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A Moving Baseline for Evaluation of Advanced Coal Extraction Systems

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ABSTRACT

This document reports results from the initial effort to establish baseline economic performance comparators for a program whose intent is to define, develop, and demonstrate advanced systems suitable for coal resource extraction beyond the year 2000. Systems used in this study were selected from contemporary coal mining technology and from conservative conjectures of year 2000 technology. The analysis was also based on a seam thickness of 6 ft. Therefore, the results are specific to the study systems and the selected seam thickness. To be more beneficial to the program, the effort should be extended to other seam thicknesses.

FOREWORD

This document is one of a series which describe systems level requirements for advanced underground coal mining equipment. These requirements are summarized in "Overall Requirements for an Advanced Underground Coal Extraction System," JPL Publication 80-39 by Martin Goldsmith and Milton L. Lavin. Five areas of performance are discussed:

(1) Production cost.

(2) Miner safety.

(3) Miner health.

(4) Environmental impact.

(5) Recovery efficiency.

The report which follows presents details of a study which extrapolates contemporary coal mining technology to the year 2000. The projections for cost and production capability comprise a so-called moving baseline which will be used to assess compliance with the systems requirement for production cost. Separate projections were prepared for room and pillar, longwall, and shortwall technology all operating under comparable sets of mining conditions.

This work is part of an effort to define and develop innovative coal extraction systems suitable for the significant resources remaining in the year 2000. Sponsorship is provided by the Office of Coal Mining, United States Department of Energy via an interagency agreement with the National Aeronautics and Space Administration. William B. Schmidt, Director of the Office of Coal Mining, is the project officer.

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The author would like to express appreciation for contributions and comments provided by members of the JPL staff. Guidance and comments received from Milton L. Lavin, project manager, are greatly appreciated. A special "thank-you" is extended to William B. Mabe for his input to the room-and-pillar analyses and his comments. Paul G. Gordon and Anthony Lynn are acknowledged for their thorough efforts directed toward the production of 1980 capital equipment costs. Other document reviewers are identified and individually praised: Frank A. Camilli, Jack Harris, and Martin Goldsmith. For her professional editorial assistance, Patricia A. South is freely applauded. Finally, I would like to personally thank Joan Winkler for her preparation of this document.

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EXECUTIVE SUMMARY

The purpose of this study was to establish baseline economic performance comparators for the evaluation of new, advanced underground coal extraction system concepts. The baseline comparators consist of contemporary, 1980 systems that have evolved through time and will exist until a new concept has matured to a commercially acceptable form. The 1980 systems were projected to their conjectural states in the year 2000 by consideration of current research and development activities, trends in the industry, and present production constraints of the selected systems.

The technologies selected for study and their representative 1980 system components are as follows: (1) continuous room-and-pillar: rotary-drum continuous miner, shuttle cars, and dual-boom roof bolter; (2) retreat longwall: double-ended ranging shearer, armored face conveyor, and chock-type hydraulic roof supports; and (3) retreat shortwall: rotary-drum continuous miner, mobile bridge carrier (MBC) conveyor haulage, and chock-type roof supports. The panel development systems selected for longwall and shortwall contain rotary-drum continuous miners, MBC units, and dual-boom roof bolters.

The proposed year 2000 system for room-and-pillar contained a rotary-drum continuous miner partnered with an MBC unit that automatically tracks the continuous miner. The roof control function of the system, which will permit breakthrough-to-breakthrough lift lengths, will be accomplished with several hydraulically powered roof-support units and dual-boom roof bolters. This same equipment was used for panel development in the year 2000 longwall and shortwall cases.

The driving force for the year 2000 longwall system was better utilization of armored face conveyor capacity. Considering the present status of longwall R&D activities, the fruition time for commercial products from these activities, and other production constraints, the proposed system has two double-ended ranging shearers equipped with vertical control systems. Complementing the shearers are roof supports, a high-capacity armored face conveyor, a stageloader, a face advancement control system, other ancillary components, and a master control system that effectively coordinates all face activities.

Because several experts have suggested narrower web widths for shortwall miners to improve ground control, the year 2000 system was configured with this notion. The shortwall system has a continuous miner with a 7-ft cut width, supports that achieve a 7-ft advance in one step, and a continuous haulage system that accommodates the face space limitations. In addition to possible strata control improvements, calculations showed that a narrower web system will increase shift production by 14% to 27%.

After identification of the systems, the performance parameters that determine shift production were identified and quantified. Two study cases were selected: ideal mining conditions and average mining conditions. The parametric values for the ideal cases were determined as the perceived design limits of equipment and operational procedures, not being influenced by the mine environment or operator ability. The values for the average cases were determined, in most instances, from actual operational data found in the literature and personal files. It was assumed that such values did not change over time for the selected systems. The following table presents the results of this study phase. The production of the 1980 cases was calculated as a check to the approach. The production range reflects the average and ideal conditions cases.

ESTIMATED SYSTEM PRODUCTIVITY IN RAW TONS PER MACHINE-SHIFT

System	Raw Tons per Machine-Shift 1980	2000
Room-and-Pillar	290-680	560-1540
Longwall and Shortwall Panel Development	450-1330	530-1390
Longwall	830-1770	1210-2530
Shortwall	520-1110	660-1260

The last phase of the study partnered the mining systems to appropriate mine plans so that discounted cash flow analyses could be performed. The analyses provided break-even production costs at a 15% return on investment. The following table shows the results in 1980 dollars. The range in values reflects the conditions cases.

ESTIMATED PRODUCTION COSTS

Technology	Production Cost per Clean Ton 1980	Production Cost per Clean Ton 2000
Room-and-Pillar	\$22.59-39.84	\$15.71-26.66
Longwall	\$17.50-29.05	\$16.48-25.71
Shortwall	\$18.53-31.36	\$18.30-29.41

SECTION I

INTRODUCTION

The purpose of this report is to establish economic performance measures against which the performance of advanced coal mining system concepts can be compared. The economic measures will be produced in the form of break-even production costs for a specified return on investment. Advanced systems refer to those which can extract significant coal resources that remain in the year 2000 and beyond, and which promise a significant improvement in economics and miner safety. Because the advanced system must compete with other underground mining systems that will exist at the time the concept has matured to its commercially acceptable design, the economic measures in this study will be developed for contemporary mining systems that have evolved over time. The projection of present systems to their conjectural states in the year 2000 will establish the moving baseline.

In order to attain the objective of this report, the scope of the study will contain several tasks. First, the baseline system will be selected and extrapolated to their year 2000 configurations. Therefore, there will be two sets of systems analyzed: present, 1980 systems and extrapolated, year 2000 systems. While the year 2000 systems will establish the economic measures for the advanced systems, the productivity and cost results of the 1980 systems will provide a check on the analysis approach. The baseline systems will be contemporary representatives of three underground coal mining technologies: continuous room-and-pillar, longwall and shortwall. The projection of the baseline systems to the year 2000 will consider current research and development activities, industrial trends, and production constraints of the baseline systems.

Secondly, the performance parameters needed to determine system productivity will be identified and quantified. Raw tons of coal per machine-shift is the measure of system productivity selected for this study. The parameters used to compute productivity will be selected for two cases: ideal conditions and average conditions. The ideal condition cases will represent, in the author's viewpoint, the production potential of the systems, since the parameters will be derived as the perceived design limits of the equipment and operational procedures. Because the average condition parameters will be developed from actual operational data when available, the average condition cases will provide a realistic estimate of system productivity.

After productivity is established, each system will be partnered with an appropriate mine plan so that a discounted cash flow analysis can be performed in order to arrive at the break-even production cost per clean ton. The cost analysis will require the identification of mine size and life, personnel requirements, equipment and construction requirements, supplies and materials cost, power cost, and other inputs needed for a discounted cash flow analysis.

The final section of the report will discuss the results of the study, both the system productivities and production costs. The 1980 results will be compared against current industrial experience and other studies to insure that reliable results have been produced. Finally, guidance will be provided regarding the use of the year 2000 results for comparisons.

This study was undertaken in support of the Advanced Coal Extraction Systems Definition Project at the Jet Propulsion Laboratory (JPL), Pasadena, California. The project is part of a program in the U.S. Department of Energy, whose purpose is to define, develop, demonstrate, and commercialize advanced coal extraction systems. The results of the moving baseline study will provide direct input to another document, in production at JPL, by Martin Goldsmith and Milton L. Lavin, entitled "Overall Requirements for an Advanced Underground Coal Extraction System." In this requirements document, JPL is trying to create a yardstick against which mining systems can be measured. To be considered as an advanced system, a concept will have to exceed in performance, against this yardstick, regardless of what existing systems or their logical derivations might offer. The standards by which a system will be judged, report to five separate attributes: (1) conservation of resources, (2) environmental effects, (3) miner health effects, (4) miner safety, and (5) production cost. Therefore, this moving baseline report will provide a standard of comparison for the production cost attribute.

Because mining regions and their mines vary greatly within the United States, it is unlikely that a universal system can be developed. Thus, the coal fields of Eastern Kentucky were selected as the initial resource target because they possess adequate coal reserves to support production well into the next century. Additionally, it is believed that markets for that coal will continue to exist; and the mining conditions pose a significant challenge to the system designer. Therefore, this study will endeavor, where possible, to be representative of Eastern Kentucky.

SECTION II

SELECTION OF SYSTEMS

A. INTRODUCTION

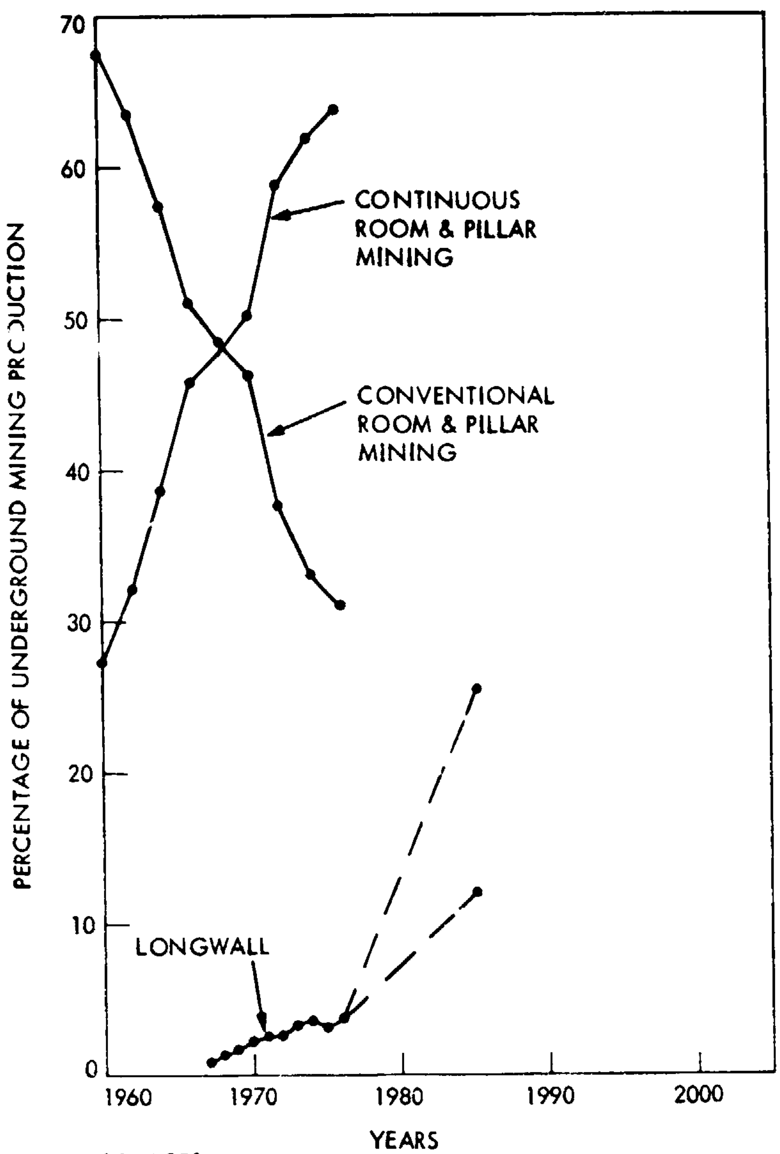
Three contemporary underground coal mining technologies have been selected for extrapolation to the target year of 2000. This section provides the reasons for the selection of the technologies and their representative systems, and for the transformation of the systems to their conjectural year 2000 configurations. The results of this section will be a description of the 1980 systems and their year 2000 counterparts, both of which comprise the moving baseline.

B. BASELINE SYSTEMS

Three of the technologies currently used by the underground coal mining industry are considered appropriate for this study. They are room-and-pillar with a continuous miner, longwall, and shortwall. Continuous room-and-pillar was selected because it accounts for over 60% of U.S. underground production today, and as Figure 2-1 shows, has made a rapid entry into the industry over the past years (Ref. 1). Although there are many possible system configurations, examination of equipment available for continuous room-and-pillar indicated that the most common scheme uses a rotary-drum continuous miner partnered with shuttle car haulage and supported by a dual-boom roof bolting machine (Ref. 2). Therefore, this system configuration was used as representative of the contemporary, 1980, case.

Longwall, which is applicable to larger mines, was selected for several reasons. First, as shown in Figure 2-1, longwall has had a consistent rate of growth in the industry. Additionally, longwall may account for a considerable portion (12% to 25%) of underground production by 1985 (Refs. 3 and 4). If these projections are extended at the same rate to the year 2000, longwall could contribute from 26% to 61% of underground production. Moreover, longwall mining systems account for the majority of underground coal production in many European countries. These factors suggest that longwall systems hold great promise for the U.S. coal industry. It has been reported that the most commonly used longwall configuration in the United States incorporates a double-drum shearer with an armored face conveyor and chock-type hydraulic roof supports (see Ref. 4). This system is chosen to represent the contemporary, 1980, case.

For longwall panel development, the system selected contains a rotary-drum continuous miner and a mobile bridge carrier (MBC) haulage unit. The system is basically room-and-pillar technology applied to panel development. The MBC unit was selected because it provides better haulage service to the continuous miner than shuttle cars, thus affording a higher potential productivity. At this time, the MBC unit is second only to the shuttle car in utilization and is increasing in popularity. Thus, it was thought appropriate to team this unit with



DATA SOURCES:

1960 - 1976: NATIONAL COAL ASSOCIATION, COAL FACTS, 1978 - 1979

HIGH LONGWALL PROJECTION: BUSINESS WEEK (1978)
LOW LONG WALL PROJECTION: KUTI (1979)

Figure 2-1. Underground Coal Production by Method the longwall. This selection widened the spectrum of mining systems in the study, also.

Shortwall technology, while producing only a small fraction of the U.S. total production, is relatively new. Shortwall holds promise for the future because, as a wall-type system, it has several advantages over longwall and continuous room-and-pillar, under proper conditions. Shortwalls require less capital than longwalls, function better at shallow depths under massive roof stratum, and are more flexible for skirting undesirable geological and man-made situations (Ref. 5). Shortwall also has better health and safety features than both longwall and R&P. The European and Australian mining establishments, too, have expressed considerable interest in the future application of shortwall technology (Ref. 6). The shortwall systems that have been tried in the United States (there have been about 11 of them) used a panel development unit in conjunction with chock-type roof supports for the shortwall system (Ref. 7). The author elected to use the same development unit for shortwall and longwall as a basis for projections. Thus, both development and production for shortwall contain a rotary-drum continuous miner and an MBC haulage unit.

Conventional room-and-pillar technology was not selected for study because it was felt that the technology has reached its maturity, and will not experience significant changes in the future. Evidence of the continuing sharp decline in conventional production, as shown in Figure 2-1, suggests that it may not be an important alternative in the future.

C. EXTRAPOLATED SYSTEMS

The projection of the contemporary systems to the year 2000 will emphasize three mining functions: coal winning, haulage, and roof support. While other mining functions have impact on system productivity, it was felt that the above-mentioned functions were most important. For all three technologies, it is anticipated that improvements will be made in dust control, gas control, equipment safety and equipment reliability. No consideration was given to improvements in equipment panel move techniques, which have a significant influence on the overall productivity of longwall and shortwall systems. Projections of the progress to be expected in each of the major technologies were based in part on a survey of current research and development activities (Ref. 8). Continuing review of published reports and journal articles, and contact with the responsible government agencies have provided supporting information.

In addition to determining the focus of research and development activities in the industry, several major production constraints for room-and-pillar, longwall, and shortwall were identified. For room-and-pillar, two major constraints were identified: the frequency of continuous miner place-change and the intermittency of shuttle-car haulage service. The constraints associated with longwall concern the capacity of armored face conveyors and their utilization, and the advance rate of the supports. An analysis of the contemporary, 1980 shortwall system identified the cutting width of the continuous miner and the mode of support advance as the major production constraints. The following paragraphs provide more detail about the constraints and the system modifications that might be expected to improve the situation in the future.

1. Extrapolated Room-and-Pillar

A statutory provision of the Federal Coal Mine Health and Safety Act of 1969 prohibits movement of personnel beyond the last permanent support unless adequate temporary support is provided. To comply with this provision, many mine operators elected to shorten continuous miner advance distances so that the locally positioned operator remained under permanent support. The advance distance is generally 18 ft to 22 ft, depending on the machine design. This option leads to the first production constraint mentioned for R&P, the frequency of continuous miner place-change. Several equipment manufacturers developed devices for remote-control operation, and also developed locally controlled miner-bolter machines that permit permanent support placement in conjunction with entry advancement. Both manufacturers' developments allow continuous miner advancement to approach the pre-1969 situation of breakthrough-to-breakthrough length lifts (60 ft to 100 ft). However, both have drawbacks. Remote-control operations are limited by operator vision, the position of haulage operators with respect to the last permanent support, the stability of the roof, and many others. Most miner-bolter machines experience roof-bolting delays that erode the potential time savings.

Another approach taken to improve the performance of R&P systems is automation combined with remote-control. The U.S. Bureau of Mines had, and, more recently, the Department of Energy, has a program to develop an automated remote-controlled continuous mining system (Refs. 9, 10, 11, and 12). The aim of the program is to develop a miner-bolter machine that can function without the aid of on-board operators. To date, the program has not demonstrated a fully automated system. Furthermore, a recent program demonstration of a locally controlled miner-bolter candidate met with many difficulties (see Ref. 12).

An interesting approach taken by Frank Stafford, a retired mine superintendent, has increased production by 27% in initial tests (Ref. 13). The approach involves hydraulically activated roof support beams that are advanced with another set of hydraulic cylinders. This roof support system allows a locally controlled continuous miner to advance further by providing adequate temporary support. While the support unit is only in the initial development phase, it has great promise because it provides a very simple, straightforward solution to the roof control problem, and it can continue to be used with present equipment as it evolves.

The second R&P constraint, the intermittent service provided by shuttle cars, may be eliminated by use of a continuous haulage system, such as the mobile bridge carrier (MBC) unit manufactured by

Long-Airdox Company. While there are several reasons why industry uses shuttle cars more extensively than MBC units, the major reasons concern surge capacity, maintainability, flexibility, and ease of operation (see Ref. 2). Chain-conveyor MBC units now manufactured by Long-Airdox have a surge bin option for their customers (Ref. 14). Additionally, a conceptual study of an automated remote-controlled continuous room-and-pillar mining system placed a surge feeder unit between the continuous miner and MBC unit (Ref. 15). This system design is part of a long-term development program in the Department of Energy (see Ref. 11). Some components of the conceptual system are undergoing additional study (Refs. 16 and 17). In order to improve the tracking and guidance of an MBC unit behind a continuous miner, the Department of Energy is developing an "auto-track" MBC unit (Ref. 18). With a feedback control system, the MBC unit will straddle and follow an induction cable that is laid on the mine bottom by the inby segment of the unit which is under local, manual control. This addition will ease guidance and control problems.

As the previous discussion indicates, there may be several future system options that directly address current R&P constraints. However, for this moving baseline, a standard rotary-drum continuous miner, partnered with an automatically tracking MBC unit, hydraulic temporary roof support units, and dual-boom roof bolters were selected. This system is seen as an obvious evolution of existing equipment that does not require a great deal of sophisticated hardware, and at the same time minimizes functional interactions between coal winning and strata control. This system will also be used for the year 2000 longwall and shortwall development cases.

2. Extrapolated Longwall

As mentioned earlier, the constraints of longwall production are the capacity of armored face conveyors, the under-utilization of face conveyor capacity, and the advance rate of roof support units. While the capacity of armored face conveyors does limit system production, it is not altogether clear that any major advances in the near future will change the situation. Conveyor capacity is governed by the cross-sectional area of the conveyor pan and the speed of the conveyor chain. The cross-sectional area is presently constrained by the design of the roof supports and the shearer, and conveyor flexibility requirements. Therefore, increase of the conveyor cross-sectional area will require system redesign. How this redesign might be accomplished is not clear.

Present chain speeds are limited in order to minimize the wear rate of chain links and pan, and to maintain acceptable noise levels. While several attempts have been made to develop lubrication systems, none seems acceptable (Ref. 19). The only feasible approach may be more abrasive-resistant materials for the links or friction-reducing liners for the conveyor pans. While it is certain that manufacturers and researchers are investigating this avenue, no positive results were found in the literature. It is evident that present conveyor capacities may be a major limiting factor for longwall production.

Other researchers in their studies of future systems have reached this same conclusion (Refs. 20 and 21).

Despite the limitation present conveyors impose on overall productivity, conveyor capacity is under-utilized. This apparently contradictory situation is due to present operational cycles for shearers which have a considerable amount of nonproductive time. This point is illustrated by the analysis of the half-face method (Appendix B), currently, the most commonly used in the United States. The nonproductive segments of the shearer cycle for the two cases examined, ranged from 30% to 47% of the total cycle time. To improve conveyor utilization, two shearers (or more) could be placed on the face. This practice is quite common in the United Kingdom (Ref. 22). Each shearer would be assigned to a particular segment of the face. With the use of an interactive control system, each shearer would cut its segment of the face in such a way that the conveyor is not overloaded and collision is avoided. Analysis of this configuration (Appendix B) showed a 33% decrease in the cycle time while obtaining the same production per cycle as the one-shearer scheme, thus improving total productivity. Although the two-shearer approach does not comply with MSHA regulation concerning one cutting machine per split of fresh air, the use of environmental sensors in the face area should permit the issuance of a regulatory variance.

Other studies examined the production increase that would result from modifications to the winning element only, such as wider-web shearers, and not to improving overall system efficiency as was done in this study (Refs. 20 and 23). Furthermore, these studies were directed primarily to respirable dust generation and methane liberation; they did not provide the overall system evaluation that is required for this analysis.

The present approach to automated longwall may improve the health and safety aspects of longwall systems by removing workers from critical areas, but may not necessarily improve production. During the preliminary design of an automated longwall (ALW) system that used a single shearer configuration, DOE contractors found that conveyor capacity constrained production (see Ref. 21). However, present automated longwall activities will definitely benefit future dual-shearer operations. The automated longwall program has identified three basic systems required for automated, remote-controlled longwall mining: a vertical control system for the shearer, a face advancement system, and a master control system (see Ref. 24). Efforts are underway to develop these systems for application to existing longwall configurations. Such developments will support the application of automation and remote-control to the dual-shearer longwall configuration, also. Additionally, British attempts at automated longwall were partially successful, but encountered labor/management problems (Ref. 25). Their experiences, nevertheless, will benefit American developments. Therefore, the availability of automated longwall options by the year 2000 does not seem unreasonable.

The third longwall production constraint is the advance rate of the roof support units. The "state-of-the-art" cycle time for a support is about 10 seconds (Ref. 26). This value transforms into a support advance rate along the face of 30 ft/min because supports are normally on 5-ft centers. Therefore, the shearer travel rate along the face should be limited to 30 ft/min.

Ideally, most roof support systems can be advanced along the face at a rate of 50 ft/min (Ref. 27). However, several factors limit support advance rates: (1) movement of the face conveyor; (2) the loss of fluid pressure and fluid flow; (3) lowering the support from the roof in preparation for advancement; and (4) raising the support to the roof after advancement. The first factor led to the development of the "one-web-back" method of face advance. This method eliminates face advance delays caused by conveyor movement needs, improves roof control, and increases the available travel space between the conveyor and supports because roof supports are advanced immediately behind the shearer. Therefore, many American operators have adopted this technique.

The second factor, the loss of fluid pressure and fluid flow, is related to the inadequacy of hydraulic power pack capacity and the buildup of back pressure in the return line. These problems can be alleviated by increasing the capacity of the hydraulic system (see Ref. 27).

The last two factors, which are support movement-related, result from the design of longwall powered supports. Therefore, in order to improve upon the situation, the basic support design must be modified. The French Collieries Research Institute has under development a crawler sliding hydraulic roof support (Ref. 28). This support design permits advancement under load, thereby eliminating the vertical roof-beam movement required with conventional longwall support designs. It is not known whether the crawler sliding support is superior to the conventional support or is cost-effective, but there are several prototypes in the field (see Ref. 28).

As discussed, heretofore, there are ongoing activities that may produce solutions to several longwall production constraints. Considering the present status of these activities and the period of time involved, the following system components are proposed for the extrapolated year 2000 longwall system:

(1) Two double-ended ranging drum shearers having vertical control systems.

(2) Chock-type roof support units.

(3) An armored face conveyor (AFC) with a peak capacity of 1500 tons per hour.

(4) A stageloader that can adequately handle peak loads from the AFC.

(5) A face advancement control system for the supports and AFC.

(6) A master control system that effectively coordinates all face activities.

3. Extrapolated Shortwall

As previously mentioned, the cutting width of continuous miners presently used in shortwall systems is usually 10 ft. Because of this width, the roof supports function in a manner which constricts production performance. These support constraints involve the rate of face advance and strata control.

Within the course of a face advance cycle, each support unit is moved forward twice, about 5 ft each time. The first advance occurs as the continuous miner cuts along the face, and results in little, if any, production interference. The second advance does not start until the continuous miner finishes cutting and starts tramming out of the face area. The resulting production interference is quite significant, accounting for 21% to 28% of the cycle time as shown in Appendix B.

Several shortwall operations have failed or have experienced many production delays because of poor roof conditions (see Refs. 7 and 18). While these poor conditions are mostly a result of natural processes, the unsupported roof area and quality of roof support that exists at the face do not help matters. While the continuous miner is cutting, the unsupported area is typically in the range of 400 sq ft (40 ft x 10 ft). After the initial support advance, a span about 5 ft wide along the entire face length (180 ft to 200 ft) is supported by the forepole devices of the supports. These devices provide little support resistance. This situation, along with the aforementioned roof span problem, promotes roof falls along the face. Roof falls not only delay production during their clean-up, but the resulting cavities also reduce support effectiveness and augment the problem.

Because the roof support constraints exist at the present cut widths, several experts have suggested narrower ones (see Refs. 5 and 6). Therefore, the extrapolated year 2000 shortwall system was projected to include this change. A review of in-house continuous miner specification pamphlets identified 7.75 ft as the narrowest miner chassis width with the cutter head minimum at 8.5 ft. A discussion with an equipment designer led to the possibility of a narrower body and cutter head (Ref. 29). Therefore, a 7-ft wide cutter head was elected for the moving baseline. To complement this narrow continuous miner, a support was designed in-house to achieve a 7-ft. advance (Ref. 30). Additionally, it was assumed that a continuous haulage system could be designed to accommodate the space limitations. An analysis of this extrapolated system determined that shift production increases range from 14% to 27%, depending upon the mining conditions.

D. SUMMARY

The moving baseline is summarized in Table 2-1, where the equipment for the room-and-pillar, longwall, and shortwall systems is listed.

Table 2-1. Description of the Moving Baseline for Room-and-Pillar, Longwall, and Shortwall

System	1980	2000
Room-and-Pillar	Continuous miner	Continuous miner
	20-ft lift length	Breakthrough length lifts
	Shuttle car haulage	Mobile bridge carrier haulage with automatic tracking
	Roof bolter	Roof bolter
		Mobile, powered temporary roof support system (MTRS)
Longwall and Shortwall Development	Continuous miner	Continuous miner
	20- to 40-ft lift lengths	Breakthrough length lifts
	Mobile bridge carrier haulage	Mobile bridge carrier haulage with automatic tracking
	Roof bolter	Roof bolter
		MTRS
Longwall	One double-ended ranging shearer (DERS)	Two DERS's
		AFC
	Armored face conveyor (AFC)	Chock-type supports
	Chock-type supports	Automatic control of DERS's, AFC, and supports
Shortwall	Continuous miner (10-ft head)	Continuous miner (7-ft head)
	Mobile bridge carrier haulage	Continuous haulage
	Powered supports	Powered supports permitting one-step advance

SECTION III

PRODUCTION ANALYSIS

A. INTRODUCTION

The next task is the identification and quantification of system performance parameters that dictate system productivity. The measure of productivity used in this study is raw tons of coal per machine-shift. The information used to quantify the necessary parameters comes from several sources: published literature, private communications, and the personal files and experience of the author.

B. APPROACH

The performance of any mining system may vary greatly over the range of mining conditions it encounters. In order to incorporate the effects of mining conditions into the analysis, it was decided to analyze all candidates in conjunction with two sets of mining conditions. These sets are designated as "ideal conditions" and "average conditions" throughout this report. In addition to the aforementioned reason, the two sets of mining conditions were selected for other reasons as well. These include the inability to accurately predict the effect of the mine environment on conceptual systems and the need to check the analysis approach against the industrial experience with contemporary underground coal mining systems. The results of the ideal conditions cases represent, in the author's viewpoint, the production potential of the systems within the constraints of the study. The performance parameters for the ideal cases were derived as the perceived design limits of the equipment and the operational procedures of the selected mining methods. In cases where actual operational data were used to develop ideal conditions parameters, the obvious effects of the mine environment were ignored. The results of the average conditions cases provided a means to check the accuracy of the analysis approach. Most of the performance parameter values of the average cases were derived from actual operational data. In other instances, the average conditions values were estimates based on the experience of the author and on the ideal values.

A "bottom-up" approach was selected for the production analysis. That is, the approach identified the basic cycles that dictate system productivity and arranged them into their hierarchical ranks. Figure 3-1 illustrates the process used to manipulate the system performance parameters within the various cycles in order to arrive at a system productivity value. Such an approach was chosen because it permits estimation of productivity for system configurations that are not currently in use by examining the performance of the system functions that control production. Additionally, the bottom-up analysis will be used to estimate the

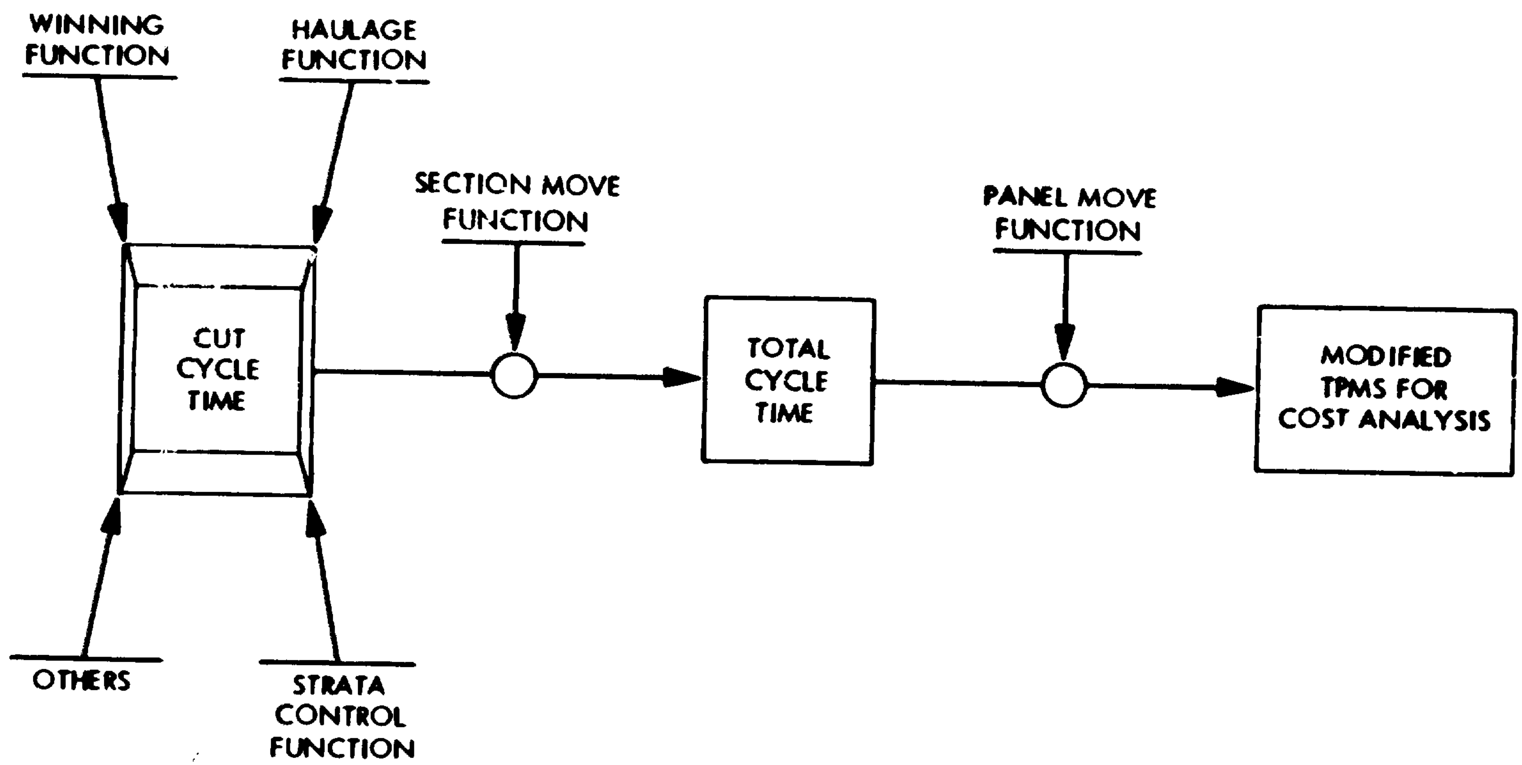


Figure 3-1. Method Used to Obtain System Productivity for Cost Analysis

production potential of new, conceptual systems. The cycles that were primarily used in this study involved the coal winning function, the section move (advance and retract) function, and the panel move function. The influence of other functions, particularly, face haulage and strata control, was considered in the appropriate cycle.

Calculation of system productivity in terms of raw tons per machine-shift (TPMS) required the quantification of the four terms in the following equation:

$$TPMS = \frac{APT \cdot A \cdot T}{C} \qquad \text{Equation (1)}$$

where APT = available productive time per machine-shift
A = system availability
T = raw tons per cycle
C = total cycle time

The term APT is defined as that portion of shift time available for productive work. In this study, APT is equal to the difference between total shift time and the assumed inherent delays such as travel time and lunch. For all cases in the study, APT is 357 min per machine-shift. Further details are provided in Appendix B.

System availability, that is A, is defined as the quotient of net work time (NWT) divided by APT, where NWT is the difference between APT and other delays. To distinguish between the two conditions cases, the definition of "other delays" was varied. For average conditions, other delays referred to maintenance time, unexpected operational delays, human-related delays, and mine-environment-related delays. The mine-environment-related delays were subtracted from the "other delays" determined for the average conditions cases, and the remaining delay time was used to arrive at a NWT for the ideal conditions cases. The values for system availability are shown in Table 3-1. These values were assumed to be constant between the 1980 and 2000 cases. In most instances, the values for NWT and APT were developed from actual operational data of systems with equipment similar to the 1980 cases (Refs. 31, 32 and 33). Further details concerning availability can be found in Appendix B.

The remaining two variables, T and C, are addressed in the following segments of this section, but a few explanatory comments are merited here. The variable T, raw tons per cycle, takes into account the volume of coal cut per cycle at a density of 85 lb/ft^3. The dimensions of this volume for the room-and-pillar systems and the panel development systems are the lift (cut) length, the entry width, and seam height. For longwall, the dimensions refer to the face length, the web width, and seam height. Shortwall dimensions include the face length, the width of the miner's cutting head, and the seam height.

Table 3-1. System Availability of Study Cases

System	System Availability Average Conditions	Ideal Conditions
Continuous Mining Machine	0.50	0.63
Longwall	0.56	0.64
Shortwall	0.60	0.66

Total cycle time, C, is developed through the proper combination of specific coal winning machine parameters and the selected operational procedure of the entire machine unit, as detailed explicitly in Appendix B. For systems that have a continuous miner, partnered with shuttle car haulage, the combination involves the winning machine parameters, the mode of lift (cut) removal, the cut sequence, the haulage machine parameters, the interaction of the haulage machines with one another, and the section move function. The computation of total cycle time for the longwall systems requires the combination of shearer parameters with the half-face method of operation and the section move function. Shortwall calculations combine continuous miner parameters with those of the supports, the guide rail, and the section move function to determine total cycle time. Because the year 2000 room-and-pillar systems and all panel development systems contain continuous haulage instead of shuttle car haulage, their total cycle time computations are more straightforward than the shuttle car systems. Combination of the continuous miner parameters with the mode of lift removal, the cut sequence, and the section move function provides the desired result.

The following three segments of this section, "Room-and-Pillar and Panel Development Systems," "Longwall Systems," and "Shortwall Systems," discuss the highlights of the procedures used to determine total cycle time for each system. Each segment also identifies the "pre-panel-move" productivity for the systems as calculated in Appendix B.

The final segment, "Cost Analysis Productivity," establishes the duration of the panel move function for each system and identifies the modified system productivities which account for idle time caused by this function. These modified productivities will be then used in the cost analysis.

C. ROOM-AND-PILLAR AND PANEL DEVELOPMENT SYSTEMS

These systems are grouped together because they contain continuous mining machines (CMM) and all are derived from room-and-pillar technology. In order to estimate the productivity for these systems, appropriate mine plans had to be established so that the remaining variables of Equation 1 could be numerically defined. This effort involved the selection of several parameters: number of entries, centerline dimensions for entries and crosscuts, crosscut angle, cut sequence, and section ventilation scheme. Information contained in a U.S. Bureau of Mines document provided input to the establishment of the parametric values (see Ref. 15). The details of the mine plan parameters are listed in Table 3-2.

Table 3-2. Mine Plan Parameters for Room-and-Pillar and Panel Development

	Parameters			
System	No. of Entries	Entry and Crosscut Centerline	Crosscut Angle	Cut Sequence and Ventilation Scheme
Room-and-Pillar:				
1980	5	100/100	90	Refer to Figures B-12 and B-14
2000	5	80/80	60	Refer to Figures B-18 and B-19
Longwall and Shortwall Development:				
1980	3	80/80	60	Refer to Figures B-1 and B-2
2000	3	80/80	60	Refer to Figure B-3

In addition to the selection of the parameters in Table 3-2, a decision was needed on the retreat mining method for the room-and-pillar cases. The partial extraction method is most common in Eastern Kentucky. Of 319 roof control plans from Eastern Kentucky reviewed in a recent study, partial extraction accounted for 76% (Ref. 34). Additionally, there has been a trend in the United States towards partial extraction (see Ref. 5). Therefore, the partial extraction method was selected for this study (see Figures B-14 and B-19).

The next step in the analyses of the systems was the identification of the elements that contribute to the total cycle time. For the 1980 Room-and-Pillar Cases, these elements included the sump cycle time, "maneuver-in-cut" time, and place-change time of the continuous miner; the change-point times, travel times, wait-point times, and dump time of the shuttle cars; and section move (advance or retract) time. Because the remaining CMM systems, both 1980 and 2000 cases, have haulage units that provide continuous service to the continuous miner, only continuous miner elements and section move time were needed for their analyses.

In order to calculate the sump cycle time of the continuous miner, the sump and shear rates were needed. For the ideal conditions cases, discussions with an expert regarding continuous miner performance revealed the theoretical limiting values for these rates to be approximately 17 and 34 ft/min, respectively (Ref. 61). Although this does not mean that this high performance can be achieved in practice in anything but the most fragile coals under excellent roof and floor conditions. The sump and shear rates for the average conditions cases were derived from actual operation data (Ref. 35). The averages for the studied systems were a sump rate of 5.66 ft/min and a shear rate of 10.50 ft/min. These ideal and average values were used not only for the 1980 Room-and-Pillar Cases, but also for the 1980 Longwall and Shortwall Panel Development Cases. Because the level of effort did not permit further study concerning the history and future of these rates, the 1980 values were used for the 2000 cases, too. Further details on the application of these rates can be found in Appendix B.

The tram rate of the continuous miner and the travel distance required identification before the "maneuver-in-cut" time could be calculated. The tram rate under ideal conditions was assumed to be 60 ft/min while the average rate was selected at 50% of the ideal rate or 30 ft/min. The distance travelled by the continuous miner during the extraction of a lift depends on the length and width of the lift, the width of continuous miner cutter head, and the mode of lift removal. While the width of the lift (20 ft) and the width of the cutter head (10 ft) remained constant for all CMM systems, the length of lift and mode of removal did not. Details concerning these two aspects can be found in the appropriate segments of Appendix B.

Place-change times for the continuous miner were governed by the tram rates and the place-change distances. In this study, an average place-change distance was established for each CMM system. Each cut sequence was followed from beginning to end in order to arrive at the total place-change travel distance. That value divided by the number of lifts in a prescribed distance of advance or retreat and combined with the aforementioned tram rates provided the average place-change time for each CMM system.

The combination of the sump cycle time, maneuver-in-cut time, the mode of lift removal, and the place-change time permitted the calculation of the average cut cycle time for the systems that use continuous haulage units. These average values are found in Table 3-3. The 1980 Room-and-Pillar cases, because they employ shuttle car haulage, required further consideration to account for the influence of face haulage on the average cut cycle time.

Table 3-3. Average Cut Cycle Time for Systems Partnered with Continuous Haulage

System	Lift Length (Ft)	Cut Cycle Time (Min)
2000 Room-and-Pillar Cases:		
Average Conditions	80/74	106/98
Ideal Conditions	80/74	45/42
1980 Longwall and Shortwall Panel Development Cases:		
Average Conditions	20	33
Ideal Conditions	40	24
2000 Longwall and Shortwall Panel Development Cases:		
Average Conditions	80	107
Ideal Conditions	80	45

1. First number in category refers to panel development and second number refers to panel retreat.

NOTE: The differences between the lift length for panel development and panel retreat are caused by geometric characteristics of the mine plans. Also, lift lengths and cycle times are rounded to the nearest whole number.

As mentioned earlier in this section, there are several shuttle car elements that influence the performance of the 1980 Room-and-Pillar cases: change-point time, travel time, wait-point time, and dump time. In order to calculate the first three elements, the shuttle car tram rates and travel distances were required. A review of the literature and manufacturer specification pamphlets identified 420 ft/min (7 MPH) as the maximum empty tram rate (Refs. 35, 36, 37, and 38). The selection of a loaded tram rate for ideal conditions depended upon the empty-to-loaded relationships found in the literature and the judgment of the author. A loaded tram rate value of 300 ft/min (5 MPH) was assumed. The average conditions values were assumed to be 50% of the ideal values - 210 ft/min empty and 150 ft/min loaded. The approach used in this study developed an average travel distance for each category of shuttle car movement. The average distances combined with the tram rates provided the necessary elements for analysis. Coupling the shuttle car elements with the appropriate winning function elements and an assumed dump time of 0.5 min produced an average cut cycle time for each 1980 room-and-pillar case. Figures B-13, B-15, B-16, and B-17 identify the element values and show how they were combined to help establish the average cut cycle times presented in Table 3-4.

Table 3-4. Average Cut Cycle Time for Systems Partnered with Shuttle Car Haulage

System	Cut Cycle Time (Min) Development	Retreat
1980 Room-and-Pillar Cases:		
Average conditions	56	67
Ideal conditions	29	35

All lifts are 20 ft long. Values rounded to nearest whole number.

There are several models available that simulate room-and-pillar operations having shuttle car haulage, but the level of effort for this study did not permit their utilization. While the simulation models exercise the continuous miner and shuttle cars through all movements required to execute a selected cut sequence, it was felt that the chosen approach provided an acceptable substitute.

The last element to be identified and to be combined with the average cut cycle time to establish the total cycle time was the section move element. This element relates to section advance time during development and section retract time during retreat. For this study, it was assumed that advance and retract times were equal. A review of the literature provides a range for move time from 1 h to 1.5 h (Ref. 39). Therefore, move times were assumed to be 1 h for the ideal conditions cases and 1.5 h for the average cases. These values, divided by the number of cuts per move provided an amortized section move time. The resulting total cycle times for the CMM systems are displayed in Table 3-5.

With the estimates of the "total cycle time" completed, the next step was the calculation of system productivity in terms of raw tons per machine-shift. The results for each system are exhibited in Table 3-6. As seen in Table 3-6, there are dramatic differences in productivity between the average and ideal conditions sets. For the 1980 R&P cases, this difference is caused by the variations in system availabilities; shuttle car travel times; section move times; and the maneuver times, cutting cycle times, and place-change times of the

Table 3-5. Total Cycle Times for R&P and Panel Development Systems

System	Lift Length (Ft)	Total Cycle Time (Min)
1980 Room-and-Pillar Cases:		
Average conditions	20/20	58/70
Ideal conditions	20/20	30/37
2000 Room-and-Pillar Cases:		
Average conditions	80/74	116/110
Ideal conditions	80/74	52/49
1980 Longwall and Shortwall Panel Development Cases:		
Average conditions	20	38
Ideal conditions	40	30
2000 Longwall and Shortwall Panel Development Cases:		
Average conditions	80	125
Ideal conditions	80	57

1. First number in category refers to panel development and second number refers to panel retreat. Values are rounded to the nearest whole number.

Table 3-6. System Productivity for R&P and Panel Development Systems

System	Raw Tons Per Machine-Shift	
	Average Conditions	Ideal Conditions
1980 Room-and-Pillar:		
Development	310	750
Retreat	260	630
2000 Room-and-Pillar:		
Development	590	1670
Retreat	530	1490
1980 Longwall and Shortwall Panel Development	460	1440
2000 Longwall and Shortwall Panel Development	550	1510

continuous miner. With the change to continuous haulage units for the year 2000 R&P cases, the productivity differences relate to system availability, continuous miner parameters, and section move time. Productivity differences for the panel development systems are caused by the same factors mentioned for the year 2000 R&P cases. Additionally, the lift length variation between the 1980 panel development systems (20 ft for average conditions and 40 ft for ideal conditions) indirectly created productivity differences.

Table 3-6 also shows a difference in productivity between development and retreat for the R&P cases. The variations in shuttle car travel distances and continuous miner place-change distances account for the productivity differences for the 1980 cases. The year 2000 situations are caused by the different lift lengths (Table 3-5), attributed to a geometrical effect.

The changes in productivities between 1980 and year 2000 for the systems represented in Table 3-6 occur for several reasons. The R&P change results from replacement of shuttle car haulage with continuous conveyor haulage, thus eliminating shuttle car change-out wait times. Additionally, the year 2000 R&P cases, because their lift lengths are 80 ft as opposed to 20 ft for the 1980 cases, experience less continuous miner place-change time. The productivity boosts associated with the year 2000 longwall and shortwall development systems relate to less place-change time, also.

The last aspect of Table 3-6 which requires explanation is the productivity difference between the R&P and panel development systems. In 1980, the selection of shuttle car haulage for the R&P systems and continuous haulage for longwall and shortwall development largely produced the productivity variance. However, the 100 ft entry centerline dimension and five-entry plan for R&P and the 80 ft dimension and three-entry plan for panel development had some influence. These mine plan variations also caused the year 2000 productivity differences between R&P and panel development.

D. LONGWALL SYSTEMS

The quantification of total cycle time for the longwall systems required the same basic approach as taken for the previous systems. A mine plan was selected to provide a panel length and face length. Also, a mode of face operation had to be selected from the several used by the American longwall establishment. From this point, appropriate performance parameters had to be selected for the equipment discussed in Section II and for the mode of face operation.

For all longwall cases, a panel length of 3000 ft was selected as being typical of American longwall practice (Ref. 40). A study to determine the optimal face length for longwall systems having a panel length of approximately the same measure as above (3024 ft) found, that little difference occurred with respect to return on investment for face lengths ranging from 200 ft to 500 ft (see Ref. 40). Therefore, a face length of 500 ft was selected for the longwall cases.

A review of the literature did not reveal much information concerning the most popular mode of face operation for a double-ended ranging shearer. However, a bulletin distributed by the Joy Manufacturing Company announced the half-face operation as being the most commonly practiced in the United States (Ref. 41). Additionally, half-face operation of a shearer is quite compatible with the "one-web-back" method of longwall face advance (Ref. 42). Finally, a recent article indicates that the half-face operation has better respirable dust characteristics than other modes of operation (Ref. 42). This factor alone is a plus for the half-face operation because respirable dust problems afflict many longwall installations (see Refs. 42 and 43). Therefore, such a mode was selected. Figure B-4 illustrates the steps of the operation, and the appropriate sections of Appendix B discuss the half-face method in more depth.

The next step in the analyses of the longwall systems was the identification and quantification of system parameters that affect the shearer cutting cycle time. The predominant parameters include shearer haulage speeds, support advance rate, and armored face conveyor capacity. A study that produced a conceptual design of an automated longwall system suggested that shearer cut haulage speeds be limited to 30 ft/min so roof support advance could keep pace (see Ref. 26). Before this speed could be used, a check was made to insure that face conveyor capacity exceeded cutting capacity. In order to do this, the web depth of the shearer was required as well as the maximum conveyor capacity. A web depth of 33 in. was found to be the present state-of-the-art and was used to determine cutting capacity (Ref. 19). A review of available literature and personal communications identified 1500 TPH (tons per hour) as the maximum available face conveyor capacity today (Ref. 44). Combination of cut speed, seam height, web depth and coal density identified a cutting capacity that was several hundred tons per hour less than the conveyor capacity. Therefore, in the ideal conditions cases, a shearer cut haulage speed of 30 ft/min was used.

Other parameters required for cutting cycle time were shearer flit speed and turn-around times. Flit speed, being the rate at which a shearer cleans a completed web cut, was assumed to be 50 ft/min for the ideal cases because it was the maximum shearer haulage speed identified (see Ref. 21 and Ref. 45).

While the cut and flit speeds of the shearer in ideal conditions were perceived as design limitations, the speeds developed for the average conditions cases were the average of operational data available to the study (see Ref. 26 and Ref. 46). The cut haulage speed was 11 ft/min and the flit speed was 28 ft/min. These values corresponded well to recent information obtained during a mine visit by the author where the shearer cut speed was 15 ft/min and the flit speed was 25 ft/min (Ref. 47). Additionally, the conceptual design of an automated longwall suggested that a flit-to-cut ratio of 2:1 be maintained (see Ref. 26). Since the real data provide a ratio of 2.5:1, it was decided to use the average values.

Turn-around elements refer to those activities on the longwall face related to directional changes in the shearer. For the half-face operation shown in Figure B-4, these elements include a cluster of activities at the tailgate and another cluster at the headgate. Tailgate activities encompass ranging of the shearer drums and reversal of the shearer and its loading devices. The time assumed for this element was 2.0 min (see Ref. 46). At the headgate, the tailgate activities are experienced again with one addition. The cutting picks are checked and replaced as required. The time assumed for the headgate element was 7.0 min (see Ref. 46). The headgate and tailgate turn-around elements were held constant for both conditions cases and for the 1980 and 2000 systems.

The combination of the shearer haulage speeds, the turn-around times, and the half-face operation provided the shearer cutting cycle times. The addition of a section move element to the cutting cycle summed to a total cycle time. This section move element, referred to in Appendix B as the amortized headgate move time, was quantified from actual operational data. For the ideal conditions cases, the smallest value found was used - 0.50 min per ft of move (see Ref. 46). The average conditions value was taken as the average of all available data - 0.64 min per ft of move (see Ref. 46 and Ref. 48). These same values were used in both the 1980 and 2000 cases. Multiplication of the above values with the move distance (web depth) for each case produced the section move times.

The resulting total cycle times for the longwall cases, combined with the tons per cycle, available productive times, and system availabilities as per Equation 1, produced the shift productions found in Table 3-7. More details are shown in Appendix B. The productivity variations found in Table 3-7 between the average and ideal conditions sets for each study year occur because of differences in shearer cut and flit speeds, availabilities, and section move times. The productivity increases experienced by the year 2000 systems in relation to the 1980 systems were caused primarily by reduction of "dead time" in the total cycle times. For the year 2000 longwall case in ideal conditions, the Gantt Chart in Figure B-6 shows how the cut travels of the headgate and tailgate shearers shadow most cycle deadtime (flit travel and turn-arounds). In the average conditions case for year 2000 longwall, the cut travels of the shearers totally dominate the total cycle time as shown in Figure B-7.

E. SHORTWALL SYSTEMS

As with the previous systems, the first step needed for the shortwall analyses was the selection of the mine plan: face length and panel length. Because there were no optimal face length studies found in the literature for the particular system configuration adopted here, typical lengths were deemed appropriate. Therefore, a face length of 180 ft and a panel length of 3000 ft were selected.

Table 3-7. Total Cycle Time and System Productivity for the Longwall Systems

System	Total Cycle Time (Min)	System Productivity (Raw tons per machine-shift)
1980 Longwall:		
Average conditions	79	890
Ideal Conditions	38	2090
2000 Longwall:		
Average Conditions	56	1350
Ideal Conditions	27	3250

NOTE: Total cycle times are rounded to the nearest whole number.

With the face length established, the total cycle time could be estimated. The parameters that required quantification include the cut and flit rates of the continuous miner, the advance rates of both the supports and guide rail, and section move rate. The sequence of operations for the continuous miner, supports (chocks), and guide rail in the 1980 shortwall systems are shown in Figure 3-2. The cut rates of the continuous miners in the shortwall systems were dictated by the sump cycles established for the conditions cases in the preceding CMM systems reporting. Review of Appendix B reveals these rates to be 0.23 min per ft for ideal conditions and 0.57 min per ft for average conditions. The flit rate, or speed at which the continuous miner retreats from the face after finishing one pass, was reported to be 20 ft/min in actual conditions (Ref. 49). Therefore this value was used for the average conditions cases. The flit rate for ideal conditions was assumed to be twice the actual value or 40 ft/min.

Roof support advance, which occurs twice during the cycle for the 1980 systems was identified at a rate of 45 seconds per unit under actual operating conditions (see Ref. 49). The design rate for these same units was specified as 30 seconds per unit (see Ref. 49). Therefore, support advance was assumed to require 30 seconds for the 1980 ideal conditions cases and 45 seconds for the 1980 average conditions cases.

For the 2000 shortwall systems, a different approach was selected for support advancement. This support design, instead of using hydraulic cylinders to advance supports by sliding as in the 1980 systems, provided a walking mechanism with specially designed

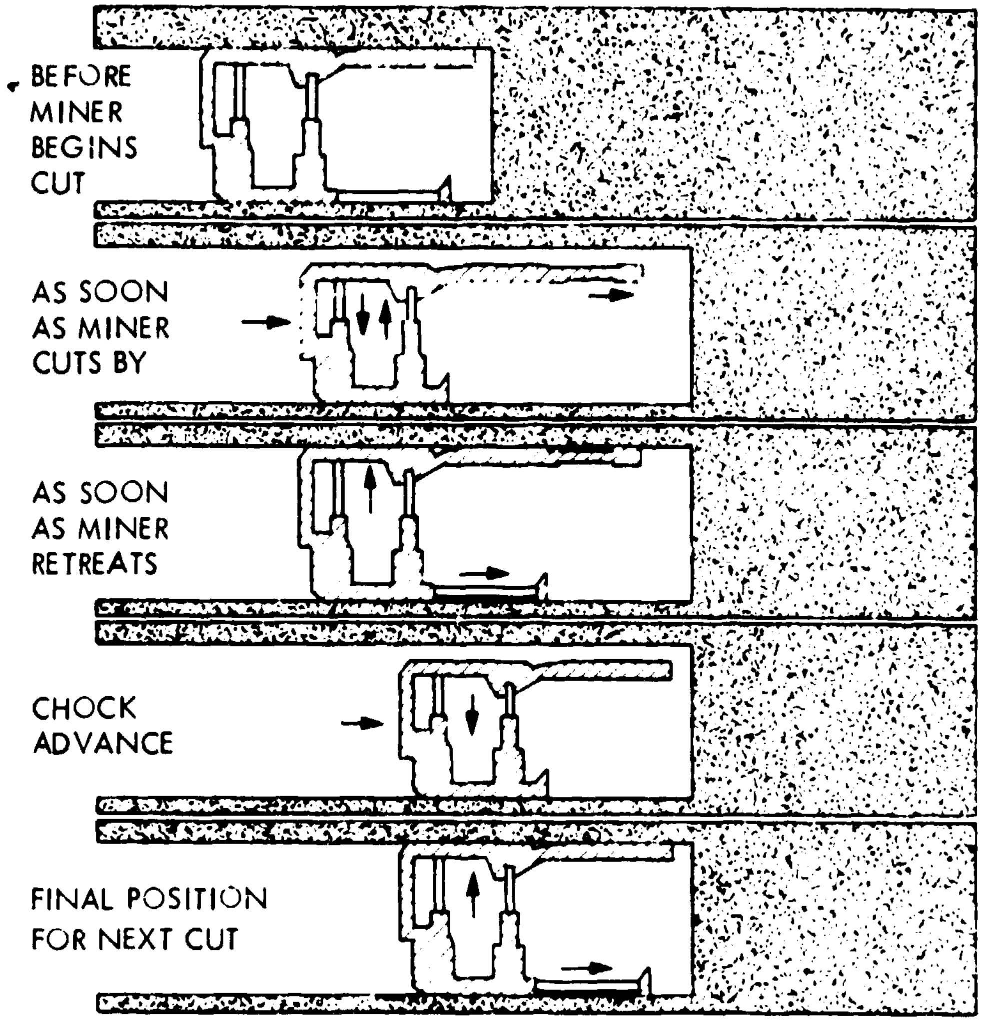


Figure 3-2. Sequence of Face Operations for 1980 Shortwall Systems (Ref. 5)

floor beams (see Ref. 30). The designer suggested an advance rate of 10 seconds per unit (for a distance of 7 ft) under ideal conditions. In average conditions, the assumed advance rate was halved to 20 seconds per unit.

Guide rail advancement, which also occurs twice per cycle for the 1980 systems and once for 2000 systems, was found to occur at a rate of 15 seconds per ram ideally (see Ref. 49). Because information was not available for rates in actual operating conditions, the ideal advance time per ram was doubled to 30 seconds for the average conditions cases.

The application of the aforementioned parameters to the properly sequenced face operations of the shortwall systems produced the cycle times for the face operations. The calculations for the analyses are presented in Appendix B. Figures B-8, B-9, B-10, and B-11 display Gantt Charts of face equipment scheduling for the shortwall cases.

The combination of the face operation cycle time with the appropriate section move elements provided the total cycle time for the shortwall cases. The move elements for shortwall were assumed equal to those used for the longwall and shortwall panel development systems.

Having identified all the variables required for Equation 1, the system productivities were calculated and are displayed in Table 3-8 along with total cycle times. All details leading to the results are shown in Appendix B. The productivity variations between the average and ideal conditions sets for both study years are caused by differences in the continuous miner parameters, support and guide rail advance times, and section move time. The small productivity increases for the year 2000 systems occurred for several reasons. The increased advanced rates for the continuous miners caused by a cutter head diameter change (3 ft for 1980 and 4 ft for 2000) and the elimination of one support/guide rail advance contributed positively. However, the narrower cutter head for the year 2000 shortwall miner had a negative effect.

F. COST ANALYSIS PRODUCTIVITY

Before the calculated productivities could be used in the Cost Analysis section, they required modification to account for the idle time caused by the panel move function. The time required for panel moves varies from system to system. For all room-and-pillar cases and the panel development units for longwall and shortwall, the panel move function time was assumed to be four shifts (Ref. 50). The time required for a longwall panel move was selected at 30 shifts (Ref. 51). Shortwall panel move time was reported to be 20 shifts (see Ref. 6). The system productivity values that result from consideration of panel move times are shown in Table 3-9. Calculations supporting these values are found in Appendix C.

Table 3-8. Total Cycle Time and System Productivity for the Shortwall Systems

System	Total Cycle Time (Min)	System Productivity (Raw tons per machine-shift)
1980 Shortwall:		
Average conditions	172	570
Ideal Conditions	80	1350
2000 Shortwall:		
Average Conditions	93	740
Ideal Conditions	48	1580

NOTE: Total cycle times are rounded to the nearest whole number.

Table 3-9. System Productivity Results After Consideration of Panel Moves

System	Productivity (raw tons per machine-shift) Average Conditions	Ideal Conditions
Room-and-Pillar		
1980 Development	310	730
1980 Retreat	260	630
2000 Development	580	1580
2000 Retreat	530	1500
Longwall and Shortwall Panel Development		
1980	450	1330
2000	530	1390
Longwall		
1980	830	1770
2000	1210	2530
Shortwall		
1980	520	1110
2000	660	1260

SECTION IV

COST ANALYSIS

A. INTRODUCTION

The final step of the analysis, the cost analysis, provides breakeven product costs at an assumed return on investment (15%) for the 12 study cases. To compute these costs, a version of a coal mining cost model for underground mines, developed by NUS Corporation for the Electric Power Research Institute and modified by JPL, was used (Ref. 52). The model, which followed a building block concept of analysis, incorporates the essential features of mining engineering, actual mining experience, and cost engineering principles which are required to analyze all cost aspects of producing coal from a new underground mine. Model selection was guided by its characteristics and availability. The structure of the model permitted the input of data that characterized the study cases. Other information required for the cost analyses and common to both the study cases and the model cases, was provided by the model. The computational provisions of the model that suited this study were the use of discounted cash flow methods to determine product cost and the ability to escalate costs from the base year of 1975 to any desired year.

The remainder of this section discusses the identification and quantification of input required for each block of the model in order to characterize the study cases. The model blocks and the flows of information between them are shown in Figure 4-1.

All costs discussed in this section are in 1980 dollars. The cost escalation factors input to the model were used to update specific cost inputs to either 1980 or year 2000 levels. Further details are presented in appropriate segments of this section.

B. PRODUCTION SIZING

This segment of the cost analysis provided the input upon which all other segments were built. The input variables were used to derive production section requirements and costs. The input supplied for this segment was placed into two categories: mine characteristics and financial aspects. The mine characteristics included the mining system, mine type, mine life, shifts per day, days per year, recovery factor, reject percentage, design capacity, existence of a preparation plant, and productivity. Tax rates, rate of return, and debt-equity ratio were included in the financial aspects.

1. Mine Characteristics

The mining system input referred strictly to the systems previously identified in the study: continuous room-and-pillar, longwall, and shortwall. A drift mine was selected as mine type since

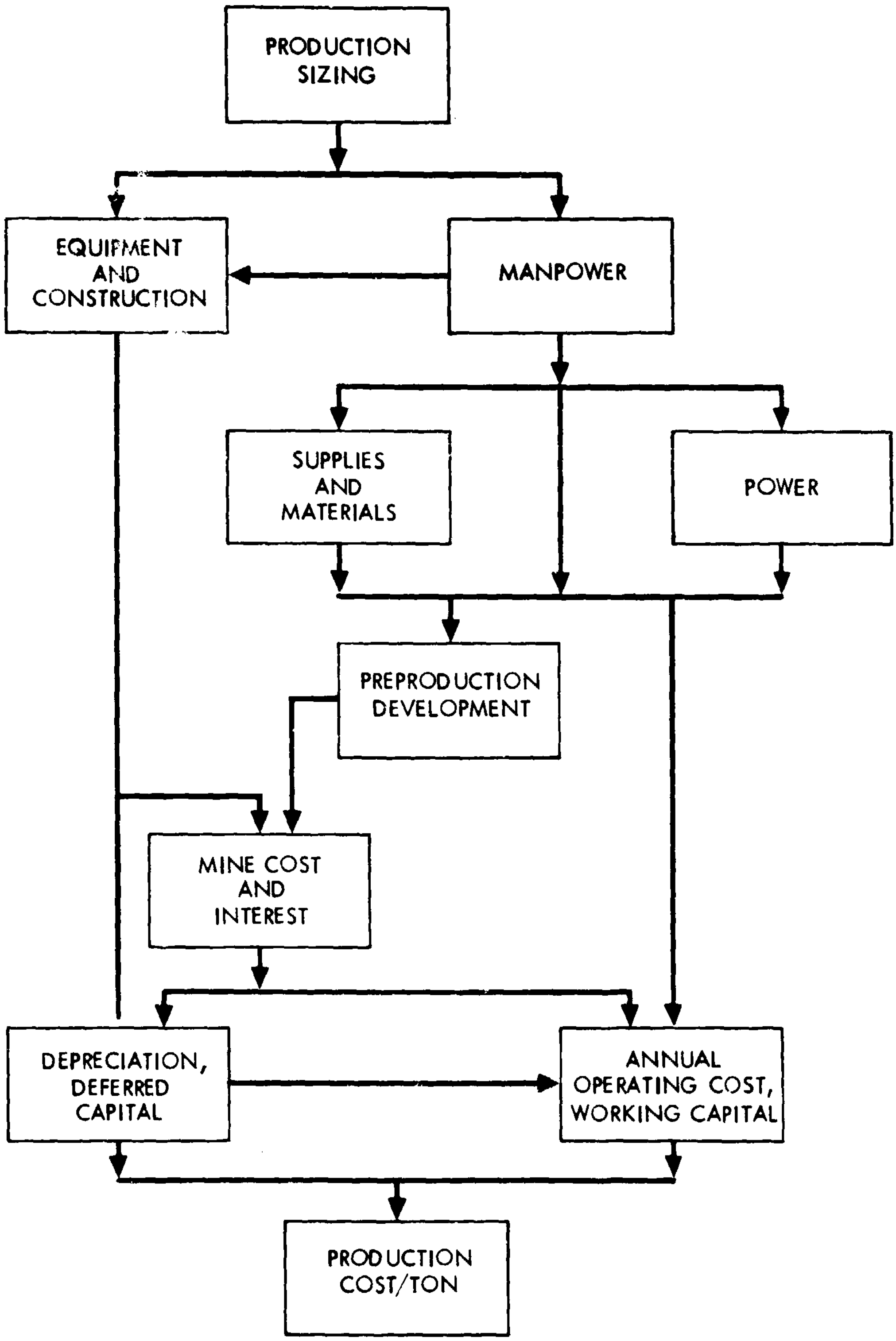


Figure 4-1. Information Flow Diagram for the NUS Underground Mine Cost Model (see Ref. 52)

the study focused on Eastern Kentucky where most mines are the drift-type (see Ref. 52). A typical mine life of 20 years not including construction and initial development was used. All cases were assumed to work three shifts per day and 220 days per year, and to have a preparation plant. In order to calculate the recovery factor for each case, a representative segment of a typical mine configuration was used. This segment included the main headings, a set of cross headings, and an appropriate number of production panels. By assuming 100% recovery from areas that were excavated, the recovery factors of Table 4-1 were achieved. Recovery factors were used in the model to calculate mine seam block size and to identify some equipment requirements.

Table 4-1. Seam Recovery Factors for Study Cases*

Technology	Recovery Rate %	
	1980	2000
Room-and-Pillar	43	54
Longwall	76	76
Shortwall	69	69

*The change in recovery from 1980 to year 2000 for room-and-pillar was caused by variations in entry and crosscut centerline dimensions and geometrical considerations of the mine plan.

The reject percent, or percent of waste material in the raw product varies with the mining system (see Ref. 52). NUS Corporation identified the reject percent to be 25% for continuous room-and-pillar and 21% for longwall. The reject percent for shortwall was assumed to be 25% because the shortwall systems use the same coal winning machine as room-and-pillar.

Design capacity, in terms of clean tons, was developed through the combination of shifts per day, days per year, productivity, reject percent, and number of machine units. The number of machine units for all room-and-pillar cases was assumed to be four. The number of units for the longwall and shortwall cases was varied, however. Each longwall and shortwall case was assumed to have one panel extraction unit. An appropriate number of panel development units, based upon the relative speed of panel development and extraction, was selected

Table 4-2. Number of Machine Units per Study Case

Technology	Number of Units	
	Average Conditions	Ideal Conditions
Room-and-Pillar		
1980	4	4
2000	4	4
Longwall		
1980	4	3
2000	4	3
Shortwall		
1980	4	3
2000	4	3

Table 4-3. Design Capacity of Study Cases

Technology	Capacity (clean tons)	
	Average Conditions	Ideal Conditions
Room-and-Pillar		
1980	574,200	1,346,400
2000	1,108,800	3,049,200
Longwall		
1980	1,136,652	2,309,802
2000	1,459,920	2,768,634
Shortwall		
1980	925,650	1,866,150
2000	1,113,750	1,999,800

also. Table 4-2 presents the number of machine units per case, and Table 4-3 identifies the resulting design capacities. Further details concerning design capacity calculations are found in Appendix C.

2. Financial Aspects

The input variables within this category were held constant for all study cases. The tax rates were selected at 48% for Federal taxes and 2% for state and local taxes combined. A 15% rate of return was used. A 100% equity investment was assumed.

C. MANPOWER

This segment provided the requirements for production section labor, support labor and salaried personnel based upon the number of production sections identified in the Production Sizing segment. Also, the cost for labor and salaried personnel required identification for input.

The salaried personnel requirements for each study case are shown in Appendix C. The job categories listed there demonstrate the basic differences from one system to another. In all 2000 cases, a control system engineer was provided because of equipment changes between the 1980 and 2000 cases. The input required for the NUS model was the number of salaried personnel and the average annual salary. Table 4-4 shows the personnel requirements for each case. The different salary requirements for the cases are related to capacity variations. The average salaries for the 2000 cases were escalated from their 1980 dollar values calculated in Appendix C to their 2000 dollar values using the cost update factor found in Appendix D.

Appendix C also contains the hourly labor requirements for each study case. Perusal of each case identifies the job category changes required between cases. Three different segments of the labor force were used: surface, underground general, and underground production crews. The number of support laborers (surface and underground general) was varied with respect to design capacity. Also, the number of production crew members changed. These hourly labor changes correspond to equipment and system configuration differences. In addition to requirements, Appendix C also provides average hourly labor costs (dollars per person per day). These costs were used to develop the manpower cost update factors found in Appendix D. The input of such factors into the model escalated the base 1975 dollar values of the model to the appropriate 1980 and 2000 dollar values. Table 4-4 also presents the hourly labor requirements as extracted from Appendix C.

D. EQUIPMENT AND CONSTRUCTION

This segment of the model provided the equipment and construction requirements, and associated capital costs. Development capital costs as well as production capital cost were computed. The transition between the development period and the production period in the NUS model occurs at the point in time when the last production unit is entered into the mine. The capital cost categories included

Table 4-4. Salaried and Hourly Personnel Requirements for Study Cases

Technology	Average Conditions		Ideal Conditions	
	Salaried	Hourly	Salaried	Hourly
Room-and-Pillar				
1980	53	222	69	288
2000	67	270	96	428
Longwall				
1980	66	293	83	353
2000	70	318	89	392
Shortwall				
1980	59	257	72	317
2000	68	285	75	323

production section equipment, haulage system auxiliary equipment, site preparation and construction, exploration, mine abandonment, and miscellaneous items.

The logic of the NUS model proved to be quite beneficial in the generation of capital requirements and the computation of capital costs. Only the production section equipment requirements and costs needed to be separately input. The remaining categories were handled internally by the model subject to the inputs of Production Sizing.

The production section equipment requirements and costs are itemized in Appendix C for all study cases. Most 1980 equipment costs were supplied by an internal staff effort that updated 1975 NUS costs to their 1980 values (Ref. 53). The 1980 costs were obtained from manufacturer quotations. Other sources of equipment costs are reported in Appendix C. Table 4-5 provides the capital cost for each particular machine unit used in this study. The unit costs for the 2000 cases are shown as 1980 dollars in Table 4-5. However, all equipment costs were escalated with the cost update factors of Appendix D by the model.

E. SUPPLIES AND MATERIALS

This model segment provided the supplies and materials cost per clean ton for each study case. The cost computation was based on an equation developed by analysis of data from a large number of mines (see Ref. 52). The equation relates supplies and materials cost to labor costs. One equation was developed for all current technologies.

Table 4-5. Machine Unit Costs in 1980 Dollars*

Unit	Capital costs (Nearest thousands of dollars)
Longwall unit	
1980	$5,411
2000	$6,935
Shortwall unit	
1980	$2,445
2000	$2,640
Continuous miner units	
R&P: 1980	$1,126
2000	$1,935
Longwall and Shortwall Development:	
Average conditions	
1980	$1,057
2000	$1,836
Ideal conditions	
1980	$1,107
2000	$1,836

*NOTE: Cost variations occur between the ideal and average conditions sets for panel development because component needs change for roof bolters, auxiliary fans, trickle dusters, and panel conveyors.

F. POWER

Although the NUS model had provisions to calculate power cost per clean ton through a labor cost approach similar to supplies and materials cost, an alternative technique suggested by NUS and used by others was adopted (see Ref. 52, and Refs. 54 and 55). Basically, the approach identified the power-consuming components and their horsepower requirements, then combined the total horsepower of each component category with an estimated operating time per day to produce the power requirements in kilowatt-hours per day. The multiplication of the daily power requirements with the operating days per year and an assumed power cost provided the annual power cost. From that point, use of the annual design capacity in clean tons identified power cost per clean ton. The resultant costs are displayed in Table 4-6.

Estimates of operating times were developed from Appendix B. Further information concerning the inputs to the power costs can be found in Appendix C. The power costs for the 2000 cases were escalated to 2000 dollars by the NUS model with the power update factor found in Appendix D.

G. PREPRODUCTION DEVELOPMENT

Preproduction development refers to the period during which the initial segment of main arteries of the mine are being developed. The period commences when the first machine unit begins operation and terminates when the final machine unit starts production. This time period is important because within the financial analysis the decision was made to capitalize all costs incurred during development. The purpose of this model segment is to identify these costs.

In order to simulate the case studies of this effort, several inputs required identification. Included were the extent of the development period in years, the tonnage produced during the period, and the length of advance in the main heading and cross-headings during the period. All four parameters were computed with the aid of the system productivities and proper scheduling of machine units into an idealized mine plan layout. The layout is shown in Figure 4-2, and the resulting input values are presented in Appendix E.

Table 4-6. Estimated Power Cost per Clean Ton for Study Cases

Technology	Cost per clean ton (1980 dollars) Average Conditions	Ideal Conditions
Room-and-Pillar		
1980	0.81	0.39
2000	0.41	0.16
Longwall		
1980	0.37	0.18
2000	0.31	0.16
Shortwall		
1980	0.46	0.22
2000	0.40	0.21

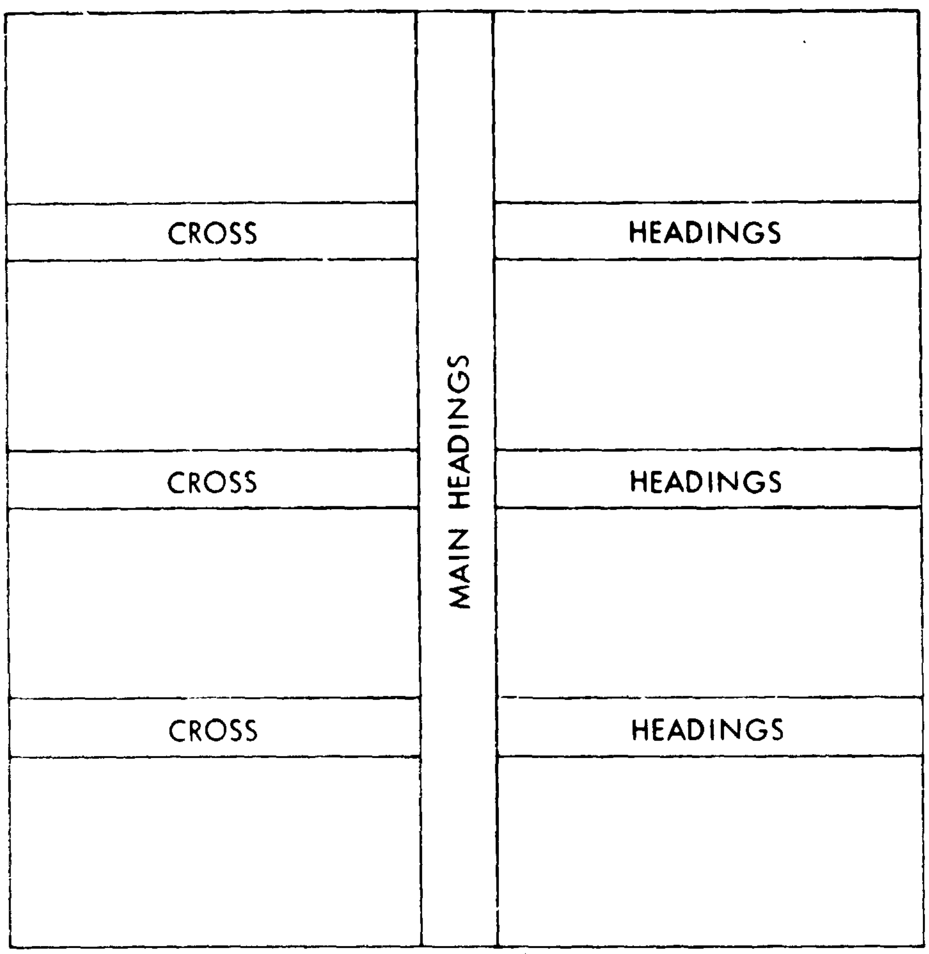


Figure 4-2. Idealized Mine Plan for Development Computations

H. OTHERS

As seen in Figure 4-1, there are four remaining segments of the model. Because none of these segments required direct input to them (they used information generated in previous segments) only a brief mention of their function will be made. The Initial Capital Investment segment uses previously calculated costs as a basis for computing the total initial capital investment. The determination of yearly depreciation charges and deferred capital investment for the life of the mine are the responsibility of the Deferred Capital Investment Depreciation segment. The straight-line method of depreciation was applied. The Annual Operating Cost, Working Capital segment includes two parts. The estimated costs of other segments are totalled and used to identify indirect and fixed costs that are functionally related to them. Also, the working capital requirement is estimated to be proportional to the annual operating costs minus depreciation. The final segment of the model, Production Cost, conducts a discounted cash flow analysis to determine the production cost per ton for a specified rate of return.

I. COST ANALYSIS RESULTS

The introduction of the aforementioned data into the NUS Model provided break-even production cost per clean ton for each study case. These costs were based on a 15% rate of return. Because the model computed the costs for the 2000 cases in 2000 dollars, they were deflated to 1980 dollars for comparison with the 1980 cases. The annual GNP deflators used to accomplish the transformation are shown in Appendix D. While the important output of the model is displayed in Appendix E for documentation purposes, the production cost results for each case are shown in Table 4-7.

Table 4-7. Production Cost per Clean Ton

Case	Cost per ton (1980 dollars)	
	Average Conditions	Ideal Conditions
Room-and-Pillar		
1980	$39.84	$22.59
2000	$26.66 (103.59)	$15.71 (61.02)
Longwall		
1980	$29.05	$17.50
2000	$25.71 (99.90)	16.48 (64.04)
Shortwall		
1980	$31.36	$18.53
2000	$29.41 (114.27)	$18.30 (71.11)

NOTE: Values in parentheses are in year 2000 dollars.

SECTION V

DISCUSSION OF RESULTS

As mentioned in previous report sections, the study approach was to fulfill two requirements. The 1980 cases were to provide a check to insure that the approach produced reliable results. The average conditions cases for the 1980 systems should provide results, both system productivity and production costs, that correspond well to current experiences. Secondly, the results of the year 2000 cases were developed to establish a measure of economic performance against which advanced system concepts will be compared. It is assumed, and quite possible, that these year 2000 systems will provide the competition to any advanced coal extraction system developed by the Jet Propulsion Laboratory through contract to the U.S. Department of Energy. In order to be competitive, the advanced system must at least match the economic performance of the industry workhorses at the time of its commercialization.

Because difficulty will be encountered in trying to estimate the effects of mining conditions on the productivity of new concepts, comparison with future competitors should be based on ideal mining conditions; hence, the year 2000 ideal conditions cases. However, should an attempt be made to establish performance levels of conceptual systems as conditions deteriorate, any comparison should consider both the average and ideal conditions cases of the extrapolated 2000 systems.

Before further discussion is presented concerning the results, two major limitations of the study should be noted. First, the data required for the production analysis approach were not easily available. Secondly, the data that were available did not always appear in consistent usable form. Some data, therefore, had to be modified. Finally, the seam height assumption proved to be quite significant. Because the "bottom-up" approach required selection of a seam height, the results of the entire study are only applicable to a 6-ft seam. Before a conceptual comparison is made, this limitation must be acknowledged or eliminated. It is suggested that similar studies be initiated for other representative seam heights.

A review of the literature showed a close correlation between the productivity values of other studies and the 1980 average conditions cases. One study that analyzed 326 continuous room-and-pillar systems, established an average productivity of 281 tons per machine-shift (TPMS) with an average seam height of 63 in.; whereas the average conditions case in this study resulted in 290 TPMS for a 6-ft seam (Ref. 56). Other room-and-pillar studies presented similar results: 300 to 310 TPMS for a 6-ft seam (see Refs. 54, 55). It is quite evident from these comparisons that the productivity estimate for the 1980 average conditions room-and-pillar case is extremely realistic.

While moving baseline study estimated 830 TPMS for the 1980 average conditions longwall system, five double-ended ranging shearer faces working 6-ft seams, obtained a combined average productivity of 790 TPMS (see Ref. 26). Another longwall study calculated an average productivity of 900 TPMS for a 7.5 ft seam (see Ref. 20). Although the productivity result of baseline study compares well with these other study results, the sample size of the comparators is too small to judge the accuracy of the baseline estimate. It is hoped that future information-gathering will permit a sound judgment on the moving baselines.

Although available shortwall studies did not report productivity with respect to seam height, the 520 TPMS estimate of this study was close to the midpoint of the ranges reported - 200 to 980 TPMS (see Refs. 6, 57). Additionally, it was suggested that current shortwall productivity should not vary, appreciably, from panel development productivity (Ref. 58). The same holds for the baseline study - 450 TPMS for the shortwall panel development and 520 TPMS for the shortwall production unit. Again, the sample size of the comparators does not warrant a sound judgment as to the accuracy of the baseline estimate. However, as with the 1980 longwall case, the initial comparison is quite encouraging.

A comparison of study cost results, that includes a 15% return on investment, with current spot market prices and long-term contract prices, established an acceptable correlation. Current spot market prices range from 20 to 43 dollars per ton, producing an average value of $31.50 per ton (Ref. 59). Also, long-term contract prices are presently in the mid-to-high twenty dollar range (Ref. 60). The 1980 average conditions case costs for longwall ($29.05) and for shortwall ($31.36) reflect favorably. However, the 1980 R&P cost ($39.84) does not. This discrepancy can be easily justified, however. The value represents the selling price requirement for a new room-and-pillar mine including plant site, development openings, and preparation plant. The result is a high initial capital investment per annual ton at a rather low labor productivity (9.5 tons per worker-day). But, the labor productivity figure is close to that experienced today. A 1976 study of a hypothetical room-and-pillar mine established a selling price of $31.50 per ton in order to achieve a 15% return on investment (see Ref. 32). The investigators recognized then that underground mines were not achieving such a high realization for their coal. Several reasons for their discrepancy were given, and in all probability, apply here since their 1976 selling price escalated to 1980 dollars ($44.61), exceeds the value presented in this study ($39.84). Operating mines were either developed before inflation escalated capital investment items to their current levels, have lower mining costs, or may not be achieving a 15% return on investment.

Scrutiny of the break-even production cost per clean ton for the year 2000 cases indicates that longwall technology will produce coal most cheaply in average conditions, but lose its supremacy to continuous room-and-pillar as ideal mining conditions are approached.

This trend can be easily seen in Figure 5-1, where production cost is plotted against mining conditions. The assumption underlying this plot is that a linear relationship exists between production cost and degree of geological difficulty. The implication of Figure 5-1, considering that ideal mining conditions rarely exist in nature, is that longwall technology should be the comparator for advanced systems technology. However, if longwall technology does not match well with the general characteristics of a selected target resource, then other technologies should be given consideration. Such is the case for the coal fields of Eastern Kentucky where mine size is generally small (less than 200,000 tons per year) and the lateral extent of coal blocks may not be appropriate for longwall technology. Therefore, attempts to develop new systems for Eastern Kentucky should recognize room-and-pillar technology as the probable competitor.

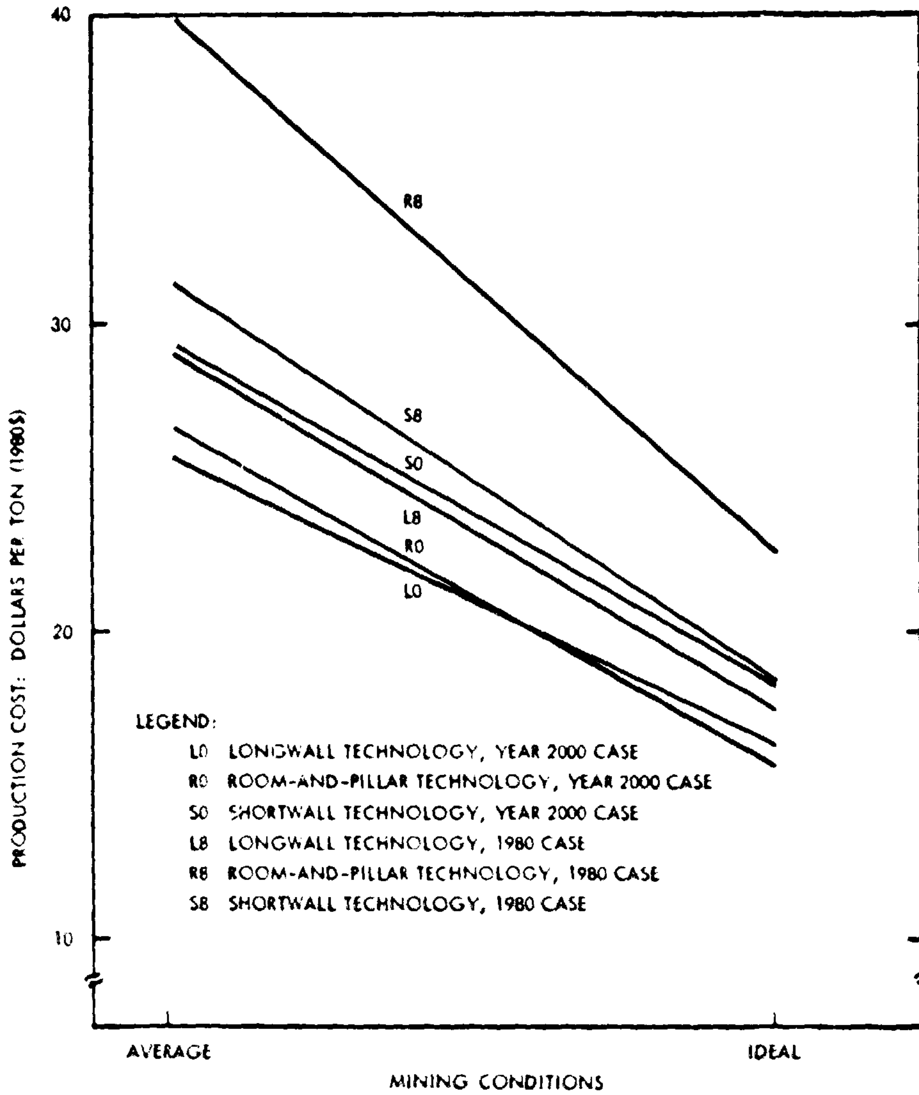


Figure 5-1. Production Cost per Clean Ton Versus Mining Conditions for Room-and-Pillar, Longwall, and Shortwall

SECTION VI

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APPENDIX A

MINING CONDITIONS DESCRIPTIONS

The purpose of Appendix A, Mining Conditions Descriptions, is to provide the reader with a description of the geological setting for the twelve cases presented in this report. The descriptions were developed from reports that discuss the variability of mining conditions and the effect the variability has on selected underground coal mining technology. The mining conditions were not used in any way to determine the value of equipment performance parameters or the mode of mining function execution. The determination of these study inputs is discussed in Section III, Production Analysis.

CONDITIONS DESCRIPTIONS

LONGWALL | IDEAL CONDITIONS

PANEL DEVELOPMENT: Description developed from information found in unpublished manuscript of Robert Stefanko, Professor of Mining Engineering, Pennsylvania State University, University Park, Pennsylvania, 1977.

- o 6-ft seam
- o Smooth, hard, dry floor with grades less than 1%
- o Roof bolts in 4 ft x 4 ft or 5 ft x 5 ft pattern with 4-ft bolts, no falls, entries to 20-ft wide
- o No methane build-up at face with minimum ventilation requirements (3000 cfm)
- o Coal easily cut by continuous miners, no special bit lacing or angles required
- o Moderate seam depth - 400 to 800 ft
- o Damp floor, but no standing water mobile equipment (rubber-tired and crawler-mounted) movement does not deteriorate floor conditions, no dust problems, crawler traction ideal

LONGWALL PRODUCTION: Developed from information in several references.

1. COMINEC, Conceptual Design of an Automated Longwall Mining System, USBM Final Report, Contract No 50241051, NTIS PB-263213, April 1976.
2. Kuti, J., "Longwall vs. Shortwall Systems," paper presented at AMC Annual Coal Convention, May 1975.
3. Stefanko, R., as before.

- o 6-ft seam
- o Floor is hard, stable, and level
- o Roof rock stays in place over supports; no cavities or caving at face; roof interface is clean, no transition; caving is immediately behind supports
- o No methane build-ups at face, in gateroads or in gob, with minimum ventilation requirements
- o Coal is cut cleanly and easily by winning machine; no face sloughing; no partings or other impurities; no discontinuities
- o Moderate seam depth - 400 to 800 ft
- o Drainage water dampens bottom, but no standing water, no floor degradation, conveyor and support advance not hindered

PANEL DEVELOPMENT: Same reference as PANEL DEVELOPMENT, IDEAL CONDITIONS.

- 6-ft seam
- Weak shale or hard fireclay occasionally interferes with equipment operations, ruts develop with regular use, grades to 7%, possibly slippery, occasional steep rolls
- Roof bolts on 4 ft x 5 ft to 5 ft patterns with long boits (6 ft to 10 ft) infrequent minor falls
- Occasional methane build-ups with properly designed ventilation system
- Coal easily cut, partings to 1-ft thick that may degrade performance of winning machine, other impurities occasionally encountered, infrequent discontinuities
- Moderate seam depth - 400 to 800 ft
- Drainage water collects in pools to depths of 6 in., pools sporadically located, occasional need for sumps

LONGWALL PRODUCTION: Same references as LONGWALL PRODUCTION, IDEAL CONDITIONS.

- 6-ft seam
- Weak shale or hard fireclay floor which adversely impacts conveyor and support advance when it softens from water presence, supports may punch into floor if roof does not cave properly or where roof is considered heavy, grades to 7%, possibly slippery, occasional steep rolls
- Roof does not always cave directly behind supports, infrequent cavities over supports, minor caves along face and in gateroads
- Occasional methane build-ups with properly designed ventilation system
- Easily cut coal with partings to 1-ft thick that degrade performance of winning machine, occasional impurities, interfaces are transitional, infrequent discontinuities
- Moderate seam depth - 400 to 800 ft
- Drainage water collects in pools to depths of 6 in., occasional need for sump in gateroads

SHORTWALL — IDEAL CONDITIONS

PANEL DEVELOPMENT: Same as LONGWALL, IDEAL CONDITIONS description.
SHORTWALL PRODUCTION: Developed from information in two references.

1. Kuti, J., as before
2. Stefanko, R., as before

- 6-ft seam
- Floor is hard, stable, and level; sandstone, hard fireclay or shale
- Roof rock stays in place after exposure; no cavities or caving at face; roof interface is definite, no transition; immediate caving at rear of supports
- No methane build-ups at face with minimum ventilation requirements (3000 cfm)
- Coal is cut cleanly and easily by winning machine; no face sloughing; no partings, impurities, or discontinuities
- Moderate seam depth - 400 to 800 ft
- Drainage water dampens bottom; no standing water; no interference with support advance or equipment movement

SHORTWALL — AVERAGE CONDITIONS

PANEL DEVELOPMENT: Same as LONGWALL, AVERAGE CONDITIONS description.

SHORTWALL PRODUCTION Same references as SHORTWALL PRODUCTION, IDEAL CONDITIONS.

- 6-ft seam
- Weak shale or fireclay floor which adversely impacts equipment movement when rutted; equipment movement degrades floor condition; grades to 7%; possibly slippery, occasionally steep rolls
- Minor roof caves along face and in gateroads; infrequent cavities over supports; roof caves behind supports most times
- Occasional methane build-ups with properly designed ventilation system
- Coal is easily cut; partings to 1-ft thick that may degrade performance of winning machine; occasional impurities and discontinuities; coal-rock interface is transitional
- Moderate seam depth - 400 to 800 ft
- Drainage water collects in poos to depths of 6 in., pools sporadically located, occasional need for active sumps in gateroads

ROOM-AND-PILLAR

Conditions descriptions correspond to those used for longwall and shortwall panel development.

APPENDIX B

PRODUCTION ANALYSIS

The purpose of Appendix B, Production Analysis is to present the approach and calculations used to determine productivity in terms of raw tons per machine-shift for the systems selected in the moving baseline study. Initially, the available productive time for all study cases is established. Secondly, the information and sources of that information used to estimate the system availabilities for the study are identified. The calculations that produce the availabilities are shown also. Finally, the available productive time and system availabilities are combined with appropriate calculations to determine the system productivities.

CONTENTS

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I. AVAILABLE PRODUCTIVE TIME

Available Productive Time = APT

APT = shift time - inherent delays

where: shift time = 8 h = 480 min

Inherent delays = 123 min

o	Travel-in	40 min
o	Safety meeting	3 min
o	Prepare to start	10 min
o	Lunch	30 min
o	Prepare to leave	10 min
o	Travel out	30 min

APT = 480 - 123 = 357 min

This value is used for all cases.

II. SYSTEM AVAILABILITY

$$A = \frac{\text{net work time}}{\text{APT}} = \frac{\text{NWT}}{\text{APT}}$$

NWT = APT - other delays

Other delays: For average conditions cases, other delays include maintenance time, unexpected operational delays, mine-environment-related delays, and human-related delays.

For ideal conditions cases, the delays considered directly related to the mine environment were factored out.

In most cases, the values for NWT and APT were developed from actual operational data found in publications.

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CONTINUOUS ROOM-AND-PILLAR CASES

(Also applies to lor wall and shortwall panel development cases)

Information for availability was extracted from: Frants, R. L., and R. H. King, Study of the Human Factor Aspects of an Automated Continuous Mining System, Final Report USBM Grant No. S0144115, March 1977.

Shift time = 480 min

APT = 320 minutes

Delays:

Average conditions = 160 min

Ideal conditions = 120 min

NWT_a = 160 min

NWT_i = 200 min

$$A_a = \frac{NWT_a}{APT} = \frac{160}{320} = 0.50$$

$$A_i = \frac{NWT_i}{APT} = \frac{200}{320} = 0.63$$

LONGWALL CASES

Information for availability was extracted from:

1. Herhal, A. J., et al., Longwall Conveyor System Study, Final Report, Contract No. U.S. DOE ET-77-C-01 8915(2), June 1978.

2. Curry, K. C., et al., Longwall Mine Availability and Delay Analysis, Jet Propulsion Laboratory Report No. 5030-46, December 1976.

$APT = 517{,}725$ min

$NWT_a = 292{,}143$ min

$NWT_i = 332{,}678$ min

$$A_a = \frac{NWT_a}{APT} = \frac{292{,}143}{517{,}725} = 0.56$$

$$A_i = \frac{NWT_i}{APT} = \frac{332{,}678}{517{,}725} = 0.64$$

SHORTWALL CASES

Information for availability was extracted from Curry, K. C., et al., Shortwall Mine Availability and Delay Analysis, JPL Internal Report No. 5030-47, December 1976.

$APT = 128{,}505$ min

$NWT_a = 77{,}205$ min

$NWT_i = 84{,}770$ min

$$A_a = \frac{NWT_a}{APT} = \frac{77{,}205}{128{,}505} = 0.60$$

$$A_i = \frac{NWT_i}{APT} = \frac{84{,}770}{128{,}505} = 0.66$$

A. 1980 LONGWALL AND SHORTWALL CASES

TONS PER MACHINE-SHIFT

1980 LONGWALL AND SHORTWALL PANEL DEVELOPMENT

IDEAL CONDITIONS

Cycle time: C

		min
o One sump cycle		
- Sump in ½ drum diameter (18 in.) at 17 fpm	=	0.088
- Shear down 36 in. at 34 fpm	=	0.088
- Trim floor 18 in. at 17 fpm	=	0.088
- Reposition drum 36 in. at 34 fpm	=	0.088
TOTAL	=	0.352 ≈ 0.35 min

		min
o One 40-ft lift cycle		
- 20-ft advance, 20 ft x (0.35 min/1.5 ft)	=	4.67
- Reposition: move 60 ft at 60 fpm	=	1.00
- 30-ft advance, 30 ft x (0.35 min/1.5 ft)	=	7.00
- Reposition: move 30 ft at 60 fpm	=	0.50
- 20-ft advance	=	4.67
- Reposition: move 30 ft at 60 fpm	=	0.50
- 10-ft advance	=	2.33
- Move to another face, amortized average place-change distance of 211 ft per lift; 211 ft at 60 fpm	=	3.50
TOTAL	=	24.17 min

- o Amortized section move-up time per 40 ft lift
  - 10 lifts per move-up
  - 1 h to accomplish a move-up
  - 6.0 min per lift
- o Cycle time per 40-ft lift
  24.17 min + 6.0 min = 30.17 min

Tons per cycle: T

- o 10 lifts per move
- o Total tons per move
  - 1927.8 tons
- o Average tons per lift
  - 192.8 tons

Shift production: TPMS

- o Tons per machine shift = $\frac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.63
  - T = 192.8 tons per cycle
  - C = 30.17 min per cycle

$$TPMS = \frac{357 \times 0.63 \times 192.8}{30.17} = 1437.28 \ (1440)$$

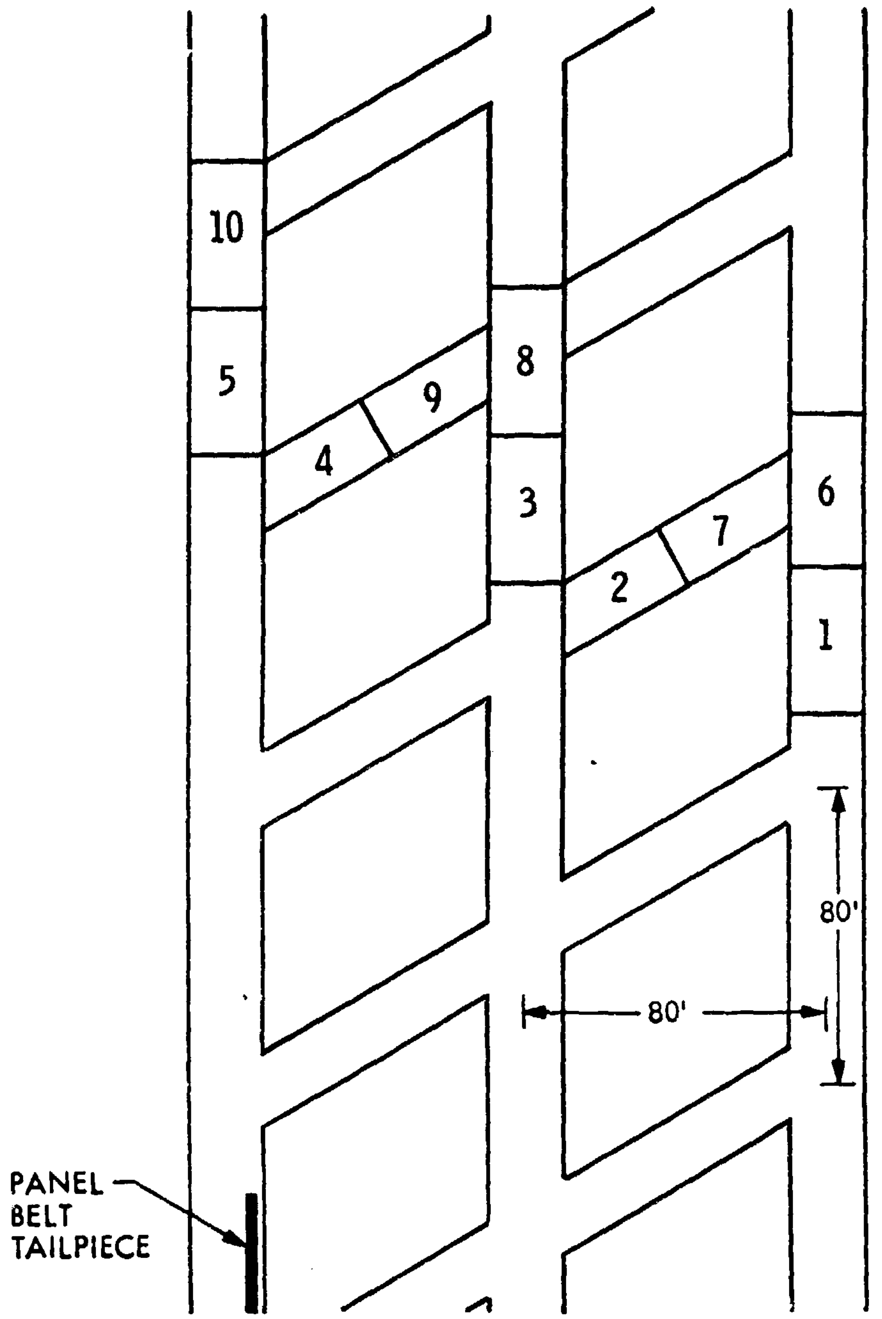


Figure B-1. Lift Sequence for 1980 Longwall and Shortwall Panel Development in Ideal Conditions

AVERAGE CONDITIONS

Cycle time: C

- o One sump cycle

	ft/min
- sump rate	5.66
- shear rate	10.50
- reverse sump rate	8.55
- reposition rate	27.50 (vertical distance)

		min
- sump in 1/2 drum diameter 18 in. at 5.66 fpm	=	0.27
- shear down 36 in. at 10.50 fpm	=	0.29
- cut cusp 18 in. at 8.55 fpm	=	0.18
- reposition drum 36 in. at 27.50 fpm	=	0.11
	TOTAL	0.85 min

- o One 20-ft long lift cycle
  - constrained by allowable roof span for mining conditions selected
  - 20-ft advance

$$20 \text{ ft} \times \frac{0.85 \text{ min}}{1.5 \text{ min}} = \underset{11.33}{\underline{\text{min}}}$$

- reposition move 60 ft at 30 fpm (average case tram speed assumed to be 1/2 ideal case)	=	2.00
- 20-ft advance		11.33
- place-change - average distance of 255.4 ft per 20-ft lift 255.4 ft at 30 fpm	=	8.50
	TOTAL	33.17

- o Amortized section move-up time per 20-ft lift
  - 20 lifts per move
  - 1.5 h to accomplish move
  - 4.50 min per lift
- o Total cycle time per 20-ft lift
  33.17 min + 4.50 min = 37.67 min

Tons per cycle: T

- o 20 lifts per move
- o Total tons per move
  - 1927.8 tons
- o Average tons per lift
  - 96.39 tons

Shift production: TPMS

- o Tons per machine shift = $\frac{APT \cdot A \cdot T}{C}$
- o Tons per machine-shift = C

  - APT = 357 min per machine-shift
  - A = 0.50
  - T = 96.39 tons per cycle
  - C = 37.67 min per cycle

  $$TPMS = \frac{357 \times 0.50 \times 96.39}{37.67}$$
  $$= 456.75\ (460)$$

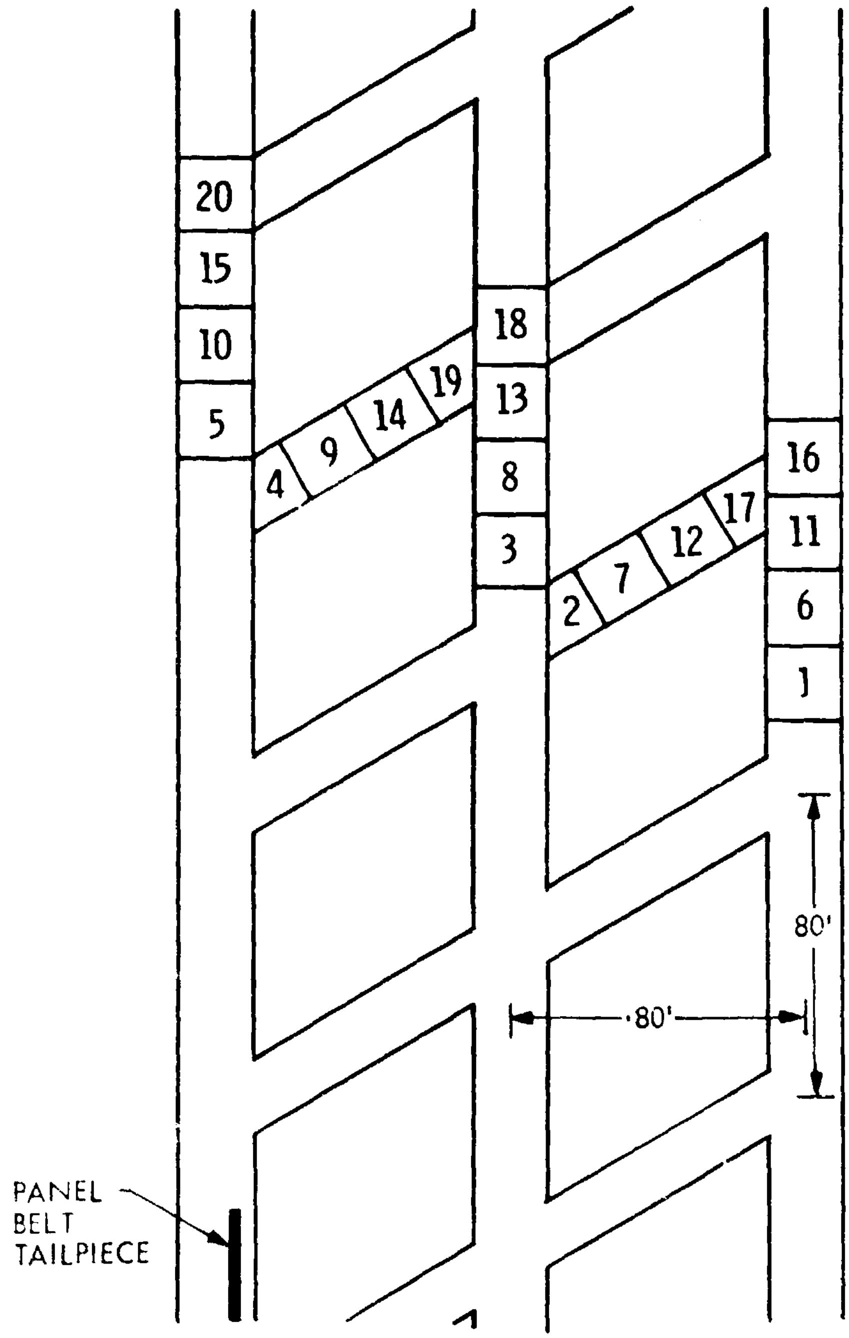


Figure B-2. Lift Sequence for 1980 Longwall and Shortwall Panel Development in Average Conditions

B. 2000 LONGWALL AND SHORTWALL PANEL DEVELOPMENT

IDEAL CONDITIONS

Cycle time: C

- o One sump cycle
  - same as 1980 case
  - 0.35 min per 1.5 ft lift

- o One lift

		min
- 20-ft advance	=	9.67
- reposition: 60 ft at 60 fpm	=	1.00
- 30-ft advance	=	7.00
- reposition: 30 ft at 60 fpm	=	0.50
- 20-ft advance	=	4.67
- reposition: 60 ft at 60 fpm	=	0.50
- 20-ft advance	=	4.67
- reposition: 30 ft at 60 fpm	=	0.50
- 20 ft advamce	=	4.67
- reposition: 30 ft at 60 fpm	=	0.50
- 20-ft advance	=	4.67
- reposition: 30 ft at 60 fpm	=	0.50
- 20-ft advance	=	4.67
- reposition: 30 ft at 60 fpm	=	0.50
- 10-ft advance	=	2.33
- place-change, average move distance of 238.34 ft 238.37 ft at 60 fpm	=	3.97
	TOTAL	45.32 min

- o Amortized section move-up time per 80-ft lift
  - 1.0 h to accomplish
  - 5 lifts per move
  - 12.0 min per lift

- o Cycle time per 80-ft lift
  45.30 min + 12.0 min = 57.30 min

Tons per cycle: T

- o 5 lifts per move
- o 1927.8 tons per move
- o 385.56 tons per cycle

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.63
  - T = 385.56 tons per cycle
  - C = 57.32 min per cycle

$$TPMS = \frac{357 \times 0.63 \times 385.56}{57.32} = 1512.85 \quad (1510)$$

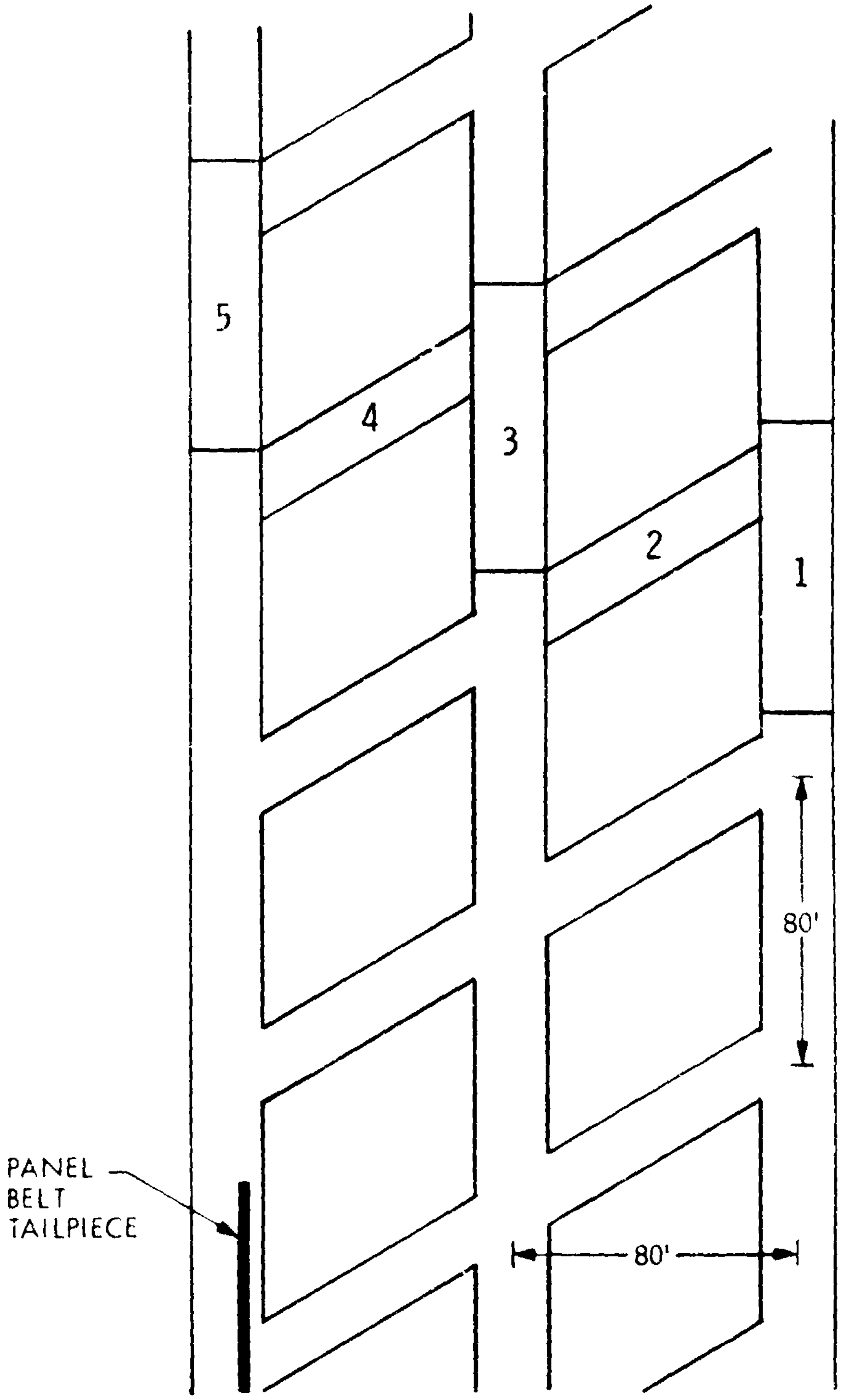


Figure B-3. Lift Sequence for 2000 Longwall and Shortwall Panel Development in Ideal and Average Conditions

AVERAGE CONDITIONS

Cycle time: C

- o One sump cycle
  - same as 1980 case
  - 0.85 min per 1.5 ft lift

- o One 80-ft lift

	min
- 20-ft advance	11.33
- reposition 60 ft at 30 fpm	2.00
- 30-ft advance	17.00
- reposition 30 ft at 30 fpm	1.00
- 20-ft advance	11.33
- reposition 30 ft at 30 fpm	1.00
- 20-ft advance	11.33
- reposition 30 ft at 30 fpm	1.00
- 20-ft advance	11.33
- reposition 30 ft at 30 fpm	1.00
- 20-ft advance	11.33
- reposition 30 ft at 30 fpm	1.00
- 20-ft advance	11.33
- reposition 30 ft at 30 fpm	1.00
- 10-ft advance	5.67
- place-change, average move distance 239.34 ft at 30 fpm	7.94
TOTAL	106.59 min

- o Amortized section move-up time per 80-ft lift
  - 5 lifts per move
  - 1.5 h to accomplish move
  - 18 min per lift
- o Cycle time per 80-ft lift
  106.59 min + 18.0 min = 124.59 min

Tons per cycle: T

- o 5 lifts per move
- o 1927.8 tons per move
- o 385.56 tons per cycle

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.50
  - T = 385.56 tons per cycle
  - C = 124.59 min per cycle

$$TPMS = \frac{357 \times 0.50 \times 385.56}{124.59} = 552.4 \quad (550)$$

C. 1980 LONGWALL PRODUCTION

IDEAL CONDITIONS

Cycle time: C

- Steps as per attached figure
  1. Shearer in position for sump
  2. Shearer sumped into face and travels to tailgate, conveyor snake finished at headgate
     - cut travel = 400 ft
  3. Range shearer drums, and reverse loading devices and shearer — 2.0 min
  4. Shearer flits towards headgate after finishing cut at tailgate, cuts remainder of coal at headgate, conveyor is snaked
     - cut travel = 20 ft + 150 ft
     - flit travel = 330 ft
  5. Set picks, range drums, and reverse loading devices and shearer — 7.0 min
  6. Shearer finishes cut at headgate, then flits to sump position
     - cut travel = 20 ft
     - flit travel = 80 ft

  TOTAL: cut travel = 590 ft; flit travel = 410 ft — 9.0 min

- Amortized headgate move time
  - 0.50 min per ft of advance
  - each cycle provides an advance of 33 in. (2.75 ft)
  - amortized headgate move time = 1.38 min

  $$2.75 \text{ ft} \times 0.50 \frac{\text{min}}{\text{ft}}$$

- Cycle time per 33-in. web

		min
- 590 ft of cut travel at 30 fpm	=	19.67
- 410 ft of flit travel at 50 fpm	=	8.20
- turn-arounds, etc.	=	9.00
- amortized headgate move time	=	1.38
TOTAL:		38.25 min

Tons per cycle: T

- One cycle: one pass across entire face
- Dimensions: 500-ft long x 6-ft high x 2.75-ft wide
- Tons: 350.63

  $$500 \text{ ft} \times 6 \text{ ft} \times 2.75 \times \frac{85\#}{\text{ft3}} \times \frac{1 \text{ ton}}{2000\#}$$

Shift production: TPMS

- Tons per machine-shift $= \dfrac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.64
  - T = 350.63 tons per cycle
  - C = 38.25 min per cycle

  $$TPMS = \frac{357 \times 0.64 \times 350.63}{38.25} = 2094.43 \quad (2090)$$

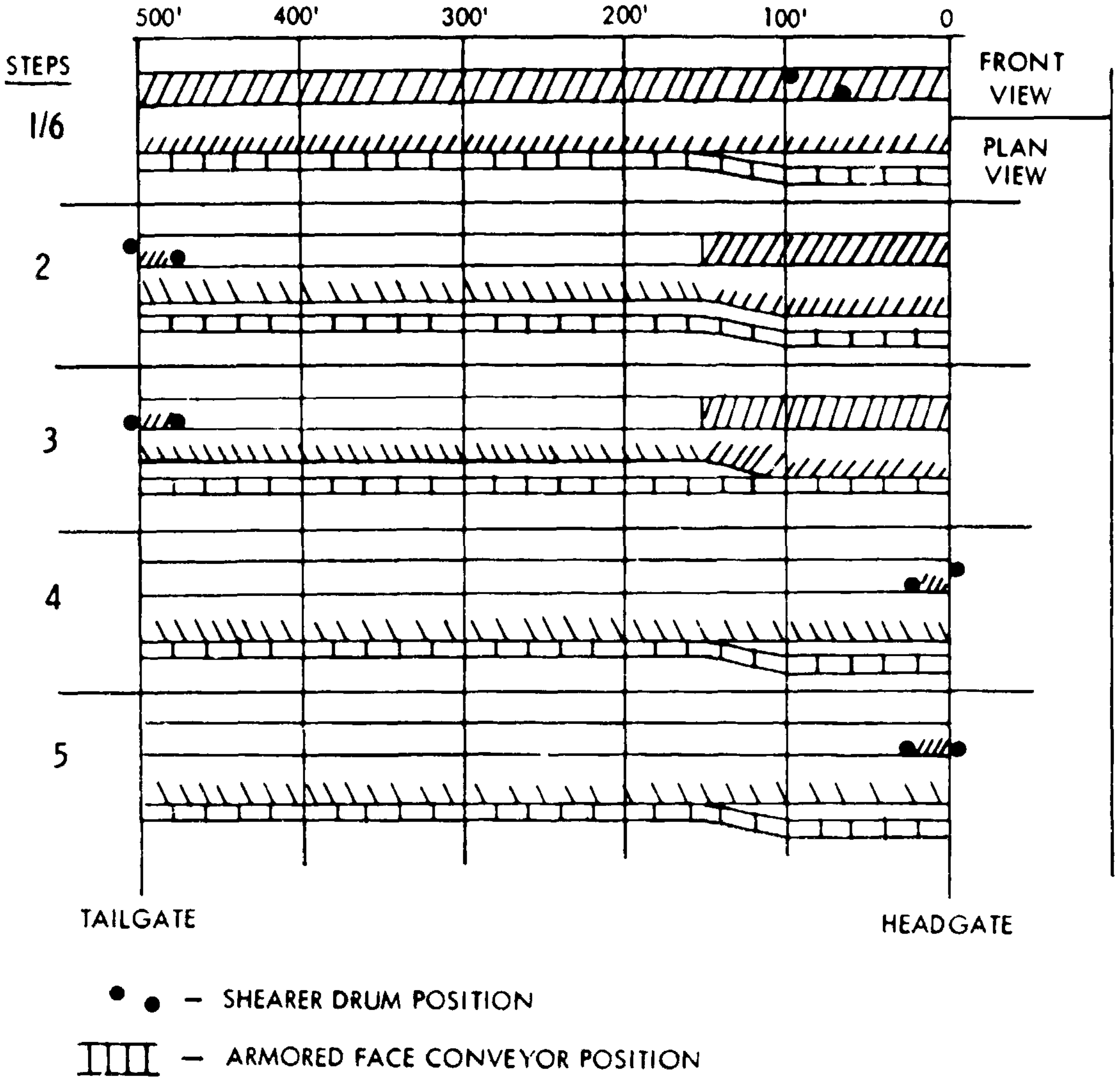


Figure B-4. Sequence of Steps for Shearer and Conveyor Movements During Removal of One Web With Half-face Operation

AVERAGE CONDITIONS

Cycle time: C

- o Same steps as "Ideal Case"

  Total: Cut travel = 590 ft
  Flit travel = 410 ft
  Turn-arounds, etc. = 9.0 min

- o Amortized headgate move time
  - 0.64 min/ft of advance
  - each cycle provides an advance of 33 in. (2.75 ft)
  - amortized headgate move time = 1.8 min

  $$2.75 \text{ ft} \times \frac{0.64 \text{ min}}{\text{ft}}$$

- o Cycle time per 33-in. web

		min
- 590 ft of cut travel at 11 fpm	=	53.6
- 410 ft of flit travel at 28 fpm	=	14.6
- turn-arounds, etc.....	=	9.0
- amortized headgate move	=	1.8
TOTAL:		79.0 min

Tons per cycle: T

- o Same as "Ideal Case"
- o 350.63 tons

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.56
  - T = 350.63 tons per cycle
  - C = 79.0 min per cycle

$$TPMS = \frac{357 \times 0.56 \times 350.63}{79.0} = 887.32 \quad (890)$$

D. 2000 LONGWALL PRODUCTION

IDEAL CONDITIONS

Cycle time: C

- o Steps as per attached figure
- o The attached Gantt Chart illustrates that the headgate shearer cycle time is in fact the cycle time for both shearers because of overlaps.

1. Headgate shearer is ready for sump into face, tailgate shearer has finished turn around and cut, and starts flit back to mid-face, snake of tailgate section of AFC is started.

2. Headgate shearer sumps into face at 300 ft and cuts to headgate cut travel: 300 ft
tailgate shearer follows into web, drums are ranged for its cut travel as conveyor snake is finished.

3. Headgate shearer drums ranged, picks checked, loading devices are reversed 7.0 min

4. Headgate shearer finished cut at gate, then flits to mid-face cut travel: 30 ft
flit travel: 270 ft
tailgate shearer finished cut and starts turn-around at gate.

5. Headgate shearer drums ranged, loading devices reversed in preparation for next cut, snake of headgate conveyor section finished 2.0 min
tailgate shearer finished turn-around and starts to finish cut. TOTAL: cut travel: 330 ft
flit travel: 270 ft
turn-arounds: 9.0 min

- o Amortized headgate move
  - 0.50 min/ft of advance
  - each cycle provides 3 ft of advance
  - move time = 1.5 min per cycle

- o Cycle time per 3-ft web

		min
- 330 ft at 30 fpm	=	11.0
- 270 ft at 50 fpm	=	5.4
- turn-around, etc.	=	9.0
- amortized headgate move	=	1.5
	TOTAL:	26.9 min

Tons per cycle: T

- o One cycle: 3-ft wide web across entire face
- o Dimension: 3-ft wide x 6-ft high x 500-ft long
- o Tons: 382.5

Shift production: TPMS

o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

- APT = 357 min per machine-shift
- A = 0.64
- T = 382.5 tons per cycle
- C = 26.9 min per cycle

TPMS: $= \frac{357 \times 0.64 \times 382.5}{26.9} = 3248.83$ (3250)

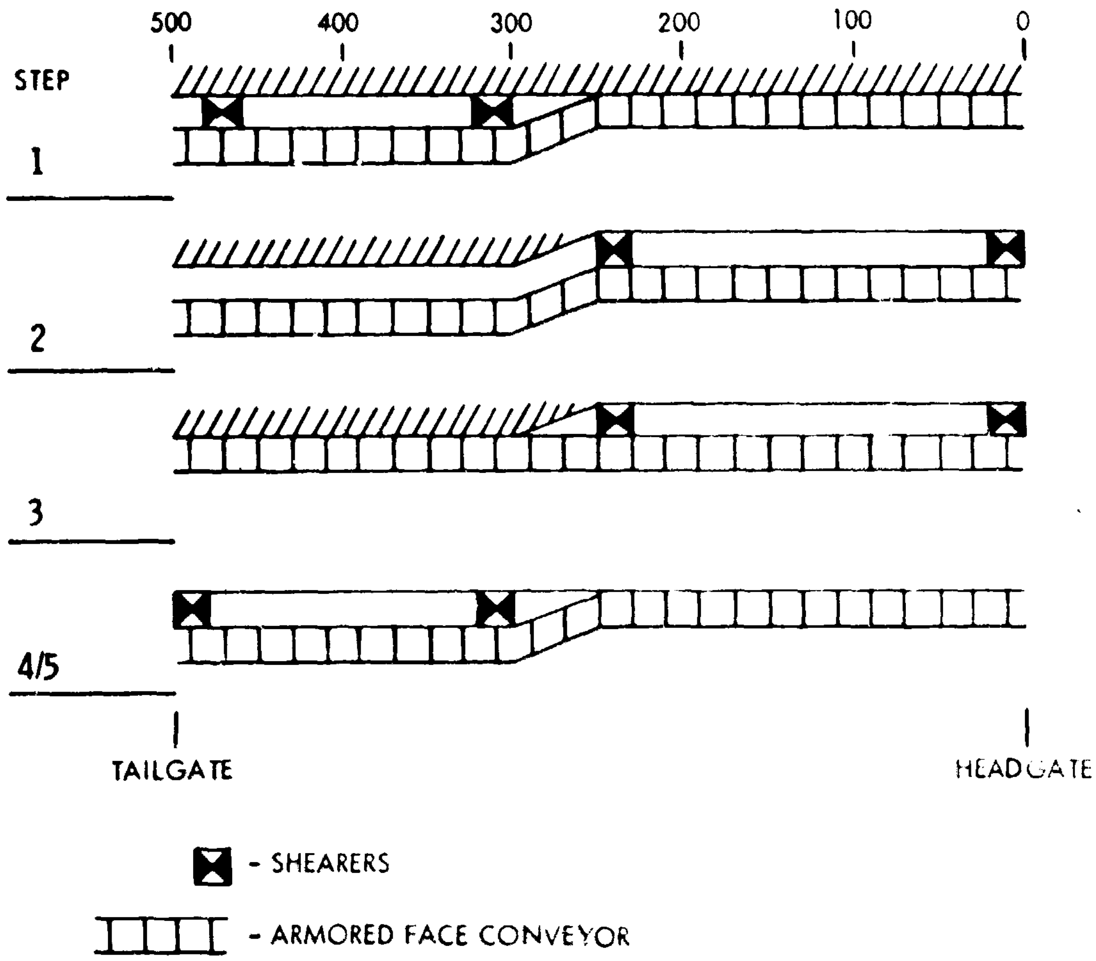


Figure B-5. Plan-view Illustrations of Cycle Steps for 2000 Longwall Systems

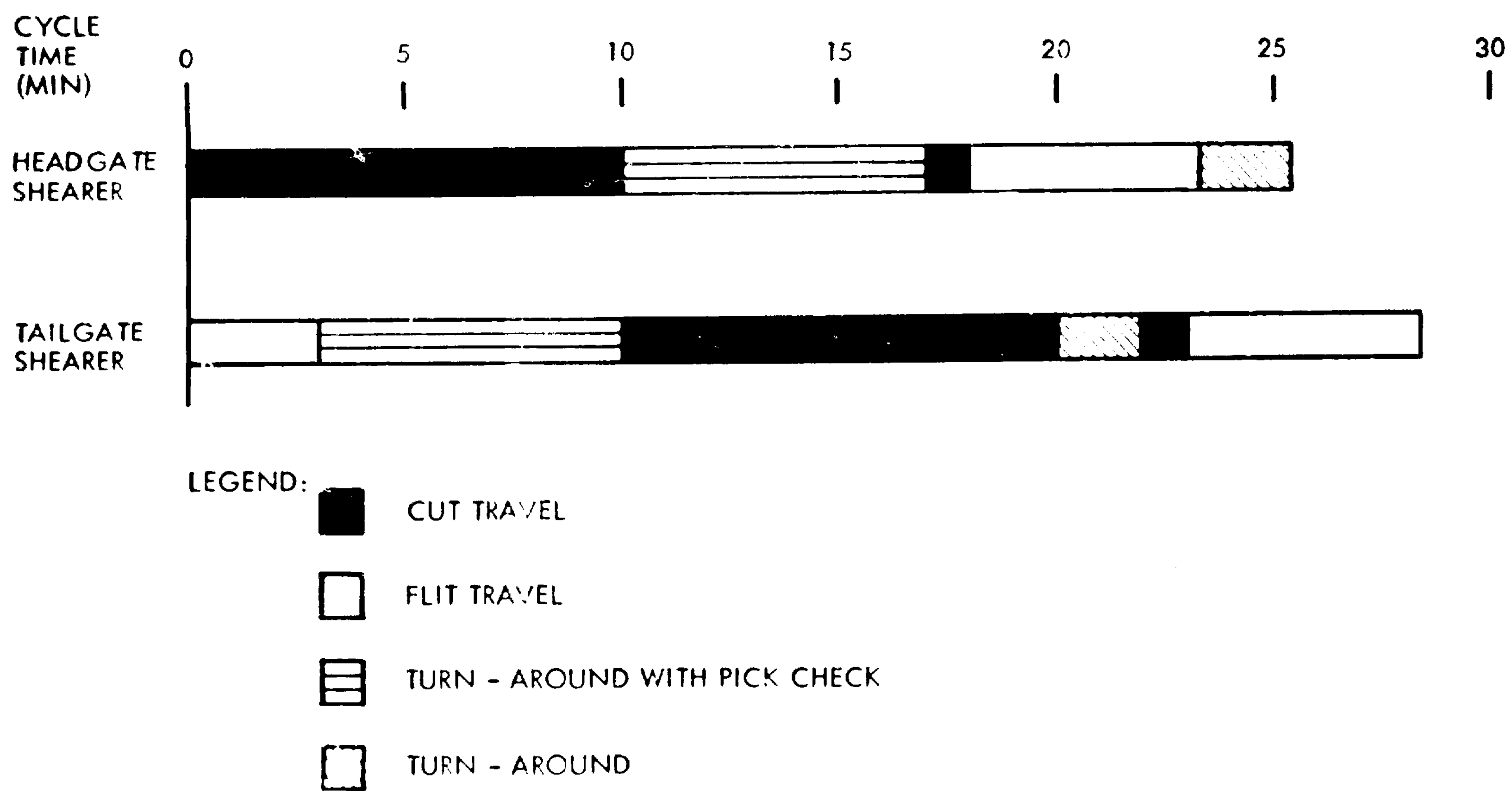


Figure B-6. Gantt Chart of Shearer Scheduling for 2000 Longwall System in Ideal Conditions

AVERAGE CONDITIONS

Cycle time: C

- o Same steps as "Ideal Conditions" case
- o The attached Gantt Chart shows that the shearers' cycle time is controlled by the sum of major cut travel time
- o Each shearer travels 300 ft during these periods
- o Amortized headgate move
  - 0.64 min/ft of advance
  - each cycle provides 3 ft of advance
  - move time + 1.92 min
- o Cycle time per 3-ft web

		min
- 600 ft of cut travel at 11 fpm	=	54.55
- amortized headgate more	=	1.92
	TOTAL	56.47 min

Tons per cycle: T

- o Same as "Ideal Conditions" case
- o 382.5 tons

Shift production: TPMS

- o Tons per machine shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.56
  - T = 382.5 tons per cycle
  - C = 56.47 min per cycle

$$TPMS = \frac{357 \times 0.56 \times 382.5}{56.47} = 1354.27 \quad (1350)$$

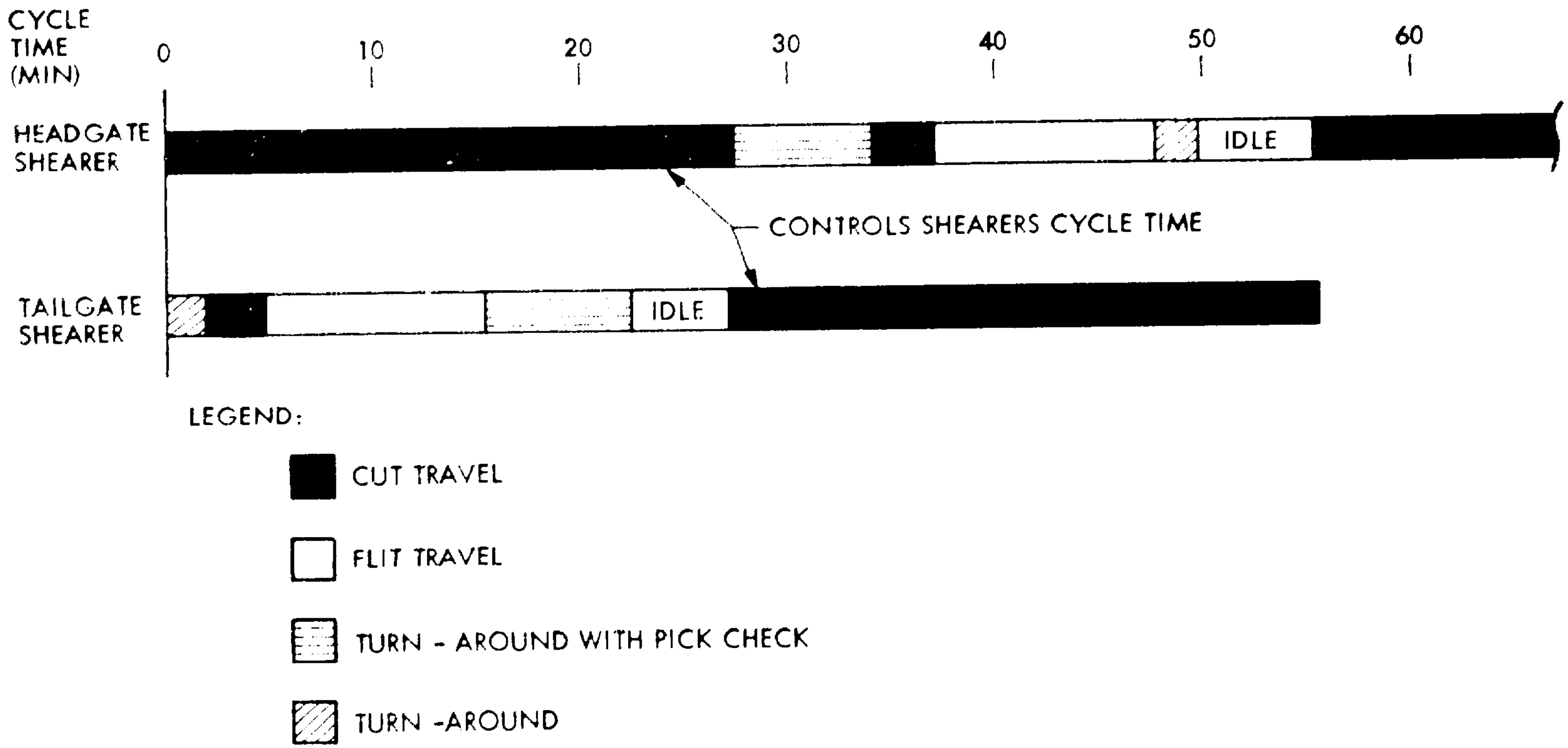


Figure B-7. Gantt Chart of Shearer Scheduling for 2000 Longwall System in Average Conditions

E. 1980 SHORTWALL PRODUCTION

IDEAL CONDITIONS

Cycle time: C

- o One sump cycle
  - same as 1980 Shortwall Development in "Ideal Conditions" case
  - 0.35 min per 1.5 ft advance
- o One 180-ft lift across face
  - $180 \text{ ft} \times \dfrac{0.35 \text{ min}}{1.5 \text{ ft}} = 42.0 \text{ min}$

  First support advance
  - supports advance 10 ft behind continuous miner
  - 10 ft + machine length (30 ft) is the amount of support advance required after machine starts its flit
  - 40 ft = 8 supports
  - support advance cycle is 30 s
  - 4.0 min of support advance before first guide rail advance is started

- o Continuous miner flit
  - under actual conditions, flit speed is 20 fpm
  - assumed twice average condition speed for ideal - 40 fpm
  - face is 180-ft long
  - 20 ft of clearance required in headgate (assumed)

  (180 ft + 20 ft)/40 fpm = 5.0 min

- o Guide rail advance
  - two advances required, one after each support advance
  - 15 s per ram
  - 1 ram per 3 supports
  - 13 rams x 15 s per ram = 3.25 min

- o Second support advance
  - 40 supports
  - 30 s per support
  - 20 min to advance wall

- o Amortized section move-up time per 10 ft of advance
  - one move every 80 ft
  - 8 lifts (10 ft each) every move
  - 1 h to accomplish move
  - 7.5 min per lift

- o Cycle time per 10-ft wide lift
  - face equipment cycle time is 72.5 min as per attached Gantt Chart

		min
- face equipment cycle	=	72.5
- amortized move-up	=	7.5
	TOTAL:	80.0 min

Tons per cycle: T

- o One cycle: 10-ft wide lift across entire face
- o Dimensions: 180-ft long x 6 ft-high x 10 ft-wide
- o Tons: 459

$$180 \text{ ft} \times 6 \text{ ft} \times 10 \text{ ft} \times \frac{85\#}{\text{ft3}} \times \frac{1 \text{ ton}}{2000\#}$$

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.66
  - T = 459 tons per cycle
  - C = 80.0 min per cycle

$$TPMS = \frac{357 \times 0.66 \times 459}{80.0} = 1351.87 \quad (1350)$$

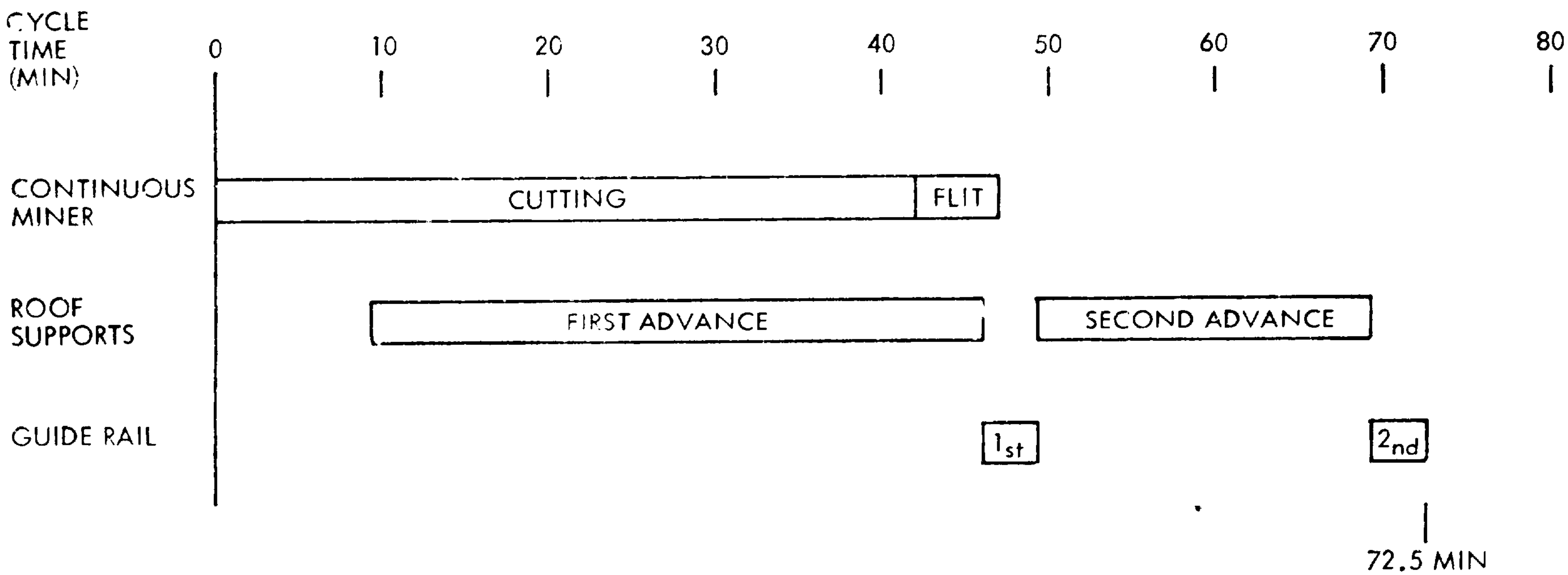


Figure B-8. Gantt Chart of Face Equipment Scheduling for 1980 Shortwall System in Ideal Conditions

AVERAGE CONDITIONS

Cycle time: C

- o One sump cycle
  - same as 1980 shortwall development in "Average Conditions" case
  - 0.85 min per 1.5-ft advance
- o One 180-ft lift across face

  $$180 \text{ ft} \times \frac{0.85 \text{ min}}{1.5 \text{ ft}} = 102.0 \text{ min}$$
- o First support advance
  - supports advance 10 ft behind continuous miner
  - 10 ft + 30 ft (machine length)
  - 40 ft = 8 supports
  - support advance cycle is 45 s for average conditions
  - 6.0 min of support advance before first guide rail advance is started
- o Continuous miner flit
  - speed is 20 fpm
  - distance is the same as "Ideal Conditions," (180 ft + 20 ft)/20 fpm = 10.0 min
- o Guide rail advance
  - two advances required, one after each support advance
  - assumed guide rail advance rate in average conditions is twice that of ideal conditions

  $$3.25 \frac{\text{min}}{\text{advance}} \times 2 = 6.5 \frac{\text{min}}{\text{advance}}$$
- o Second support advance
  - 40 supports
  - 45 s per support
  - 30 min to advance wall
- o Amortized section move-up time per 10 ft of advance
  - one move every 80 ft
  - 8 lifts every move
  - 1.5 h to accomplish move
  - 11.25 min per lift
- o Cycle time per 10-ft wide lift
  - face equipment cycle time per attached Gantt Chart

		min
- face equipment cycle		161.0
- amortized move-up		11.25
	TOTAL	172.25 min

Tons per cycle: T

- same as "Ideal Conditions" case
- 459 tons per cycle

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.60
  - T = 459 tons per cycle
  - C = 172.25 min per cycle

  $$TPMS = \frac{357 \times 0.60 \times 459}{172.25} = 570.79 \quad (570)$$

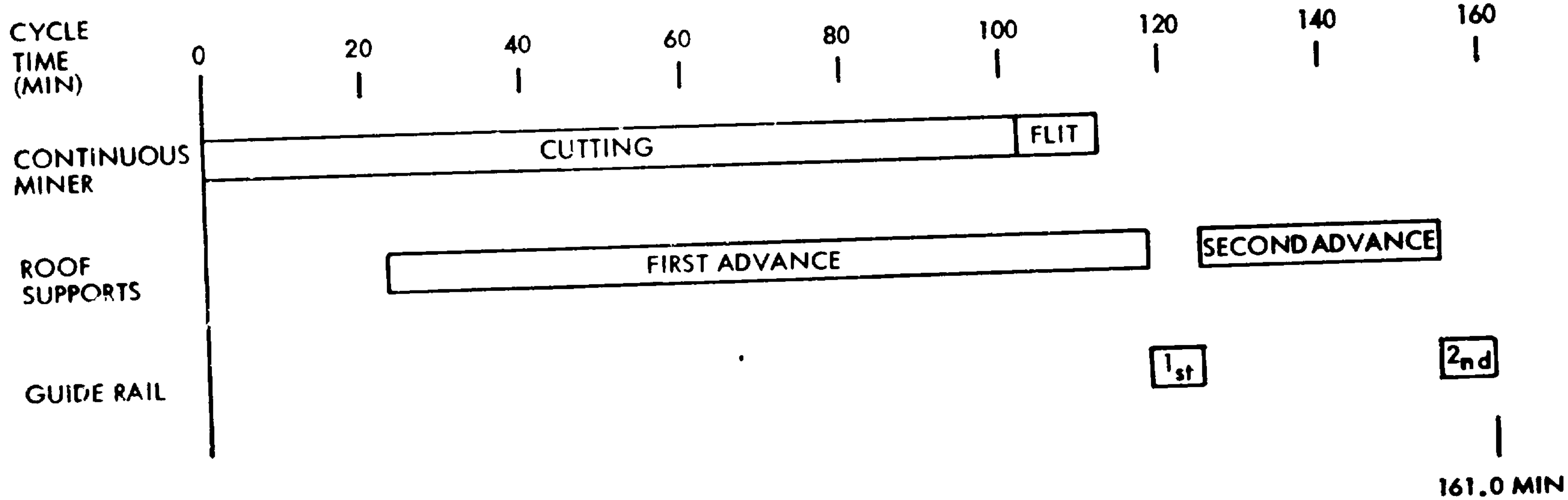


Figure B-9. Gantt Chart of Face Equipment Scheduling for 1980 Shortwall System in Average Conditions

F. 2000 SHORTWALL PRODUCTION

IDEAL CONDITIONS

Cycle time: C

- o One sump cycle
  - rates are the same, but cutter head is 4 ft in diameter rather than 3, therefore deeper sump

		min
- sump head 2 ft at 17 fpm	=	0.12
- shear down 2 ft at 34 fpm	=	0.06
- trim floor 2 ft at 17 fpm	=	0.12
- reposition 2 ft at 17 fpm	=	0.12
	TOTAL	0.42 min

- o One 180-ft long lift across face
  - $180 \text{ ft} \times \frac{0.42 \text{ min}}{2 \text{ ft}} = 37.80 \text{ min}$
- o Support advance
  - supports advance 10 ft behind continuous miner cut
  - 10 ft + machine length (30 ft) is the amount of support advance required after machine starts its flit
  - 40 ft = 8 support
  - support advance cycle is 10 s
  - 1.33 min of support advance before guide rail advance is started
- o Continuous miner flit
  - same speed and time as 1980 Shortwall, Ideal Conditions case
  - 40 fpm for 200 ft
  - 5 min to flit
- o Guide rail advance
  - advance is started as miner flits and is finished as miner enters headgate
- o Amortized section move-up time per 7 ft of advance
  - one move every 80 ft
  - 11.43 lifts every move
  - 1.0 h to accomplish move
  - 5.25 min per lift
- o Cycle time per 7-ft wide lift
  - face equipment cycle time as per attached Gantt Chart

		min
- face equipment cycle	=	42.8
- amortized move-up	=	5.25
	TOTAL	48.05 min

Tons per cycle: T

- o One cycle: 7-ft wide lift across entire face
- o Dimensions: 180-ft long x 6-ft high x 7-ft wide
- o Tons: 321.3

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.66
  - T = 321.3 tons per cycle
  - C = 48.05 min per cycle

  $$TPMS = \frac{357 \times 0.66 \times 321.3}{48.05} = 1575.54 \quad (1580)$$

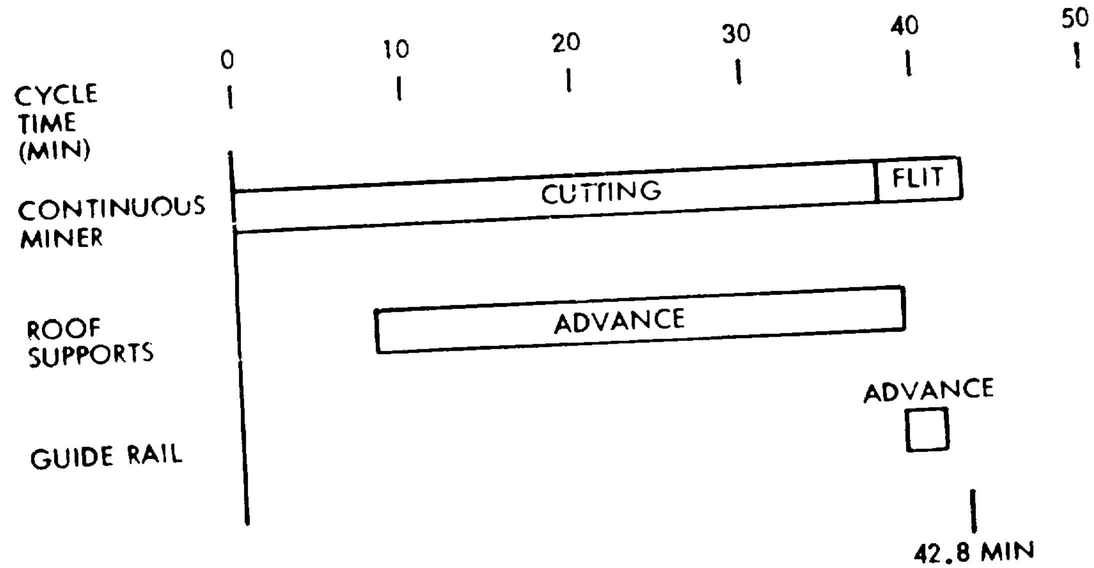


Figure B-10. Gantt Chart of Face Equipment Scheduling for 2000 Shortwall System in Ideal Conditions

AVERAGE CONDITIONS

Cycle time: C

- o One sump cycle
  - same rates as 1980 Shortwall Development, Average Conditions case

		min
- sump 2 ft at 5.66 fpm	=	0.35
- shear 2 ft at 10.50 fpm	=	0.19
- reverse sump 2 ft at 8.55 fpm	=	0.23
- reposition 2 ft at 27.50 fpm	=	0.07
	TOTAL	0.84 min

- o One 180-ft long lift across face
  - $180 \text{ ft} \times \frac{0.84 \text{ min}}{2 \text{ ft}} = 75.60 \text{ min}$
- o Support advance
  - supports advance 10 ft behind continuous miner
  - supports advance in average case assumed twice that of ideal case, 20 s per support
  - 8 supports at 20 s each = 2.67 min
  - 2.67 min of support advance at tailgate before guide rail advance started
- o Continuous miner flit
  - 20 fpm speed
  - 200 ft of travel
  - 200 ft at 20 fpm = 10 min
- o Guide rail advance
  - advance is started as miner flits and is finished as miner enters headgate

- o Amortized section move-up time per 7 ft of advance
  - one move every 80 ft
  - 11.43 lifts every move
  - 1.5 h to accomplish move
  - 7.87 min per lift
- o Cycle time per 7-ft wide lift
  - face equipment cycle time as per attached Gantt Chart

		min
- face equipment cycle	=	85.6
- amortized move-up	=	7.87
	TOTAL	93.47 min

Tons per cycle: T

- same as 2000 Shortwall, Ideal Conditions Case
- 321.3 tons per cycle

Shift production: TPMS

- Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.60
  - T = 321.3 tons per cycle
  - C = 93.47 min per cycle

  $$TPMS = \frac{357 \times 0.60 \times 321.3}{93.47} = 736.31 \quad (740)$$

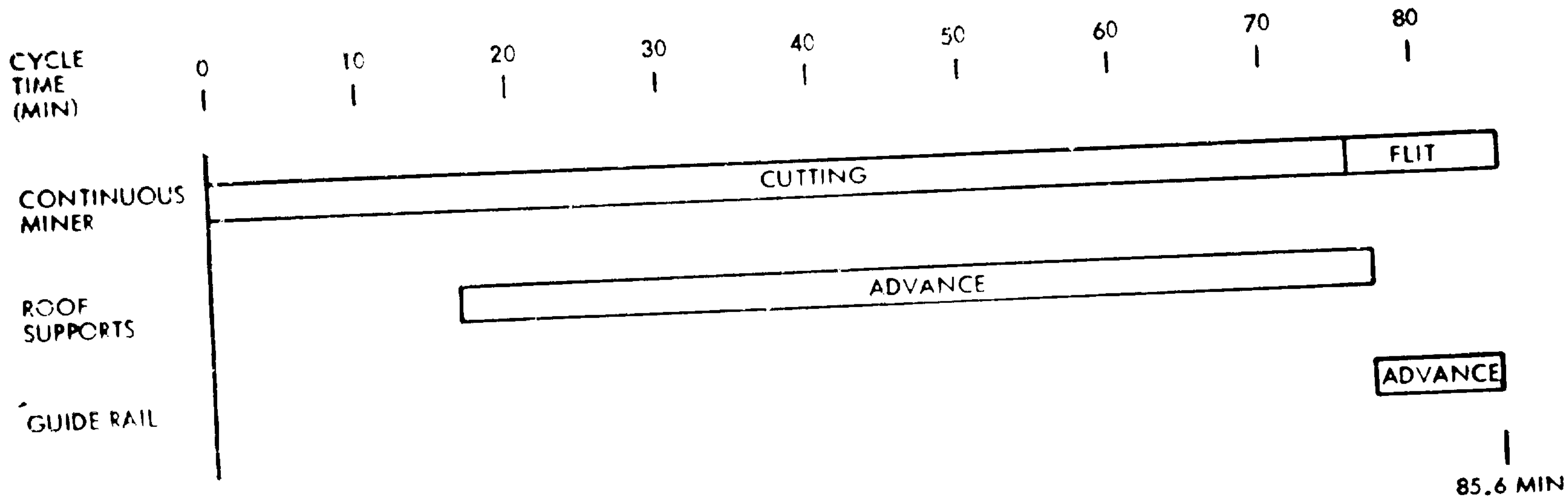


Figure B-11. Gantt Chart of Face Equipment Scheduling for 2000 Shortwall System in Average Conditions

G. 1980 ROOM-AND-PILLAR

IDEAL CONDITIONS - DEVELOPMENT

Cycle time: C

			min
o	One sump cycle		
	- sump in 1/2 diameter (18 in.) at 17 fpm	=	0.088
	- shear down 36 in. at 34 fpm	=	0.088
	- trim floor 18 in. at 17 fpm	=	0.088
	- reposition drum 36 in. at 34 fpm	=	0.088
	TOTAL		0.352 ≈ 0.35 min

- o Shuttle car loading time
  - capacity is 277 cubic ft
  - one sump cycle produces 135 loose cubic ft of coal
  - two cycles per shuttle car or 0.70 min
- o Shuttle car travel time
  - the average shuttle car change-point distance from the change-point distance for 100 ft of coal advance is 75.61 ft
  - the average shuttle car travel distance from the change-point to the wait-point is 250.73 ft for the standard shuttle car and 187.32 ft for the off-standard shuttle car
  - the tram rates of the shuttle cards in ideal conditions were assumed to be 7 ft per s (420 fpm) empty, and 5 ft per s (300 fpm) leaded
  - change point travel times are as follows

    empty = 0.18 min

    loaded = 0.25 min
- o Shuttle car unloading time
  - assumed to be 0.5 min
- o Maneuver time
  - after a 20-ft advance, 10-ft wide cut is made, the continuous miner must maneuver to the other side of the lift
  - with a travel distance of 60 ft at a rate of 60 fpm, maneuver time is 1.0 min
- o Shuttle car wait point time at dump
  - 50 ft of travel, full
  - 50 ft of travel, empty
  - 0.15 min + 0.17 min = 0.29 min
- o Cut cycle time
  - attached figure illustrates diagram and values used in the analysis which determined the cut cycle time
  - cut cycle time - 23.56 min
- o Place change time
  - average move is 325.37 ft
  - tram rate for continuous miner is 60 fpm
  - 5.42 min per place change
- o Amortize section move-up time per 100 ft of section advance
  - 41 cuts per move
  - 1 h to accomplish move
  - 1.46 min per cut
- o Total cut cycle time per 100 ft of section advance

  23.56 min + 5.42 min + 1.46 min = 30.44 min

Tons per cycle: T

- o One cut is 20 ft x 20 ft x 6 ft
- o 102 tons per cycle

Shift production: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.63
  - T = 102 tons per cycle
  - C = 30.44 min per cycle

$$TMPS = \frac{357 \times 0.63 \times 103}{30.44} = 753.64\ (750)$$

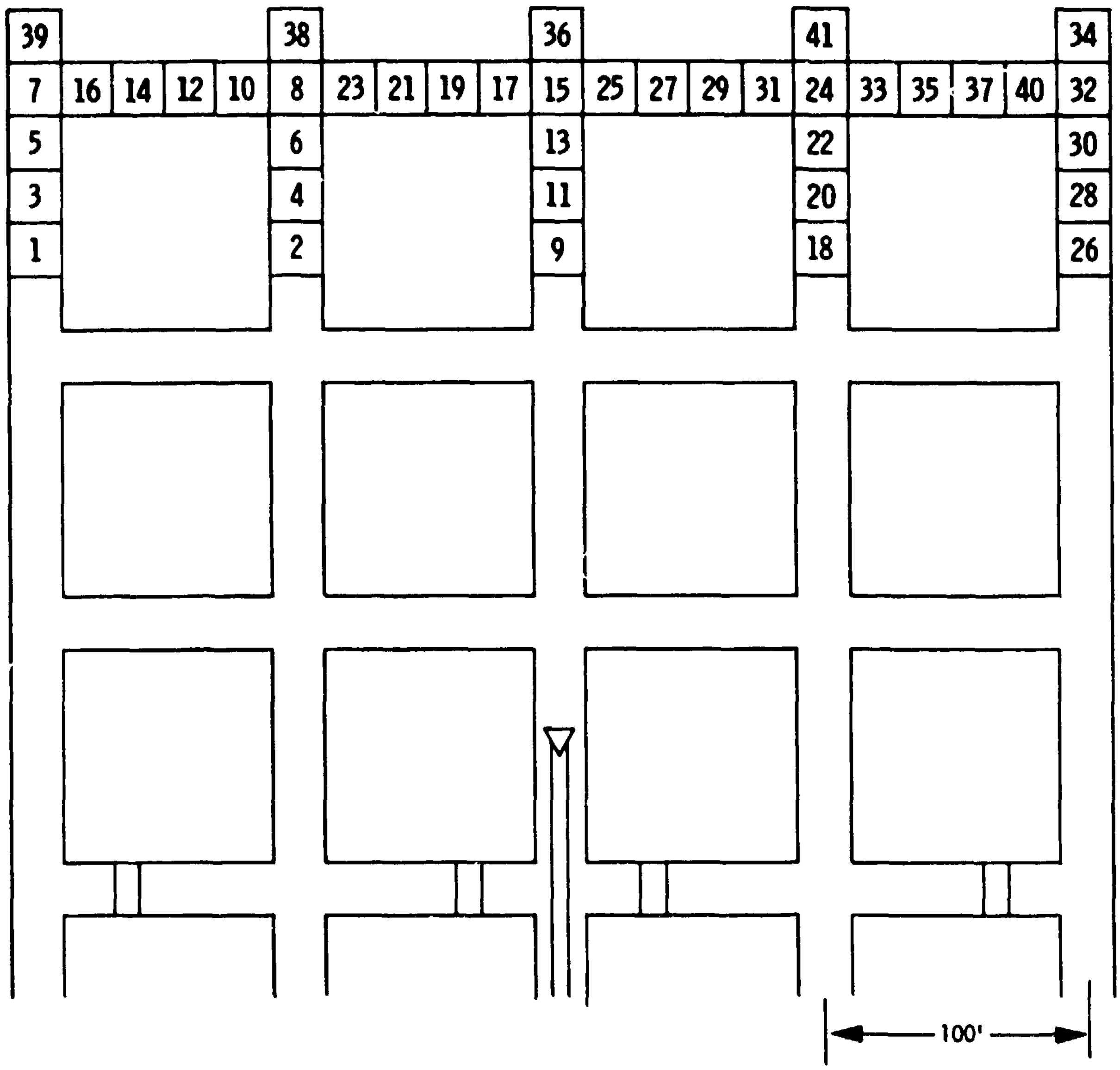
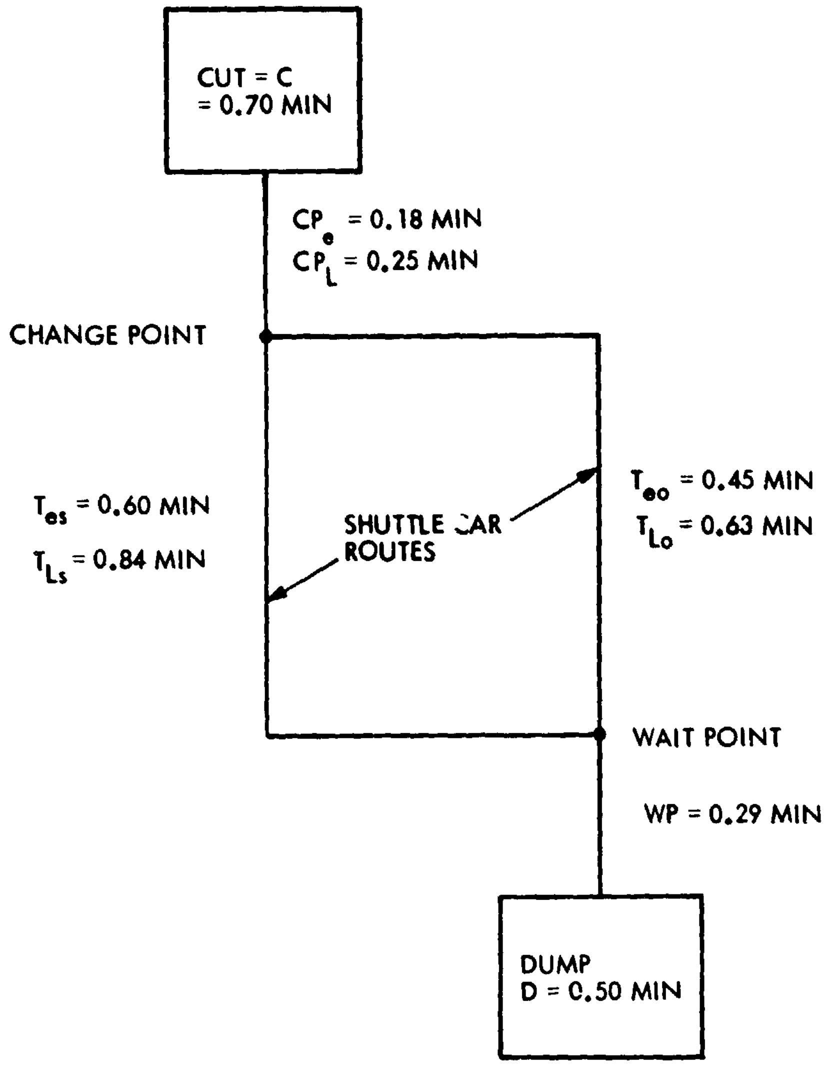


Figure B-12. Lift Sequence for 1980 Room-and-Pillar Cases, Development



C = CUT TIME
CP_e = CHANGE POINT TRAVEL TIME, EMPTY
CP_L = CHANGE POINT TRAVEL TIME, LOADED
T_{es} = TRAVEL TIME, STANDARD CAR, EMPTY
T_{Ls} = TRAVEL TIME, STANDARD CAR, LOADED
T_{eo} = TRAVEL TIME, OFF-STANDARD CAR, EMPTY
T_{Lo} = TRAVEL TIME, OFF - STANDARD CAR, LOADED
WP = WAIT POINT TRAVEL TIME
D = DUMP TIME

Figure B-13. Elemental Times Used in 1980 Room-and-Pillar Ideal Conditions Case to Determine Cycle Time for Development

IDEAL CONDITIONS - RETREAT

Cycle time: C

- o One sump cycle
  - same as Development
  - 0.35 min per 1.5 ft of advance
- o Shuttle car loading time
  - same as Development
  - 0.70 min per shuttle car
- o Shuttle car travel time
  - average shuttle car travel from change-point to a cut is 172.5 ft
  - average shuttle car travel distance from the change-point to the wait-point is 330 ft
  - shuttle car travel distance from the wait-point to the dump and back is 100 ft
  - the tram rates for the shuttle cars in ideal conditions were assumed to be 420 fpm empty and 300 fpm loaded
  - travel times are as follows:

	empty	loaded
change-point to cut	0.41	0.58
change-point to wait-point	0.79	1.10
wait-point to dump and return	0.12	0.17

- o Shuttle car unloading time
  - assumed to be 0.5 min
- o Continuous miner maneuver time
  - after the first 10-ft wide advance is made, continuous miner is maneuver to open cut to 20 ft wide
  - a travel distance of 60 ft is required for maneuvering
  - at a rate of 60 fpm, maneuver time is 1.0 min
- o Cut cycle time
  - attached figure illustrates diagram and values used in the analysis which determined the cut cycle time
  - cut cycle time = 31.26 min
- o Place change time
  - average move is 203.75 ft
  - tram rate for continuous miner is 60 fpm
  - 3.40 min per place change
- o Amortize section move time per 100 ft of section retreat
  - 32 cuts per move
  - 1 h to accomplish move
  - 1.88 min per cut
- o Total cycle time
  31.26 min + 3.40 min + 1.88 min = 36.54 min

Tons per cycle: T

- o Average cut is 20-ft long, 30-ft wide, and 6-ft high
- o 102 tons per cycle

Shift production: TPMS

- Tons per machine-shift $= \frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.63
  - T = 102 tons per cycle
  - C = 36.54 min per cycle

$$TPMS = \frac{357 \times 0.63 \times 102}{36.54} = 627.83\ (630)$$

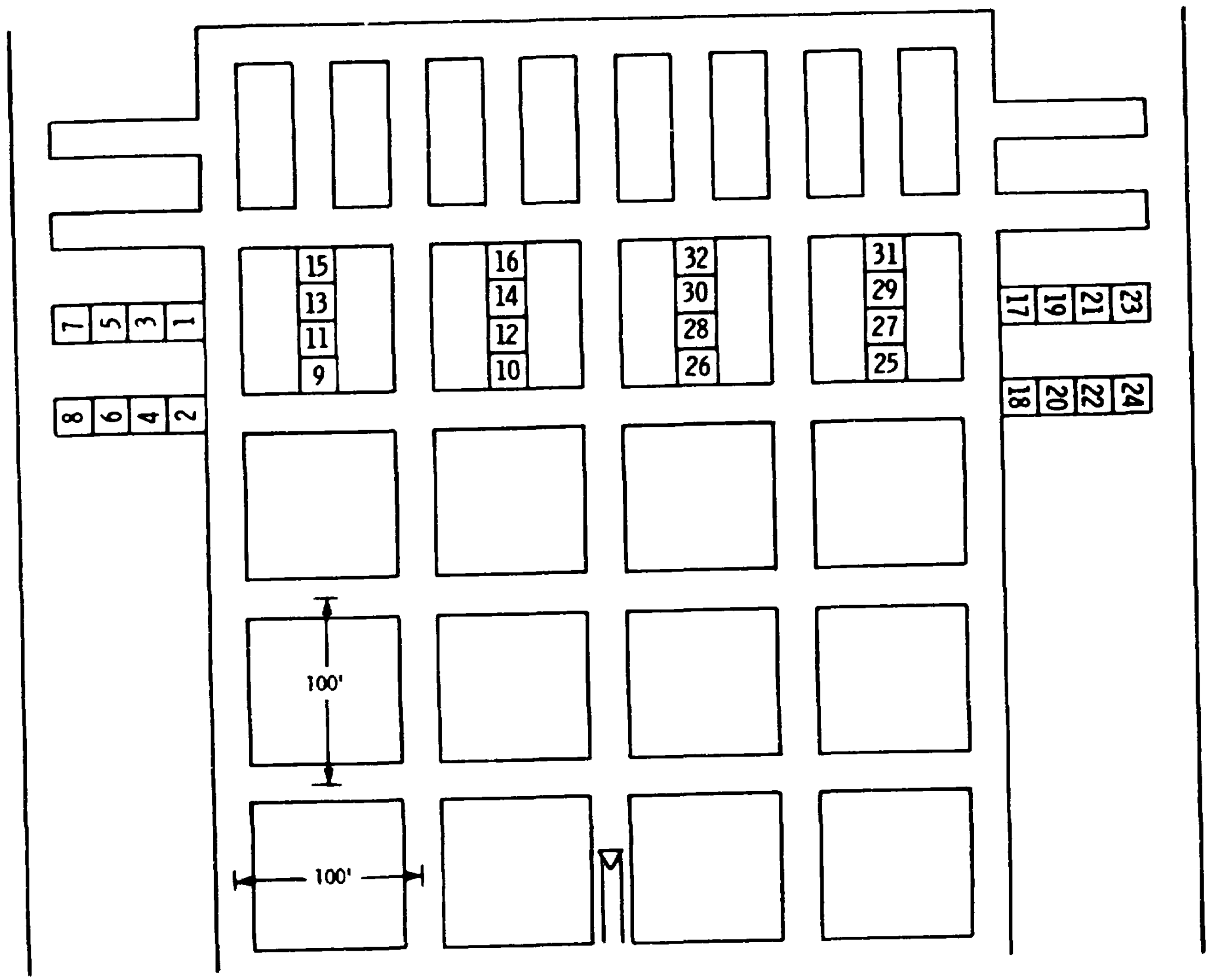
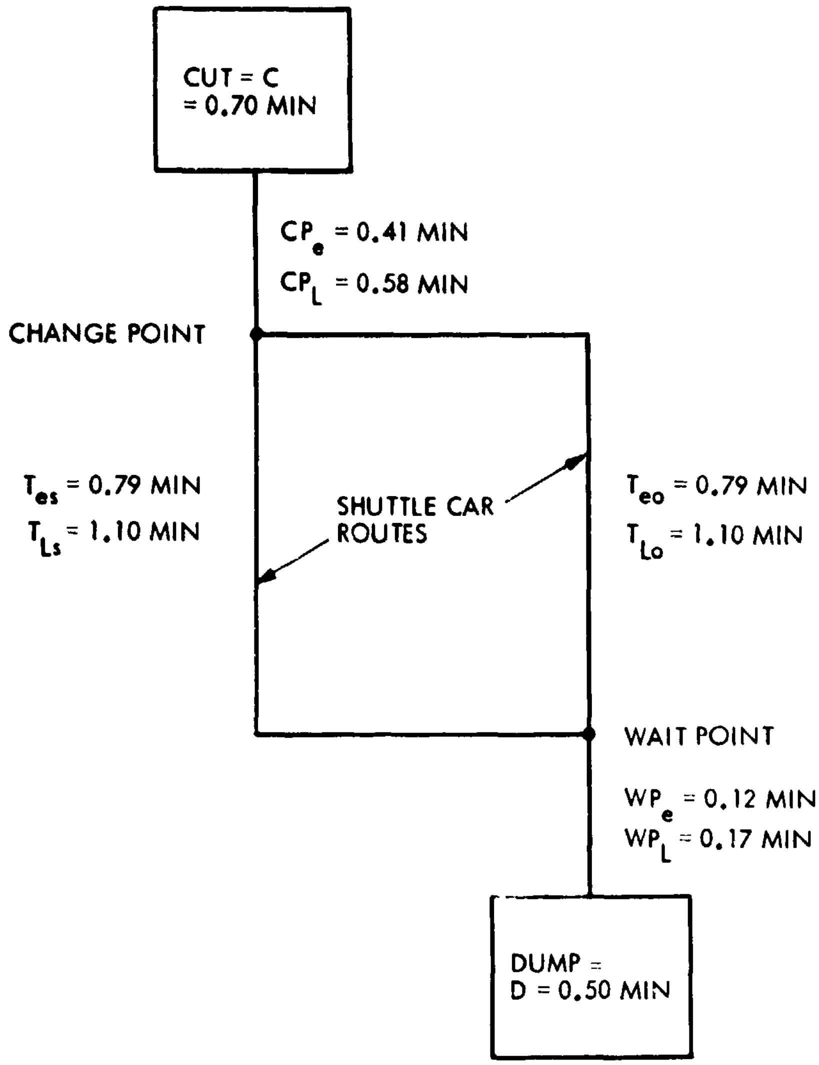


Figure B-14. Lift Sequence for 1980 Room-and-Pillar Cases, Retreat



C = CUT TIME

CP_e = CHANGE POINT TRAVEL TIME, EMPTY

CP_L = CHANGE POINT TRAVEL TIME, LOADED

T_{eo}, T_{es} = TRAVEL TIME (S - STANDARD CAR, O - OFF - STANDARD CAR), EMPTY

T_{Lo}, T_{Ls} = TRAVEL TIME, LOADED

WP_e = WAIT POINT TRAVEL TIME, EMPTY

WP_L = WAIT POINT TRAVEL TIME, LOADED

D = DUMP TIME

Figure B-15. Elemental Times Used in 1980 Room-and-Pillar Ideal Conditions Case to Determine Cycle Time for Retreat

1980 ROOM-AND-PILLAR

AVERAGE CONDITIONS - DEVELOPMENT

Cycle time: C

- o One sump cycle:
  - same as 1980 longwall case, average conditions
  - 0.85 min per 1.5 ft lift
- o Shuttle car loading time
  - two cycles per shuttle car or 1.70 min
- o Shuttle car travel time
  - average shuttle car change-point distance for 100 ft of advance is 75.61 ft
  - the average shuttle car travel distance from the change point to the wait point is 250.73 ft for the standard car and 187.32 ft for the off-standard car
  - tram rates of the shuttle cars in average conditions were assumed to be one-half those of ideal conditions: 210 fpm empty and 150 fpm loaded
  - change point travel times are as follows:

    empty = 0.36 min
    loaded = 0.50 min

  - travel times are as follows:

	empty	loaded
Standard	1.19	1.67
Off-standard	0.89	1.25

  - shuttle car wait point travel time at dump
    50 ft of loaded travel = 0.33 min
    50 ft of empty travel = 0.24 min
    total time = 0.57 min
- o Shuttle car unloading time
  - assumed to be 0.50 min
- o Maneuver time
  - assumed to be twice that in ideal conditions, 2.00 min
- o Cut cycle time
  - attached figure illustrates diagram and values used in the analysis which determined the average cut cycle time
  - cut cycle time = 45.45 min
- o Place-change time
  - average move is 325.37 ft
  - tram rate for continuous miner is 30 fpm
  - 10.85 min per place change
- o Amortized section move-up time per 100 ft of section advance
  - 41 cuts per move
  - 1.5 h to accomplish move
  - 2.20 min per cut
- o Total cycle time per 100 ft of section advance
  45.45 min + 10.85 + 220 min = 58.50 min

Tons per cycle: T

- o Same as ideal case
- o 102 tons per cycle

Shift production: TPMS

- o Tons per machine shift = $\frac{APT \cdot A \cdot T}{C}$

  - APT = 357 min per machine-shift
  - A = 0.50
  - T = 102 tons per cycle
  - C = 58.50 min per cycle

$$TPMS = \frac{357 \times 0.50 \times 102}{58.50} = 311.23\ (310)$$

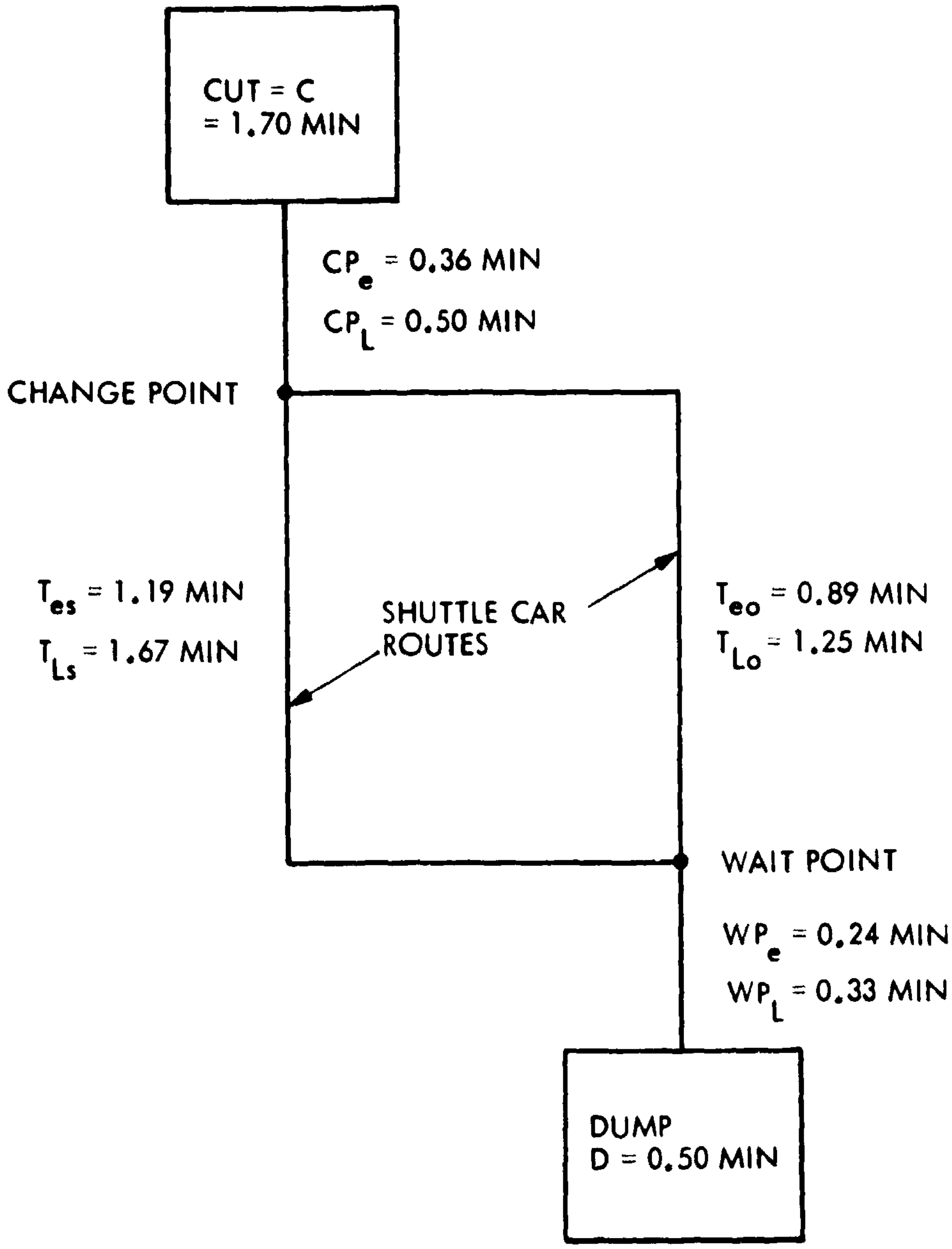


Figure B-16. Elemental Times Used in 1980 Room-and-Pillar Average Conditions Case to Determine Cycle Time for Development

AVERAGE CONDITIONS - RETREAT

Cycle time: C

- o One sump cycle
  - same as Development
  - 0.85 mine per 1.5 ft of advance
- o Shuttle car loading time
  - same as Development
  - 1.70 min per shuttle car
- o Shuttle car travel time
  - average travel distance from change-point to a cut is 172.5 ft
  - average travel distance from change-point to wait-point is 330 ft
  - travel distance from wait-point to dump and back is 100 ft
  - tram rates for average conditions were assumed to be 210 fpm, empty and 150 fpm, loaded
  - travel times are as follows:

	empty	loaded
change-point to cut	0.82	1.15
change-point to wait-point	1.57	2.20
wait-point to dump and return	0.24	0.33

- o Shuttle car unloading time
  - assumed to be 0.5 min
- o Continuous miner maneuver time
  - same as Development
  - 60 ft at 30 fpm
  - 2.0 min
- o Cut cycle time
  - attached figure illustrates diagram and values used in the analysis to determine cycle time
  - cut cycle time = 60.59 min
- o Place-change time
  - average move is 203.75 ft
  - tram rate of continuous miner is 30 fpm
  - 6.79 min per place-change
- o Amortized section move time per 100 ft of section advance
  - 32 cuts per move
  - 1.5 h to accomplish move
  - 2.81 min per cut
- o Total cycle time

  60.59 min + 6.79 min + 2.81 min = 70.19 min

Tons per cycle: T

- o Average cut is 20-ft long, 20-ft wide, and 6-ft high
- o 102 tons

Shift production: TPMS

- o Tons per machine-shift $= \dfrac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.50
  - T = 102 tons per cycle
  - C = 70.19 min per cycle

$$TPMS = \frac{357 \times 0.50 \times 102}{70.19} = 259.39 \quad (260)$$

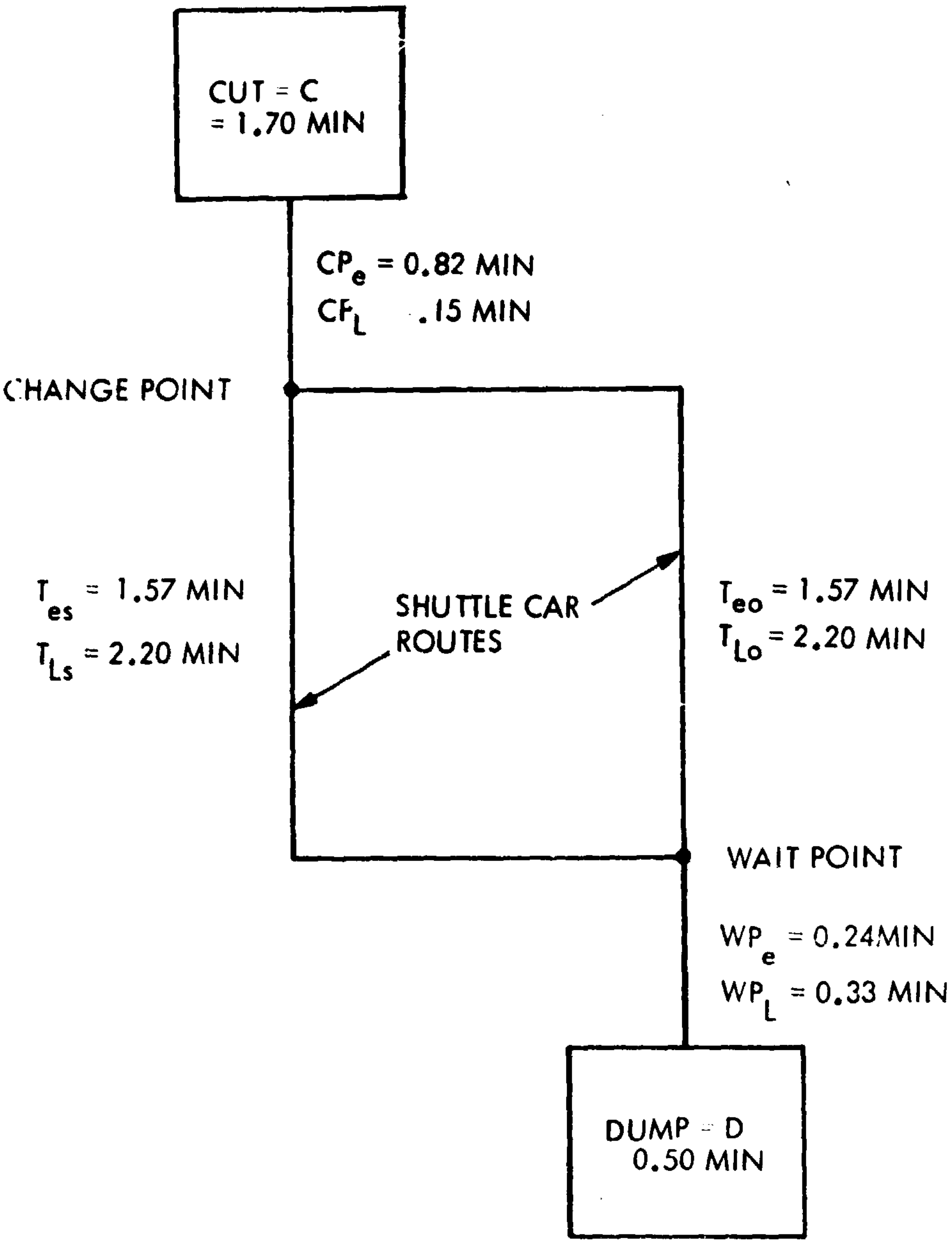


Figure B-17. Elemental Times Used in 1980 Room-and-Pillar Average Conditions Case to Determine Cycle Time for Retreat

H. 2000 ROOM-AND-PILLAR

IDEAL CONDITIONS - DEVELOPMENT

Cycle time: C

- o One sump cycle
  - same as 1980 Room-and-Pillar case
  - 0.35 min per 1.5 ft of advance

- o One lift
  - same as 2000 Longwall Development case but with different place-change time
  - 2000 R&P place-change
    average distance is 228.46 ft at 60 fpm, therefore
    3.81 min per move
  - cycle time for lift

$$\begin{array}{ll} +45.32 \text{ min} & \text{(2000 Longwall case)} \\ -\ 3.97 \text{ min} & \text{(2000 Longwall place-change time)} \\ \underline{+\ 3.81 \text{ min}} & \text{(2000 R\&P place-change time)} \\ \ \ 45.16 \text{ min} & \end{array}$$

- o Amortized section move-up time per 80-ft lift
  - 1.0 h to accomplish move
  - 9 lifts per move
  - 6.67 min per lift
- o Total cycle time per 80-ft lift
  45.16 min + 6.67 min = 51.83 min

Tons per cycle: T

- o 9 lifts per move
- o 3455.8 tons per move
- o 383.98 tons per cycle

Shift production: TPMS

- o Tons per machine-shift $= \dfrac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.63
  - T = 383.98 tons per cycle
  - C = 51.81 min per cycle

$$TPMS = \frac{357 \times 0.63 \times 383.98}{51.83} = 1666.23 \quad (1670)$$

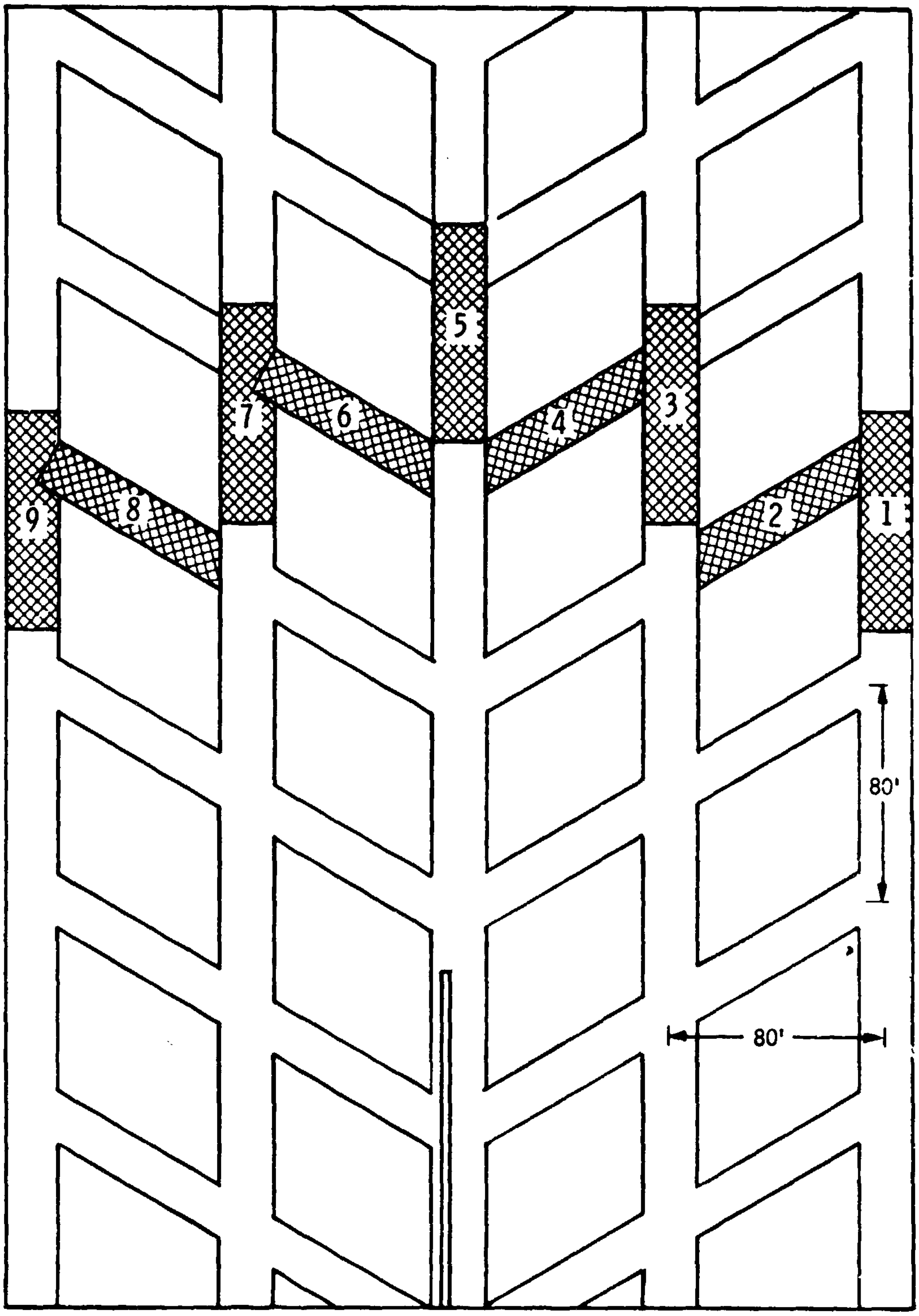


Figure B-18. Lift Sequence for 2000 Room-and-Pillar Cases, Development

IDEAL CONDITIONS - RETREAT

Cycle time: C

- o One sump cycle
  - same as 2000 R&P Ideal Conditions case, Development
  - 0.35 min per 1.5 ft of advance
- o One lift length
  - average lift length is 74.25 ft
  - average place-change distance is 194.76 ft
  - cycle

			min
o	20-ft advance	=	4.67
o	reposition: 60 ft at 60 fpm	=	1.00
o	30-ft advance	=	7.00
o	reposition: 30 ft at 60 fpm	=	0.50
o	20-ft advance	=	4.67
o	reposition: 30 ft at 60 fpm	=	0.50
o	20-ft advance	=	4.67
o	reposition: 30 ft at 60 fpm	=	0.50
o	20-ft advance	=	4.67
o	reposition: 30 ft at 60 fpm	=	0.50
o	20-ft advance	=	4.67
o	reposition: 30 ft at 60 fpm	=	0.50
o	14.25 ft advance	=	3.33
o	reposition: 24.25 ft at 60 fpm	=	0.40
o	4.25 ft advance	=	0.99
o	place-change, average move distance of 194.76 ft 194.76 ft at 60 fpm	=	3.25
		TOTAL	41.82 min

- o Amortized section move-up time per shift
  - 1.0 h to accomplish move
  - 8 lifts per move
  - 7.50 min
- o Total cycle time per lift

  41.82 min + 7.50 min = 49.32 min

Tons per cycle time: T

- o 8 lifts per move
- o 2616.3 tons per move
- o 327.04 tons per lift

Shift producton: TPMS

o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

- APT = 35 min per machine-shift
- A = 0.63
- T = 327.04 tons per cycle
- C = 49.43 min per cycle

$$TPMS = \frac{357 \times 0.63 \times 327.04}{49.32} = 1491.37 \quad (1490)$$

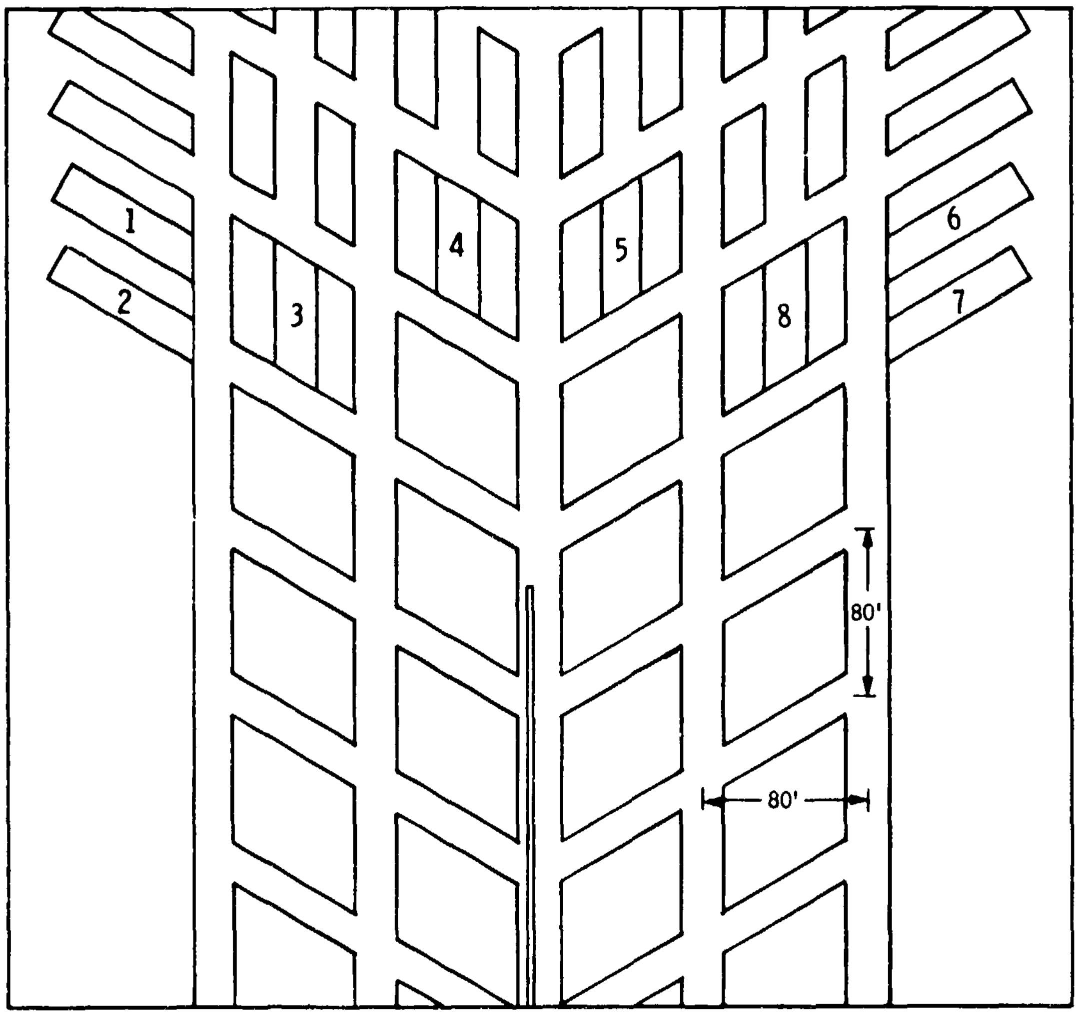


Figure B-19. Lift Sequence for 2000 Room-and-Pillar Cases, Retreat

2000 ROOM-AND-PILLAR

AVERAGE CONDITIONS - DEVELOPMENT

Cycle time: C

- o One sump cycle
  - same as 1980 R&P, Average Conditions case
  - 0.85 min per 1.5 ft of advance
- o One lift
  - same as 2000 Longwall Development case, but with different place-change time
  - +106.59 min (2000 Longwall case)
  - − 7.94 min (2000 Longwall place-change time)
  - − 7.62 min (2000 R&P place-change time)

    106.27 min
- o Amortized section move time per 80-ft lift
  - 1.5 hr per move
  - 9 lifts per move
  - 10 min per lift
- o Cycle time per 80-ft lift

  106.27 min + 10.00 min = 116.27 min

Tons per cycle: T

- o 9 lifts per move
- o 3455.8 tons per move
- o 383.98 tons per cycle

Shift Production: TPMS

- o Tons per machine-shift $= \dfrac{APT \cdot A \cdot T}{C}$
  - APT = 357 min per machine-shift
  - A = 0.50
  - T = 383.98 tons per cycle
  - C = 116.27 min per cycle

$$TPMS = \frac{357 \times 0.50 \times 383.98}{116.27} = 589.49\ (590)$$

AVERAGE CONDITIONS - RETREAT

Cycle time: C

- o One sump cycle
  - same as 2000 R&P Development
  - 0.85 min per 1.5 ft of advance
- o One lift length
  - average lift length is 74.25 ft
  - average place-change distance is 194.76 ft
  - cycle

			min
o	20-ft advance	=	11.37
o	reposition: 60 ft at 60 fpm	=	2.00
o	30-ft advance	=	17.00
o	reposition: 30 ft at 60 fpm	=	1.00
o	20-ft advance	=	11.33
o	reposition: 30 ft at 60 fpm	=	1.00
o	20-ft advance	=	11.33
o	reposition: 30 ft at 60 fpm	=	1.00
o	20-ft advance	=	11.33
o	reposition: 30 ft at 60 fpm	=	1.00
o	20-ft advance	=	11.33
o	reposition: 30 ft at 60 fpm	=	1.00
o	14.25 ft advance	=	8.08
o	reposition: 24.25 ft at 60 fpm	=	0.81
o	4.25 ft advance	=	2.41
o	place-change, average move distance of 194.76 ft 194.76 ft at 60 fpm	=	6.49
		TOTAL	98.44 min

- o Amortized section move-up time per shift
  - 1.5 h to accomplish move
  - 8 lifts per move
  - 11.25 min per lift
- o Cycle time per lift

  98.44 min + 11.25 min = 109.69 min

Tons per cycle time: T

- o 8 lifts per move
- o 2616.3 tons per move
- o 327.04 tons per lift

Shift producton: TPMS

- o Tons per machine-shift = $\frac{APT \cdot A \cdot T}{C}$

- APT = 357 min per machine-shift
- A = 0.50
- T = 327.04 tons per cycle
- C = 109.69 min per cycle

$$\text{TPMS} = \frac{357 \times 0.50 \times 327.04}{109.69} = 532.20 \quad (530)$$

APPENDIX C

COST ANALYSIS INPUTS

The purpose of Appendix C, Cost Analysis Inputs is to identify the inputs that were required for the NUS coal costing model in order to simulate the case studies of this moving baseline effort. The items included in Appendix C are the calculations that modified the system productivities of the Production Analysis task for input to the NUS model, design capacity calculations, and the requirements and costs for section equipment, electric power, salaried personnel, and hourly laborers.

INPUTS FOR THE

1980 and 2000 LONGWALL CASES

MODIFIED SHIFT PRODUCTION

	1980 LONGWALL CASES		2000 LONGWALL CASES	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
<u>TPMS</u>:				
Panel Development:	460	1440	550	1510
Longwall Production:	890	2090	1350	3250
<u>Tons per panel</u>:				
Development:	69,039 tons		69,039 tons	
Production:	341,700 tons		341,700 tons	
<u>Shifts to mine one panel</u>:				
Development:	150.08	47.94	125.52	45.72
Production:	383.93	163.49	253.11	105.14
<u>Shifts to move to next panel</u>:				
Development:	4	4	4	4
Production:	30	30	30	30
<u>Total Shifts per panel</u>:				
Development:	154.08	51.94	129.52	49.72
Production:	413.93	193.49	283.11	135.14
<u>Modified TPMS</u>:				
Development:	448.07 (450)	1329.21 (1330)	533.04 (530)	1388.55 (1390)
Production:	825.50 (830)	1765.98 (1770)	1206.95 (1210)	2528.49 (2530)

DESIGN CAPACITY

Design Capacity =

$$[(N(d) \times TPMS(d)) + (N(P) \times TPMS(P))]\ S \times D \times (1 - R)$$

N (d) = Number of development units per shift
TPMS (d) = Productivity of development units
N (P) = Number of productions per shift
TPMS (P) = Productivity of production units
S = Shifts per day
D = Days per week
R = Reject precentage

	1980 LONGWALL		2000 LONGWALL	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
N (d)	3	2	3	2
TPMS (d)	450	1330	530	1390
N (P)	1	1	1	1
TPMS (P)	830	1770	1210	2530
S	3	3	3	3
D	220	220	220	220
R	21%	21%	21%	21%
Design Capacity in clean tons	1,136,652	2,309,802	1,459,920	2,768,634

EQUIPMENT COSTS

		1980 LONGWALL CASES			
		Average Conditions		Ideal Conditions	
Continuous Miner Units	Cost/unit[1]	No.	Total Cost	No.	Total Cost
Continuous Miner	$412,000	3	$1,236,000	2	$ 824,000
Mobile Bridge Carrier System[2]	242,000	3	726,000	2	484,000
Roof Bolter	76,700	3	230,100	4	306,800
Auxiliary Fan	18,000	5	90,000	3	54,000
Scoop with Batteries & Charger	77,200	3	231,000	2	154,400
Bantam Duster	18,000	3	54,000	2	36,000
Trickle Duster	15,000	5	75,000	3	45,000
Section Power Center & Cables	53,000	3	159,000	2	106,000
Parts Car	10,500	3	31,500	2	21,000
Oil Storage Car	12,000	3	36,000	2	24,000
Section Tools	7,700	3	23,100	2	15,400
Ventilation Tubes	2,000	3	6,000	2	4,000
Section Welder	1,700	3	5,100	2	3,400
Section Haulage Belt (42 in., 3000 ft)	$42.5/ft	2	255,000	1	127,500
Fire Suppression	4,200	3	12,600	2	8,400
1980 Dollars	TOTAL		$3,171,000		$2,213,900

1. All costs were found or were updated from information in IOM FF-345-79-239 (P. G. Gordon), JPL, 12/17/79
2. The 1977 cost extracted from Conceptual Design of a Fully Automated Continuous Mining System Operating Under Remote Supervisory Control, USBM Contract 50166007, Final Report, Arthur D. Little, Inc., 1977.

EQUIPMENT COSTS

		2000 LONGWALL CASES			
		Average Conditions		Ideal Conditions	
Continuous Miner Units	Cost/unit [1]	No.	Total Cost	No.	Total Cost
Continuous Miner	$412,000	3	$1,236,000	2	$ 824,000
Mobile Bridge Carrier System [2]	242,000	3	726,000	2	484,000
Roof Bolter	76,700	3	230,100	4	306,800
Auxiliary Fan	18,000	5	90,000	3	54,000
Scoop with Batteries & Charger	77,200	3	231,000	2	154,400
Bantam Duster	18,000	3	54,000	2	36,000
Trickle Duster	15,000	5	75,000	3	45,000
Section Power Center & Cables	53,000	3	159,000	2	106,000
Parts Car	10,500	3	31,500	2	21,000
Oil Storage Car	12,000	3	36,000	2	24,000
Section Tools	7,700	3	23,100	2	15,400
Ventilation Tubes	2,000	3	6,000	2	4,000
Section Welder	1,700	3	5,100	2	3,400
Section Haulage Belt (42 in., 3000 ft)	$42.5/ft	2	255,000	1	127,500
Fire Suppression	4,200	3	12,600	2	8,400
1980 Dollars	TOTAL		$3,171,000		$2,213,900

1. All costs were found or were updated from information in IOM FF-345-79-239 (P. G. Gordon), JPL, 12/17/79
2. The 1977 cost extracted from Conceptual Design of a Fully Automated Continuous Mining System Operating Under Remote Supervisory Control, USBM Contract 50166007, Final Report, Arthur D. Little, Inc., 1977.

1980 LONGWALL CASES

AVERAGE & IDEAL CONDITIONS	EQUIPMENT COST		1980 DOLLARS
Longwall Unit	No.	Cost/Unit (1)	Total Cost
Longwall Face Equipment:	1	$5,170,000	$5,170,000
- Shearer			
- Face Conveyor			
- Self-Advancing Supports (500-ft face)			
- Stageloader			
- Electrical Equipment			
- Controls			
Section Haulage Belt (42 in. - 3000 ft)	1	$42.5/ft	127,500
Fire Suppression	1	4,200	4,200
Parts Car	1	10,500	10,500
Oil Storage Car	1	12,000	12,000
Section Tools	1	7,700	7,700
Section Welder	1	1,700	1,700
Scoop w/Batteries & Charges	1	77,200	77,200
		TOTAL	$5,410,800

1. All costs were found or updated from information in IOM FF-345-79-239 (P. G. Gordon), JPL, 12/17/79

2000 LONGWALL CASES

AVERAGE & IDEAL CONDITIONS	EQUIPMENT COST		1980 DOLLARS
Longwall Unit	No.	Cost/Unit (1)	Total Cost
Longwall Face Equipment: - One Shearer - Face Conveyor - Self-Advancing Supports (500 ft face) - Stageloader - Electrical Equipment - Controls	1	$5,170,000	$5,170,000
Shearer (for Dual-Shearer Face)	1	824,000 (2)	824,000
Automatic Control System - Vertical Control Subsystem - Face Alignment Subsystem - Master Control System	1	700,000 (3)	700,000
Section Haulage Belt (42 in. - 3000 ft)	1	$42.5/ft	127,500
Fire Suppression	1	4,200	4,200
Parts Car	1	10,500	10,500
Oil Storage Car	1	12,000	12,000
Section Tools	1	7,700	7,700
Section Welder	1	1,700	1,700
Scoop w/Batteries & Charger	1	77,200	77,200
		TOTAL	$6,934,800

1. All costs were found or updated from information in IOM FF-345-79-239 (P. G. Gordon), JPL, 12/17/79, except where noted.
2. 1978 cost from Interoffice memo, 11/16/78, J. Harris to M. Lavin, JPL; updated to 1980 by IOM FF-345-79-239.
3. Estimate developed by JPL personnel with aid of private communications, March 1980.

1980 LONGWALL CASES

AVERAGE CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Longwall Unit	1	625	625	10	466	4,660
Continuous Miner	3	600	1800	9	1342	12,078
MBC Unit	3	120	360	9	268	2,412
Roof Bolter	3	50	150	12	112	1,344
Auxiliary Fan	5	15	75	18	56	1,008
Mantrip Jeep	6	15	90	4	67	268
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	4	80	320	12	239	2,868
Bantam Duster	4	30	120	12	89	1,068
Trickle Duster	6	10	40	18	30	540
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fans	1	1000	1000	24	746	17,904
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
					TOTAL	63,729

Power at \$0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{63{,}729 \text{ kW-h}}{\text{day}} \times \frac{220 \text{ days}}{\text{year}} = \frac{\$420{,}611.4}{\text{year}}$$

$$\text{Power Cost per Clean Ton} = \frac{\$420{,}611.4}{\text{year}} \times \frac{1 \text{ year}}{1{,}136{,}652} = \frac{\$0.37}{\text{tons}}$$

1980 LONGWALL CASES

IDEAL CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Longwall Unit	1	625	625	12	466	5,592
Continuous Miner	2	600	1200	12	895	109,740
MBC Unit	2	120	240	12	179	2,148
Roof Bolter	4	50	200	7	149	1,043
Auxiliary Fan	3	15	45	18	34	612
Mantrip Jeep	5	15	75	4	56	224
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	3	80	240	12	179	2,148
Bantam Duster	3	30	90	12	67	804
Trickle Duster	4	10	40	18	30	540
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fans	1	1000	1000	24	746	17,904
Outside Electricals			600	14	447	6,256
Miscellaneous			400	10	298	2,980
					TOTAL	61,334

Power at $0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{61{,}334\ \text{kW-h}}{\text{day}} \times \frac{220\ \text{days}}{\text{year}} = \frac{\$404{,}804.4}{\text{year}}$$

$$\text{Power Cost per Clean Ton} = \frac{\$404{,}804.4}{\text{year}} \times \frac{1\ \text{year}}{2{,}309{,}802} = \frac{\$0.18}{\text{ton}}$$

2000 LONGWALL CASES

AVERAGE CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Longwall Unit	1	850	850	10	634	6,340
Continuous Miner	3	600	1800	9	1342	12,078
MBC Unit	3	120	360	9	268	2,412
Roof Bolter	6	50	300	6	224	1,344
Powered Temp. Supports	12	40	480	2	358	716
Ventilation System	12	15	180	18	134	2,412
Mantrip Jeep	6	15	90	4	67	268
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	4	80	320	12	239	2,868
Bantam Duster	4	30	120	12	89	1,068
Trickle Duster	16	10	60	18	45	805
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fans	1	1000	1000	24	746	17,904
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
					TOTAL	67,794

Power at \$0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{67{,}794\ \text{kW-h}}{\text{day}} \times \frac{220\ \text{days}}{\text{year}} = \frac{\$447{,}440.4}{\text{year}}$$

$$\text{Power Cost per Clean Ton} = \frac{\$447{,}440.4}{\text{year}} \times \frac{1\ \text{year}}{1{,}459{,}920} = \frac{\$0.31}{\text{ton}}$$

2000 LONGWALL CASES

IDEAL CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Longwall Unit	1	850	850	12	634	7,608
Continuous Miner	2	600	1200	12	895	10,740
MBC Unit	2	120	240	12	179	2,148
Roof Bolter	4	50	200	6	149	894
Powered Temp. Supports	10	40	400	2	298	596
Ventilation System	10	15	150	18	112	2,016
Mantrip Jeep	5	15	75	4	56	224
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	3	80	240	12	179	2,148
Bantam Duster	3	30	90	12	67	804
Trickle Duster	4	10	40	18	30	540
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fans	1	1000	1000	24	746	17,904
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
					TOTAL	65,201

Power at $0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{65{,}201\ \text{kW-h}}{\text{day}} \times \frac{220\ \text{days}}{\text{year}} = \frac{\$430{,}326.6}{\text{year}}$$

$$\text{Power Cost per Clean Ton} = \frac{\$430{,}326.6}{\text{year}} \times \frac{1\ \text{year}}{2{,}768{,}634} = \frac{\$0.16}{\text{ton}}$$

SALARIED PERSONNEL REQUIREMENTS AND COSTS

		1980 LONGWALL CASES				2000 LONGWALL CASES			
		Average Conditions		Ideal Conditions		Average Conditions		Ideal Conditions	
Personnel	Salary	No.	Yearly Total	No.	Yearly Total	No.	Yearly Total	No.	Yearly Total
Superintendent	$33,000	1	$ 33,000	1	$ 33,000	1	$ 33,000	1	$ 33,000
General Mine Foreman	27,500	1	27,500	1	27,500	1	27,500	1	27,500
Assist. Mine Foreman	22,500	3	67,500	3	67,500	3	67,500	3	67,500
Section Foreman	20,000	12	240,000	9	180,000	12	240,000	9	180,000
Construction Foreman	21,000	11	231,000	23	483,000	14	294,000	27	567,000
Longwall Foreman	22,000	1	22,000	1	22,000	1	22,000	1	22,000
Haulage Foreman	21,000	2	42,000	4	84,000	2	42,000	5	105,000
Supply Foreman	21,000	6	126,000	9	189,000	6	126,000	9	189,000
Maint. Superintendent	22,500	1	22,500	1	22,500	1	22,500	1	22,500
General Maint. Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Shop Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Chief Mine Engineer	28,000	1	28,000	1	28,000	1	28,000	1	28,000
Draftsman	16,000	2	32,000	3	48,000	2	32,000	3	48,000
Surveyors	18,000	4	72,000	6	108,000	4	72,000	6	108,000
Safety Director	20,000	1	20,000	1	20,000	1	20,000	1	20,000
Safety Inspector	19,000	1	19,000	1	19,000	1	19,000	1	19,000
Training Instructor	18,000	1	18,000	1	18,000	1	18,000	1	18,000
Noise/Dust Technician	16,000	1	16,000	1	16,000	1	16,000	1	16,000
Office Personnel Manager	20,000	1	20,000	1	20,000	1	20,000	1	20,000
Time/Bookkeeper	16,000	1	16,000	1	16,000	1	16,000	1	16,000
Warehouse Clerk	14,000	3	42,000	3	42,000	3	42,000	3	42,000
Industrial Engineer	23,800	1	23,000	1	23,000	1	23,000	1	23,800
Electrical Engineer	24,000	1	24,000	1	24,000	1	24,000	1	24,000
Prep. Plant General Foreman	25,000	1	25,000	1	25,000	1	25,000	1	25,000
Prep. Plant Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Control Systems Engineer	27,000	-	------	-	------	1	27,000	1	27,000
1980 DOLLARS	TOTAL	66	$1,346,500	83	$1,695,500	70	$1,436,500	89	$1,827,500
Average Salary: per person per year			$20,401		$20,428		$20,521		$20,534

1980 LONGWALL CASES

AVERAGE CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	6	71.18	427.08
Conveyor Attendants	6	71.18	427.08
Equipment Movers	17	71.76	1,219.92
Trackman	17	71.18	1,210.06
Greaser/Oiler	3	71.18	213.54
Mason	6	71.18	407.08
Laborer	34	71.18	2,420.12
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	6	79.72	478.32
SUBTOTAL	124		$9,000.84
Underground - Continuous Miner Crew			
Miner Operator	9	$79.72	$717.48
Miner Helper	9	76.48	688.32
Mobile Bridge Carrier Operator	36	73.48	2,645.28
Roof Bolt Operator	18	79.72	1,434.96
Utility Person	9	71.18	640.62
Mechanic	9	79.72	717.48
SUBTOTAL	90		$6,844.14
Underground - Longwall Crew			
Shearer Operator	6	79.72	478.32
Support Operator	6	79.72	478.32
Headgate Attendant	6	71.18	427.08
Tailgate Attendant	6	71.18	427.08
Utility Person	3	71.18	213.54
Mechanic	6	79.72	478.32
SUBTOTAL	30		$2,502.66
Hourly Total	293		$21,788.45

1980 LONGWALL CASES

IDEAL CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	30	71.78	2,135.40
SUBTOTAL	55		$3,867.89
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	11	71.18	782.98
Conveyor Attendants	6	71.18	427.08
Equipment Movers	34	71.76	2,439.84
Trackman	34	71.18	2,420.12
Greaser/Oiler	3	71.18	213.54
Mason	11	71.18	782.98
Laborer	69	71.18	4,911.42
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	11	79.72	876.92
SUBTOTAL	208		$15,052.52
Underground - Continuous Miner Crew			
Miner Operator	6	$79.72	$478.32
Miner Helper	6	76.48	458.88
Mobile Bridge Carrier Operator	24	73.48	1,763.52
Roof Bolt Operator	12	79.72	956.64
Utility Person	6	71.18	427.08
Mechanic	6	79.72	478.32
SUBTOTAL	60		$4,562.76
Underground - Longwall Crew			
Shearer Operator	6	79.72	478.32
Support Operator	6	79.72	478.32
Headgate Attendant	6	71.18	427.08
Tailgate Attendant	6	71.18	427.08
Utility Person	3	71.18	213.54
Mechanic	6	79.72	478.32
SUBTOTAL	30		$2,502.66
Hourly Total	353		$25,985.83

2000 LONGWALL CASES

AVERAGE CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	No.	Wages/Person/Day	Daily Total
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	7	71.18	498.26
Conveyor Attendants	6	71.18	427.08
Equipment Movers	22	71.76	1,578.72
Trackman	22	71.18	1,565.96
Greaser/Oiler	3	71.18	213.54
Mason	7	71.18	498.26
Laborer	43	71.18	3,060.74
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	7	79.72	558.04
SUBTOTAL	146		$10,598.24
Underground - Continuous Miner Crew			
Miner Operator	9	$79.72	$717.48
Miner Helper	9	76.48	688.32
Mobile Bridge Carrier Operator	9	73.48	661.32
Roof Bolt Operator	36	79.72	2,869.92
Utility Person	9	71.18	640.62
Mechanic	9	79.72	717.48
Mobile TRS Operator	9	79.72	717.48
SUBTOTAL	90		$7,012.62
Underground - Longwall Crew			
Cutting Technician	3	79.72	239.16
Support Technician	6	79.72	478.32
Headgate Attendant	6	71.18	427.08
Tailgate Attendant	6	71.18	427.08
Utility Person	3	71.18	213.54
Mechanic	6	79.72	478.32
Control System Technician	3	79.72	239.16
SUBTOTAL	33		$2,502.66
Hourly Total	318		$23,554.33

2000 LONGWALL CASES

IDEAL CONDITIONS		HOURLY LABOR	1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	30	71.18	2,135.40
SUBTOTAL	55		$3,867.89
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	14	71.18	996.52
Conveyor Attendants	6	71.18	427.08
Equipment Movers	41	71.76	2,942.16
Trackman	41	71.18	2,918.38
Greaser/Oiler	3	71.18	213.54
Mason	14	71.18	996.52
Laborer	82	71.18	5,836.76
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	14	79.72	1116.08
SUBTOTAL	244		$17,644.68
Underground - Continuous Miner Crew			
Miner Operator	6	$79.72	$478.32
Miner Helper	6	76.48	458.88
Mobile Bridge Carrier Operator	6	73.48	440.88
Roof Bolt Operator	24	79.72	1913.28
Utility Person	6	71.18	427.08
Mechanic	6	79.72	478.32
Mobile TRS Operator	6	79.72	478.32
SUBTOTAL	60		$4,675.08
Underground - Longwall Crew			
Cutting Technician	3	79.72	239.16
Support Technician	6	79.72	478.32
Headgate Attendant	6	71.18	427.08
Tailgate Attendant	6	71.18	427.08
Utility Person	3	71.18	213.54
Mechanic	6	79.72	478.32
Control System Technician	3	79.72	239.16
SUBTOTAL	33		$2,502.66
Hourly Total	392		$28,690.31

INPUTS FOR THE

1980 and 2000 SHORTWALL CASES

MODIFIED SHIFT PRODUCTION

	1980 SHORTWALL CASES		2000 SHORTWALL CASES	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
TPMS:				
Panel Development:	460	1440	550	1510
Shortwall Production:	570	1350	740	1580
Tons per Panel:				
Development:	69,039 tons		69,039 tons	
Production:	123,012 tons		123,012 tons	
Shifts to Mine One Panel:				
Development:	150.08	47.94	125.52	45.72
Production:	215.81	91.12	166.23	77.86
Shifts to Move to Next Panel:				
Development:	4	4	4	4
Production:	20	20	20	20
Total Shifts per Panel:				
Development:	154.08	51.94	129.52	49.72
Production:	235.81	111.12	186.23	97.86
Modified TPMS:				
Development:	448.07 (450)	1329.21 (1330)	533.04 (530)	1388.55 (1390)
Production:	521.66 (520)	1107.02 (1110)	660.53 (660)	1257.08 (1260)

DESIGN CAPACITY

Design Capacity =

$$[(N (d) \times TPMS (d)) + (N (P) \times TPMS (P))]\ S \times D \times (1 - R)$$

N (d) = Number of development units per shift
TPMS (d) = Productivity of development units
N (P) = Number of productions per shift
TPMS (P) = Productivity of production units
S = Shifts per day
D = Days per week
R = Reject precentage

	1980 SHORTWALL		2000 SHORTWALL	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
N (d)	3	2	3	2
TPMS (d)	450	1330	530	1390
N (P)	1	1	1	1
TPMS (P)	520	1110	660	1260
S	3	3	3	3
D	220	220	220	220
R	25%	25%	25%	25%
Design Capacity in clean tons	925,650	1,866,150	1,113,750	1,999,800

1980 SHORTWALL CASES

Equipment Cost	1980 Dollars

Continuous Miner Development Units

Average Conditions: Same as longwall case
$3,171,000 for three units

Ideal Conditions: Same as longwall case
$2,213,900

Shortwall Production Unit - Average and Ideal Conditions

	No.	Cost/Unit	Total Cost
Continuous Miner	1	$ 412,000	$ 412,000
MBC Unit	1	242,000	242,000
Roof Bolter	1	76,700	76,700
Support, Pumps, etc.(1)	lot	1,245,240	1,245,240
Scoop with Batteries & Charger	1	77,200	77,200
Bantam Duster	1	18,000	18,000
Trickle Duster	2	15,000	30,000
Section Power Center & Cables	1	53,000	53,000
Parts Car	1	10,500	10,500
Oil Storage Car	1	12,000	12,000
Section Tools	1	7,700	7,700
Section Welder	1	1,700	1,700
Section Haulage Belt (42 in. x 3000 ft)	1	$42.5/ft	255,000
Fire Suppression	1	4,200	4,200
		TOTAL	$2,445,240

1. Cost is updated from "Analysis of United States Shortwall Mining Practice", Mining Congress Journal, Katen, Kenneth P., January 1979. Update factor from IOM FF-345-79-239 (P. G. Gordon), JPL, 12/17/79.

2000 SHORTWALL CASES

Average Conditions	Equipment Cost	1980 Dollars

Development Units:

- Same as 2000 Longwall case
- 3 units
- $5,218,900

Shortwall Unit:

- Same as 1980 case except for shortwall supports
- 1978 shortwall supports, 5-ft stroke, $6,000/ft
- 2000 shortwall supports, 7-ft stroke, $8000/ft
- 180-ft face
- 2000 supports = $1,440,000
- 1978 supports = $1,245,240
- Total unit cost in 1980 = $2,445,240
- Unit cost in 2000 = $2,445,240 - $1,245,240 + $1,440,000
  = $2,640,000

Ideal Conditions	Equipment Cost	1980 Dollars

Development Units:

- Same as 2000 longwall case
- 2 units
- $3,671,900

Shortwall Unit:

- Same as 2000 shortwall, average conditions case
- $2,640,000

1980 SHORTWALL CASES

Average Conditions	Equipment Cost	1980 Dollars

Power consumption for shortwall unit assumed to be the same as for a development unit in U.S. Bureau of Mines IC 8757, 1977, Green, L. E. and Palowitch, E. R., Comparative Shortwall and Room-and-Pillar Mining Costs.

Therefore, the 1980 Longwall Development Power Consumption was modified for shortwall.

All consumption remains the same except that associated directly with development equipment - continuous miner, MBC unit, roof bolter, and trickle duster. The consumption of these components was increased by 33% to account for the shortwall unit.

Total power consumption for 1980 shortwall, average conditions is therefore 64,513 kW-h per day.

$0.46 per clean ton

Ideal Conditions	Equipment Cost	1980 Dollars

The same approach was taken as in the average conditions case. Therefore, total power consumption is 62,977 kW-h per day.

$$\text{CLEAN: } 62{,}977 \frac{\text{kW-h}}{\text{day}} \times 220 \frac{\text{day}}{\text{year}} \times \frac{\$0.03}{\text{kW-h}} \times \frac{1 \text{ year}}{1{,}866{,}150 \text{ clean tons}} = \frac{\$0.22}{\text{ton}}$$

2000 SHORTWALL CASES

Average Conditions	Power Cost	1980 Dollars

Power requirements adapted from longwall case. Assumed that a shortwall unit consumed as much power as a development unit. Therefore, components directly related to development units were inflated by factor of 1.33 to account for shortwall consumption.

	kW-h
Continuous Miner	16,064
MBC Unit	3,208
Roof Bolter	1,788
Temporary Roof Supports	716
Ventilation System	2,412
Jeeps	982
Supply Motor	2,868
Dusters	2,139
42-in. Conveyors	5,595
48-in. Conveyors	4,832
Ventilation Fan	17,904
Outside Electricals & Miscellaneous	9,238
	66,946 kW-h

Per clean ton: $\frac{\$0.03}{\text{kW-h}} \times 66{,}946\ \frac{\text{kW-h}}{\text{day}} \times 220\ \text{days} \times \frac{1\ \text{year}}{1{,}113{,}750\ \text{tons}} = \$0.40/\text{ton}$

Average Conditions	Power Cost	1980 Dollars

Same assumptions as average conditions case.

	kW-h
Continuous Miner	16,110
MBC Unit	3,222
Roof Bolter	1,341
Temporary Roof Supports	596
Ventilation System	2,016
Jeeps	938
Supply Motor	2,148
Dusters	1,614
Conveyors	9,627
Ventilation Fans	17,904
Outside Electricals & Miscellaneous	9,238
	64,546 kW-h

Per clean ton: $\frac{\$0.03}{\text{kW-h}} \times 64{,}754\ \frac{\text{kW-h}}{\text{day}} \times 220\ \text{days} \times \frac{1\ \text{year}}{1{,}999{,}800\ \text{tons}} = \$0.21/\text{ton}$

SALARIED PERSONNEL REQUIREMENTS AND COSTS

		1980 SHORTWALL CASES				2000 SHORTWALL CASES			
		Average Conditions		Ideal Conditions		Average Conditions		Ideal Conditions	
Personnel	Salary	No.	Yearly Total	No.	Yearly Total	No.	Yearly Total	No.	Yearly Total
Superintendent	$33,000	1	$ 33,000	1	$ 33,000	1	$ 33,000	1	$ 33,000
General Mine Foreman	27,500	1	27,500	1	27,500	1	27,500	1	27,500
Assist. Mine Foreman	22,500	3	67,500	3	67,500	3	67,500	3	67,500
Section Foreman	20,000	12	240,000	9	180,000	12	240,000	9	180,000
Construction Foreman	21,000	10	210,000	19	399,000	12	252,000	21	441,000
Shortwall Foreman	22,000	1	22,000	1	22,000	1	22,000	1	22,000
Haulage Foreman	21,000	2	42,000	3	63,000	2	42,000	3	63,000
Supply Foreman	21,000	3	63,000	6	126,000	6	126,000	6	126,000
Maint. Superintendent	22,500	1	22,500	1	22,500	1	22,500	1	22,500
General Maint. Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Shop Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Chief Mine Engineer	28,000	1	28,000	1	28,000	1	28,000	1	28,000
Draftsman	16,000	1	16,000	2	32,000	2	32,000	2	32,000
Surveyors	18,000	2	36,000	4	72,000	4	72,000	4	72,000
Safety Director	20,000	1	20,000	1	20,000	1	20,000	1	20,000
Safety Inspector	19,000	1	19,000	1	19,000	1	19,000	1	19,000
Training Instructor	18,000	1	18,000	1	18,000	1	18,000	1	18,000
Noise/Dust Technician	16,000	1	16,000	1	16,000	1	16,000	1	16,000
Office Personnel Manager	20,000	1	20,000	1	20,000	1	20,000	1	20,000
Time/Bookkeeper	16,000	1	16,000	1	16,000	1	16,000	1	16,000
Warehouse Clerk	14,000	3	42,000	3	42,000	3	42,000	3	42,000
Industrial Engineer	23,000	1	23,000	1	23,000	1	23,000	1	23,800
Electrical Engineer	24,000	1	24,000	1	24,000	1	24,000	1	24,000
Prep. Plant General Foreman	25,000	1	25,000	1	25,000	1	25,000	1	25,000
Prep. Plant Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Control Systems Engineer	27,000	-	------	-	------	1	27,000	1	27,000
1980 DOLLARS	TOTAL	66	$1,210,500	83	$1,475,500	68	$1,394,500	75	$1,544,500
Average Salary: per person per year			$20,517		$20,493		$20,507		$20,593

1980 SHORTWALL CASES

AVERAGE CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	18	71.18	1,281.24
SUBTOTAL	43		$3,013.73
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	5	71.18	355.90
Conveyor Attendants	6	71.18	427.08
Equipment Movers	14	71.76	1,004.64
Trackman	14	71.18	996.52
Greaser/Oiler	3	71.18	213.54
Mason	5	71.18	355.90
Laborer	29	71.18	2,064.22
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	5	79.72	398.60
SUBTOTAL	110		$8,014.04
Underground - Continuous Miner Crew			
Miner Operator	9	$79.72	$717.48
Miner Helper	9	76.48	688.32
Mobile Bridge Carrier Operator	36	73.48	2,645.28
Roof Bolt Operator	18	79.72	1,434.96
Utility Person	9	71.18	640.62
Mechanic	9	79.72	717.48
SUBTOTAL	90		$6,844.14
Underground - Shortwall Crew			
Miner Operator	3	79.72	239.16
Miner Helper	3	79.72	229.44
MBC Operator	6	73.48	440.88
Chock Operator	6	79.72	478.32
Mechanic	3	79.72	239.16
Utility Person	3	71.18	213.54
SUBTOTAL	24		$1,840.50
Hourly Total	267		$23,176.83

1980 SHORTWALL CASES

IDEAL CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	10	71.18	711.80
Conveyor Attendants	6	71.18	427.08
Equipment Movers	29	71.76	2,081.04
Trackman	29	71.18	2,064.22
Greaser/Oiler	3	71.18	213.54
Mason	10	71.18	711.80
Laborer	58	71.18	4,128.44
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	10	79.72	797.20
SUBTOTAL	184		$13,332.76
Underground - Continuous Miner Crew			
Miner Operator	6	$79.72	$478.32
Miner Helper	6	76.48	458.88
Mobile Bridge Carrier Operator	24	73.48	1,763.52
Roof Bolt Operator	12	79.72	956.64
Utility Person	6	71.18	427.08
Mechanic	6	79.72	478.32
SUBTOTAL	60		$4,562.76
Underground - Shortwall Crew			
Miner Operator	3	79.72	239.16
Miner Helper	3	79.72	229.44
MBC Operator	6	73.48	440.88
Chock Operator	6	79.72	478.32
Mechanic	3	79.72	239.16
Utility Person	3	71.18	213.54
SUBTOTAL	24		$1,840.50
Hourly Total	317		$23,176.83

2000 SHORTWALL CASES

AVERAGE CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	No.	Wages/Person/Day	Daily Total
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	6	71.18	427.08
Conveyor Attendants	6	71.18	427.08
Equipment Movers	17	71.76	1,219.92
Trackman	17	71.18	1,210.06
Greaser/Oiler	3	71.18	213.54
Mason	6	71.18	427.08
Laborer	35	71.18	2,491.30
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	6	79.72	478.32
SUBTOTAL	125		$9,092.02
Underground - Continuous Miner Crew			
Miner Operator	9	$79.72	$717.48
Miner Helper	9	76.48	688.32
Mobile Bridge Carrier Operator	9	73.48	661.32
Roof Bolt Operator	36	79.72	2,869.92
Utility Person	9	71.18	640.62
Mechanic	9	79.72	717.48
Mobile TRS Operator	9	79.72	717.48
SUBTOTAL	90		$7,012.62
Underground - Shortwall Crew			
Miner Operator	3	79.72	239.16
Miner Helper	3	79.72	229.44
MBC Operator	3	73.48	220.44
Chock Operator	6	79.72	478.32
Mechanic	3	79.72	239.16
Utility Person	3	71.18	213.54
SUBTOTAL	21		$1,620.06
Hourly Total	285		$21,165.51

2000 SHORTWALL CASES

IDEAL CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	10	71.18	711.80
Conveyor Attendants	6	71.18	427.08
Equipment Movers	31	71.76	2,224.56
Trackman	31	71.18	2,206.58
Greaser/Oiler	3	71.18	213.54
Mason	10	71.18	711.80
Laborer	63	71.18	4,484.34
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	10	79.72	797.20
SUBTOTAL	193		$13,974.54
Underground - Continuous Miner Crew			
Miner Operator	6	$79.72	$478.32
Miner Helper	6	76.48	458.88
Mobile Bridge Carrier Operator	6	73.48	440.88
Roof Bolt Operator	24	79.72	478.32
Utility Person	6	71.18	427.08
Mechanic	6	79.72	478.32
Mobile TRS Operator	6	79.72	478.32
SUBTOTAL	60		$4,675.08
Underground - Shortwall Crew			
Miner Operator	3	79.72	239.16
Miner Helper	3	79.72	229.44
MBC Operator	3	73.48	220.44
Chock Operator	6	79.72	478.32
Mechanic	3	79.72	239.16
Utility Person	3	71.18	213.54
SUBTOTAL	21		$1,620.06
Hourly Total	323		$23,710.49

INPUTS FOR THE

1980 and 2000 ROOM-AND-PILLAR CASES

MODIFIED SHIFT PRODUCTION

	1980 ROOM-AND-PILLAR		2000 ROOM-AND-PILLAR	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
TPMS:				
Development:	310	750	590	1670
Production:	260	630	530	1500
Tons per Panel:				
Development:	117,096 tons		120,953 tons	
Production:	84,864 tons		91,570 tons	
Shifts to Mine One Panel:				
Development:	377.73	156.13	205.01	72.43
Production:	326.40	134.70	172.77	61.05
Shifts to Move to Next Panel:				
Development:	4	4	4	4
Production:	0	0	0	0
Total Shifts per Panel:				
Development:	381.73	160.13	205.01	76.43
Production:	326.40	134.70	172.77	61.05
Modified TPMS:				
Development:	306.75 (310)	731.26 (730)	578.69 (580)	1582.53 (1580)
Production:	260	630	530	1500

DESIGN CAPACITY

Design Capacity =
$$[(N\ (d) \times TPMS\ (d)) + (N\ (P) \times TPMS\ (P))]\ S \times D \times (1 - R)$$

N (d) = Number of development units per shift
TPMS (d) = Productivity of development units
N (P) = Number of productions per shift
TPMS (P) = Productivity of production units
S = Shifts per day
D = Days per week
R = Reject precentage

	1980 ROOM-AND-PILLAR		2000 ROOM-AND-PILLAR	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
N (d)	2	2	2	2
TPMS (d)	310	730	580	1580
N (P)	2	2	2	2
TPMS (P)	260	630	530	1500
S	3	3	3	3
D	220	220	220	220
R	25%	25%	25%	25%
Design Capacity in clean tons	575,200	1,346,400	1,108,800	3,049,200

1980 ROOM-AND-PILLAR CASES

AVERAGE & IDEAL CONDITIONS	EQUIPMENT COST		1980 DOLLARS
Component	No.	Cost/Unit	Total Cost
Continuous Miner	4	$412,000	$1,648,000
Shuttle Cars	8	105,000	840,000
Roof Bolter	4	76,700	306,800
Auxiliary Fan	8	18,000	144,000
Scoop with Batteries & Charger	4	77,200	308,800
Bantam Duster	4	18,000	72,000
Trickle Duster	8	15,000	120,000
Section Power Center & Cables	4	53,000	212,000
Parts Car	4	10,500	42,000
Oil Storage Car	4	12,000	48,000
Section Tools	4	7,700	30,800
Ventilation Tubes	4 lots	2,000	8,000
Section Welder	4	1,700	6,800
Section Haulage Belt (42 in. x 3000 ft)	3	$42.5/ft	382,500
Fire Suppression	4	4,200	16,800
Ratio Feeder	4	79,200	316,800
		TOTAL	$4,503,300

2000 ROOM-AND-PILLAR CASES

AVERAGE & IDEAL CONDITIONS	EQUIPMENT COST		1980 DOLLARS
Component	No.	Cost/Unit	Total Cost
Continuous Miner	4	$412,000	$1,648,000
MBC System	4	400,000	1,600,000
Roof Bolter	8	76,700	613,600
Powered Temp. Support System	22	100,000	2,200,000
Scoop with Batteries & Charger	4	77,200	308,800
Bantam Duster	4	18,000	72,000
Trickle Duster	8	15,000	120,000
Section Power Center & Cables	4	53,000	212,000
Parts Car	4	10,500	42,000
Oil Storage Car	4	12,000	48,000
Section Tools	4	7,700	30,800
Ventilation System	22	20,000	440,000
Section Welder	4	1,700	6,800
Section Haulage Belt (42 in. x 3000 ft)	3	$42.5/ft	382,500
Fire Supression	4	4,200	16,800
		TOTAL	$7,741,300

1980 ROOM-AND-PILLAR CASES

AVERAGE CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Continuous Miner	4	600	2400	9	1776	15,984
Shuttle Cars	8	115	920	9	681	6,129
Roof Bolter	4	50	200	12	148	1,776
Auxiliary Fans	8	15	120	18	89	1,602
Mantrip Jeep	6	15	90	4	67	268
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	4	80	320	12	239	2,868
Bantam Duster	4	30	120	12	89	1,068
Trickle Duster	8	10	80	18	59	1,062
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fan	1	1000	1000	24	746	17,902
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
Ratio-Feeder	4	100	400	9	296	2,664
					TOTAL	70,902

Power at \$0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{70{,}902 \text{ kW-h}}{\text{day}} \times \frac{220 \text{ days}}{\text{year}} = \frac{\$467{,}953.20}{\text{year}}$$

Power Cost per Clean Ton = \$0.81

1980 ROOM-AND-PILLAR CASES

IDEAL CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Continuous Miner	4	600	2400	12	1776	21,312
Shuttle Cars	8	115	920	12	681	8,172
Roof Bolter	4	50	200	14	148	2,072
Auxiliary Fans	8	15	120	18	89	1,602
Ratio-Feeder	4	100	400	12	296	3,552
Mantrip Jeep	6	15	90	4	67	268
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	4	80	320	12	239	2,868
Bantam Duster	4	30	120	12	89	1,068
Trickle Duster	8	10	80	18	59	1,062
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fan	1	1000	1000	24	746	17,902
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
					TOTAL	79,557

Power at \$0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{79{,}557\ \text{kW-h}}{\text{day}} \times \frac{220\ \text{days}}{\text{year}} = \frac{\$525{,}076.20}{\text{year}}$$

Power Cost per Clean Ton = \$0.39

2000 ROOM-AND-PILLAR CASES

AVERAGE CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Continuous Miner	4	600	2400	9	1776	15,984
MBC Unit	4	120	480	9	358	3,222
Roof Bolter	8	50	400	6	298	1,788
Powered Temp. Supports	22	40	880	2	656	1,312
Ventilation System	22	15	330	18	246	4,428
Mantrip Jeep	6	15	90	4	67	268
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	4	80	320	12	239	2,868
Bantam Duster	4	30	120	12	89	1,068
Trickle Duster	8	10	80	18	59	1,062
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fan	1	1000	1000	24	746	17,902
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
					TOTAL	69,481

Power at \$0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{69{,}481\ \text{kW-h}}{\text{day}} \times \frac{220\ \text{days}}{\text{year}} = \frac{\$458{,}574.60}{\text{year}}$$

Power Cost per Clean Ton = \$0.41

2000 ROOM-AND-PILLAR CASES

IDEAL CONDITIONS			POWER COST			1980 DOLLARS
Component	No.	hp/Unit	Total hp	h/day	kW	Total kW-h Req.
Continuous Miner	4	600	2400	12	1776	21,312
MBC Unit	4	120	480	12	358	4,296
Roof Bolter	8	50	400	6	298	1,788
Powered Temp. Supports	22	40	880	2	656	1,312
Ventilation System	22	15	330	18	246	4,428
Mantrip Jeep	6	15	90	4	67	268
Mechanic Jeep	3	15	45	15	34	510
Personnel Jeep	3	7.5	22.5	12	17	204
Supply Motor	4	80	320	12	239	2,868
Bantam Duster	4	30	120	12	89	1,068
Trickle Duster	8	10	80	18	59	1,062
42-in. Conveyor	4	125	500	15	373	5,595
48-in. Conveyor	2	150	300	18	224	4,032
Ventilation Fan	1	1000	1000	24	746	17,902
Outside Electricals			600	14	447	6,258
Miscellaneous			400	10	298	2,980
					TOTAL	75,883

Power at $0.03 per kW-h

$$\text{Total Power Cost} = \frac{\$0.03}{\text{kW-h}} \times \frac{75{,}883 \text{ kW-h}}{\text{day}} \times \frac{220 \text{ days}}{\text{year}} = \frac{\$500{,}827.80}{\text{year}}$$

Power Cost per Clean Ton = $0.16

SALARIED PERSONNEL REQUIREMENTS AND COSTS

		1980 ROOM-AND-PILLAR CASES				2000 ROOM-AND-PILLAR CASES			
		Average Conditions		Ideal Conditions		Average Conditions		Ideal Conditions	
Personnel	Salary	No.	Yearly Total	No.	Yearly Total	No.	Yearly Total	No.	Yearly Total
Superintendent	$33,000	1	$ 33,000	1	$ 33,000	1	$ 33,000	1	$ 33,000
General Mine Foreman	27,500	1	27,500	1	27,500	1	27,500	1	27,500
Assist. Mine Foreman	22,500	3	67,500	3	67,500	3	67,500	3	67,500
Section Foreman	20,000	12	240,000	12	240,000	12	240,000	12	240,000
Construction Foreman	21,000	6	126,000	15	315,000	12	252,000	32	672,000
Haulage Foreman	21,000	1	21,000	2	42,000	2	42,000	5	105,000
Supply Foreman	21,000	3	63,000	6	126,000	6	126,000	9	189,000
Maint. Superintendent	22,500	1	22,500	1	22,500	1	22,500	1	22,500
General Maint. Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Shop Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Chief Mine Engineer	28,000	1	28,000	1	28,000	1	28,000	1	28,000
Draftsman	16,000	1	16,000	2	32,000	2	32,000	3	48,000
Surveyors	18,000	2	36,000	4	72,000	4	72,000	6	108,000
Safety Director	20,000	1	20,000	1	20,000	1	20,000	1	20,000
Safety Inspector	19,000	1	19,000	1	19,000	1	19,000	1	19,000
Training Instructor	18,000	1	18,000	1	18,000	1	18,000	1	18,000
Noise/Dust Technician	16,000	1	16,000	1	16,000	1	16,000	1	16,000
Office Personnel Manager	20,000	1	20,000	1	20,000	1	20,000	1	20,000
Time/Bookkeeper	16,000	1	16,000	1	16,000	1	16,000	1	16,000
Warehouse Clerk	14,000	3	42,000	3	42,000	3	42,000	3	42,000
Industrial Engineer	23,000	1	23,000	1	23,000	1	23,000	1	23,800
Electrical Engineer	24,000	1	24,000	1	24,000	1	24,000	1	24,000
Prep. Plant General Foreman	25,000	1	25,000	1	25,000	1	25,000	1	25,000
Prep. Plant Foreman	20,000	3	60,000	3	60,000	3	60,000	3	60,000
Control Systems Engineer	27,000	-	------	-	------	1	27,000	1	27,000
1980 DOLLARS	TOTAL	53	$1,083,500	69	$1,408,500	67	$1,372,500	96	$1,970,500
Average Salary: per person per year			$20,412		$20,413		$20,485		$20,526

1980 ROOM-AND-PILLAR CASES

AVERAGE CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	No.	Wages/Person/Day	Daily Total
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	18	71.18	1,281.24
SUBTOTAL	43		$3,013.73
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	3	71.18	213.54
Conveyor Attendants	6	71.18	427.08
Equipment Movers	9	71.76	645.84
Trackman	9	71.18	690.62
Greaser/Oiler	3	71.18	213.54
Mason	3	71.18	213.54
Laborer	18	71.18	1,218.24
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	3	79.92	239.16
SUBTOTAL	83		$6,009.20
Underground - Continuous Miner Crew			
Miner Operator	12	$79.72	$956.64
Miner Helper	12	76.48	917.76
Shuttle Car Operator	24	73.48	1,763.52
Roof Bolt Operator	24	79.72	1,913.28
Utility Person	12	71.18	854.16
Mechanic	12	79.72	956.64
SUBTOTAL	96		$7,362.00
Hourly Total	222		$16,384.93

1980 ROOM-AND-PILLAR CASES

IDEAL CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	3	71.18	498.26
Conveyor Attendants	6	71.18	427.08
Equipment Movers	21	71.76	1,506.96
Trackman	21	71.18	1,494.78
Greaser/Oiler	3	71.18	213.54
Mason	7	71.18	498.26
Laborer	42	71.18	2,989.56
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	7	79.72	558.04
SUBTOTAL	143		$10,384.12
Underground - Continuous Miner Crew			
Miner Operator	12	$79.72	$956.64
Miner Helper	12	76.48	917.76
Shuttle Car Operator	24	73.48	1,763.52
Roof Bolt Operator	24	79.72	1,913.28
Utility Person	12	71.18	854.16
Mechanic	12	79.72	956.64
SUBTOTAL	96		$7,362.00
Hourly Total	288		$21,186.93

2000 ROOM-AND-PILLAR CASES

AVERAGE CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Conveyor Attendent	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	24	71.18	1,708.32
SUBTOTAL	49		$3,440.81
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	3	71.18	427.08
Conveyor Attendants	6	71.18	427.08
Equipment Movers	17	71.76	1,219.92
Trackman	17	71.18	1,210.06
Greaser/Oiler	3	71.18	213.54
Mason	6	71.18	427.08
Laborer	35	71.18	2,491.30
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	6	79.72	478.32
SUBTOTAL	125		$9,092.02
Underground - Continuous Miner Crew			
Miner Operator	12	$79.72	$956.64
Miner Helper	12	76.48	917.76
Shuttle Car Operator	12	73.48	881.76
Roof Bolt Operator	24	79.72	1,913.28
Utility Person	12	79.72	956.64
Mechanic	12	79.72	956.64
SUBTOTAL	96		$7,436.88
Hourly Total	270		$19,969.71

2000 ROOM-AND-PILLAR CASES

IDEAL CONDITIONS	HOURLY LABOR		1980 DOLLARS
Surface	**No.**	**Wages/Person/Day**	**Daily Total**
Hoistman	3	$68.05	$204.15
Electrician	1	71.87	71.87
Shop Mechanic	6	71.87	431.22
Shop Electrician	3	71.87	215.61
Supply Handler	6	67.47	404.82
Lampman	3	67.47	202.41
Warehouse Laborer	3	67.47	202.41
Prep. Plant Crew	30	71.18	2,135.40
SUBTOTAL	55		$3,867.89
Underground - General			
Motorman	12	$71.76	$861.12
Fireboss	3	79.72	239.16
Electrician	3	79.72	239.16
Mechanic	6	79.72	478.32
Scoop Operator	3	73.48	220.44
Pumper	16	71.18	1,138.88
Conveyor Attendants	6	71.18	427.08
Equipment Movers	48	71.76	3,444.48
Trackman	48	71.18	3,416.64
Greaser/Oiler	3	71.18	213.54
Mason	16	71.18	1,138.88
Laborer	95	71.18	6,762.10
Stopper Operator	2	79.72	159.44
Auxiliary Equip. Operators	16	79.72	1,257.52
SUBTOTAL	277		$19,801.22
Underground - Continuous Miner Crew			
Miner Operator	12	$79.72	$956.64
Miner Helper	12	76.48	917.76
Shuttle Car Operator	12	73.48	881.76
Roof Bolt Operator	24	79.72	1,913.28
Utility Person	12	79.72	956.64
Mechanic	12	79.72	956.64
Mobile TRS Operator			
SUBTOTAL	96		$7,436.88
Hourly Total	428		$31,105.99

APPENDIX D

COST UPDATE FACTORS

This information was provided by M. Dean Westerfield, and originally appeared in I.O.M. 311.2-969 of June 9, 1980.

The following cost update factors were used as input, or to modify input to the costing analysis of baseline mining technologies.

All Cases	1975-1980	1975-2000	1980-2000
Equipment Cost, General	.6757	5.2432	--
Production Section Equip.	--	--	2.7258
Power Cost/Clean Ton	--	--	1.0345
Preparation Plant	.2753	1.5899	--
Salaries	--	--	3.9299
General Inflation (GNP Deflator)	--	--	2.8849

Manpower Costs	1975-1980		1975-2000	
	Average Conditions	Ideal Conditions	Average Conditions	Ideal Conditions
Shortwall	.3414	.3356	5.6778	5.6329
Continuous	.3420	.3420	5.6482	5.6482
Longwall	.3576	.3542	5.6733	5.6311

The cost update factor is defined such that

Base year cost x (1 + cost update) = Future Cost

All of the update factors with the exception of the manpower cost update and general inflation factors were derived from data generated by M. Gyamfi (IOM 311.3-104, 10/16/78). This data is reproduced in Table D-1.

The manpower cost update was found as follows

1975 to 1980:

$$\frac{\text{Cost/Man-shift}_{1980}}{\$55/\text{Man-shift}_{1975}} - 1 = \text{Update factor}_{1975-80}$$

1975 to 2000:

$$\frac{\text{Cost/Man-shift}_{1980}}{\$55/\text{Man-shift}_{1975}} - 1 = \text{Update factor}_{1975-80}$$

The 4.92989 was derived from M. Gyamfi's manpower cost data. The $55.00 base year cost was necessitated by the internal programming of the NUS model. Manpower cost update factors were different in each case because of the different mixes of labor employed in each mining scenario.

The general price deflator was derived from Data Resources Inc. data (DRI Energy Review, Autumn 1979, pg. 106). DRI projected the inflation rates for the years indicated in the table below. Using linear interpolation for each year not included in the data below, the inflation rate for each year from 1980 to 2000 was projected. The product of these numbers was used as the general price deflator for 1980 to 2000.

The general price deflator was used to project the cost of royalties to the year 2000 and deflate year 2000 coal costs to 1980 levels.

ANNUAL RATE OF CHANGE IN GNP DEFLATOR

1980	1981	1982	1985	1990	1995	2000
8.7%	8.7	8.2	7.4	6.4	5.8	5.3

Source: DRI Energy Review, Autumn 1979

Table D-1. Projected Price Indexes

Year	Chemical Plant and Equipment Price Index	Mining Equipment Price Index	Electric Power Price Index	Coal Miners Daily Wages (Current Dollars) $/Day
1975	178	185	195	60.00
1976			205	
1977				67.68
1978	212	246	226	75.52
1979				83.84
1980	227	310	232	91.44
1981				101.68
1982	258	364		110.40
1983				119.60
1984	294	428		132.24
1985	314	464	268	143.52
1986	323	505		154.56
1987				170.40
1988	341	599		184.72
1989				192.32
1990	360	700	319	218.72
1991				
1992	378	800	345	252.76
1993				
1994	397	900		292.09
1995	407	950	388	314.00
1996	417	1000		337.55
1997				
1998	439	1090	436	390.08
1999				
2000	461	1155	472	450.79

Source: M. Gyamfi IOM 311.3-104, 10/16/78

APPENDIX E

COST ANALYSIS RESULTS

The purpose of Appendix E, Cost Analysis Results is to document the results of the cost analysis performed by the NUS coal costing model for the twelve study cases.

LONGWALL CASES

	Output	1980 Systems Average Conditions	1980 Systems Ideal Conditions	2000 Systems Average Conditions	2000 Systems Ideal Conditions
		(costs in 1980 dollars)		(costs in 2000 dollars)	
I.	Production sizing				
	Reject percentage	21.0	21.0	21.0	21.0
	Design capacity - raw tons	1,438,800	2,923,800	1,848,000	3,504,600
	Tons per machnie shift				
	Room-and-Pillar	450	1330	530	1390
	Longwall unit	830	1770	1210	2530
	Shortwall unit	-	-	-	-
	Machine sections per shift	4	3	4	3
	Design capacity - clean tons	1,136,652	2,309,802	1,459,920	2,768,634
II.	Manpower				
	Hourly Labor				
	Daily requirement (persons)	293	353	318	392
	Direct cost per year	4,793,245	5,716,537	25,546,390	31,117,004
	Total cost per year	9,375,791	12,895,924	48,527,450	64,067,640
	Cost per ton	8.25	5.62	33.24	23.14
	Salaried				
	Daily requirement (persons)	66	83	70	89
	Direct cost per year	1,820,511	2,269,221	7,081,790	9,009,702
	Total cost per year	2,548,716	3,176,910	9,914,506	12,613,583
	Cost per ton	2.24	1.38	6.79	4.56
III.	Equipment and Construction				
	Section equipment costs				
	Continuous R&P units	3,171,000	2,213,900	19,471,403	13,707,581
	Longwall units	5,410,800	5,410,800	25,801,930	25,801,930
	Shortwall units	-	-	-	-
	Belt haulage system cost	3,172,062	2,696,169	11,818,377	10,045,309
	Track haulage system cost	-	-	-	-

LONGWALL CASES (Continuation 1)

	Output	1980 Systems: Average Conditions (costs in 1980 dollars)	1980 Systems: Ideal Conditions (costs in 1980 dollars)	2000 Systems: Average Conditions (costs in 2000 dollars)	2000 Systems: Ideal Conditions (costs in 2000 dollars)
	Underground auxiliary equip. costs				
	Dewatering	323,406	159,190	1,204,938	593,104
	Fire and safety	149,136	145,784	555,645	543,158
	Communications	251,352	251,352	936,480	936,480
	Surface auxiliary equip. costs				
	Heavy equip.	1,808,059	1,808,059	6,736,413	6,736,413
	Personnel vehicles	36,865	36,865	137,350	137,350
	Ventilation	268,109	201,082	998,912	749,184
	Miscellaneous	268,109	268,109	998,912	998,912
	Site prep. & const. costs				
	Mine entries	127,528	127,528	258,990	258,990
	Other surface const.	8,330,294	15,509,149	20,986,519	37,471,302
	Exploration costs	106,000	213,900	276,500	524,400
	Abandonment costs	127,528	127,528	258,900	258,900
	Reserve deposit size (tons)	37,863,157	76,421,051	48,631,578	92,226,314
IV.	Supplies and Materials				
	Annual cost	8,767,843	9,860,553	-	-
	Cost per ton	7.71	4.30	24.36	15.56
V.	Power				
	Annual cost	443,294	435,890	950,408	902,575
	Cost per ton	0.39	0.19	0.65	0.33
VI.	Preproduction Development				
	Direct and indirect costs	20,075,497	10,845,340	82,281,809	49,881,255
	Development time (years)	1.21	.44	1.03	.45
	Coal production for period (tons)	428,478	395,658	429,057	374,408

LONGWALL CASES (Continuation 2)

	Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
VII.	Initial Capital Investment				
	Total plant & equip. cost	25,987,558	32,212,964	100,082,000	109,291,870
	Initial capital investment	34,243,692	36,416,623	141,561,180	136,178,960
	Interest rate (%)	15.0	15.0	15.0	15.0
	Const. & devel. time (years)	3.0	3.0	3.0	3.0
VIII.	Deferred Capital Depreciation				
	Total salvage value	967,401	1,961,055	4,627,944	8,755,605
	Average yearly depreciation	2,542,104	2,694,859	10,954,996	10,789,977
	Replacement capital	15,416,002	15,083,718	71,822,370	68,919,282
	Additional capital	2,149,780	4,357,900	10,284,320	19,456,900
IX.	Annual Operating Cost, Working Capital				
	Annual operating cost				
	Direct	21,135,643	26,369,277	94,958,064	120,661,603
	Indirect	2,875,566	3,824,027	13,064,247	17,857,364
	Fixed	3,061,119	3,339,119	12,956,636	12,975,814
	Cost per ton	23.82	14.62	82.87	54.72
	Working capital required	6,132,740	7,709,391	27,505,987	35,176,200
X.	Production Cost				
	Federal tax rate (%)	48.0	48.0	48.0	48.0
	State and local tax rate (%)	2.0	2.0	2.0	2.0
	Total capital investment	50,969,600	54,024,714	219,358,914	216,058,526
	Present worth of capital investment	43,843,113	47,954,950	185,336,532	188,720,654
	Debt/equity ratio	0.0	0.0	0.0	0.0

LONGWALL CASES (Continuation 3)

Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
Required net annual cash flow	7,004,440	7,661,353	29,609,637	30,150,289
Annual gross profit	5,949,782	6,621,992	24,872,854	25,813,750
Annual sales revenue	33,022,845	40,154,414	145,851,800	177,308,530
Production cost ($/ton)	29.05	17.50	99.90	64.04
Production cost components				
Labor	8.15	4.87	31.29	20.29
Supplies	7.71	4.30	24.36	15.56
Power	0.39	0.19	0.65	0.33
Union	2.34	2.13	8.74	7.41
Royalties	0.50	0.50	1.94	1.94
Capital related	9.96	5.51	32.92	18.52

SHORTWALL CASES

	Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
I.	Production Sizing				
	Reject percentage	25.0	25.0	25.0	25.0
	Design capacity - raw tons	1,234,200	2,488,200	1,485,000	2,666,400
	Tons per machine shift				
	Room-and-Pillar	450	1330	530	1390
	Longwall unit	-	-	-	-
	Shortwall unit	520	1110	660	1260
	Machine sections per shift	4	3	4	3
	Design capacity - clean ton	925,650	1,866,150	1,113,750	1,999,800
II.	Manpower				
	Hourly labor				
	Daily requirement (persons)	267	317	285	323
	Direct cost per year	4,336,762	5,090,681	22,954,078	25,716,848
	Total cost per year	8,278,124	11,144,860	42,592,445	51,504,340
	Cost per ton ($)	8.94	5.97	38.24	25.75
	Salaried				
	Daily requirement (persons)	59	72	68	75
	Direct cost per year	1,624,931	1,961,333	6,874,468	7,613,787
	Total cost per year	2,274,903	2,745,866	9,624,256	10,659,302
	Cost per ton ($)	2.46	1.47	8.64	5.33
III.	Equipment and Construction				
	Section equipment costs				
	Continuous R&P units	3,171,000	2,213,900	19,471,403	13,707,581
	Longwall units	-	-	-	-
	Shortwall units	2,445,240	2,445,240	9,836,112	9,836,112

SHORTWALL CASES (Continuation 1)

Output	1980 Systems Average Conditions	1980 Systems Ideal Conditions	2000 Systems Average Conditions	2000 Systems Ideal Conditions
	(costs in 1980 dollars)		(costs in 2000 dollars)	
Belt haulage system cost	3,172,100	2,696,201	11,818,377	10,045,309
Track haulage system cost	-	-	-	-
Underground auxiliary equipment costs				
Dewatering	323,410	159,191	1,204,938	593,104
Fire and safety	149,137	145,786	555,645	543,158
Communications	251,355	251,355	936,480	936,480
Surface auxiliary equipment costs				
Heavy equipment	1,808,080	1,808,080	6,736,413	6,736,413
Personnel vehicles	36,865	36,865	137,350	137,350
Ventilation	268,112	201,084	998,912	749,184
Miscellaneous	268,112	268,112	998,912	998,912
Site preparation and construction costs				
Mine entries	127,530	127,530	258,990	258,990
Other surface construction	7,000,020	12,829,671	16,647,359	27,747,256
Exploration costs	100,200	201,900	244,700	439,400
Abandonment costs	127,530	127,530	258,990	258,990
Reserve deposit size (tons)	35,773,913	72,121,739	43,043,478	77,286,956
IV. Supplies and Materials				
Annual cost	8,047,285	8,929,504	-	-
Cost per ton	8.69	4.78	29.16	17.92
V. Power				
Annual cost	425,799	410,553	906,593	853,915
Cost per ton	0.46	0.22	0.81	0.43

SHORTWALL CASES (Continuation 2)

	Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
VI.	Preproduction Development				
	Direct and indirect costs	18,825,511	10,066,337	77,909,973	43,479,928
	Development time (years)	1.21	0.42	1.03	0.45
	Coal production for period (tons)	405,164	406,175	407,333	355,451
VII.	Initial Capital Investment				
	Total plant and equipment cost	21,213,466	25,935,047	77,471,967	80,651,409
	Initial capital investment	27,845,260	28,691,086	110,431,246	99,579,418
	Interest rate (%)	15.0	15.0	15.0	15.0
	Construction and development time (years)	3.0	3.0	3.0	3.0
VIII.	Deferred Capital, Depreciation				
	Total salvage value	914,832	1,809,054	4,101,678	7,492,212
	Average yearly depreciation	2,067,774	2,142,574	8,537,022	8,012,524
	Replacement capital	12,392,100	11,949,329	55,296,037	51,513,918
	Additional capital	2,032,960	4,020,120	9,114,840	16,649,360
IX.	Annual Operating Cost, Working Capital				
	Annual operating cost				
	Direct	19,026,110	23,230,783	85,599,655	98,850,453
	Indirect	2,564,172	3,331,503	11,508,639	14,258,141
	Fixed	2,492,044	2,661,275	10,086,461	9,625,552
	Cost per ton	26.02	15.66	96.25	61.37
	Working capital required	5,503,638	6,770,246	24,664,433	28,680,405

SHORTWALL CASES (Continuation 3)

	Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
X.	Production Cost				
	Federal tax rate (%)	48.0	48.0	48.0	48.0
	State and local tax rate (%)	2.0	2.0	2.0	2.0
	Total capital investment	41,483,018	42,979,011	170,999,434	160,509,472
	Present worth of capital investment	36,176,011	38,592,859	147,692,836	141,607,158
	Debt/equity ratio	0.0	0.0	0.0	0.0
	Required net annual cash flow	5,779,533	6,165,652	23,595,625	22,623,368
	Annual gross profit	4,949,011	5,364,104	20,078,137	19,481,125
	Annual sales revenue	29,031,337	34,587,664	127,272,890	142,215,270
	Production Cost ($/ton)	31.36	18.53	114.27	71.11
	Production cost components				
	Labor	9.02	5.30	37.49	23.33
	Supplies	8.69	4.78	29.16	17.92
	Power	0.46	0.22	0.81	0.43
	Union	2.38	2.15	9.39	7.75
	Royalties	0.50	0.50	1.94	1.94
	Capital Related	10.31	5.59	35.48	19.74

CONTINUOUS ROOM-AND-PILLAR CASES

	Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
I.	Production Sizing				
	Reject percentage	25.0	25.0	25.0	25.0
	Design capacity-raw tons	765,600	1,795,200	1,478,400	4,065,600
	Tons per machine shift				
	Room-and-Pillar	290	680	560	1540
	Longwall unit	-	-	-	-
	Shortwall unit	-	-	-	-
	Machine sections per shift	4	4	4	4
	Design capacity-clean tons	574,200	1,346,400	1,108,800	3,049,200
II.	Manpower				
	Hourly labor				
	Daily requirement (persons)	222	288	270	428
	Direct cost per year	3,604,880	4,661,408	21,658,119	33,737,965
	Total cost per year	6,517,827	9,542,431	40,465,242	69,597,668
	Cost per ton	11.35	7.09	36.49	22.82
	Salaried				
	Daily requirement (persons)	53	69	67	96
	Direct cost per year	1,454,029	1,883,231	6,765,998	9,715,033
	Total cost per year	2,035,640	2,636,524	9,472,397	13,601,046
	Cost per ton	3.55	1.96	8.54	4.46
III.	Equipment and Construction				
	Section equipment costs				
	Continuous R&P units	4,503,300	4,503,300	28,842,585	28,842,585
	Longwall units	-	-	-	-
	Shortwall units	-	-	-	-

CONTINUOUS ROOM-AND-PILLAR CASES (Continuation 1)

	Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
	Belt haulage system cost	1,699,140	1,699,140	6,330,605	21,559,434
	Track haulage system cost	-	-	-	-
	Underground auxiliary equipment costs				
	Dewatering	323,406	169,244	1,204,938	630,563
	Fire and safety	149,136	149,136	555,645	555,645
	Communications	251,352	251,352	936,480	936,480
	Surface auxiliary equipment costs				
	Heavy equipment	1,808,059	1,808,059	6,736,413	6,736,413
	Personnel vehicles	36,865	36,865	137,350	137,350
	Ventilation	268,109	268,109	998,912	998,912
	Miscellaneous	268,109	268,109	998,912	998,912
	Site Preparation and Construction Costs				
	Mine entries	127,528	127,528	258,990	258,990
	Other surface construction	4,809,540	9,626,731	16,568,730	41,043,077
	Exploration costs	99,700	233,800	311,300	856,110
	Abandonment costs	127,528	127,528	258,990	258,990
	Reserve deposit size (tons)	35,609,302	83,497,674	54,755,555	150,577,778
IV.	Supplies and Materials				
	Annual cost	6,870,083	8,272,281	-	-
	Cost per ton	11.96	6.14	27.41	15.42
V.	Power				
	Annual cost	465,102	525,096	916,978	984,892
	Cost per ton	0.81	0.39	0.73	0.25

CONTINUOUS ROOM-AND-PILLAR CASES (Continuation 2)

	Output	1980 Systems Average Conditions	1980 Systems Ideal Conditions	2000 Systems Average Conditions	2000 Systems Ideal Conditions
		(costs in 1980 dollars)		(costs in 2000 dollars)	
VI.	Preproduction Development				
	Direct and indirect costs	9,998,864	5,505,416	26,307,560	14,758,791
	Development time (years)	1.36	0.56	0.73	0.25
	Coal production for period (tons)	249,861	249,318	255,490	254,037
VII.	Initial Capital Investment				
	Total plant and equipment cost	15,911,142	21,221,203	70,873,509	114,851,279
	Initial capital investment	16,386,580	21,332,668	71,749,669	114,683,594
	Interest rate (%)	15.0	15.0	15.0	15.0
	Construction and development time (years)	3.0	2.0	2.0	2.0
VIII.	Deferred Capital, Depreciation				
	Total salvage value	1,605,330	3,616,344	9,109,899	9,104,220
	Average yearly depreciation	1,446,202	1,928,025	6,890,939	9,798,414
	Replacement capital	10,573,389	12,807,849	54,934,799	70,157,319
	Additional capital	3,567,400	8,036,320	20,244,220	20,231,600
IX.	Annual Operating Cost, Working Capital				
	Annual operating cost				
	Direct	15,888,652	20,976,332	81,242,351	131,200,026
	Indirect	2,076,449	2,895,738	10,975,067	19,491,959
	Fixed	1,764,425	2,352,449	8,308,410	12,095,440
	Cost per ton	34.36	19.48	90.66	53.39
	Working capital required	4,570,831	6,074,123	23,408,722	38,247,252

CONTINUOUS ROOM-AND-PILLAR CASES (Continuation 3)

Output	1980 Systems Average Conditions (costs in 1980 dollars)	1980 Systems Ideal Conditions (costs in 1980 dollars)	2000 Systems Average Conditions (costs in 2000 dollars)	2000 Systems Ideal Conditions (costs in 2000 dollars)
X. Production Cost				
Federal tax rate (%)	48.0	48.0	48.0	48.0
State and local tax rate (%)	2.0	2.0	2.0	2.0
Total capital investment	29,051,567	38,688,021	138,077,778	196,227,282
Present worth of capital investment	23,820,908	31,729,384	110,425,991	170,560,754
Debt/equity ratio	0.0	0.0	0.0	0.0
Required net annual cash flow	3,805,663	5,069,133	17,641,819	27,249,037
Annual gross profit	3,145,948	4,188,145	14,334,505	23,267,496
Annual sales revenue	22,875,474	30,412,662	114,860,332	186,054,920
Production cost ($/ton)	39.84	22.59	103.59	61.02
Production cost components				
Labor	12.33	6.81	35.89	19.95
Supplies	11.96	6.14	27.41	15.42
Power	0.81	0.39	0.83	0.32
Union	2.56	2.24	9.15	7.33
Royalties	0.50	0.50	1.94	1.94
Capital related	11.67	6.51	28.38	16.05