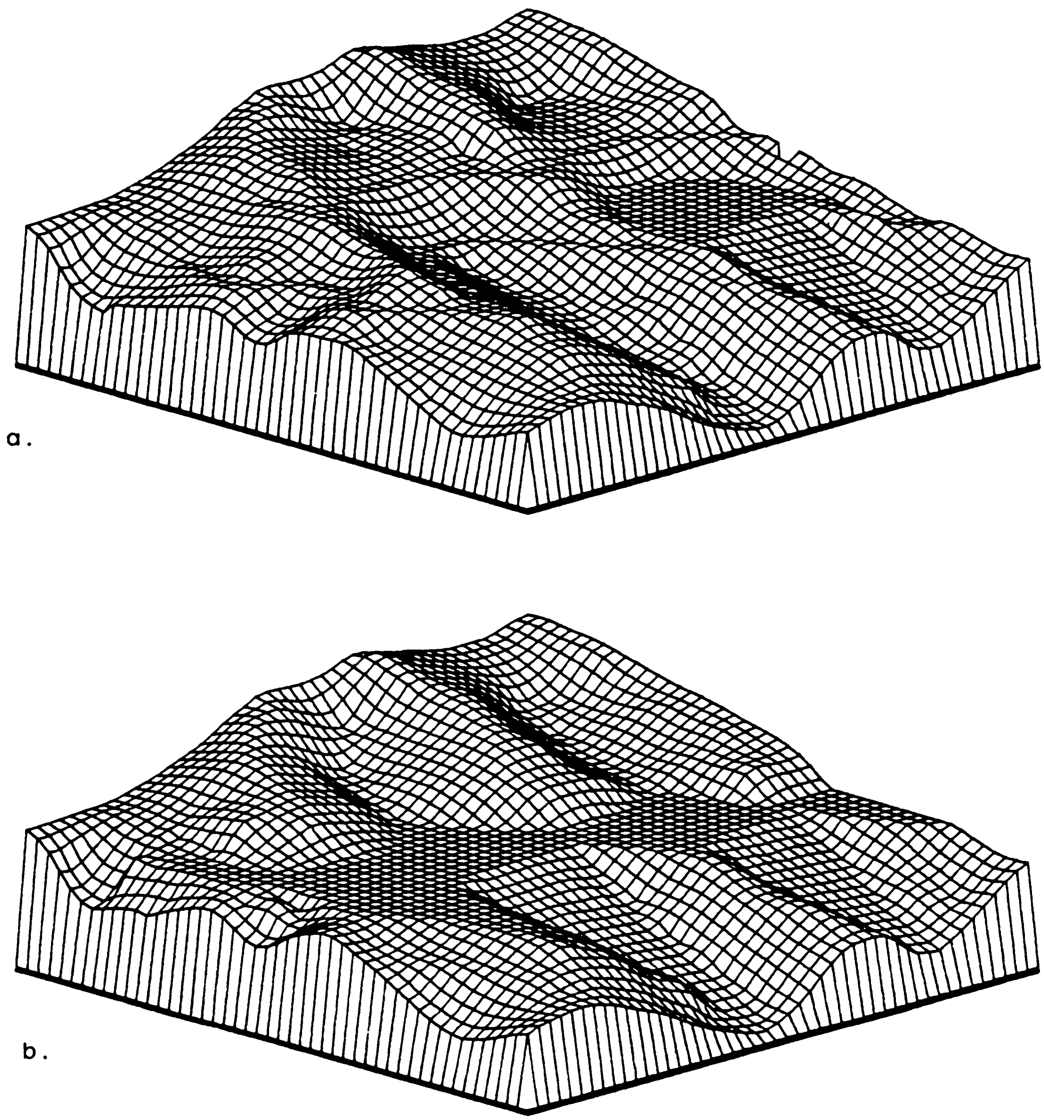


Figure 4. Isometric representations of possible post-mining topographies at Amos Ridge.
(a). A likely post-mining landform had the site been mined using conventional methods, the approximate original contour (AOC) case. An access road is located approximately 40 feet above the outcrop of the Lower Marker coal. Excess spoil is disposed in the first hollow above the mining bench and in the fill constructed in the second hollow.
(b). The postmining landform currently under construction, the Landform Alteration (LA) case. The outslopes of hollow fills 1 and 2 are represented by the triangular-shaped surfaces between the finger ridges at the lower right hand side of the image.

on the near-level upper surfaces. The mining permit calls for all hollow fills to be monitored for water quality, stability, compliance to construction specifications, and cost. The final landform will include a broad, near-level bench area extending over the stripped fingers and filled hollows (Fig. 4). An agricultural post-mining land (hayland and pasture) is planned for the site.

Amos Ridge Coal Company is a small firm by modern standards. Privately owned and operated, the firm employs 7 men on a full time basis. Machinery owned by the firm and operated at the site includes three haulers (two Terex 33-09, and one Caterpiller 773B used primarily for backup), three overburden loaders (two Caterpiller 992B and a Caterpillar 988), a coal loader, two Caterpiller D9G dozers, a Fiat Allis FL-14 dozer that is used strictly for reclamation grading, a blasthole drill, a coal auger, and an assortment of minor machines including a Finn hydromulcher. Thus, the machinery is never fully utilized but the operation is highly flexible. The machinery is conscientiously maintained on site, and most is being used beyond what would generally be considered its useful economic life.

Three seams of flat bedded coal are present (Fig. 5). All are low ash premium grade bituminous. The majority of the overburden is sandstone. Except where they have weathered over the points, the sandstones below the Upper Marker are hard enough to meet OSM's durable rock criteria. The three seams are contour mined in sequence, from top down, and the final cuts are augered. The location of a deep mine in the Low Splint close to the stripping operation indicates that the final landform is not likely to be disturbed by re-mining in future years.

Site Data Collection

The objective of the economic monitoring program at Amos Ridge was to determine the costs of mining at this site (Zipper and Daniels, 1984). Information collected at the site included data on machinery operation, mining progress, and unit operations.

In order to estimate mining costs, it is necessary to know the hours and locations of operation, and tasks performed, for each piece

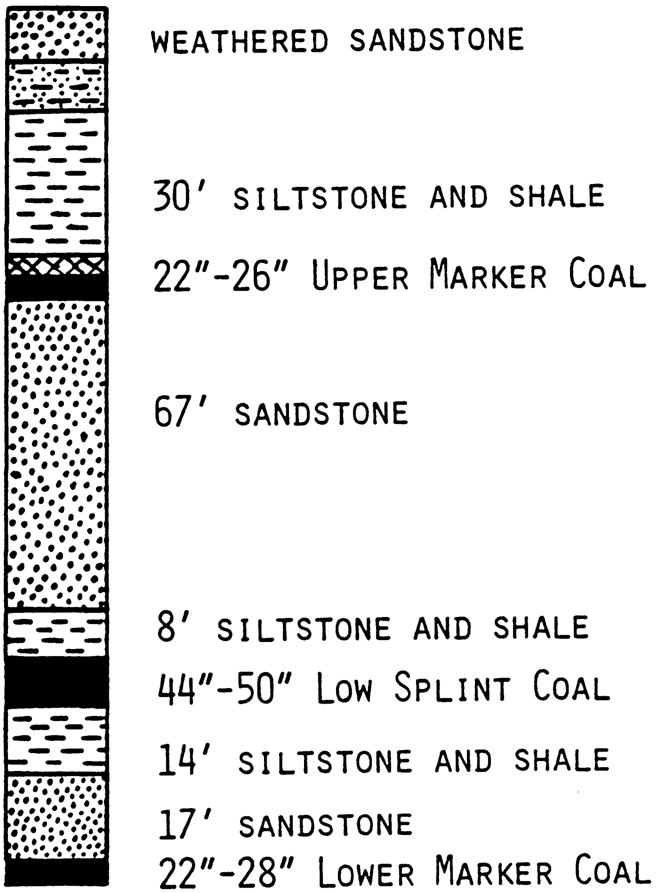


Figure 5. The overburden sequence at Amos Ridge. Strata thicknesses represent average values.

of machinery on the site each working day. Forms were designed specifically for that purpose (Fig. 6). The objective for designing the forms as shown was to provide a format for easy, non-time consumptive recording of appropriate information by personnel who are at the site each day. Locations are specified with reference to the mining blocks, which are defined and numbered as appropriate at the site (Fig. 7). A map of the portion of the site currently being worked and a series of symbols to represent the most common operations are attatched to the data recording form. Each day's operations are located on a map using these symbols.

Mr. Andy Hall, the owner of Amos Ridge Coal Company and job foreman, consented to complete these forms on a daily basis. He estimated that approximately 10 minutes of his time per day were required to complete the forms. He spent this time at the close of the day, and recorded his recollection of what occurred. An excellent record of daily operations was compiled, mainly due to his attention and diligence. On those rare days when he was not at the site, he recorded the data based upon information received from those who were.

All operations requiring 20 minutes or longer were recorded; times of operation were estimated to within one quarter hour, whenever possible. A problem with the unit operations approach was the many occasional activities which did not fit listed categories. The ample comment spaces allowed these activities to be recorded and provided a means for choosing an appropriate category for the recorded time based upon the objectives of the monitoring program. Another problem was encountered when multiple operations occurred more or less simultaneously. For example, if hauled material was being segregated, no single destination applied. In these situations, the person recording the data was asked to use his judgement regarding the distribution of the total time period among the tasks performed. If no judgement was recorded or could be recalled, a 50%-50% distribution was assumed.

Additional daily data have been recorded by the hauler drivers since early August, 1984. Each hauler was outfitted with pencils, a clipboard, a calendar, and a hand operated counter. The drivers were asked to use the equipment provided to count and record the number of

DAILY RECORD OF MACHINERY OPERATION AND
MINING PROGRESS - AMOS RIDGE COAL CO.

Date:____________________ Day: M Tu W Th F S (check one)

Person filling out form: ____

Weather: ____

1. HAULERS

Machine	NH	Haul From 1	Haul From 2	Haul From 3	Haul To Block #	Distribution	Distance	Cycle Time	Loads
No. 1									
No. 2									

Comments:

NH: No. of hours hauling to each location or set of locations.

Haul From: 1 = 992 #1 2 = 992 #2 3 = 992 #3 or 988 or 945B

Haul To: Please write Block # or #'s; if a Block number does not describe the location, please describe as comment.

Distribution: If hauler(s) haul to more than one location during "NH" time period, please indicate proportion hauled to each.

Distance: Please estimate average one-way haul distance, in feet.

Cycle Time: If you estimate an average round trip cycle time (including time to load) please record.

Loads: Number of loads hauled (to be filled in by CZ from data recorded by drivers.)

Figure 6. An example of the forms used for recording daily data at Amos Ridge.

2. LOADERS

				Operation							Material					
Mach-ine	NH	Block No.	Lift No.	L1	L2	LC	CD	R	DS	O	S	TSS	SO	Tx	C	O
992																
945 B																
988																

Carry & Dump: Approx. one way carry distance = _____ft. to Block_____

Comments:

NH: Number of hours machine was operated, production time only.

Block No.: Location of Operation (H1 = first hollow, H2 = second hollow, DS = Dump Site.)

Bench No.: (For L1, L2, CD only) 1 = top bench, 2 = second bench, X = To Coal

Operation: If "NH" time period is used for more than one operation, please distribute NH hours among operations, or indicate time distribution using fractions or percentages.

L1 = Load 1 hauler

L2 = Load 2 haulers

LC = Load Coal, clean coal, prepare coal for loading

CD = Carry and Dump

R = Road work (no need to record routine haul road scraping)

DS = Dumpsite

Material:

TS = Top Soil

TSS = Top Soil Substitute

SO = Shot Overburden

Tx = Toxic material, special handling

C = Coal

O = Other

Figure 6. Continued.

3. DOZER

Machine	NH	Location	ST	PD	FO	PO	DS	Rec	HB	HF	HD	HT	R	O
D9#1														
D9#2														
FL 14														

PO: Approx. push distance = ______ft. from______Lift of Block______ to Block ______. (1 = Top Lift, 2 = second, X = To Coal)

Comments:

NH: Number of Hours of operation, production time only

Location: Please write Block Number or Block Numbers

Operation: If "NH" time period is used for more than one operation, please distribute "NH" hours among operations, or indicate time distribution using fractions or percentages. If this is impossible, please explain.

ST = Strip Topsoil

PD = Prepare Drill Bench

FO = Feed Overburden to loader

PO = Push Overburden from one location to another

DS = work Dump Site

Rec = Reclamation; grade or prepare surface for seeding

HB = Hollow fill, work Body

HF = Hollow fill, work Face

HD = Hollow fill, work Drainage

HT = Hollow fill, work Toe

R = Road work within mining area (no need to record routine haul road scraping)

O = Other (please comment)

Figure 6. Continued.

4. DRILLING AND BLASTING

		X if To Coal	Drill Hours:
Block. No._____	Lift No.______1 (top)	_______	_________ = warmup
	______2	_______	_________ = drill
	______3	_______	_________ = move

Number of Holes Drilled:____________ Average Depth:_____________ft.

Diameter (if not 5 $\frac{7}{8}$):_______in.

Lb. ANFO:____________ Hrs. Labor to fetch, load, and shoot:_______

5. MISCELLANY

Labor:______________men,_______________hours each, plus_____________

Additional machinery
and hours of use

Did anything unusual
happen today to
disrupt normal
operation?

6. AUGER

_____________ = Location _____________ Hours of operation

_____________ = Number of man hours to operate or set up

_____________ = Number of holes _____________ = hole diameters

_____________ = (feet) (sections) of average hole depth

7. COAL LOADED TODAY - Please record pit dimensions on map

_____________ = average thickness of seam (inches)

_____________ = number of loads stripped from Block _____________

_____________ = number of loads stripped from Block _____________

_____________ = number of loads augured from Block _____________

8. LONG TERM RECORDS

Fuel Delivered today:__________________gallons

Long Term Coal Production Figures:________________ Tons produced

From_______________ To_______________

Figure 6. Continued.

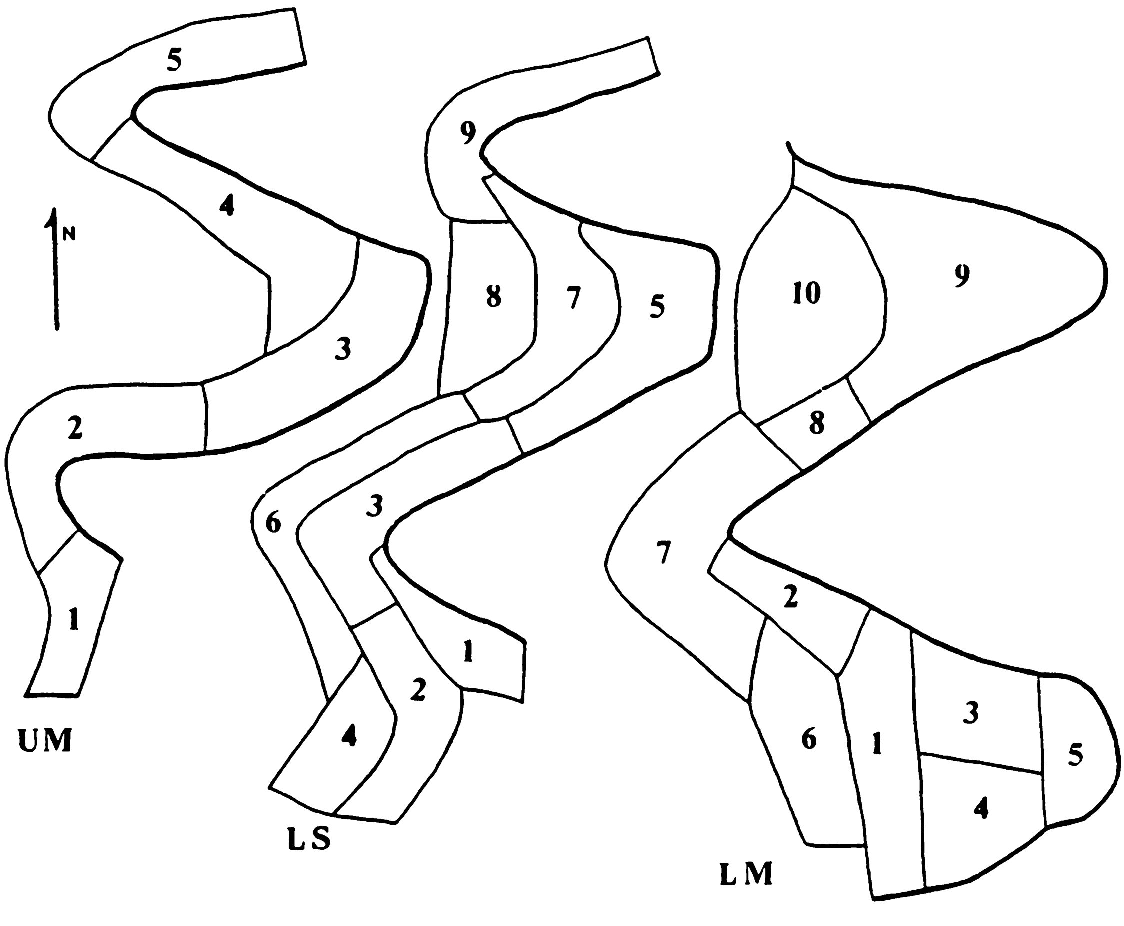


Figure 7. The mining blocks at Amos Ridge. Upper Marker blocks are on the left, and Lower Marker blocks are on the right. The manner in which the blocks overlie each other is not represented.

loads hauled each day. Monthly visits were made to the site each month to pick up the machine operation and hauler load forms and to provide fresh forms for the succeeding month.

During monthly visits to the site, photographs and selected site measurements were taken. The objective for taking site measurements was to provide data for estimating overburden volumes. Any mining block which was more or less empty was measured during each site visit. The lateral dimensions were measured with a tape and/or a measuring wheel. Corner points were located on a rough sketch and dimensions were recorded. This task was made considerably more difficult if site visits did not coincide with the period after overburden removal but before substantial backfilling. Mr. Hall measured the blocks that would have been exposed and partially backfilled between site visits. Highwall heights were estimated using an abney level to estimate bearings to the top and bottom from a point separated from the base by a measured distance.

Mining block measurements were supplemented by periodic surveys, which allowed the current mining blocks to be located on the site map. The previously recorded measurements then were used to estimate the locations of the non-surveyed blocks, through reference to survey data and coal outcrops. The surveys also were used to locate the coal seams at various points, and to assist with definititon of the post-mining topography.

Data on unit operations also were recorded, as appropriate. The primary operations of interest were those modeled by OPSIM: drilling of blastholes, and overburden movement by dozers, loaders, and haulers. Of these, hauling was by far the most critical to the objectives of the study. Loading and hauling times and haul road profiles were recorded for 19 separate haul routes, the majority on the Amos Ridge site (Appendix B).

Analysis of Site Data

Daily operations data collected at the site between January 1, 1984 and August 1, 1985 form the basis for this study. These data were supplemented with records of reclamation operations performed in disposal areas utilized during the period of study whose reclamation was completed afterwards.

Hourly owning and operating costs for the machines on site are based upon the fiscal 1984 Financial Statement of the Amos Ridge Coal Company, industry average costs, and the record of daily data collected during fiscal 1984. All expenditures made during the 1984 fiscal year (1 November 1983 through 31 October 1984) were allocated to the following budget categories: labor, machinery, drilling and blasting supplies, and overhead. The account for drilling and blasting supplies was distributed to the mining blocks directly on the basis of the daily data. The labor account and the year's total labor hours were used to develop a per hour labor cost figure. Wages paid for machinery operation were treated as machinery operation costs. Residual labor hours (hours paid but not spent operating machinery) were considered to have been spent repairing and maintaining the machinery and wages paid for those hours were added to the machinery account; that account was then distributed to per hour costs for each machine on the basis of industry average ownership and operating costs (Dataquest, Inc., 1984) and the number of hours recorded as operational for each machine. During fiscal 1984, hourly machine costs calculated as outlined above were 80% of the industry average. The cost figures reported below assume that fiscal 1984 costs remained constant through the first nine months of fiscal 1985 (Table 3).

All costs incurred by the firm were distributed to the mining blocks using the COSTSUM programs. The spoil handling cost figures reported here do not include coal handling (augering, cleaning, loading, hauling) costs.

Block volume estimates were prepared on the basis of site measurements (survey data and dimensions of coal exposed in each mining block) and a contour map. Survey data included the coordinates of points on the coal seams at various locations. Since each coal seam approximates a plane, least squares regression analysis converted these coordinates into second order polynomial equations approximating the locations of the coal seams. Radian Corporation's CPS/PC was used on an IBM-AT microcomputer to transform the digitized site contours and the coal seam equations into contoured plots of the site with

Table 3. Unit costs used to estimate overall mining and reclamation cost at Amos Ridge.

Cost item[a]	Estimate
Terex 33-09 and Cat 773B haulers	$75.73
Cat 992B loader	114.17
Cat 988 loader	82.11
Fiat Allis 945B loader	72.19
Cat D9G dozer	69.47
Fiat Allis FL-14 dozer	46.64
Chicago Pneumatic Model 650 blasthole drill	73.12
Drilling and blasting, per hole	2.45
Blasthole drilling, per foot of hole	0.07
Blasting agent (ANFO), per lb.	0.11
Mack Model 800 coal truck	50.63
Water truck (modified Mack Model 600)	48.93
Huber road grader	41.52
Finn Model 80 hydromulcher	36.80
Backhoe	36.77
Salem Tool Co. MBT-18 coal auger	72.55
Foreman labor, per hour	8.00
Overhead, per ton of coal produced	2.89
Temporary employee labor hour	8.00
Contract coal hauling, per ton	1.50
Seeding supplies, per hydromulcher batch[b]	90.00

[a]: All costs per machine operating hour, including operator wage, unless otherwise noted.
[b]: Approximately 4 batches required per acre.

traces of the coal seam outcrops. The survey data and coal block measurements were used to draw the coal blocks onto the output contour map, thereby delineated the coordinates of polygons describing the coal blocks. CPS/PC was then used to calculate the first approximation of the block and lift volumes; these were adjusted to reflect deviations of the coal seam coordinates from the polynomial approximations, deviations of lift thicknesses from the average "slice" volumes computed by the software, and a few obvious inaccuracies in the contour map as evidenced by the blasting and survey data. In retrospect, additional time in surveying would have been well spent as it would have improved the accuracy of the volume estimates. I believe that the estimates reported here are within 10% of the true volumes, with the greater uncertainties for the earlier mining blocks.

Estimating Overburden Movement

The quantity of spoil moved from each mining block to each fill cell and overburden swell were estimated through the use of the program MOVE of COSTSUM.

After determining block volumes and making an initial estimate of swell for each lift based upon rock type and characteristics and tables prepared by Caterpiller Tractor Co. (1985) and Terex Corp. (1981), an initial estimate of overburden movement was prepared. A first approximation was made using an assumed overall dozer movement rate of 275 lcy/hr and loader carry rate of 330 lcy/hr (33 loads per hour, 10 yard bucket). These factors were derived from on-site observations and comments of the operator regarding his methods for deciding when to move spoil by dozer push and/or loader carry. Hauler movement estimates were based upon the output file from program HAULER containing the recorded loads-per-hour (lph) hauler rates since August 1, 1985. Rates were estimated for routes run previous to that date, based upon the overall recorded 8.5 lph rate, distance, change in elevation, and route-specific recorded rates. The moved:topo ratios for each lift were assumed to correspond to loader and dozer bucket fill factors, i.e. the proportion of total bucket (blade) capacity utilized on average while removing spoil from a particular lift. The next step was to vary the assumed movement rates on a route-specific basis, referring to distances and conditions recorded on the daily forms. Swell factors were also varied, when necessary, within the listed ranges for the various rock types. When these modified assumptions successfully yielded moved:topo ratios within the ranges listed as typical bucket fill factors for the appropriate rock types by Caterpiller (1985) and Terex (1981) the results of the operation were assumed to approximate the actual movement rates at the site. This is a very crude method of estimation, but it is the best that could be done under the circumstances.

The next problem was the realization that the estimated movement rates were dependent upon the estimates of swell, which were themselves ballpark estimates. Therefore, site measurement data was referenced to construct a model of the post-mining topography using CPS/PC. That model was used to estimate the total volume backfilled into areas mined during the study period and placed in HF1. Separate totals were calculated for Upper Marker spoil and for Low Splint - Lower Marker spoil, since these materials were, for the most part, kept separate at the site. Finally, these two totals were compared to the corresponding estimates of spoil volume prepared using program MOVE. The results allowed swell factors to be revised. The resultant moved volume estimates are consistent with mining block, swell, and fill cell volumetric estimates and were used as the basis for simulation studies.

III.E. SIMULATION OF MINING ALTERNATIVES

The objective of simulation was to estimate the cost of producing the post-mining landform at Amos Ridge (the Landform Alteration, or LA, case). The method chosen was to consider the cost of landform production as an incremental cost, given that the post-mining landform is actually a byproduct of the mining process. That incremental cost was estimated in reference to the mining cost of the most likely alternative: returning the land to its approximate original contour (the AOC case) as required by law.

General Assumptions

Since the estimation of a cost difference was the objective of the modeling procedures, only those factors assumed to differ between the two cases were considered. A primary assumption was that the major cost difference between the two landforms is spoil handling cost. Therefore, only spoil handling operations were modeled. These include drilling, dozer push, loader carry, and load and haul. Strictly speaking, it was unnecessary to model drilling due to the assumption that identical volumes of overburden and coal would be removed in both cases. Non-modeled costs judged to have an effect upon the cost comparison were added to the analysis manually at the conclusion of modeling procedures.

Another important assumption was that only the costs of operating machinery would be considered; in other words, the mining firm bears no cost if a piece of machinery sits idle for any period of time. This assumption seems to hold true at the Amos Ridge site where there is an excess of machinery over manpower and all machines are owned outright. Therefore, costs of insurance and other costs which accrue whether the machine is used or not were considered as overhead rather than machinery ownership costs. The assumption that the operating costs of these machines remained constant over time, i.e. that the cost of operation during fiscal 1984 could be carried over into 1985, was made on the basis of convenience since fiscal 1985 data were not available at the time of this study.

Finally, it was assumed that identical areas would be mined regardless of post-mining landform. Again, this was an assumption of convenience. However, looking at the LA case from a standpoint of strictly-defined profitability, the mining firm would have been better off not taking some of the cuts that they did take. Therefore, assuming that the AOC mining case would only take the most profitable cuts would have been an unequal comparison.

The major portion of the simulation process was performed blind to cost. Spoil handling assumptions were made, spoil handling plans were developed to implement those assumptions, output was checked and spoil handling plans modified in order to develop the desired simulation strategy. In the LA case, the desired strategy was a near-duplication of the COSTSUM results; in the AOC case, the desired strategy minimized long up-grade hauls to the greatest degree possible while preserving the most cost-effective elements of the LA strategy (disposal of maximum spoil in adjacent areas) whenever possible. Finally, when the mining plans were judged to have been successfully implemented, the cost output was obtained and analyzed.

Mining Plan Assumptions

The first step in defining spoil movement was to produce models of the post-mining landform alternatives using CPS/PC (Fig. 4; Zipper and Daniels, 1986b). Due to spoil handling considerations, the AOC contours were assumed to vary from the original contours. The primary consideration was disposal of excess spoil. At least one hollow fill would be necessary to handle the excess spoil generated from this site if an AOC mining plan were to be implemented. Since the second hollow is centrally located to what is presently the experimental practice area, this would be the most likely hollow for fill construction. Also, it was assumed that the operator would desire to keep hollow fills to a minimum, due to their expense. Therefore, it was assumed that additional excess spoil would be disposed of by building up the AOC backfill in the area above the present HF1. This is a common practice. An access road approximately 35 feet wide was located approximately 40 feet above the level of the Lower Marker coal seam where it could perform the additional functions of interupting the long downslopes, as required by law, and giving access to the steeply sloping areas above and below for seeding. Finally, it was assumed that the slopes of the rebuilt point above blocks 1LM and 3-5LM (Fig. 7) would be built up to grades approaching 2:1 (27°), in order to allow for maximum spoil disposal, since the operation was threatened with a "spoil-bound" condition during simulated mining of the 2LS - 4LS - 6LM - 7LM area. Slopes in the hollow above the present HF1 were assumed to approximate 2.5:1 grades.

The next step was to define fill cells and to estimate fill cell volumes for both cases. Fill cells in the mining areas were defined as laterally identical to the mining blocks in order to simplify notation and to remain consistent with the Amos Ridge record of daily operations. Fill cells in the areas mined on the experimental practice area but previous to initiation of detailed data recording on Jan. 1, 1984 were also defined in reference to the coal seams. Fill cells above the former Low Splint seam were defined in three 26.3 foot lifts, to the base of the Upper Marker seam. Upper Marker fill cells were defined in 25 foot vertical segments, and Lower Marker cells were defined as a single 33 foot thickness. CPS/PC was used to estimate the volume of each of these fill cells for each post-mining topography. Cells which had a volume of 0.0 cubic yards for both topographies were discarded. Some of the AOC cells had 0.0 cubic yard volumes in the LA topography.

Identical spoil movement assumptions were used in both cases. Lift thicknesses and drilling rates, rock characteristics, drilling fractions, dozer assist fractions, and bucket correction factors were kept constant for the two cases and based upon the results of analysis of site data using the COSTSUM programs (Appendix A). The proportions of each lift moved by dozer push and loader carry modes (generally the least expensive way to move spoil), and rates of movement by these modes, were also kept constant between the two cases and consistent with the COSTSUM data. The only exceptions to the above occurred in situations of the AOC case where such assumptions would be clearly inappropriate. For example, since HF1 was not constructed in the AOC case, material could not be carried or pushed to HF1. Also, spoil disposal difficulties in the 2LS - 4LS - 6LM - 7LM area (Fig. 7) indicated that a reduction in the amount of spoil moved by carry and push was required. Similarly, the quantities of hauled spoil disposed of in blocks of origin or in adjacent blocks were kept constant, when possible. Quantities, origins, and rates of movement of spoil to HF2 were also kept constant for the two cases.

The primary load-and-haul assumption was that hauler efficiency and other load-and-haul cycle components remain consistent with estimates made on the basis of analysis of the hauler cycle data set (Appendix B). It was also necessary to assume a series of hauler routes for each mining plan. Observed hauler routes were approximated and held constant between spoil handling plans where appropriate. Again, site geometry made this assumption inappropriate for certain routes of the AOC case. Observed hauler routes traversing the upper surface of HF1 could not be defined to model AOC reclamation procedures. In addition, the spoil movement pattern over the 1LM-3LM-4LM-5LM point had to be assumed as greatly altered due to the different post-mining landform being constructed over this area.

Common assumptions were used in defining the haul routes. In accordance with on-site observations, the maximum grade used for routine hauling was 8 %. Only in situations which required rapid vertical ascent was this slope exceeded. In all cases, the maximum hauling grade was 15 %; and the maximum downhill hauling grade was 13 %. When defining hauler routes, horizontal and vertical distances were conserved through reference to a site map with coal seam elevations. The loading point was assumed to be in the center of the mining block at the level of the top of the coal seam. Backfilling was assumed to occur in lifts 25 to 33 feet in thickness as described previously; the spoil delivery point was placed in the geometric center of the fill cell approximately 20 feet above its base (25 feet for Lower Marker fill cells). The only exceptions to this rule occurred early in the backfilling process, where the 8 % grade rule would be exceeded by adhering to the spoil delivery point rule and in the final stages of the backfilling process, when delivering to a cell located near the final surface that was not filled to anything near its capacity. The minimum hauling distance used was 250 feet, since 150 - 200 feet is the range of distance where loader carry operations generally are cut off. When constructing roads, the only speed limits used were entry limits; a 5 mph limit was used to define hairpin curves, a 9 mph limit for sharp right angle turns, and a 12 mph limit for broad turns that would require limitation of speed (Caterpiller, 1984). Occasionally, a 7 mph limit was used for points which did not clearly conform to either of the first two situations.

A final assumption had to do with the construction of surface media. Due to the generally favorable overburden characteristics at the site, no special topsoil or surface medium isolation procedures were assumed for the AOC case. All topsoil hauling operations recorded in the site data were duplicated in the LA mining plan.

Implementation of Mining Plans: Problems

The LA mining plan was designed to reproduce observed conditions as closely as possible. The first problem in implementing this design was a contradiction between the site data and the specifications of OPSIM, which only allows one destination to be specified per mode of transport for each lift. The data clearly showed that materials were hauled from each lift to multiple destinations. This problem was solved by breaking each lift into as many sub-lifts as necessary to distribute the mining spoil to fill cells in accordance with the observed spoil handling strategy. Each sub-lift retained the loading characteristics (rock type, drilling rate, swell, bucket fill factors, fraction of loading time assisted by dozer) of the original lift; thickness was also maintained while total area was partitioned among the sub-lifts in proportion to the quantity of material. In some cases, the original block had to be defined as two or three simulation blocks, with coal defined only for the final simulation block, due to the multitude of spoil movement destinations and the OPSIM limitation of an individual mining block to 5 lifts.

There were similar problems in specifying destinations. Early in 1984, most of the spoil was disposed of in an area directly south of the experimental practice area being reclaimed to AOC. The volume control was poor for spoil disposed in this area due to a lack of hauler load counts and detailed area measurements. Information recorded on the daily forms included only distance estimates and the approximate elevation of the disposal area, defined relative to the coal seam being backfilled. Assumptions about spoil movement rates to these areas were required in order to estimate overburden movement with the program MOVE. The simulation strategy for these areas was to define haulroads to implement the assumed movement rates and to keep the amount of spoil moved to these areas constant between the two cases. A problem with implementing this strategy was the difficulty of excess spoil disposal from the 2LS-4LS-6LM-7LM area (Fig. 7), due to the lack of HF1 and unavailability of HF2 at that stage of the mining process. This problem was solved by moving an additional 10000 lcy back to the AOC from blocks 3-5LM at rates similar to those assumed during overburden movement estimation procedures; delivery of this spoil was assumed to be evenly split between the Low Splint and Upper Marker coal seam elevations.

Another problem in implementing the LA mining plan was the disparity between LA fill cell volumes and estimates of overburden quantities moved to the fill cells on the basis of daily records using program MOVE. This disparity is quite understandable, since identification of former mining blocks during backfilling after elimination of landmarks through mining is quite difficult. The problem was solved by making another series of assumptions sufficient to cause the excesses from blocks recorded as being "overfilled" to be placed in adjacent "underfilled" locations.

In order to implement an AOC mining plan, another series of spoil movement assumptions had to be made to distribute the total quantity of available mining spoil to the fill cells. The observed lifts were maintained, although the pattern of distributing these lifts to sub-lifts was altered. The biggest problem in implementing this plan was near spoil-bound condition of the site while mining the 2LS-4LS-6LM-7LM area, as already discussed.

IV. RESULTS

IV.A. SPOIL HANDLING AND RECLAMATION PROCEDURES AND COSTS

The average cost of spoil handling at Amos Ridge between January 1, 1984, and August 1, 1985 is estimated at $1.90 per bank cubic yard (Table 4). However, spoil handling costs varied widely between blocks (Table 5). The major factors influencing spoil handling cost were ease of overburden handling, as related to landscape position, the proportion of total spoil moved by hauler, and distance to hauler disposal areas. The data indicate that the smallest spoil movement costs occurred in the first cut point blocks, while the largest costs were recorded for first cut blocks in the hollows where hard sandstone was encountered.

On the points, weathered rock was sometimes moved without blasting; the remaining overburden was generally drilled and shot less expensively and more cleanly than the harder, less weathered materials encountered in subsequent cuts on the points and in the hollows. After shooting, these softer materials were moved more efficiently, due to the lack of large boulders and general ease of handling. Landscape position also contributed to more efficient material movement from the first cuts on the points: loaders and dozers had better access to the overburden and to adjacent spoil disposal areas. In the hollows, the opposite situation occurred. The harder rock did not shoot as cleanly or efficiently and was not as easy to work; greater proportions had to be moved by hauler, resulting in larger spoil movement costs. In the second and third cut point blocks, spoil handling costs were found to be dependent upon mining practices. Generally, the greater the proportion of these materials moved by carry and dump operations and the shorter the haul, the more efficiently the spoil could be handled.

Appendix C contains a detailed account of spoil handling practices at Amos Ridge in which observed cost differences are related to landscape and overburden characteristics and spoil handling strategies. The fact that spoil handling costs can be related to observed practices in a rational manner indicates success in developing cost monitoring and analysis procedures.

Table 4. Components of average spoil handling cost at Amos Ridge.

Component		$ per bcy[a]
Clear and bench		0.01
Drill and blast		0.41
Carry and push		0.20
Load and haul		
Dozer feed	0.07	
Loading	0.35	
Hauling	0.38	
Dumpsite	0.09	
Total		0.89
Reclamation		0.08
Overhead		0.31
Total		$1.90

[a]: a bank cubic yard is a measure of overburden volume before blasting and associated swell.

Table 5. Estimated costs of spoil handling at Amos Ridge.

Mining Area	Stripping Ratio[a]	$ per bcy	$ per ton
1LM	11.4	1.90	21.64
2LM	11.8	2.36	27.73
3LM	9.8	2.16	21.20
4LM	10.0	2.14	21.31
5LM	14.3	1.15	16.40
6LM	11.3	2.47	27.93
7LM	6.8	2.12	14.52
8LM	9.3	1.76	16.31
9LM	12.0	1.79	21.36
1LS	5.8	2.60	15.07
2LS	13.5	2.09	28.15
3LS	5.3	2.09	11.13
4LS	14.6	1.55	22.64
5LS	6.0	1.08	6.54
6LS	6.7	2.09	14.00
7LS	9.0	1.71	15.42
1UM	21.8	2.03	44.35
2UM	11.5	1.82	20.87
3UM	10.3	1.20	12.42
4UM	11.0	1.77	19.35
Totals			
LM seam	10.3	1.98	20.54
LS seam	8.4	1.87	15.82
UM seam	12.0	1.69	20.30
Average	9.6	1.90	18.38

[a]: Bank cubic yards (bcy) spoil handled per ton of coal mined. Coal was augered in blocks 6LM, 7LM, 8LM, 6LS, 2UM, and 4UM. Blocks 10LM, 9LS, and 5UM were partially mined during the period of study.

Reclamation Costs

Four primary areas were reclaimed during the reporting period. These were hollow fill HF1, the relatively flat area on the point above blocks 1LM and 3LM through 6LM (PTA), and two areas where highwalls were backfilled. Operations considered as reclamation costs for the mined areas include surface grading, seeding, and special handling of topsoil or topsoil substitute materials; the entire cost of HF1 construction was considered as reclamation. As calculated, reclamation did not appear to be an important spoil handling cost variable, accounting for less than 5% of the total cost (Table 4). The block with the largest reclamation cost component was the block from which the largest proportion of material was placed into fill HF1, 3LS ($0.17).

Approximately 225,000 loose cubic yards (lcy) of spoil were placed in fill HF1 for a direct cost of approximately $31,000 (Tables 6 and 7). Thus, the cost of spoil disposal in HF1 is approximately $0.19 per bcy disposed. The major costs of constructing HF1 were rough surface grading and reclamation (36%), cleaning topsoil and trees from the hollow previous to spoil disposal (25%), and rehandling the dumped spoil (22%). The major purpose of the latter operation was to move the larger boulders that segregated at the base of the dump piles into the blanket drain.

The PTA surface was constructed close to the level where the Low Splint seam outcropped on the point. However, since the Low Splint dips down into the hollow at a steeper grade than the PTA surface, that surface intersects the unmined topography at levels between 10 and 20 feet above the Low Splint seam. The estimated spoil disposal volume in the roughly 2.5 acre area is 160,000 lcy. Reclamation was accomplished by end dumping topsoil and well shot spoil on the rough graded subsurface in closely spaced piles, followed by grading and seeding. The reclamation cost is calculated at $4900, or $0.04 per bcy. This figure would be increased if rough grading were to be considered as a reclamation cost rather than a hauler dumpsite cost. However, the cost of working the dumpsite per bcy dumped in this area was approximately 70 % of the job average.

Table 6. Estimates of reclaimed areas and spoil volumes, and costs of reclamation at Amos Ridge.

	Reclamation Area			
	HF1	PTA	AOC83[a]	ATOP
Disposal volume				
- lcy	225000	160000	330000	220000
- bcy	160000	115000	230000	150000
Area (acres)	3.5	2.5	4.0	2.5
Direct Costs[b]				
Remove topsoil and trees	$8423			
Shape dumped material	8049			
Rough grade	1772			
Other	1000			
Revegetation				
Haul in topsoil[c]	$2861	$1128	$1348	$ 431
Finish grade	5492 (+700)	1344	5651	4178
Seed	1700 (+1000)	2400	3000	3084
Indirect costs				
Work roads and dumpsites[c]	0	0	9840	500
Excess haul time[c]	0	0	?	1400
Total revegetation	11752	4885	10000	7693
Total direct	31000	4885	10000	7693
Total indirect	0	0	9840	1900
Total	$31000	$4885	$19840	$9593

[a]: Costs for AOC area may be underestimated; these costs were recorded during the period of cost monitoring procedure developement. Lack of hauler load count data during this period prevented estimation of cost of excess haul time, relative to site average.

[b]: Costs in parentheses are estimates for work not yet completed.

[c]: Calculated as difference in cost between routine disposal and disposal in specified area.

Table 7. Per-acre and per-bcy reclamation costs at Amos Ridge by reclamation area.

Reclamation Cost	Reclamation Area HF1	PTA	AOC83	ATOP
Per-acre:				
Haul topsoil	$817	$ 451	$ 337	$ 172
Finish grade	1770	537	1412	1671
Seeding	770	965	750	1233
	----	----	----	----
Revegetation	$3357	$1950	$2500	$3076
Per-bcy:				
Revegetation	$0.07	$0.04	$0.04	$0.05
Work roads and dumpsites	0.00	0.00	0.04[a]	0.003
Total Reclamation	$0.19	$0.04	$0.086	$0.06

[a]: Application of 1984 cost solely to material disposed of in 1984 results in $0.09/bcy road and dumpsite maintenance cost.

Estimating the costs of highwall backfill reclamation was not as straightforward. There was poor volume control over the area mined in 1983 and returned to AOC (AOC83) since work was performed there before initiation of hauler load counting procedures and because a portion of the area reclaimed in 1984 was backfilled during 1983. However, an estimated 170,000 lcy were placed in the AOC83 area during 1984. A rough estimate of the amount of spoil placed the AOC83 reclamation area in 1983 would be 130,000 lcy. Reclamation cost was approximately $10,000. Thus, the direct costs of reclaiming can be estimated as approximately $0.04 per bcy disposed in the area. However, there are a number of indirect costs associated with AOC83 reclamation. One which can be quantified is the cost of working haulroads and dumpsites, which was disproportionately large for the amount of spoil moved to the AOC83 area during 1984 relative to patterns observed on the rest of the site, partially because this area was backfilled during the winter months. Considering the excess haul road and dumpsite maintenance time as an indirect cost of reclamation raises the estimated reclamation cost to approximately $.09 per bcy. Also, hauling costs in all mining blocks where significant amounts of spoil were disposed in the AOC83 area (1LM through 4LM, 1LS, 1UM) appear to equal or exceed the site average. However, lack of hauler load counts to specified areas during early 1984 prevent an approximate estimation the cost of excess hauling time.

The fourth reclamation area (ATOP) included the Upper Marker highwall backfill at 1UM, spoil disposal on the Upper Marker east of 1UM and 2UM to the 4LS and 6LS highwalls, and the Low Splint highwall backfill below. An estimated 220,000 lcy of spoil were disposed in the area. This is, again, a rough estimate of disposal volume. All spoil disposed on the Upper Marker seam was overburden from that seam. Approximately 60 vertical feet of the Low Splint Highwall required backfilling. Backfilled spoil was hauled from blocks 6LS, 7LS, 9LM, and 10LS; the latter two blocks provided the spoil for covering the upper portions of the highwall. The backfilled spoil was covered with topsoil from 5UM placed on the Upper Marker bench and graded down from above. The direct reclamation cost for this area is calculated at $7700, or $0.05 per bcy. Quantifiable indirect costs include the six hours spent by the Cat 988 loader at the dumpsite the day the top ramp was dumped full, and the incremental cost of haulage, given the average 7.3 loads per hour (lph) rate from 9LS and 10LM to 6LS and the 8.55 lph site average. Calculating the indirect cost as above adds $1900 to the direct cost total, bringing the per bcy disposed cost to $0.06.

In terms of differences among the areas, one obvious observation is that the highwall backfill areas experienced indirect costs while the HF1 and PTA areas did not. Increased hauling times to the highwall backfills reflect movement of material upwards from the point of origin to the disposal site, which is generally more time consuming than moving it downward or laterally. Increased haul road and dumpsite maintainance costs are also a function of mine site geometry. The dumpsites were smaller on the AOC backfills than upon the PTA and HF1 disposal sites, because the flat areas suitable for dumping on slopes were narrow, terrace like formations. Therefore, keeping them clear required more of a constant effort than the broader, more capacious dumpsites at HF1 and PTA; these larger areas could be dumped during the course of a day, then taken care of more efficiently, with one or two hours of constant effort by a loader or dozer towards the close of the day. Also, in general, haulroads leading to the highwall backfills are more critical and more difficult to keep clean. Because climbing steep grades with a full load requires near-optimum hauler performance, decline in road quality due to bad weather and other factors affects machine performance more strongly than upon flat surfaces. These other factors often include rocks rolling downslope from the dumpsites above, a problem that becomes more acute as the backfill grows steeper. Thus, the roads require more attention. Also, the longer hauls mean that, when hauler performance is affected loader performance is also affected. On short hauls, a slight hauler delay will not affect loader performance because, when haulers are working in tandem (the usual case), the returning hauler often has to wait for the second hauler to be loaded. Therefore, slight hauler delays do not affect overall job performance.

Another observation is that the most expensive spoil disposal occurred in HF1, primarily due to area preparation, drainage structure construction, and other operations required in this area that were not required in other reclamation areas.

In terms of revegetation costs, topsoil isolation expenses were greater, on a per-acre basis, for the near level areas than for the highwall backfills. Again, this can be explained generally by referring to site geometry. First of all, more care was taken with the areas being prepared for improved use, apparently because greater value was placed upon reclamation of these areas by the operator. Also, there was a greater opportunity to isolate surface materials upon these areas. The period when highwall backfills are available for surface material placement is brief, relative to the near-level areas, since the passage of time causes the earlier roads to be covered by roads required to get higher up on the slope. Also, as the backfill gets higher and material is dumped from higher levels, the previously dumped materials are covered. Thus, if topsoil were to be isolated and placed early in the construction on a highwall backfill, it would likely be buried with less favorable material. Finally, when grading does begin, haul roads are obliterated and there is no way to place subsequently isolated materials.

In terms of observations at Amos Ridge, these generalizations apply more directly to the AOC83 area than to the ATOP area, because, in the latter case, topsoil from the Upper Marker seam was stockpiled for a considerable period of time on the Upper Marker bench directly above the Low Splint highwall, as backfills were being constructed in 4LS and 6LS. However, the longer hauling distance to the AOC83 reclamation areas with topsoil from areas mined in 1984 (sometimes as far as 1500 feet, one way, upgrade) resulted in a greater topsoil haulage cost to the AOC area. HF1 topsoil isolation cost was greater than the PTA cost because of the short time available for topsoil placement, which required that more special measures be taken to isolate the material. The PTA surface was covered with a layer of topsoil 3 to 6 feet in depth over a six month period. Significant quantities of topsoil were hauled to all reclamation areas during the course of routine spoil disposal.

Grading costs showed a distinct pattern; it was approximately $1000 per acre more expensive to grade the sloping highwall backfill areas than the PTA area. A large portion of the expense of grading HF1 was due to the necessity to work drainage structures into the final reclaimed surface, including terraces and the blanket drain.

Seeding costs also show variation. However, that variation does not appear to be a function of site geometry. The greatest expense occurred in the ATOP area, portions of which had to be re-seeded because of dry weather.

IV.B. POST-MINING LANDFORM COST COMPARISONS

Simulating mining according to the assumptions discussed in section III.E resulted in the data of tables 8, 9, and 10. A detailed comparison of the two mining plans is contained in Appendix D.

The simulated coal production figures are inaccurate by a factor of approximately 13 % (actual strip production was 60,621 tons) indicating a problem with the coal thickness and/or density specifications (Table 8). However, this error has no bearing on simulation results. The fact that coal tonnages were equal in both cases indicates that the total areas of the blocks where mining was simulated was equal in both cases. The drilled volume and total volume estimates also indicate that the total volume of materials treated by simulated mining in each case was approximately equal to the actual estimated overburden volume. Agreement between these two sets of numbers is no coincidence: the data processed to produce these volume estimates was transferred directly from the COSTSUM output files by a series of auxiliary programs designed for that purpose. The 200 bcy disparity between the COSTSUM and OPSIM figures is a result of rounding errors.

The pushed and carried volume estimates produced by the OPSIM simulation of the LA landform total to an amount close to the COSTSUM combined estimate. This is as expected, since the LA mining plan was designed to emulate observed practices. As discussed earlier, the quantity of material pushed and carried in the AOC case was less than the quantity pushed and carried in the LA case, due to the unavailability of HF1 as a nearby disposal area; also, excess spoil disposal difficulties in the 2LS-4LS-6LM-7LM (Fig. 7) area were assumed to prevent some of the push and carry operations from these areas to adjacent blocks. As a result, a greater quantity of material was hauled in the AOC case. The cost per-bcy carried and pushed was less in the AOC case than the LA case because carrying and pushing to the hollow fill was more expensive, on average, than other carry and push operations. Often, the materials moved to the hollow fill consisted of large rocks and boulders suitable for core drain construction. These materials were more difficult to handle than

Table 8. Comparison of estimates of production at Amos Ridge: actual (COSTSUM - LA) vs. simulated (OPSIM), by postmining landform (LA vs. AOC).

Measure of Production	COSTSUM - LA	OPSIM - LA	OPSIM - AOC
	Estimation Method		
Coal production (tons)	70679[a]	52473	52473
Overburden (bcy):			
Drilled	723084	723257	723248
Pushed	102673	102687	86581
Carried	117597	117558	94222
Hauled	506414	506581	546027
Total[b]	726684	726858	726849
Overburden (lcy):			
Drilled	1105994	1105980	1105969
Pushed	157773	157569	133460
Carried	183634	182756	146414
Hauled	769587	770114	830570
Total[b]	1110494	1110484	1110472

[a]: Includes augered production; auger operation was not modeled with OPSIM.
[b]: Rounding errors prevent totals from equaling sum of pushed, carried, and hauled volumes.

Table 9. Comparison of estimates of spoil handling costs at Amos Ridge: actual (COSTSUM - LA) vs. simulated (OPSIM), by postmining landform (LA vs. AOC).

Cost Type	Estimation Method: COSTSUM - LA	OPSIM - LA	OPSIM - AOC
Total			
Drill	$149026	$150041	$149998
Carry and push	141659	140648	108531
Carry		48378	38757
Push		92270	69774
Load and haul	574184	483070	563370
Total cost	$864869	$773765	$821904
Per-bcy			
Drill	$0.206	$0.207	$0.207
Carry and push	0.643	0.639	0.598
Push		0.471	0.447
Carry		0.785	0.735
Load and haul	1.134	0.954	1.034
Total	$1.190	$1.065	$1.131
Per-lcy			
Drilled	$0.135	$0.136	$0.136
Carry and push	0.416	0.413	0.386
Carry		0.307	0.290
Push		0.505	0.472
Load and haul	0.746	0.627	0.679
Total	$0.779	$0.697	$0.740

Table 10. OPSIM-simulated spoil handling costs, with volumes hauled and per-bcy hauling costs by mining block and post-mining landform.

Block	Spoil handling cost[a]		Volume hauled (bcy)		Per-bcy haul cost	
	(LA)	(AOC)	(LA)	(AOC)	(LA)	(AOC)
1LM	$21935	$21824	24231	24232	$ 0.79	$ 0.79
2LM	20030	21154	10978	16670	1.12	1.18
3LM	25592	23164	19077	21689	1.06	0.97
4LM	30547	33426	22597	24430	1.05	1.17
5LM	6173	6173	5709	5710	0.59	0.59
6LM	31913	40091	30418	37300	0.89	1.07
7LM	37166	43369	35587	37444	0.95	1.13
8LM	3844	5603	3881	7161	0.61	0.78
9LM	36752	49449	46602	49496	0.76	1.00
10LM	31089	31160	29409	29410	0.96	0.97
1LS	24828	25142	17361	18103	1.40	1.39
2LS	50800	48514	18115	28121	1.14	1.03
3LS	20678	24442	20001	23101	0.96	1.06
4LS	35675	34983	16969	16977	0.88	0.83
5LS	9777	11346	10229	10296	0.78	0.92
6LS	65967	68192	35762	35784	1.30	1.36
7LS	35152	36881	24756	25426	0.91	0.97
9LS	34728	37132	30430	30439	1.01	1.09
1UM	18282	18239	11989	11757	0.97	0.97
2UM	21247	20929	26086	26086	0.80	0.79
3UM	21722	21537	23286	23286	0.79	0.78
4UM	35416	41444	36332	36333	0.95	1.12
5UM	6099	7501	6776	6777	0.79	0.99

[a]: Total cost of dozer push, loader carry, and load and haul operations.

smaller sized fragments and they were often moved by carry and push operations to HF1 over longer than otherwise routine distances. In the AOC case, it was assumed that these materials would be handled routinely. Thus, the major cost increase of implementing the AOC mining plan, relative to the LA plan, is hauling.

Reasons for Hauling Cost Differences

There are two primary reasons for the large difference in hauling cost between the mining procedures used to construct the LA and AOC landforms.

Number one is the difference in quantities of material hauled. Since less material was able to be carried and pushed, more had to be hauled in order to construct the AOC landform. This effect was not concentrated within a few mining blocks, but occurred throughout the mining area (Table 10). The AOC mining plan caused in significant increases (>1000 lcy) in hauled volumes from 8 of 23 mining blocks.

Second is the difference in cost per bcy hauled. In an overall sense, it is easy to see the reason for this effect. The elimination of HF1 altered the entire spoil handling plan. The major difference between the two mining plans is that spoil which was disposed in HF1 in the LA case had to be used, for the most part, to reconstruct the mined point above blocks 1-6LM in the AOC case. Spoil which had been moved laterally and downward to HF1 would have been moved upward had the AOC mining plan been implemented. Per-bcy cost differences are most distinct for blocks where large quantities of material had to be hauled to the highwall backfill and rebuilt point areas. Again, this difference was fairly well distributed throughout the mining area, as significant per-bcy hauling cost increases (>$.05/bcy) were estimated for 12 of 23 mining blocks.

Adjustment of Spoil Handling Cost Differences

Primary influences upon the difference in hauling cost derived from modeling procedures are the assumptions made with regard to the AOC landform and the roads used to construct it. The costs of Tables 9 and 10 are based upon a particular set of assumptions.

Changing the haulroads in the rebuilt AOC point area resulted in a $4000 simulated mining cost reduction. This change consisted of eliminating some of the 8% grades with switchbacks in favor more direct but steeper haul routes. On-site observations indicate that these steeper routes are not favored by the operator due to increased wear and tear on equipment and the vulnerability of hauling efficiencies on such routes to bad weather, factors that were not modeled.

Also, it seems reasonable to assume that the operator might attempt to gain some cost savings by reducing spoil hauled to some of the surface fill cells at higher elevations when rebuilding the point above 1-6LM and east of 1UM. However, the degree to which such changes could be implemented would be restricted by the spoil disposal requirements of future blocks. At the completion of AOC simulation, the only disposal areas available were hollow fill HF2, and blocks 9LM-10LM along with the overlying blocks 5LS and 7LS. However, at this point, it was necessary to plan ahead for the future 8LS, a third-cut block to be mined from the point area bounded by 6LS, 7LS, and 9LS. The preferred method of mining such a block is to have a spoil disposal area available at its base to allow the majority of its material to be carried and pushed into the pit below (Appendix C). An additional block needed to be taken from below 9LS (11LM) before 8LS could be mined in this fashion. The upper lift of 11LM was non-durable rock and therefore unsuitable for placement in the basal portion of HF2 being constructed at the time; the only disposal area available adjacent to 8LS would be 9LM. However, this block was already over half filled. Therefore, it is not consistent with site realities to assume that the point backfill above 1-6LM could be reduced in volume to any significant extent. It is likely, however, that in an actual mining situation, the volume placed in the point backfill above and east of 1UM would have been reduced in order to reduce hauling distances from 4UM and 5UM. Thus, an additional assumption was that approximately $4000 (one half of the 4UM and 5UM hauling cost difference between the two landforms) could be saved in this fashion. Thus, the spoil handling cost difference between the two mining plans is estimated at $39,000 to $47,000.

The primary quantifiable costs not included in the simulation procedures were reclamation costs and permitting costs associated with HF1 (Table 11). The major cost adjustment is the result of HF1 not being constructed in the AOC case. Thus, all costs required to design, permit, and reconstruct HF1, aside from the cost of movement of material into the fill, were subtracted from the simulated cost difference.

However, this cost adjustment itself requires modification, due to the greater area of steep slopes and associated direct reclamation expenses on the AOC landform.

In terms of indirect costs of reclamation, differences in hauling costs due to the spoil placement requirements of the post-mining landforms were estimated by the modeling procedures. However, the costs of maintaining haul roads and dumpsites were not. An estimate of a reasonable range of road and dumpsite maintenance costs can be developed from the recorded time requirements of performing these operations in the AOC83 and ATOP reclamation areas (Tables 6 and 7).

The final adjustment is weather related hauling delays. This adjustment comes from the observation that the modeled hauling cost and the actual hauling cost differed by approximately $100,000, in spite of the hauler-simulator "calibration" procedure undertaken with site data (Appendix B). Visits to the site tended to coincide with fair weather. The data indicate that hauler efficiencies tend to drop drastically in damp, foggy weather (Appendix B, Table B-1). Such weather is not uncommon at elevations of between 2700 and 2900 feet, near the top of Amos Ridge. An attempt to determine the determine reasons for poor hauler performance over certain routes (as indicated by low lph rates relative to those estimated by OPSIM) by looking back at the daily data forms showed that poor weather conditions were frequently recorded simultaneous with low hauling rates. Another possible explanation for the cost difference would be loading difficuties do to poorly shot rock. Although such situations are assumed to be represented on average by the 25% overall delay factor (Appendix B, Table B-2), they also could have been underrepresented in the load-and-haul cycle observations.

Table 11. Adjustment of OPSIM-estimated cost difference between landform alternatives due to non-modeled factors, liberal and conservative assumptions.

Factor	Liberal Assumption	Conservative Assumption
Spoil handling cost difference (AOC > LA)	$47000	$39000
HF1 costs:		
Design (3 engineer work days plus one draftsman work day)	- 1000	- 1000
Bonding	- 200[a]	- 2000[b]
Construction and reclamation	- 31000	- 32000
Reclamation of mined area covered by HF1 (approx. 1.3 acres, $2500 per acre)	+ 3250	+ 3250
Additional grading costs, AOC slopes above PTA (approx. 2.5 acres, $1000 per acre)	+ 2500	+ 2500
Work road and dumpsites, AOC slopes above PTA (approx. 130000 bcy)	+ 5500	+ 400
Weather-related hauling delays	+ 15000	
Total	$41000	$10000
Total per ton	$0.58	$0.14

[a]: partipation in Virginia Surface Mine Reclamation Fund (bonding pool) at $1500 per acre, assuming 60% release after 3 years, 80% release after 5 years, full release after 8 years; $12.50/$1000/annum bonding fee.

[b]: standard performance bond, $15000 per acre, other assumptions as above.

The $15,000 estimate of Table 11 assumes the entire difference in cost between the OPSIM estimated hauling cost on the LA landform and the COSTSUM estimate to be the result of weather-related delays; in projecting the weather-related delay cost effect to the AOC mining plan, it was simply assumed to be proportionate to the total hauling cost. Thus, the fact that steep upward hauls of the AOC landform would likely be effected by bad weather to a greater degree than the slopes of the LA landform (partially because the LA landform moves less material upwards; partially because of greater spoil handling flexibility which allows steep hauls to be avoided on bad days) is not considered.

The results of this analysis indicate that the observed mining operation would have required greater expense, had the land been reclaimed to the AOC landform rather than the LA. The cost difference is estimated at $0.14 to $0.58 per lcy (Table 11). Thus, a logical conclusion is that, if the mining operation had been conducted in more conventional fashion, the economics of spoil handling would have restricted the mining to the more favorable cuts; coal that was mined in order to produce the LA landform likely would have been left in the ground.

V. DISCUSSION

Site Data

The efforts to develop a site data collection system for use at Amos Ridge were successful. The data forms developed for use at the site (Fig. 6) provided an effective and efficient means for recording data at the site on a daily basis. Mr. Hall estimates that 5 to 10 minutes of his time were required per day by the data recording procedures, while another 10 minutes per day were required of personnel at VPI & SU to enter the data from the forms into computer files for processing. Once the COSTSUM system was developed, analysis of these data were also performed efficiently; the detailed cost data presented here would not have been available without the analyses provided by COSTSUM.

The data collected, and on-site observations, indicate that spoil was handled in cost-effective fashion at Amos Ridge in spite of this operation's departure from conventional mining and reclamation practices. The $1.90 per bcy total spoil handling cost (Table 4) compares favorably with costs quoted privately by industry sources in spite of the fact that the landform being produced is capable of a greater variety of uses than a typical AOC landform.

The LA landform appeared to offer a number of cost advantages to the mining operator. Highwall backfilling was accomplished relatively easily in the ATOP area in comparison with the AOC83 area and and in comparison with AOC backfill construction procedures observed on auxiliary sites. The location of the near-level area at the base of the ATOP backfill reduced the ramping required to get to the top of the highwall and the small area to be covered gave the operation the flexibility to construct the backfill at the most convenient time.

With the exception of hollow fill HF1, reclamation requirements also appeared to have been eased by the LA landform. The least expensive reclamation area on the site was the near-level area extending from above block 5LM to the base of the 4LS highwall. The cost of reclaiming the ATOP area was less than the cost of reclaiming AOC83. The cost of constructing HF1 might have been reduced, had this not been the operator's first experience with hollow fill construction.

The case study site is representitive of a number of other mining sites in the central Appalachian region. The steeply-sloping points and hollows topography occurs commonly. The cost structure at Amos Ridge is not typical of all firms. However, many of the smaller firms operating primarily on a contract basis appear to have cost structures similar to that of the Amos Ridge Coal Co. Some firms use newer equipment which is not owned outright, maintain machine:operator ratios close to 1:1, and attempt to maximize machine use time. At such job sites, the zero idle time cost assumption used in this analysis would not be appropriate. However, some of the larger firms in the area maintain multiple job sites in close proximity, switching machinery from site to site daily. The cost structure of such firms would, in many respects, be like that observed at Amos Ridge. Therefore, it is likely that the results of analyses performed upon data gathered from Amos Ridge will apply in a general sense to a many other mining sites in the central Appalachian region.

Landform Cost Comparisons

The results of the modeling procedures confirm that the LA landform construction procedures are cost effective.

With respect to the modeling procedures, the obvious question is: how valid are they? In a general sense, they form a very rough approximation of what actually goes on at the site. Steep slope surface mining is a difficult process to model. There are a wide variety of site conditions that influence mining operations and decisions on a day-to-day basis. Reduction of these operations to assumptions and equations do not do them justice. With the number of assumptions required for this analysis, some of them must have missed the base.

On the other hand, the modeling procedures were firmly grounded upon site data; the OPSIM-simulated LA hauling costs were 84 % of the recorded hauling costs while the detailed input specifications allowed the combined carry-and-push cost estimate to come within 1%. Industry sources indicate that any mining cost estimate within 90 % of actual cost is considered to be exceptionally accurate. Also, the results conform to to common sense: it seems likely that it would be more expensive to move 250,000 lcy upwards, into the 1-6LM point backfill and the 6LS highwall backfill, than to move them laterally and downward into HF1. The increased cost effect of altered spoil handling procedures seemed to pervade the entire operation; they were not confined to a few areas where large cost changes are estimated.

In evaluating these cost effects, another logical question is: if it is true that the mining firm can produce such a landform at reduced cost, given that the mining conditions at Amos Ridge appear to be representitive of more-or-less typical conditions throughout the region, why are these procedures not widespread? The law would not have prohibited the procedures used at Amos Ridge in the absence of an experimental paractice permit. A variance from its AOC requirements may be obtained if the land is being prepared for an improved land use, such as the agricultural use planned for the site at Amos Ridge. Hollow fill HF1 was produced using procedures prescribed for construction of durable rock fills by section V816.74 of the Virginia code. The only experimental procedures used at the site are the construction techniques used in hollow fills 2 and 3, which had no bearing upon the cost comparison reported here.

The direct implication of the results reported here are that, given identical mining volumes, the LA landform could be produced at less cost than the AOC landform. However, these results do not translate directly into a potential mining cost savings since the operator has the potential to control stripping ratio and, thus, level of profit. Therefore, the primary benefits of the potential cost alteration would likely fall to the land owner, whose costs are approximately the same regardless of mining strategy but whose profits are greater in the LA case due to potentially increased coal recovery.

An analysis of tne non-quantified effects of producing the LA landform, rather than implementing conventional AOC mining and reclamation practices, show that the LA landform appears to offer additional benfits to the mining operator, the land owner, and residents of surrounding areas (Table 12).

Non-Quantified Effects

From the point of view of the operator, the major non-quantified advantage to the AOC case is that it is the conventional practice. Thus, mining to reconstruct the AOC landform requires less engagement with unfamiliar (for some operators) hollow fill construction techniques and, perhaps more importantly, invites less regulatory scrutiny.

However, the LA case also appears to offer a number of advantages to the operator including increased operational flexibility. The near-level areas are used for equipment storage and maintenance during the course of mining; thus, equipment can be maintained and parked in a contiguous area proximate to the working areas of the mine; no provision was made for maintenance of such an area during the AOC simulation procedures. Also, due to the opportunity to haul, carry, and push laterally into the first hollow fill, it gives greater opportunity to avoid steep uphill hauls on bad weather days, when slick roads reduce hauler efficiency. In addition, construction of the LA landform will require less wear and tear on machinery due to the reduction in steep uphill hauling and steep slope grading. Finally, there is a safety factor to consider, due to the inherent danger of operating large machines at the top of steeply sloping unconsolidated spoil banks, a situation which occurs less frequently during construction of the LA landform.

From the point of view of the landowner, there are also advantages to the LA case. A primary advantage will be the improved use potentials of reclaimed areas. In the immediate future, the near-level areas and hollow fill faces at Amos Ridge will be used as pasture and hayland. Its soils will be productive, relative to other reclaimed and natural pastured areas in the central Appalachian coal mining region. In the long term, a variety of other uses will be possible due to favorable topography. Although it is doubtful that this particular site would be re-mined using surface methods, the ease of re-mining a site reclaimed to a topography similar to the LA case,

Table 12. Non-quantified benefits of LA and AOC postmining landforms, by concerned party.

Landform	Concerned Party	Benefits
LA	Mining firm	Greater operational flexibility - area for equipment maintainance - able to avoid steep hauls on bad days - able to isolate favorable surface materials conveniently Reduced employee occupational danger Less wear and tear on equipment
	Land owner	Improved post-mining land use potential Less disturbance required to re-mine Greater coal recovery
	Regulatory agency, general public, and above	More favorable environmental impact - hydrologic - stability - vegetative production - reduced chance of re-mining
AOC	Mining firm	Smaller perceived risk in conventional practice
	All	Smaller acreage directly disturbed by mining

relative to the AOC case where far more spoil would have to be moved in order to expose the coal seam, would be a definite advantage to a landowner with a continued interest in mineral exploitation and substantial remaining mineral deposits. However, the presumed ability of the LA mining procedures to recover a greater quantity of coal economically, and thus reduce the possibility that re-mining would be required, is a greater advantage.

From the standpoint of society, there also appear to be advantages to the LA case. Primary among these would be a more favorable environmental impact. The potential for construction of unstable highwall backfills is minimized through use of the LA technique, since highwall backfills are themselves minimized. In the event of a failure, the existence of the near level area constructed below the backfill would localize the adverse impacts. Although it is also possible that the outslope of the near-level terrace above the mined point could be constructed unstably, the consequences of the resultant slope failure would be limited due to the reduced vertical height of the terrace outslope relative to an AOC backfill on the same site. It also seems that the LA case would have more favorable hydrologic consequences. The steep reconstructed slopes of the AOC landform are interrupted only by the access road and narrow terraces. In contrast, the broad terrace of the LA landform will have the effect of dispersing, rather than concentrating, water moving downslope from the terrain above, and it should be far more effective at reducing the reclaimed landscape's sediment yield particularly if deep, productive soils are established. In a broader sense, it seems that the LA landform will also have the effect of redistributing downward moving water in time, since some would be temporarily stored in the soils and spoils on the near-level area, reducing storm related peak flows which cause flooding in steeply sloping areas. Again, this effect would be enhanced if deep, uncompacted surface soils were to be constructed at the site. An offsetting factor could be the smaller area directly disturbed by the mining/backfilling procedures of the AOC landform.

In addition, widespread use of landform alteration techniques will have a more favorable economic impact as well, by producing lands capable of supporting improved land uses. In the future, the widespread availablility of such landforms would enhance the area's potential to support a more diverse industrial, commercial, and residential development pattern.

Another non-quantified factor to consider is the possibility that the price of coal might change during the course of mining. In one sense, the land owner and the operator have a common interest relative to coal prices, in that high prices stimulate production and bring greater profits to both parties. However, the operator will be affected to a greater extent by changes in price due to the necessity to plan mining cuts well in advance, so as to be able to coordinate stripping ratios and spoil movement activities. The post-mining landform will affect the operator's ability to respond to changes in price by adjusting stripping ratios. Either landform should allow a response to small changes in price. In the LA case, the final near-level surface could be adjusted upwards or downwards in order to absorb variations in stripping ratios and, thus, spoil quantities; in the AOC case, the final surfaces of the excess spoil disposal areas in the first and second hollows could also be allowed to vary slightly from initially planned elevations without having a major affect upon operations.

However, large changes in price will be a different matter. A large price increase may be accomodated more easily by the LA landform; there is no physical reason why additional cuts could not have been taken from the points and the near-level surface elevation raised; in contrast, the near spoil-bound conditions that would have been encountered during AOC mining would also have prevented the taking of additional cuts. However, a large price decrease would have an opposite effect. Once a hollow fill is initiated, the operator has made a substantial commitment to mining in future months. Initiation of hollow filling activities commits the operator to continue mining over the period required to complete the fill (approximately one year at Amos Ridge) and to take stripping ratios close to planned ratios during that time. Completion of the fill will require that substantial spoil volumes be produced; significant departures from planned stripping ratios will increase average spoil movement distances and costs. Generally, the operator has posted a substantial performance bond to ensure the completion of reclamation activities, including hollow fill construction. Thus, the early stage of hollow fill construction is a time when the mining operation is very sensitive to coal price. In contrast, AOC mining does not produce such long-term commitments: hollow fills are minimized and backfilling generally follows mining more closely. Thus, stripping ratio can be more responsive to price and the operation can be shut down on shorter notice, if necessary.

Thus, it appears that non-quantified effects also favor mining procedures such as those used at the Amos Ridge site. However, it also appears that the case for the LA landform is much more clear cut from the point of view of the land owner than from the point of view of the mining operator, particularly in situations where stripping ratio is easily controlled. From the operator's perspective, two unpredictable factors with large potential effects upon profitablity are regulatory enforcement and coal price; the potential for both of these effects to decrease profits is enhanced by the LA landform.

The results of this analysis indicate that the use of landform alteration mining and reclamation techniques may allow coal to be produced more economically without compromising the environmental concerns of SMCRA. Yet, these procedures are not being implemented in any widespread fashion, at present. The reasons for this lack of implementation in spite of favorable economics may lie in the institutional context of mining and reclamation in the central Appalachian region. There are many factors which influence the behavior of individuals responsible for decisions affecting mining and reclamation activities. Possible reasons discussed below are based upon the obsevations of the author, rather than empirical measurements or modeling procedures. However, if correct, they indicate that the reasons for lack of implementation could be easily overcome by a sustained, cooperative effort including industry and regulatory personnel.

Institutional Constraints to Improved Reclamation.

The general context provided by the current law is not the major constraining factor to improved reclamation. As discussed above, the mining procedures at Amos Ridge are, in large part, being performed according to current laws and regulations. The results and analyses reported above indicate that situations exist where it would be in the interest of the mining industry to produce stable, productive landscapes, given the commonly practiced AOC option as an alternative. However, in the absence of law, it was in the interest of individual mining firms to push the majority of mined spoil over the outslope. Development of effective laws and regulations is an evolutionary process, and it is only with experience of the effects of SMCRA that possibilities for improvement of the law and implementation regulations have become apparent.

The law does provide a means for obtaining a variance from the AOC provisions in cases where a higher land use is planned than that which precedes mining. However, this provision is seldom taken advantage of. In order to obtain such a variance, documentation must submitted with the initial permit application, since the permit defines the post-mining land use. Depending upon the land use, such documentation may be quite costly. The applicant is required to submit materials substantiating stated intentions and an ability to carry out that proposed land use, as well as design specifications. The cost of producing this documentation must be borne previous to the permit application. The permit application generally precedes the onset of mining by six months to two years, and the mining process itself may take a few years to complete. There is often an additional time lag between reclamation and land use implementation; thus, many years separate the initial application from the financial returns of the improved land use. Given the time value of money, and assuming that the land owner/permit applicant expects the profits from the post mining land use to pay for the permitting and design costs, this time lag, in effect, handicaps the potential for profitability. The permittee faces another penalty, if the procedures required to prepare the land for its post mining use is a more expensive procedure than that which is commonly practiced, since the amount of performance bond must be sufficient to cover reclamation to the permittees declared postmining use.

The uncertainty factor also tends to inhibit improved land use permit applications. Unforeseen fluctuations in the price of coal often cause mining firms to abandon mining plans after they've been permitted but before mining begins. Similarly, unforeseen fluctuations in demand for the product of the proposed land use may alter the potential profitability of the investment. The net result of these factors is that mining firms have a number of reasons for avoiding the expense of improved post mining land use planning, design, and documentation with the initial permit application, if that expense is of significant magnitude. Thus, it may be that the procedures for obtaining variances from the law's AOC provisions, rather than the AOC provisions themselves, could be most profitably addressed in order to stimulate production of more favorable post-mining landforms. Perhaps the situation would be improved by allowing the permit applicant to document the intention and a proposed method for preparing a reclaimed lancform with increased land use potential and more favorable environmental impact, relative to the AOC alternative, as a condition for receiving an AOC variance. Such a provision could give the landowner greater flexibility by allowing decisions regarding post-mining land use implementation to be deferred while continuing to require that all environmental performance standards be met.

The structure of the coal industry in the Appalachian region also tends to inhibit the development of improved uses on reclaimed lands. Generally, the mining firm is not the owner of the land being mined. Thus, the firm will not benefit financially from reclamation practices which go beyond the minimum standards. It may be that there are no established precedents for contractual arrangements between operators and land owners, to require that specified reclamation practices be carried out in exchange for (presumably) a share of the expected profits or benefits of the improved reclamation practice. Thus, the development and/or publication of legal documents containing such clauses would assist landowners with a desire for altered post-mining landforms to establish the necessary contractual arrangements without the expense of paying a law firm to develop precedent.

The financial condition of the coal industry adds rigidity to the status quo. There is a risk to embarking upon unfamiliar mining methods. Given the current low price of coal and the unfavorable financial condition of many firms, additional risk is not of interest to those firms. However, there are situations where it appears that landform alteration mining methods may be implemented at a cost savings and/or a profit increase. Accurate, time-efficient, reliable cost estimating methods can help a firm to recognize such situations. Thus, a firm that invests in the development and/or use of accurate cost assessment and cost estimating techniques will gain an increased ability to recognize opportunities to profit through altering landforms. During the course of this research, it was observed that the majority of firms had a very poor mechanisms for determining the cost of operating their own machinery.

The owners of mined lands in the Appalachian region are largely corporate. Since many of the primary land owning corporations are primarily in the mineral business, they may have little experience with land development for purposes other than mineral extraction and are often unwilling to develop any surface use which will conflict with future deep mining. Given today's technologies, the presence of housing, for example, on the surface would preclude deep mining of underlying coal due to the potential for subsidence, structural damage, and liability suits. Land owners may identify surface coal deposits which are not underlain by deep reserves for special attention in their long-term planning procedures. The sale of these lands for non-mineral use after mining will remove the burden of managing non-mineral enterprises from the mineral land owner. The establishment of long-term contracts for the potential products of extensive post-mining land uses, such as managed timber or beef cattle production, can also serve to remove the risk of engagement in unfamiliar business practices from the land owning company. Long term contracts and extensive use will only be feasible in areas where large acreages of favorable landforms can be produced.

Another factor to consider is the mining industry's perception of being overregulated, which makes individual firms hesitant to attract regulatory scrutiny through variance application. If the regulatory agencies were to encourage such applications in favorable areas, it would help to remove the current stigma from variance applications. If additional firms such as Amos Ridge Coal Co. implement simililar mining plans, the use of landform alteration techniques to prepare mined lands for improved use would no longer be an unconventional practice. It may be in the interest of regulatory agencies to encourage landform alteration practices, given the unstable tendencies of AOC backfills in steeply sloping topography. Economic development would be enhanced if one area were to be identified where a number of proximate favorable landforms could be produced, particularly if near-surface deposits were not underlain by deep reserves.

Finally, there is the effect of technological unknowns which limit landowner's abilities to forecast profits from and/or implement improved mined land use opportunities. What is the site index of mine soils constructed using controlled overburden placement techniques for southern pine timber production? How does one establish a functional and legally acceptable septic drainfield on a mine soil site? What will be the effect of deep mining on future groundwater quantity and quality in the Appalachian coalfields? The continued exchange of information and cooperation between the mining industry and the research community will facilitate the development of improved reclamation and uses on coal surface mined lands.

VI. CONCLUSIONS

The purpose of this study was to estimate the cost of improved mined land reclamation in southwestern Virginia. In order to meet this purpose, a number of objectives were established. These objectives were:

1. To study active haulback mining operations so as to determine the costs associated with spoil handling and reclamation activities.

2. To develop computerized methods capable of estimating spoil handling and reclamation costs in steep slope topography.

3. To use those methods to estimate the cost of reclaiming surface mined land is southwestern Virginia in a fashion that will favor improved land uses.

4. To determine if cost is, in fact, the major factor preventing implementation of improved reclamation practices.

The improved reclamation practice chosen for study was the production of a stable post-mining landform capable of agricultural use in an area of steeply sloping pre-mining topography. This practice was studied through observation and data recording at a case-study site in southwestern Virginia where the practice was being implemented on an experimental basis. The data were collected and analyzed using methods developed specifically for use in this study. The results of analyses of data from the case study site were used for a simulation study of mining cost, designed to estimate the cost of the reclamation practice under study through an overall mining cost comparison using conventional AOC practice as an alternative. The conclusions of this research are:

1. It is possible to perform contour mining operations in steeply sloping Appalachian regions in a cost effective, environmentally beneficial manner. A landform with extensive near-level areas capable

of supporting an improved land use was produced while mining coal profitably. This landform was produced at per-bcy spoil handling costs close to those experienced by other mining operations using more conventional but less environmentally favorable techniques. The near-level areas of this landform were covered with soils that will likely prove to be more productive, due to increased depth, than the majority of the natural soils of this sloping terrain. The future stability of this landform should prove superior to landforms produced during more conventional (AOC) mining procedures. Stability is enhanced by the 3:1 slope of the face of the hollow fill and the broad terrace-like area extending over the top of the stripped points and filled hollows below the majority of the backfilled highwalls.

2. The efforts to develop a site data collection and analysis system were successful. Once daily forms and COSTSUM were developed and in use, data was collected on a daily basis and analyzed in time-efficient fashion. The results of these procedures have provided data on the cost of mining and reclamation at the Amos Ridge site with greater detail than would have been available otherwise.

3. The OPSIM spoil handling cost estimating model appears to predict mining costs with reasonable accuracy. It is capable of accepting inputs sufficient to reproduce the observed costs of drilling, dozer push, and loader carry operations on a block-by-block basis. Hauling cost was estimated at 84 % of actual hauling cost, in a first attempt. This accuracy could likely be enhanced by additional hauler observations, load-and-haul cycle time data collection, and model calibration procedures.

4. The estimated cost of mining so as to produce a landform capable of improved use is less than the estimated cost of a conventional AOC mining plan, assuming identical cuts would be taken in each case. This indicates that, had the site been mined in more conventional fashion, fewer cuts would have been taken and coal recovery would not have been as great. Thus, the environmental disturbance caused by the mining operation would not have produced the associated benefit of maximum

economical coal recovery, and the chances of future environmental disturbance (re-mining) of the reclaimed site would be enhanced.

5. The primary benefits of using the surface mining process to produce landforms with increased stability, soil productivity, and land use potential, relative to conventional AOC post-mining landforms, will accrue primarily to land owners rather than minimg operators. Therefore, it would benefit the owners of coal bearing lands to take the initiative in improving conventional reclamation practices.

6. Since, apparently, it is possible for coal surface mining operations to produce near-level lands capable of improved use in a landscape where level land is scarce at a possible cost savings, there exists the potential for coal surface mining operations to have a large future impact upon local land uses. However, in contrast to past impacts (and, possibly, future impacts of present practices, given potential AOC backfill stability problems) that impact may be of positive nature, from both an environmental and an economic perspective.

7. Although mining at this site was exempted from certain provisions of SMCRA as an experimental practice, that exemption had only a minor influence upon the permitting, mining, and reclamation requirements within the study area. The pre-mining topography at Amos Ridge is typical of other mined landforms in the area, yet mining procedures such as those demonstrated at Amos Ridge have not been used previously in Virginia and are infrequently used (if at all) in neighboring states. Since these procedures appear to be economically advantageous, there are apparently some non-economic factors acting to restrict mining firms from engaging in these procedures. An attempt to identify these "institutional constraints" to improved surface mine reclamation in the central Appalachian coal region indicates that they could be easily overcome by a sustained, cooperative effort between industry and regulatory agencies.

VII. SUMMARY

The Appalachian coal mining region is subject to a number of environmental and economic problems; many are a result of the steeply sloping topography. The extensive surface mining activities in the area appear to offer the opportunity to produce more favorable landforms at minimal marginal costs. Yet, despite this apparent opportunity and the success of research efforts to develop improved mine soil construction and revegetation techniques, the majority of the mining and reclamation activities in the Virginia coal region are carried out using conventional methods: reconstructing steeply sloping mining areas to their approximate original contours.

The purpose of this research was to estimate the costs of coal surface mine reclamation methods designed to prepare mined lands for improved use in areas of steeply sloping topography. The practice chosen for study was the production of a post-mining landform containing extensive near-level areas and surface soils suitable for agricultural use. Since production of this landform required an alteration of the mining process, a computer simulation approach was chosen. Due to the fact that the most widely used mining and reclamation methods return sloping lands to their approximate original contours (AOC), the cost of the practice studied was estimated as the change in overall mining cost resulting from implementation of the practice under study rather than conventional AOC techniques.

In order to carry out this research, a computer-based mining and reclamation cost estimating system was developed. COSTSUM is a set of seven programs designed to analyze data from active surface mining sites, to determine spoil handling and reclamation costs. OPSIM is a surface mining simulator designed to estimate the differences in spoil handling costs among reclamation and postmining landform alternatives. These techniques were applied to a case study, a surface mine at Amos Ridge in Wise County, Virginia, where the improved reclamation techniques under study were put into practice.

The pre-mining landform at the case study site consisted of a series of finger ridges protruding from a central "spine", Amos Ridge. Nearly all of the land being mined had slopes in excess of 20°; this type of topography is common throughout the region. During the course of mining, three hollow fills are being constructed so that their upper surfaces are contiguous with flat areas on the tops of the finger ridges, which are not being returned to their approximate original contours. The final landform will include a broad, near-level bench extending over the stripped fingers and filled hollows; an agricultural land use is planned at the completion of mining. During the period of study, the first hollow fill was completed, associated areas were mined and reclaimed, and the second hollw fill was initiated.

During this study, data on machinery operation at the case-study site was recorded on a daily basis and analyzed using the COSTSUM programs, which were developed for use in this study. The average cost of spoil handling at Amos Ridge between 1 January 1984 and 1 August 1985 was estimated to be $1.90 per bank cubic yard. However, spoil handling costs varied widely between blocks. The detailed output of the COSTSUM programs allowed identification of high cost components and low cost components of spoil handling within each block, and thus the reasons for the sharp variations in spoil handling costs among the various mining blocks.

The data collected at the site formed a basis for simulation procedures using the OPSIM model, which was also developed for use in this study. The first step of simulation was to duplicate the actual mining plan as closely as possible. Completion of this step showed that the OPSIM program was able to simulate the drilling, dozer push, and loader carry operations so as to produce costs nearly identical the COSTSUM cost figures. The OPSIM-estimated hauling cost was 84% of the COSTSUM figure.

The second step was to simulate mining of the area using conventional AOC techniques, without varying the input variables defining spoil movement rates unless such changes were clearly dictated by the change in mining plan. The simulated drilling cost did not change, due the assumption that identical areas would be mined in both cases. The total cost of dozer push and loader carry operations were estimated to decline with implementation of the AOC mining plan, since smaller quantities of spoil could be pushed or carried to disposal in areas adjacent to the mining block of origin. However, the hauling cost was judged to increase with implementation of the AOC mining plan, due to the increase in the quantity of material hauled and the increase in per-bank-cubic-yard hauling cost. The simulated cost of the AOC mining and reclamation regime was $47,000 greater than the simulated cost of observed practices.

The third step of the cost comparison procedures was to manually adjust the simulated cost difference to compensate for non-modeled factors. This procedure was carried out using both liberal and conservative assumptions; the estimated affect of post-mining landform on mining cost is between $0.14 and $0.58 per ton of coal produced; apparently, it is less expensive to mine using the improved reclamation techniques under study than conventional AOC techniques. Also, the techniques under study appear to offer a number of non-quantified benefits to mining operators, land owners, and residents of local communities.

This study concludes that it is possible to perform economically and environmentally sound mining and reclamation procedures in steeply sloping topography, in spite of departure from conventional AOC practices. The efforts to develop site data collection and analysis procedures, and surface mine operation modeling procedures, appear to have been successful. Had mining at the Amos Ridge site been performed using conventional practices, it is likely that less coal would have been recovered due to the increased costs of spoil disposal, relative to the observed practice. Since the pre-mining landform at the case study site is similar to many others in the area, it is possible that additional opportunites for improved coal surface mining exist in the steeply-sloping central Appalachian coal mining region. Many of the factors preventing widespread implementation of these improved reclamation technologies could be eliminated through cooperation between the various firms of the industry and regulatory agencies.

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Appendix A. OPSIM input variables and simulation values.

Table A-1: Selected input variables to program OPSIM, with quantities represented and/or simulation strategies. Legend follows table.

Variable name/type	Value/ strategy	Quantity represented
OVERALL JOB:		
MRED(I)	1,1,2,1,2,1	MAXIMUM NUMBER OF MACHINES TO BE USED IN ANY ONE LIFT SIMULTANEOUSLY. I=1: DRILL 2: DOZER 3: O. TRUCK 4: O. LOADER 5: C. TRUCK 6: C. LOADER
SEQCL(I)	v	BACKFILL SEQUENCE; I = 1,NFILCL
FILBRK(I)	v	BREAKS IN BACKFILLING SEQUENCE; FINAL CELL IN A SUB-SEQUENCE; SHIFTS SPOIL DISPOSAL SEARCH TO EXCESS SPOIL STORAGE AREAS; ENTER 0 IN UNUSED LOCATIONS; i = 1,9
SEQBB(I)	v	STRIPPING SEQUENCE; I = 1,NBLOCK
NBREAK	n/a	NUMBER OF 'BREAK POINTS', STRIP BLOCKS WHOSE COMPLETION SIGNALS PROGRAM TO MOVE TEMPORARY STORAGE SPOIL TO BACKFILL CELLS.
NCEL(I)	n/a	BREAK POINT SEQUENCE; I = 1,NBREAK
NBLOCK	31	NUMBER OF MINING BLOCKS.
NFILCL	85	NUMBER OF BACKFILL CELLS.
NHOLLW	v	NUMBER OF EXCESS SPOIL DISPOSAL AREAS.
NEX	n/a	NUMBER OF TEMPORARY SPOIL STORAGE AREAS.
CTIM	n/a	SHIFT TIME (MIN.).
CDEN	n/a	BANK DENSITY OF COAL (TONS/CU.FT.).
NSDY	n/a	NUMBER OF SHIFTS PER DAY.

Table A1: Continued.

Variable name/type	Value/ strategy	Quantitiy represented
NDMH	n/a	NUMBER OF WORKING DAYS PER MONTH.
CREVR	n/a	PROPORTION OF COAL RECOVERABLE.
ROCK TYPES:		
NROCK	3	NUMBER OF TYPES OF ROCK.
PFAC(I)	n/a	POWDER FACTOR OF ITH TYPE OF ROCK (LB/BCY)
DENS(I)	1.05, 1.16,[a] 1.17	LOOSE DENSITY OF ITH TYPE ROCK (TONS/LCY)
SWELF(I)	1.25, 1.45,[a] 1.65	SWELL FACTOR, ROCK TYPE I (E.G. 1.3)
DRILL:		
NPDR	1	NUMBER OF DRILLS
DB	10.5[c]	AVERAGE BURDEN (FT.)
DS	10.5[c]	AVERAGE SPACING (FT.)
TMOVE	0	AVERAGE TIME FOR DRILL TO MOVE FROM ONE HOLE TO ANOTHER (MIN).
DOZER: (CAT D9G)		
NPBL	2	NUMBER OF DOZERS
TRATE1	91.7	TRAVELING RATE FOR LOADED DOZER (FT/MIN).
TRATE2	91.7	TRAVELLING RATE FOR EMPTY DOZER (FT/MIN).

Table A1: Continued.

Variable name/type	Value/ strategy	Quantitiy represented
LTIME	0	DOZER LOADING TIME (MIN).
DTIME	0	DOZER DUMPING TIME (MIN).
CONST	0	TRAM CONSTANT (MIN).
CAPB	10.0	BUCKET CAPACITY (CU.YD).
COAL TRUCKS:		
NC1	n/a	NUMBER OF COAL TRUCKS.
HCTR	n/a	TOTAL HAUL TIME PER LOAD, COAL TRUCK (MIN.)
GVC	n/a	WEIGHT OF EMPTY COAL TRUCK(TONS)
TCTR	n/a	CAPACITY OF COAL TRUCK (TONS)
OVERBURDEN TRUCKS: (TEREX 33-09)		
N1	2	NUMBER OF OVERBURDEN TRUCKS.
GVW	46.6[b]	WEIGHT OF EMPTY OVERBURDEN TRUCK (TONS).
TOTR	33.[b]	CAPACITY OF OVERBURDEN TRUCK (CU.YD.)
TONTR	55.[b]	RATED TONNAGE CAPACITY OF OVERBURDEN TRUCK
TEX	0.33[e]	MANEUVERING TIME FOR TRUCK AT LOADING POINT (MINUTE).
DMP	0.85[e]	MANEUVERING TIME AT DUMPING POINT (MINUTE).

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
NGEAR	6	NUMBER OF GEARS IN OVERBURDEN TRUCK
NPPG	6	NUMBER OF (RIMPULL,SPEED) POINTS TO BE USED FOR INPUT OF HAULER POWER CURVE
SHFTIM	0.0[b]	TIME REQUIRED FOR SHIFTING BETWEEN GEARS
TIMINC	3.0	MAXIMUM TIME INCREMENT TO BE USED WHEN SIMULATING HAULER PERFORMANCE; MINIMUM TIME INCREMENT = 0.01*TIMINC (SECONDS)
DA	5.0[a]	DEACCELERATION WHEN BRAKE USED (FEET/SECOND SQUARE)
RORI	5.0[a]	AVERAGE ROLLING RESISTANCE (%)
DGR	15.0	MAXIMUM DOWNHILL GRADE (PERCENT)
AMAX	10.0	MAXIMUM ACCELERATION (FEET/SECOND SQUARED)
VMAX	38.0	MAXIMUM VELOCITY (M.P.H)
DHILL(I,J)	i[a]	MAXIMUM DOWNHILL SPEED AT GRADE J (M.P.H.) I = 1: FULL I = 2: EMPTY; RETURN TRIP
SPEED(J)	i[b]	SPEED LIMIT AT JTH GEAR (M.P.H.)
RP(J)	i[b]	CORRESPONDING RIMPULL (LBS).
VMCF(J)	i[a]	VEHICLE MASS CORRECTION FACTOR, GEAR J (CORRECTION FACTOR FOR ROTATIONAL INERTIA, AS IN VEHSIM MANUAL
OPFAC	0.45[e]	FACTOR USED FOR ADJUSTMENT OF HAUL TIMES - TO ACCOUNT FOR NON-OPTIMUM SITE CONDITIONS FACTOR LESS THAN 1.00 INCREASES SIMULATED HAUL TIME PROPORTIONATELY

Table A1: Continued.

Variable name/type	Value/ strategy	Quantitiy represented
COAL LOADER:		
NC2	n/a	NUMBER OF LOADERS (COAL).
CCPL	n/a	BUCKET CAPACITY OF COAL LOADER (TONS)
CSV	n/a	COAL LOADER CYCLE TIME (MINUTES)
OVERBURDEN LOADER (CAT 992B)		
N2	2	NUMBER OF LOADERS
CAPL	10.0	BUCKET CAPACITY OF OVERBURDEN LOADER (LCY)
NBUCK	3	NUMBER OF BUCKETS PER HAULER LOAD
AVSPD	390.0	AVERAGE SPEED FOR OVERBURDEN LOADER, LOADER CARRY OPERATION (FT/MIN).
PSV	1.05[e]	OVERBURDEN LOADER CYCLE TIME (MINUTES)
BLOCK: (I = 1,NBLOCK)		
NLIFT(I)	v	NUMBER OF LIFTS OF ITH BLOCK
CAREA(I)	n/a	AREA OF COAL OF BLOCK I (SQ.FT.)
CTIK(I)	n/a	COAL THICKNESS OF BLOCK I (FEET)
PSBLK(I)	v	BLOCK NUMBER THAT TO BE FINISHED BEFORE WORK ON I BLOCK CAN START.
PSLFT(I)	v	LIFT NUMBER THAT HAS TO BE FINISHED BEFORE WORK ON I BLOCK CAN START. (THIS LIFT IS IN PSBLK(I))

Table A1. Continued.

Variable name/type	Value/ strategy	Quantity represented
BPREP(I)	n/a	PREPARATION TIME FOR BLOCK I (HR.), DOZER
TPREP(I)	n/a	TIME TO PREPARE BLOCK I FOR COAL LOADING (HR.), COAL LOADER
LIFT: (I = 1,NBLOCK; J = 1,NLIFT(I))		
BCFL(I,J)	i[c]	BUCKET CORRECTION FACTOR FOR LOADER; FACTOR LESS THAN 1.00 REDUCES MATERIAL HANDLED PER BUCKET PROPORTIONATELY; FACTOR GREATER THAN 1 INCREASES.
BFLH(I,J)	v	PRIMARY DISPOSAL AREA - LOAD AND HAUL IF = 98: PRIMARY HAULER DIPOSAL AREA IS EXCESS SPOIL AREA HOLH(I,J) IF = 99: PRIMARY HAULER DISPOSAL AREA IS TEMPORARY STORAGE AREA HOLH(I,J) OTHERWISE: PRIMARY HAULER DISPOSAL AREA IS BACKFILL CELL BFLH(I,J)
HOLH(I,J)	v	SECONDARY DISPOSAL AREA - LOAD AND HAUL SPOIL HAULING DESTINATION IF BFLH = 98 OR 99 UNLESS FILLED TO CAPACITY (THEN - NEXT CELL IN SEQUENCE IS DESTINATION) OTHERWISE: EXCESS SPOIL DISPOSAL AREA ACCESSED IF BACKFILL SEQUENCE SEARCH INITIATED AT BFLH(I,J) ENCOUNTERS ANY BACKFILL CELL DESIGNATED AS FILBRK OTHERWISE: PRIMARY HAULER DISPOSAL AREA IS
BFCD(I,J)	v	PRIMARY DISPOSAL AREA - LOADER CARRY IF = 98: PRIMARY LOADER DIPOSAL AREA IS EXCESS SPOIL AREA HOLH(I,J) IF = 99: PRIMARY LOADER DISPOSAL AREA IS TEMPORARY STORAGE AREA HOLH(I,J) OTHERWISE: PRIMARY LOADER DISPOSAL AREA IS BACKFILL CELL BFCD(I,J)

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
HOCD(I,J)	v	SECONDARY DISPOSAL AREA - LOADER CARRY LOADER CARRY DESTINATION IF BFCD = 98 OR 99 UNLESS FILLED TO CAPACITY (THEN - NEXT CELL IN SEQUENCE IS DESTINATION) OTHERWISE: EXCESS SPOIL DISPOSAL AREA ACCESSED IF BACKFILL SEQUENCE SEARCH INITIATED AT BFCD(I,J) ENCOUNTERS ANY BACKFILL CELL DESIGNATED AS FILBRK
BFPO(I,J)	v	PRIMARY DISPOSAL AREA - DOZER PUSH IF = 98: PRIMARY DOZER DISPOSAL AREA IS EXCESS SPOIL AREA HOPO(I,J) IF = 99: PRIMARY DOZER DISPOSAL AREA IS TEMPORARY STORAGE AREA HOPO(I,J) OTHERWISE: PRIMARY DOZER DISPOSAL AREA IS BACKFILL CELL BFPO(I,J)
HOPO(I,J)	v	SECONDARY DISPOSAL AREA - DOZER PUSH DOZER PUSH DESTINATION IF BFPO = 98 OR 99 UNLESS FILLED TO CAPACITY (THEN - NEXT CELL IN SEQUENCE IS DESTINATION) OTHERWISE: EXCESS SPOIL DISPOSAL AREA ACCESSED IF BACKFILL SEQUENCE SEARCH INITIATED AT BFPO(I,J) ENCOUNTERS ANY BACKFILL CELL DESIGNATED AS FILBRK
PRBLK(I,J)	v	BLOCK NUMBER THAT HAS TO BE STRIPPED BEFORE WORK ON LIFT J OF BLOCK I CAN BE STARTED.
PRLFT(I,J)	v	LIFT NUMBER THET HAS TO BE STRIPPED BEFORE WORK ON LIFT J OF BLOCK I CAN BE STARTED. THIS LIFT IS IN BLOCK PRBLK(I,J).
ADEP(I,J)	i^c	DEPTH OF THE LIFT J OF BLOCK I (FT.)
BLAREA(I,J)	i	AREA OF THE LIFT J OF BLOCK I (SQ. FT.)
DRATE(I,J)	i^c	DRILLING RATE, LIFT J OF BLOCK I (FT./MIN.)

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
FTIME(I,J)	i^c	FRACTION OF LOADER TIME DOZER IS NEEDED FOR FEEDING OVERBURDEN TO LOADER (LOADER CARRY MODE).
FTIME2(I,J)	i^c	FRACTION OF LOADER TIME DOZER IS NEEDED FOR FEEDING OVERBURDEN TO LOADER (LOAD AND HAUL MODE).
FBLAS(I,J)	v^c	FRACTION OF LIFT TO BE BLASTED
FDOZ(I,J)	v^c	FRACTION TO BE PUSHED BY DOZER
FRCD(I,J)	v^c	FRACTION TO BE CARRIED BY LOADER
FLNH(I,J)	v^c	FRACTION TO BE LOADED AND HAULED
FROCK(I,J,K)	i^c	FRACTION OF LIFT(I,J) COMPOSED OF ROCK TYPE K
BCF(I,J)	i^c	BLADE CAPACITY CORRECTION FACTOR FOR DOZER PUSH
FILL CELLS: (I = 1,NFILCL)		
CFILCL(I)	v	CAPACITY OF BACKFILL CELL I (LCY)
BTIME(I)	n/a	DOZER TIME NEEDED TO GRADE BACKFILL CELL I (HR.)
NAFIL(I)	i	NUMBER OF BLOCKS THAT MUST BE MINED BEFORE THIS BACKFILL CELL IS AVAILABLE. DO NOT SET EQUAL ZERO. IF NO PREREQUISITES, SET = 1 AND SET NBFIL(I,1) = 0.
NBFIL(I,J)	i	BLOCK NUMBERS OF BLOCKS THAT MUST BE MINED BEFORE BACKFILL CELL I

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
FILSEG(JJ,K)	v	NUMBER OF SECTIONS IN HAUL ROAD FROM BLOCK JJ TO BACKFILL CELL K 5 = MAX.
FILGR(JJ,K,L)	v	GRADE OF SEGMENT NUMBER L OF BLOCK JJ AND BACKFILL CELL K (%) IF GRADE OF FIRST SEGMENT = 99, GRADE OF SECOND SEGMENT INTERPRETED AS 2 WAY HAUL TIME (SEC.), EXCLUSIVE OF DUMP TIME.
FILDS(JJ,K,L)	v	LENGTH OF SEGMENT L IN HAUL ROAD FROM BLOCK JJ TO BACKFILL CELL K (FEET)
POFIL(JJ,K)	i	PUSH OVERBURDEN DISTANCE FROM BLOCK JJ TO BACKFILL K
CDFIL(JJ,K)	i	LOADER CARRY DISTANCE FROM BLOCK JJ TO BACKFILL K
TEMPORARY STORAGE AREAS (EXT CELLS) (I = 1,NEX)		
CEXST(I)	n/a	CAPACITY OF SPOIL TEMPORARY STORAGE AREA I (CU.YD.).
NEXT(I,J)	n/a	BACKFILL CELLS WHERE SPOIL FROM TEMPORARY CELL I SHOULD BE TAKEN DURING REHANDLE, IN ORDER OF PRIORITY; J=1,5 5 DESTINATIONS MUST BE SPECIFIED FOR EACH EXT CELL; ENTER 0'S IF NO DESTINATION PREFER PREFERRED.

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
EXTSEG(JJ,I)	n/a	NUMBER OF SECTIONS IN HAUL ROAD FROM BLOCK JJ TO TEMPORARY STORAGE AREA I. 5 = MAX.
EXTGR(JJ,I,L)	n/a	GRADE OF SEGMENT NUMBER L OF HAUL ROAD FROM BLOCK JJ TO TEMPORARY STORAGE AREA I (%) IF GRADE OF FIRST SEGMENT = 99, GRADE OF SECOND SEGMENT INTERPRETED AS 2 WAY HAUL TIME (MIN.), EXCLUSIVE OF DUMP TIME.
EXTDS(JJ,I,L)	n/a	LENGTH OF SEGMENT NUMBER L OF HAUL ROAD FROM BLOCK JJ TO TEMP STORAGE AREA I (FEET)
POEXT(J,I)	n/a	PUSH OVERBURDEN DISTANCE FROM BLOCK J TO TEMP CELL I (FEET - ONE WAY)
CDEXT(J,I)	n/a	LOADER CARRY DISTANCE FROM BLOCK J TO TEMP CELL I (FEET - ONE WAY)
EXCESS SPOIL DISPOSAL AREAS (HOL CELLS) (K = 1,NHOLLW)		
CHOLLW(K)	v^d	CAPACITY OF EXCESS SPOIL DISPOSAL AREA K (CU.YD.).
HOLSEG(JJ,K)	v^d	NUMBER OF SECTIONS IN HAUL ROAD FROM BLOCK JJ TO HOL CELL K 5 = MAX.
HOLGR(JJ,K,L)	v^d	GRADE OF SEGMENT NUMBER L OF ROAD FROM BLOCK JJ TO HOL CELL K (%) IF GRADE OF FIRST SEGMENT = 99, GRADE OF SECOND SEGMENT INTERPRETED AS 2 WAY HAUL TIME (MIN.), EXCLUSIVE OF DUMP TIME.

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
HOLDS(JJ,K,L)	v^d	LENGTH OF SEGMENT NUMBER L OF ROAD FROM BLOCK JJ TO HOL CELL K (FEET). HAULER ROAD IF NHAUL = 0
POHOL(I,K)	v^d	PUSH OVERBURDEN DISTANCE FROM BLOCK I TO HOL CELL K
CDHOL(I,K)	v^d	LOADER CARRY DISTANCE FROM BLOCK I TO HOL CELL K
REHANDLE:		
REHSEG(JJ,K)	n/a	NUMBER OF SECTIONS IN HAUL ROAD FROM TEMP STORAGE AREA JJ TO BACKFILL CELL K 5 = MAX.
REHGR(JJ,K,L)	n/a	GRADE OF SEGMENT NUMBER L OF HAUL ROAD FROM TEMP STORAGE AREA JJ TO BACKFILL CELL K (%) IF GRADE OF FIRST SEGMENT = 99, GRADE OF SECOND SEGMENT INTERPRETED AS 2 WAY HAUL TIME (MIN.), EXCLUSIVE OF DUMP TIME.
REHDS(JJ,K,L)	n/a	LENGTH OF SEGMENT NUMBER L OF HAUL ROAD FROM STORAGE AREA JJ TO BACKFILL CELL K (FEET)
SPEED LIMITS (I = 1,NLIM; NLIM = TOTAL NUMBER OF SPEED LIMITS)		
LIMTYP(I)	v	TYPE OF SPEED LIMIT 1 = ENTRY LIMIT 2 = FULL SEGMENT LIMIT

Table A1: Continued.

Variable name/type	Value/ strategy	Quantity represented
LIMCOD(I)	v	CODE FOR TYPE OF ROAD WHERE SPEED IS LIMITED 1 = BLOCK TO BACKFILL 2 = BLOCK TO HOL 3 = BLOCK TO EXT 5 = REHANDLE
LIMFRM	v	CELL NUMBER, SOURCE
LIMTO	v	CELL NUMBER, DESTINATION
LIMSEG	v	ROAD SEGMENT NUMBER
SPDLIM(I)	v	LIMITING SPEED (MILES PER HOUR)
COSTS:		
OPCOS(I)	73.12 69.47 75.73 114.17 82.11	OPERATING COST ($/HR) I = 1 DRILL I = 3 OVERBURDEN TRUCK I = 2 DOZER I = 4 OVERBURDEN LOADER I = 5 COAL LOADER
OWCOS(I)	0.00 0.00 0.00 0.00 0.00	OWNING COST ($/HR) I = 1 DRILL I = 3 OVERBURDEN TRUCK I = 2 DOZER I = 4 OVERBURDEN LOADER I = 5 COAL LOADER COST PER HOUR, APPLIED TO OPERATING TIME AND IDLE TIME

Legend:

Numerical value: value used for simulation of LA and AOC spoil handling plans.

n/a : value assigned does not influence spoil handling costs.

i : values assigned to array were identical, AOC and LA cases.

v : values assigned to array varied with spoil handling strategy.

a : Source: Caterpiller Tractor Co., 1984 and 1985; Terex, 1981.

b : Source: manufacturer's literature.

c : Values transferred directly from COSTSUM output files.

d : Spoil movement distances and volumes transported to HF2 identical, LA and AOC cases.

e : Source: Load and haul cycle time data set.

Appendix B. Analysis of Load-and-Haul Cycle Time Data Set.

The purpose for recording load-and-haul cycle time data was to develop a data base to verify hauling simulation procedures. This data allowed estimation of average times for cycle components assumed as job averages by OPSIM (loader cycles, truck maneuvering and dumping times). Also, the data set allowed the hauler simulation procedure utilized by OPSIM, which assumes optimum machine and operator performance, to be "calibrated" with reference to a record of hauler performance.

The portion of the data set containing timed load-and-haul cycles considered appropriate for use included 409 cycles over 15 routes at the Amos Ridge site, and 78 cycles over 3 routes at auxiliary sites. The Amos Ridge data was supplemented because of a shortage of observations of the steep upgrade hauling necessary for construction of AOC backfills, due to the spoil handling plan utilized at that site. The procedures for recording this data included dividing the cycles into components and recording the times of transition between cycle components. Obvious non-routine delays in any cycle component were recorded with an asterisk and (if time allowed) a short note defining the reason for the delay. At the close of the day, the haul road profile was measured and sketched, including the approximate average grade and length of each segment. Load and haul cycle components recorded include: the loader cycles, truck time between final loader bucket and the start of motion (considered as a maneuvering time), the time in motion hauling to the dump site, dump site maneuvering and dumping time, the time required to return to the loading site, and the truck maneuvering time at the loading point previous to the first loader bucket.

The first step of analysis consisted of entering the observed transition times, with delay notations, into computer files. A SAS program was written to calculate the elapsed time of each observed cycle component. Next, for each component, all non-delay elapsed times were plotted by hauler route using the PROC UNIVARIATE procedure. The resulting box plots allowed identification of additional outlying points; these were considered to be the result of non-recorded non-routine delays and were identified as delayed components. In addition, those cycle components considered as constant for the entire job (loader cycles, truck maneuvering and dumping times) were plotted as an entire data set, again for the purpose of identifying outlying points.

The next step was to check the assumption that loader cycle times and truck maneuvering and dumping times were, in fact, constant for the entire operation. No statistics were run to check these assumptions. However, the frequency plots for these cycle components over the entire operation appeared to have a near-normal distributions, skewed towards the high side. In general, those differences which appeared to exist between the route means could not be related to any predictable factors, but appeared to be related to factors such as haul road and loading site conditions and weather. The one exception is dumpsite time, which is quite definitely related to the characteristics of the dumpsite; dumpsites high on AOC backfills require greater times, since trucks are often required to back for longer distances from the stopping point to the dumping point. None of these factors were incorporated into the analysis and average times were assumed.

Finally, load-and-haul cycle efficiencies were estimated. The overall efficiency was assumed to have two distinct components: those associated with non-routine delays, and those associated with haulers operating at less than peak efficiencies.

Hauler efficiencies were estimated by simulating hauler performance over the observed hauler routes, and comparing the simulated to the observed times (Table B-1). Only the loaded hauling times were used for this comparison, due to the observed tendencies of the drivers to time their returns to the departure of the tandem hauler at the loading site, rather than to "hurry and wait." For each route, an efficiency factor was calculated as a ratio of the simulated time to the observed time. The overall efficiency factor was calculated as a number-of-observations weighted average of all route efficiency factors, at 62.14 %.

Delay factors were also calculated (Table B-2). Only the Amos Ridge observations were used in this procedure. For each hauler route, the total non-delay cycle time was estimated as the sum of the means of all non-delayed cycle components. Estimates of average time spent waiting at the loading point for the departure of the other hauler during non-delay tandem cycles were included in these totals. To determine the actual mean for each route, the total time of observation was divided by the number of loads hauled per hauler over the time period. Again, the route-specific factors were calculated as the ratio of the non-delay time divided by the observed mean cycle time, and a weighted average of 74.76 % was computed for the entire site. This estimate compares favorably to the Caterpiller Tractor Co. (1985) estimate of a "typical" job efficiency factor of 45 to 50 minutes per working hour.

The major reasons for delays appeared to be large rocks in the loader pit, and hauler travel delays due to narrow road passage and interference with other machines working the haul roads or the dumpsites. Single hauler route effuiciencies were estimated as greater than tandem hauler routes (89.1 % vs. 71.3 %). These factors were not applied to the simulation procedures. If they had been, it would appear that they would have a larger positive effect on spoil handling cost in the AOC case, rather than the LA case, due to longer, narrower haul routes, increased frequency of haul road and dumpsite maintainance operations, and fewer opportunities to run single haulers.

The hauler efficiency and cycle delay factors were incorporated into the analysis as effects upon cycle components. The 25% delay factor was applied by increasing the observed non-delay means of the loader cycle time, truck maneuver time, and dumping time observations by a factor of 1.33. The OPFAC variable, which is used to modify the calcualated haul time, was set at 0.45, the product of the cycle delay and hauler efficiency factors. The result of these procedures is OPSIM-estimated hauler loads-per-hour factors between 6.0 and 11.0, with the majority falling between 7.5 and 10.0. This range agrees closely with the range represented in the COSTSUM output, excepting those few routes where extrmely low rates result from abnormal delays.

Table B-1. Data used for calculation of hauler efficiency factors, by hauler route.

Route number	Number of observations	One way distance	Obs. time	Pred. time	Efficiency
		(feet)	(sec)	(sec)	(%)
2	8	295	34.4	21.4	62.2
3	3[a]	740	71.6	44.0	61.5
4	8	800	64.1	58.7	91.6
5	11	265	19.6	18.2	92.9
6	14	300	34.6	19.2	55.6
8	26	470	57.5	29.4	51.1
9	18	384	38.1	25.4	66.6
10	23	584	55.0	41.8	75.9
15	49	939	99.1	56.4	56.8
15A[b]	31	809	70.9	44.4	62.5
17	39	1039	86.0	63.4	73.7
19[c]	14	1311	114.4	72.0	62.9
20[c]	29	1174	118.9	80.0	67.2
21[c]	25	638	80.2	45.6	56.8
31[d]	20	380	61.0	20.7	33.9
32[d]	18	250	40.5	13.7	33.8
33	5	555	75.6	41.7	66.6

Average					62.14

[a]: Terex hauler only; Cat hauler running on same route ran approx 20 seconds slower; poor repair of Cat hauler caused data to be discarded for this analysis.

[b]: Same dates as route 15, alternate dump site.

[c]: Cat haulers on steep upgrades at auxiliary site; all others Terex haulers at Amos Ridge.

[d]: Poor weather and road conditions; fog, rain, mud.

Table B-2. Data used for calculation of load-and-haul cycle delay factors, by hauler route.

Route number	Haulers per route	Number observ.	Non-delay average	Delayed Average	Efficiency
			(sec)	(sec)	(%)
1	1	12	243	277	87.7
2	1	9	228	292	78.1
3	2	12	320	326	98.0
4	2	14	262	391	66.9
5	1	13	209	247	84.5
6	1	14	244	262	93.1
8	2	32	290	536	54.0
9	2	23	204	283	72.3
10	2	38	250	349	71.2
15[a]	2	37	317	497	63.8
15[a]	2	60	317	423	75.0
17	2	40	306	405	75.7
31	2	23	264	338	78.3
32	1	19	226	244	92.8
Average					74.76

[a]: Hauler routes 15 and 15A were recorded on two successive dates.

APPENDIX C. Analysis of Mining Block Cost Differences

The general mining method used at Amos Ridge is typical of that used at other haulback contour sites in southwestern Virginia. Mining activities generally take place in two or three areas simultaneously. These are usually non-adjacent for safety, operational efficiency, and ease of spoil handling purposes. Thus when a block of coal is exposed, mining can proceed in other areas without disrupting coal removal activities. When the coal is removed, mining in the adjacent block normally begins. By mining in this fashion, the maximum amount of material can be moved by dozer push and loader carry operations into the adjacent block after coal removal. It is less expensive to push and carry material for short distances than to load and haul. An estimated 20-25 % of the spoil handled at Amos Ridge was pushed and carried. Generally, if a push destination is available, material will be pushed to that area until the pushing distance required to dispose of the material becomes too great for economical operation (generally about 100 feet). Material movement is then continued with a loader carry operation, again filling the closest available disposal areas until the economical distance (about 175 feet) is exceeded. The remainder of the material is hauled, with destinations chosen so as to balance two separate objectives: maintaining the quickest haul route while attaining the desired post mining land form and surface. The above is an idealized sequence, of course, which requires near-perfect coordination between drilling and blasting, overburden removal, and coal removal operations.

First Cut Blocks in the Hollows

Blocks 1LS, 3LS, 2LM, and 7LM (Fig. 7) were all outcrop blocks in the first hollow. The initial cuts into the hollow (1LS and 2LM) resulted in high spoil handling costs. The reasons for the expense include the general difficulties of removing shot material from the hollows, hard rock and difficult access. The hard, unweathered sandstone resulted in higher drilling and blasting costs; it also tended to break into larger fragments which were more time consuming to handle and often resulted in the loader working with less than a full bucket. Other loading difficulties included problems with breaking out loadable material from the shot due to tendency of the shot rock to wedge into the highwall corner from the force of the explosive. Block 1LS had the highest cost component (Table 4) for using the dozer to feed shot material to the loader ($0.23 per block bcy) on the entire job. Working material from the shot was a big problem on the lower lift of 2LM, where the final shot did not break up the rock as desired. This occurred in spite of drilling on a closer-than-average hole spacing, which resulted in the job's highest blasting cost component ($0.56). Considerable loader time was spent trying to move these materials out; the location of the highwalls on two sides prevented the dozers from getting behind and pushing, which would have been the most convenient method. The high spoil handling cost of 2LM was recorded in spite of being able to place up to 25% of the spoil handled, including the large boulders, into the top end of hollow fill HF1. Lack of access to adjacent mined blocks required that large proportions of both 1LS and 2LM had to be hauled back to areas contour mined in 1983 which were being reclaimed to AOC; the cost component for working the dumpsites of 1LS ($0.23) was also the highest recorded.

Spoil handling costs in blocks 3LS and 7LM were also affected by the hard rock and restricted access. Drilling and blasting costs were near or above average for both blocks, in spite of considerable loose soil deposits that were moved directly. Restricted access to adjacent blocks and the fact that the construction of hollow fill HF1 was well advanced by the time these blocks were mined caused the proportion of material carried and pushed to be less than 15% for each. Loading costs were affected by the inside corners in the highwalls which caused difficulties when breaking out the shot rock. Although the primary haulage destinations were close by for each of these blocks (HF1 for 3LS, HF1 and 6LM for 7LM), haulage costs per bcy moved exceeded the site average.

Point Outcrops

The opposite situation is represented by the three lowest cost blocks on the site: 5LM, 5LS, and 3UM. All are outcrop blocks on the points. The softer, weathered materials on the points minimized drilling and blasting cost components (less than $0.30 for each) by making it possible to move substantial material without blasting and by offering minimum resistance to drilling in the portion that was drilled. The point position generally offered excellent access to material (it is easier to push material away from a point highwall than to push out of a corner) and to adjacent mining pits (>90% of each was moved to adjacent blocks). Approximately 50% of 5LM and 35% of 5LS were moved by carry and push operations. Softer materials were loaded rapidly and easily, which allowed haulers to carry loads that were closer to their capacity. However, a related problem was that, in some cases, a portion of the exposed coal was found to be sufficiently weathered as to be unmarketable. The high stripping ratios of 5LM and 9LM resulted from this low rate of coal recovery.

Block 9LM was the first cut point block with the highest spoil handling cost. A considerable portion of 9LM had been covered by block 5LS. Hence, the material was not as well weathered. Also, material from 5LS occupied the adjacent disposal areas. Over 90% of block 9LM hauled, the majority to H1 and 7LM.

Second and Third Cuts

Spoil handling costs in blocks 2LS, 4LS, and 6LM showed the effects of different spoil handling practices. Blocks 2LS and 4LS are the second and third cut point blocks, the first cut having been taken during 1983; they are composed of near identical materials; both were moved initially by carrying and pushing from the top lift over the highwall into the pit below (1LM and 2LM from 2LS; 6LM and 7LM from 4LS). The difference was that mining had been completed at the base of 4LS before it was mined. One of the purposes for pushing 2LS material over the edge was to to segregate hard sandstone rocks and boulders suitable for use in the HF1 blanket drain. Thus, much of the material required rehandling. Also, a smaller proportion of 2LS was pushed and carried, since the subsequent mining of 6LM and 7LM would have been hindered by excess spoil. As it was, it appears that the spoil pushed over from 2LS (along with high blasting cost, other problems associated with hard rock, and dumpsite cost component of $0.17) was partially responsible for the high cost of spoil handling in 6LM. Due to the accumulated 2LS material, spoil movement by push and carry operations was minimal while haulage costs were high in spite of the proximity of the primary dumping locations (HF1, 4LM, 5LM). This block had to be mined as if it were in a hole; material had to be hauled out of the hole in order to access disposal areas. However, the result of mining this block and 7LM was that over 50% of 4LS was pushed and carried directly to the 6LM-7LM pit, with no recorded rehandling.

Block 6LS was mined in a fashion which shared some of the characteristics of 2LS and the first cut hollow blocks; 30-40% of its material was pushed into the mining pits below, with minimal rehandling. However, the high cost component of spoil movement was haulage, partially due to the loading difficulties associated with hard rock. The blockier portions of the material hauled from 6LS were moved a long distance to hollow fill HF2 for purposes of core drain construction, while finer materials were dumped in 4LS.

Blocks 3LM and 4LM also were relatively high cost blocks in spite of the facts that the drilling and blasting cost components were below the site average and that proportions of materials from each carried or pushed to adjacent blocks were close to the site average. The high cost component for these blocks was haulage. Virtually all of the hauled materials were taken back to the AOC area, which was being readied for reclamation at that time. A significant amount was taken for one of the longest hauls recorded, which included a switchback route to a backfill area on the Upper Marker bench.

Upper Marker Blocks

Relative spoil handling costs on the Upper Marker bench displayed a pattern similar to that observed in the blocks mined below. However, the average cost of moving spoil on this seam was lower, mainly due to softer rock which was easier to shoot and load. For the same reason, the strong cost effect of working a first cut (4UM) into the hollow was not evident. Drilling and blasting cost components for all UM blocks were below $0.40 per block bcy; 3UM had the lowest drilling and blasting cost component for the entire job ($0.16). The low spoil handling cost for 3UM occurred in spite of the low percentage of material carried and pushed (<20%) due to ease of hauling. Over 95% of the hauled material was taken to 2UM. The high hauler loads per hour (lph) over this route (10.8, vs. 8.55 for the entire job) reflected ease of loading as well as rapid hauler cycles.

The highest cost Upper Marker block was 1UM, a second cut block taken from the point. Approximately 75% of 1UM was moved by hauler; the cost of haulage was high, as most of this material was ramped up to cover the Upper Marker highwall in the AOC area to the southwest.

Blocks 2UM and 4UM are both first cut hollow blocks; less than 5% of each was carried or pushed to a disposal area. Haulage distances were similar. Spoil from 2UM was hauled to 1UM and to the first cut block directly east, while 4UM was hauled primarily to 2UM; much material from 4UM was dumped over the highwall into the northern wing of 6LS below.

APPENDIX D. Mining Plan Comparison

The following is a detailed comparison of the simulated AOC mining plan to the LA mining plan as observed and simulated. The LA case (Fig. 4a) is referred to as actual mining while the AOC case (Fig. 4b) is referred to as simulated mining. In order to ease the verbal demands of this discussion, AOC mining is referred to as if it had actually occurred.

As in actual mining, simulated mining was initiated in blocks 1LM, 1UM, 1LS, and 2LM (Fig. 7). Minor variations in cost between the two mining plans resulted from the AOC plan's redirection of material placed in the hollow fill and from failure to move topsoil from the UM seam the the Lower Marker area for reclamation purposes during AOC mining. The vast majority of spoil from these areas was hauled back to the AOC83 area in both cases.

Next, blocks 3LM, 4LM, and 5LM were mined in both cases. Most of the materials from 3LM and 4LM were taken to the AOC83 area; 18,000 loose cubic yards (lcy) placed in HF1 during actual mining was redirected to AOC83 during simulation. The reduction of per-bcy cost of hauling material from 3LM is not a result of the change in post-mining landform; it is compensated, for the most part, by the increase in per-bcy hauling cost from 4LM, as I took the liberty of redirecting 3LM material from the upper disposal area of AOC83 area to a lower elevation, lower cost disposal area, in order to accomodate an the 18000 lcy from 4LM while remaining consistent with the original assumptons. The most likely disposal area for the "extra" 10000 lcy directed to the AOC83 area during AOC mining is the upper region of the hollow immediately south of the experimental practice area. Eight thousand lcy of material from subsequent blocks, including most of the topsoil isolated for surface placement in the AOC83 area during actual mining, were redirected to more localized disposal areas during the AOC simulation. Material from 5LM was disposed in adjacent blocks (3LM and 4LM) in both cases.

During spoil removal from the LM point (3LM, 4LM, and 5LM), block 2LS was also being mined. During actual mining, the majority of this material (nearly 50,000 of 82,000 lcy) was taken to HF1, including approximately 20,000 rehandled lcy. This was the most difficult mining block to simulate, in terms of developing reasonable spoil handling assumptions, for three reasons: HF1, the major disposal area for carried and pushed materials, was no longer available, the rehandling of material from the base of the carried and pushed spoil in the 2LM pit to HF1 increased both the cost of material handling and the amount of material that could be carried and pushed, and the loader carry costs from this area were extremely high due to poorly shot, extremely hard material and the large fraction of loader time assisted by the dozer. It was assumed that 35,000 lcy would be carried and pushed from this area in the AOC case, with no rehandling, and that the majority of this spoil was taken to 1-5LM.

In both cases, blocks 6LM and 2UM were mined shortly after 2LS. During actual mining, the majority of 6LM spoil was taken to the LM point (blocks 1LM and 3-5LM) and to HF1, including nearly 10,000 lcy which were carried and pushed. However, in the AOC case, the backfill cells coinciding with mining blocks 1LM and 3-5LM had already been filled with spoil from 2LS. Therefore, there was nowhere to carry or push material from 6LM to, so all 6LM material was hauled to higher elevations in the 1LM and 3-5LM area; this began the process of filling the "second level" of backfill cells on the point, those defined at elevations 33 to 59 feet above the LM seam. All material from 2UM was disposed in the 1UM vicinity in the AOC case similar to observed disposal procedures except that topsoil was hauled to reclamation areas below during actual mining.

At this point, the two mining plans diverged. During actual mining, 3LS and 7LM were the next blocks taken as the operation moved into the hollow; large quantities of spoil from these two blocks (approximately 33,000 lcy) were placed in HF1, with most of the remainder going to 6LM. However, this was not possible during simulated mining, due to a near spoil-bound condition where the spoil pile above 1LM and 3-5LM could be visualized as looming over the 6LM pit below and restricting available spoil disposal options. The necessity to plan for disposal of a large quantity of material from 4LS made this situation a problem. The quickest solution was to immediately complete the mining of the eastern end of 7LM (that portion that could be mined without first excavating 3LS) by disposing of most of this material in the adjacent 6LM, and then to mine 4LS by carrying and pushing most of this material (68%) into 6LM and 7LM pits below. The strategy for handling the 4LS material was identical to that used during actual mining but it occurred at a different point in the sequence. The effect of moving 4LS forward in the sequence was to make the entire area east of 4LS available for spoil disposal since all mining in the area was complete, thus eliminating spoil disposal difficulties. The next steps in the AOC mining plan were to remove spoil from 3LS to the rebuilt point area above 1-6LM, and to complete the mining of 7LM by taking advantage of the newly exposed 4LS disposal area.

In both cases, block 3UM was worked into the mining sequence near this point with the majority of its spoil hauled to 2UM and the 1UM vicinity. Block 8LM was also mined at this time; most of its material was actually hauled to 7LM but the AOC mining plan called for it to be added to the rebuilt point above 1-6LM.

Block 5LS was mostly disposed of in adjacent areas of 7LM and 8LM during actual mining. However, during simulation, it also was hauled to the upper areas of the backfill at the 1-6LM point so as to reduce the amount of materials that would be required to be hauled from the lower-lying blocks 9 and 10 LM to these higher elevations.

Blocks 6LS and 9LM were large blocks mined in tandem, with nearly half of 6LS (48,000 of 105,000 lcy) being carried and pushed into areas below (including 3LS, 4LS, and 7LM) in both cases. Also in both cases, approximately 27,000 lcy were hauled from 6LS to hollow fill HF2. During actual mining, 16,000 lcy from 6LS were used to initiate highwall backfilling activities at 4LS. In the AOC case, these 16,000 lcy were hauled to the the point backfill above 1-6LM. The abnormally high hauling cost from 6LS was in part due to hard rock and loading difficulties (high dozer assist ratios) in both cases. The hardness of the rock made it an excellent material for initiation of HF2 core drain construction.

There was a large difference between the spoil handling costs estimated for block 9LM in the two cases. During actual mining, the majority of 9LM spoil was disposed of in nearby areas (HF1, and the

HF1-7LM interface), with the remaining 15,000 lcy being hauled to the 4LS backfill. In the AOC case, 30,000 lcy were taken to the 1-6LM point backfill. The large increase in per-bcy hauling costs stems from increased amounts taken to the backfill, the AOC backfill's requirement for increased (relative to the LA backfill) amounts of spoil at higher elevations, and the unavailability of the HF1 upper surface as a hauler route. At this point, the point backfill was being "topped off"; these were some of the most expensive simulated hauler routes on the site.

In both cases, the majority of spoil from block 7LS was disposed of in 6LS. However, in the AOC case, increased quantities were required at a relatively high elevation. Block 4UM was also handled similarly in both cases. However, in the LA case, approximately 9000 lcy were dumped over the highwall to 6LS, a relatively low cost method of building up the backfill below. Due to increased the spoil requirements of the rebuilt point above and east of 1UM, this option was not available during AOC mining procedures; the majority of the 4UM spoil was used to build up this point in the AOC mining plan.

Large quantities of spoil were moved from blocks 10LM and 9LS to HF2 (approximately 30,000 lcy each) in both cases, with most of the remainder going to adjacent areas (mostly 9LM) and to the highwall backfill at 6LS. The higher per-bcy hauling cost from 9LS was, again, the result of the AOC landform's requirement for larger quantities of spoil at the higher elevations. The removal of spoil from block 5UM completed mining activity during the period of study. During actual mining, a large proportion of this spoil was able to be placed in nearby areas (3UM and 4UM), whereas it was disposed in 2UM during simulated mining, another cost effect of the large quantity of spoil consumed by the requirement to rebuild the point above and east of block 1UM.

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